

THE UNIVERSITY OF ZAMBIA

SCHOOL OF MINES

DEPARTMENT OF METALLURGY AND MINERAL PROCESSING

MM 590-FINAL YEAR PROJECT REPORT

**CHARACTERISATION OF DIFFERENT ORE TYPES AND
DETERMINATION OF OPTIMUM OPERATING CONDITIONS
FOR TREATING KANSANSHI ORES THROUGH THE LEACHING
CIRCUIT**

BY

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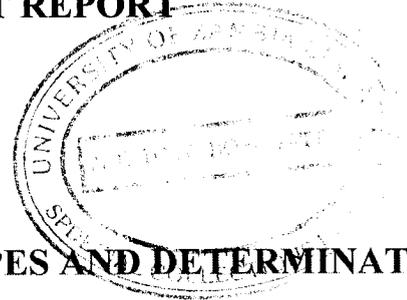
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**PROJECT REPORT SUBMITTED TO PARTIALLY FULFIL REQUIREMENTS
FOR THE AWARD OF A BACHELORS DEGREE OF MINERAL SCIENCES IN
METALLURGY AND MINERAL PROCESSING OF THE UNIVERSITY OF
ZAMBIA**

BY

CHILANDO CHANDA ANDREW

FEBRUARY, 2008.

DEDICATION

I take delight in dedicating this piece of work to my parents Mr. and Mrs. Chilando for their love, patience and support.

EXAMINERS

INTERNAL EXAMINER..... 

Dr. J.B. MWALULA

HEAD OF DEPARTMENT..... 

Mr. C. V LUNGU

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I am so glad that my final year project has been completed. But this would not have been possible without the invaluable assistance of the following people.

I wish to start by extending my gratitude to the staff of Kansanshi Mining Company for giving me the opportunity to be attached to their Company during the 2006 July vacation. This enabled me to understand the plant operations and hence made my project quite easy, meaningful and very interesting. Special thanks go to Mr. Pyokera Kawonga (Senior Metallurgist at Kansanshi Mine) for his guidance and patience during the time I was doing the experiments.

I m grateful to Dr J.B. Mwalula my project supervisor for the guidance he provided.

My gratitude also goes to Dr N.R. Nyirenda whom I often consulted for any clarification that I needed.

May God the Almighty bless you all.

LIST OF ABBREVIATIONS

TCu	total copper
AS.Cu	acid soluble copper
HFCu	hydrofluoric acid soluble copper
GPL or gpl	grams per litre
pH	potential of hydrogen
Hrs	hours
Wt%	weight percent
kg	kilograms
µm	micrometer
m ³ /hr	cubic metre per hour
SX	Solvent Extraction
U/F	underflow
HGG	HIGH GRADE GREEN
LGG	LOW GRADE GREEN
HGB	HIGH GRADE BLUE
LGB	LOW GRADE BLUE
HGR	HIGH GRADE RED
LGR	LOW GRADE RED

LIST OF SYMBOLS

O	oxygen
S	sulphur
Si	silicon
Fe	iron
C	carbon
Cu	copper
%	percent

ABSTRACT

Different ore types react differently when leached under the same leaching conditions. The extent to which minerals go into solution is a function of several parameters. In this project, emphasis was on determining the mineralogical characteristics of different ores, the optimum residence time, the quantity of acid needed to leach the different ore types and the dissolution of some impurities in the leach solutions.

The leaching parameters that were studied were:

- (a) % solids of the slurry (28%), (b) particle size distribution (80% passing 150 μ m), (c) pH about 1.60, and, (d) mass of ore leached (1.5kg) and were kept constant for different types of ore.

The mineralogy of the ores was investigated and the ore were found to be of predominately composed of malachite, pseudomalachite and Chrysocolla.

The optimum residence time for HGB, LGB and HGG was found to be 2hrs. The highest acid consumption was observed in LGB and was 93kg H₂SO₄ per tonne of ore.

The acid consumption can be reduced by increasing the concentration of the solids in the feed to the leaching circuit. Increasing the concentration of solids from 55% to 65% solids in the feed the consumption would reduce acid consumption by 4.2kg H₂SO₄/ tonne ore. From 55% to 75% and 85% solids the consumption would reduce by 7.3 and 9.4 kg / tonne of ore respectively.

CHAPTER ONE

1 INTRODUCTION

An ore can be regarded as an aggregate of minerals from which a metal or metals can be extracted at a profit. It comprises both valuable minerals and gangue minerals. Different ore types have different characteristics and mineralogy and therefore respond differently when leached under the same conditions.

The residence times for an ore determines the working capacity of a processing plant and can have an effect on the amount of impurities which go into solution.

Due to the variety of ore types which are mined at the Kansanshi open pits and subsequently treated at the processing plant, it became necessary to study on a laboratory scale the leaching characteristics of these ore types.

1.1 BACKGROUND

The Hydrometallurgical section (Back End) of the Kansanshi Processing Plant treats about 15000 tonnes of dry oxide ore per day. The Kansanshi processing plant treats a variety of oxide ores which are predominantly composed of malachite, Pseudomalachite and Chrysocolla as disseminated particles in fine grained, siliceous rock. Other minerals present in small proportions are chalcopyrite, chalcocite and bornite.

The project was formulated because of the variations in the ores which are processed at the plant. It became necessary to investigate the leaching characteristics of different ore types. Therefore it was suggested to study the ores with respect to the mineralogy of the ores, residence time, and acid consumption of the different ore types.

1.2 OBJECTIVES OF THE PROJECT

The aims of the project were to;

- investigate the mineralogical characteristics of the six ore types,
- determine the minimum leaching time for a specific copper extraction efficiency (leachability) for each ore type by leaching all the ore types under the same conditions of pH, temperature and %solids,
- determine the quantity of acid required for leaching the different ore types, and
- determine the amount of impurities, notably (Fe) that dissolved and reported in the pregnant solution.

The above steps were taken in order to establish the optimum operating parameters for extracting acid soluble copper (AS.Cu) from the different ore types at a minimum leaching time and thus minimising the co-dissolution of impurities in the pregnant solution which would otherwise adversely affect the subsequent treating processes. The particle size was optimized to 80% passing 150 μm . The pH used for all the experiments was between 1.56 and 1.64, the average pH being 1.60. The % solid of the pulp was 28%.

1.3 BRIEF BACKGROUND OF THE PLANT

Kansanshi Mining Company is jointly owned by the First Quantum Minerals and the Zambia Consolidated Copper Mines (ZCCM) Investment Holdings. The mining company is located in Solwezi in the North Western Province of Zambia. The Kansanshi mining project officially commenced production in the year 2005 with the aim of treating copper oxide and sulphide ores. The Company is made up of two departments; these are the Mining and the Processing Departments.

The Company produces finished copper cathodes as well as copper sulphide concentrates mostly from porphyry ores. Most of the copper concentrates are sold to other mining companies where there are processed usually through the pyrometallurgical route into finished copper. The rest of the concentrates are leached using pressure leaching method at the Kansanshi Pressure Leach Plant. The copper oxide ores are leached using agitation leaching.

1.3.1 THE MINING DEPARTMENT

The Mining Department operates two main open pits. These are; the Main pit and the North West pit. The Main pit is the major source of the sulphide ores while the North West pit is the major source of oxide ores as it is in its initial stage of mining. At the beginning of the project, the measured and indicated mineral resource at a 0.5% cut off totalled 302 million tonnes with an average grade of 1.17% copper.

1.3.2 THE PROCESSING DEPARTMENT

The Processing department consists of the front end and the back end sections.

The Processing plant treats about 24000 tones of oxide ore and about 24000 tones of sulphide ore per day. Ore treatment is flexible to allow for variations in ore types, and includes conventional milling operations (crushing and grinding), flotation, acid leaching, Solvent Extraction, and Electrowinning to produce copper cathodes of about 99.99% copper and copper sulphide concentrates.

1.4 PLANT OPERATIONS

The plant consists of the following subsections;

- Crushing,
- Grinding and Classification,
- Flotation,
- Thickening and filtration,
- Leaching,
- Counter Current Decantation (CCD),
- Tailings Disposal, and
- Solvent Extraction and Electrowinning.

1.4.1 THE CONCENTRATOR

The front end of the processing plant is what can be described as a concentrator and is made up of the following subsections;

Crushing, Grinding and Classification, Flotation, and the Thickening and filtration subsections.

Crushing Section

The basic function of the crushing circuit is to reduce the run-off mine (ROM) ore to a size suitable for feeding to the grinding circuit. Size reduction (comminution) in the crushing circuit is achieved through the use of primary crushers.

Grinding and Classification Section

The basic function of the grinding circuit is to reduce the size of the crushed ore to fines (80% passing 175 mm to the final product size of 80% passing 150 microns) in order to liberate the valuable mineral from the gangue minerals prior to leaching or flotation.

The classification circuit optimises the particle size distribution and thickens the resultant slurry prior to; leaching for oxide ores and flotation for sulphide ore.

Classification

Classification is the process of separating particles according to their settling rate in a fluid, in this case water. The classifying device used in the Kansanshi grinding circuit is the hydrocyclone.

The hydrocyclone is a classifying device that makes use of centrifugal force to increase the settling rate of particles.

Hydrocyclones have replaced mechanical classifiers because they are more efficient, especially in the finer size ranges. They require less floor space and consume less power. Cyclones have relatively short particle residence times allowing better circuit control. The hydrocyclone is used for classification of between 5 - 150 microns, although coarser particles can also be classified.

Flotation

Froth Flotation is a process in which the valuable minerals are separated from the waste material (gangue) and are floated by means of attachment to air bubbles. Due to the chemical and physical nature on the surface of the minerals, and by the use of certain chemical reagents, it is possible to recover the valuable mineral.

1.4.2 THE HYDROMETALLURGICAL SECTION

The slurry from the mills are thickened in the preleach thickener to about 55% solids and pumped to the repulp tank of the leaching circuit. The thickened slurry in the repulp tank is diluted with the addition of high grade raffinate and concentrated sulphuric acid (H_2SO_4) to about 28 % solids. The residence time is controlled by the flow rate of the slurry which depends on the pump power.

From the leach tanks the slurry gravitates to the Counter Current Decantation (CCD) circuit for solid-liquid separation. In the CCDs leaching continues as the solid residues still contain some acid soluble copper and the solution is acidic (low pH). The leach residues are pumped to the neutralization tank to neutralize the acid and later pumped to the tailings dam.

The clear solution is sent to the pond from where it is pumped for solvent extraction section for concentration and purification. From solvent extraction plant, the resulting solution is pumped to the Electrowinning tankhouses for copper recovery as copper cathodes.

The Flow Sheet for the Leaching Section at Kansanshi Processing Plant

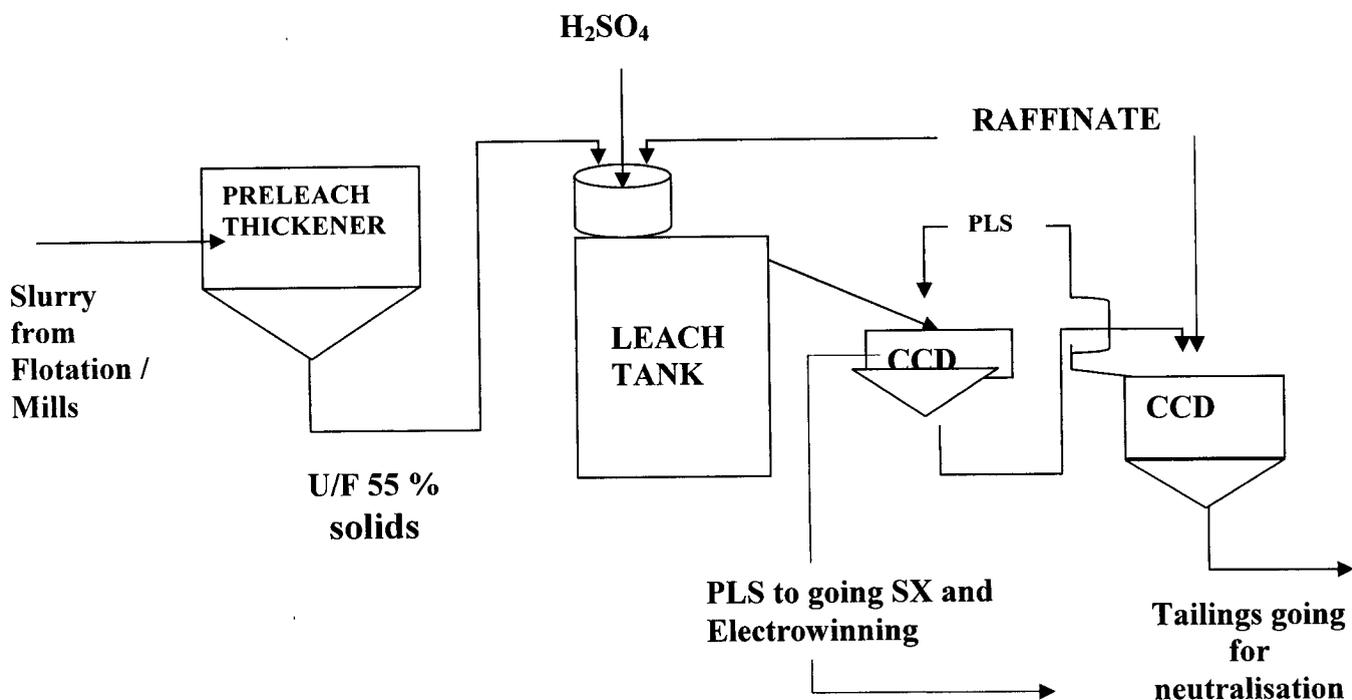


Figure 2: Flow sheet of the Kansanshi leaching circuit

CHAPTER TWO

2 LITERATURATURE REVIEW

2.1 THEORY ON LEACHING

Hydrometallurgical extraction is a process used to produce metal(s) from non-renewable reserves by utilising reactions in, or associated with, aqueous media. Hydrometallurgical reclamation applies to those same processes used to recover metals from recyclable reserves. The sequential use of these processes to produce metal from either source is termed hydrometallurgical processing. It consists usually of leaching followed by the separation of pregnant solution from the leach residue (solid), concentration and purification of the pregnant solution and the recovery of the metals by electrowinning or precipitation.

Leaching is removal of materials by dissolving them away from solids. It can also be defined as the selective dissolution of one or more components from a metal bearing solid material due to the action of a leachant (lixiviant or leach liquor).

The theory and practice of leaching are well-developed because for many years leaching has been used to separate metals from their ores and to extract sugar from sugar beets.

Ideally, leaching yields a metal-rich or pregnant solution from which the metal(s) can be separated and recovered in a pure state and a gangue residue sufficiently free of value minerals which can be discarded.

Exceptionally, impurities may be removed from a concentrate by leaching. This process is known as **hydrometallurgical beneficiation** and is a means of upgrading the concentrate.

Once an ore has passed through the crushing and grinding circuits, it will be sufficiently liberated to allow leaching of the metal. Agitation leaching is the extraction of a metal from the ore and involves mixing the slurry with leachant in a series of agitation tanks.

The method chosen for leaching an ore or waste material will depend mainly upon the value of the metal content of the material, the cost of mining and of bringing the ore to the surface, the cost of any milling and other preleaching treatment such as roasting or melting, and the easy of dissolution. The preferred method is likely to be the one with the lowest capital and operating costs, commensurate with both speed of processing and high percentage recovery. The easy of dissolution of the material is generally closely related to its chemical and mineralogical constitution, and in the case of a metallic ore, to its geological form. Among the copper minerals; malachite ($\text{CuCO}_3 \cdot \text{Cu}(\text{OH})_2$) and azurite ($2\text{CuCO}_3 \cdot \text{CuCO}_3$) dissolve

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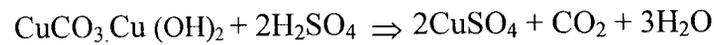
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readily in dilute sulphuric acid whereas covellite (CuS), chalcocite (Cu₂S) and chalcopyrite (CuFeS₂) require an oxidizing agent and dissolve less readily particularly chalcopyrite which is very refractory.

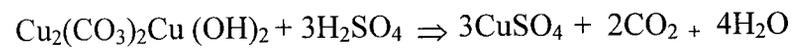
The leaching equation for some leachable copper ores;

The leaching reactions for common copper oxide minerals which are readily leacheable in dilute sulphuric acid (H₂SO₄) can be represented by the following equations:

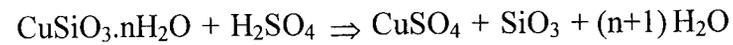
Malachite



Azurite



Chrysocolla



2.1.1 MECHANISM OF LEACHING AND LEACHING PARAMETERS

Thermodynamic data can only indicate the tendency of a reaction to occur. It provides no information on the rate of reaction or the detailed mechanism.

This is particularly so in the case of leaching reactions which are heterogeneous in character and consist of several sequential stages. Although leaching is basically the reaction between a solid and the leaching solution, three phases become involved when a gaseous reaction such as oxygen also participates in the dissolution process. This situation is represented in the Figure below in which the stages may be considered to consist of;

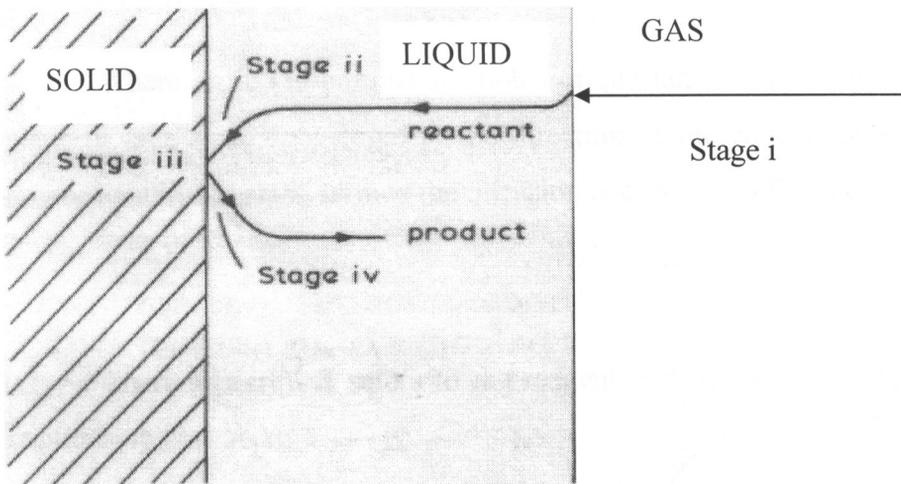


Figure 3: Stages involved in leaching

Stage 1

This stage involves the transfer of the gaseous reactant from the gaseous phase to, and dissolution in, the solution.

Stage 2

Transport of the reactant through the solution to the solid/liquid (solution) interface.

Stage 3

Reaction at the interface, which may be chemical or electrochemical and may involve adsorption and desorption and, in the case of an electrochemical reaction, transfer of electrons and ions across an electrical double-layer.

Transport of the products from the interface into the bulk solution.

Although it is possible for any one of these stages to be slowest step to control the rate of the overall process, it is most frequently stage 2 and sometimes stage 4, each of which involves diffusion.

Processes of this type are usually said to be diffusion-controlled. However under some circumstances no single stage is rate controlling and more than one stage may then be involved in determining the overall rate of reaction.

For such a process, transport of a reactant to the solid-solution interface involves passage through a diffusion layer in which the concentration of the reactants diminishes with the increasing distance from the surface due to reaction at the interface.

The simplest treatment of this involves the assumption of a linear concentration gradient with the application of Fick's first law of diffusion. This law may be expressed in the form:

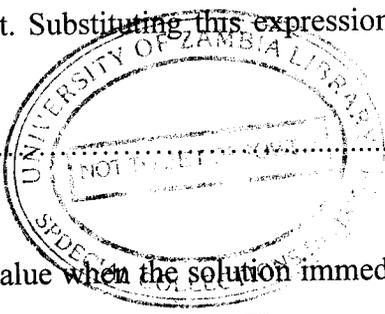
$$\frac{dn}{dt} = -D_B A \frac{dc}{dx} \dots\dots\dots (2.1)$$

Where dn is the number of moles of substance B flowing during a time interval dt through a plane of area A at right angle to the direction of the concentration gradient $-dc/dx$, and D_B is the proportionality constant, termed the diffusion coefficient of B. that

$$-\left(\frac{dc}{dx}\right)_R = \frac{C_R - C_{R0}}{\delta} \dots\dots\dots (2.2)$$

where subscript R refers to the reagent. Substituting this expression in the equation (2.2), the relationship becomes

$$\left(\frac{dn}{dt}\right) = -D_R A \frac{C_R - C_{R0}}{\delta} \dots\dots\dots (2.3)$$



The rate of flow will attain a limiting value when the solution immediately adjacent to the solid-liquid interface is totally depleted of reagent, i.e. when C_{R0} =zero. Then

$$\left(\frac{dn}{dt}\right)_R = \frac{D_R A C_R}{\delta} \dots\dots\dots (2.4)$$

This indicates that, for a diffusion-controlled reaction, the rate is directly proportional to the concentration of reagent in the bulk of solution, to the interfacial area and to the value of the diffusion

coefficient whilst it is inversely proportional to the thickness of the diffusion layer. For maximizing the rate of leaching, this suggests that the most effective conditions would involve an ore or material of small particle size to create a large interfacial area, with vigorous agitation to minimize the diffusion layer thickness. This thickness, in the absence of agitation may be ~0.5mm, but probably decrease to a value approaching 0.01mm when vigorous stirring (agitation) is used. This represents an increase in the rate by a factor of up to fifty times, and emphasizes the importance of agitation in leaching practice. The effect of increasing the concentration of a non-gaseous reagent beyond a certain level is not always greatly advantageous, because the solubility of a solid from the solution. There may also be a greater tendency for impurities to dissolve. However, this is rarely the case when oxygen is the gaseous reagent. Its solubility at atmospheric pressure is so low that the reaction rate is frequently directly proportional to oxygen partial pressure over a considerable range of pressure. Some limiting values are eventually reached where a change may occur in the rate-controlling process.

Temperature to a lesser extent also affects the rate of leaching.

2.1.2 LEACHING PERFORMANCE

Leaching performance is measured by the proportion of the metal in the feed slurry that is dissolved and therefore reports into the liquid or solution phase. Leaching performance is often termed leach recovery or extraction efficiency and can be measured across individual tanks as well as over the entire leach circuit.

The general formula for leach or extraction efficiency is shown below:

$$\text{Leaching or extraction efficiency} = \frac{\text{Mass of metal leached}}{\text{Mass of metal in the feed material}} * 100\% \dots\dots\dots (2.5)$$

2.1.3 FACTORS THAT AFFECT LEACHING

- Material preparation
- Selection of leachant
- Selection of leaching conditions
- Selection of leaching method

2.1.3.1 MATERIAL PREPARATION

Crushing and grinding of the ore is done in order to liberate valuable mineral from the gangue material. This increases the surface area for the acid and valuable ore contact.

Classification or sizing is done to optimize the particle size for leaching.

Concentration of the feed material has the following advantages

- Richer leach liquor
- Higher productivity for the same size capacity of the plant
- Solid-liquid separation is easier
- Residue washing is easier
- Loss of leachant entrapped in the residue is minimized
- Less use of leachant as most gangue which contributes to gangue acid consumption has been removed from the material

2.1.3.2 SELECTION OF LEACHANT

When using a leachant attention should be paid to the following:

- selectivity of the leachant
- adequacy of leaching kinetics
- should preferably react in low stoichiometric amounts
- must permit impurity removal
- must permit solid-liquid separation
- must permit value recovery
- must permit leachant regeneration
- must be easily washed from the residues

2.1.3.3 SELECTION OF LEACHING CONDITIONS

- leaching time (duration or residence time)
- temperature
- concentration of leachant
- pH of leaching
- solid-liquid ratio
- optimize red-ox potential of the leachant by addition of oxidizing or reducing agents
- degree of agitation or turbulence of leaching method
- partial pressure of gaseous reactant(s) (if gas is required by the reaction) must be optimized

Residence time

Residence time is an important parameter in the determination of leach performance as it represents the time that the slurry is in contact with the leachant. If the leach circuit residence time is reduced, lower leach recovery will generally result.

Residence time will depend on the total live slurry volume in each leach tank and the average slurry flowrate through the leach circuit and can be calculated as:

$$\text{Residence time (hrs)} = \text{live slurry volume (m}^3\text{)} / \text{slurry flow rate (m}^3\text{/hr)}$$

Residence time in the leach circuit will be reduced if;

- The dry solid rate and slurry flowrates are increased,
- The slurry % solids is reduced (for the same dry solid rate)
- Live slurry volume in any tank is reduced
- One or more leach tanks are by passed.

Increasing the residence time generally increases the recovery but may also cause large amounts of impurities to be leached.

Concentration of leachant

The concentration of the leachant in the slurry affects the pH of the slurry and the rate of leaching. The more concentrated the leachant the higher the rate of reaction. But too high the leachant concentration may cause high levels of impurities in the leach solution.

Solid-Liquid ratio (% solids of the slurry)

The slurry % solid is an important parameter in the leach circuit, as it determines the relative quantities of the solid and liquid (solution) phases. If the % solid is allowed to become too high, the following problems can occur;

- Reduction in copper leaching rate due to lack of fluidity of the slurry, and
- Increase in slurry viscosity, producing flow and blockage problems

There is an optimum % solids range for leaching typically in the operating range 20-35% solids.

The feed particle size distribution

The feed solids (in the slurry) must be sufficiently liberated in order to expose the copper minerals. This allows good contact between the solid and the leachant. The finer the ore, the faster the leaching rate. If the ore particles are too coarse, the copper minerals may only be partially liberated and leaching will occur very slowly. There is an optimum mesh of grind for which ore type

Temperature

Temperature increases the rate of leach by increasing the rates of reactions.

Agitation

Slurry agitation is important in each leach tank to ensure that the slurry is well mixed and maintain maximum contact between various reactants. Agitation is generally provided by air (Pachucas) or mechanical (impellers) means.

2.1.4 LEACHING METHODS

The method chosen for leaching an ore or waste material will depend mainly upon the value of the metal content of the material, the cost of mining and of bringing the ore to the surface, the cost of any milling and other preleaching treatment such as roasting or melting, and the easy of dissolution. The preferred method is likely to be the one with the lowest capital and operating costs, commensurate with both speed of processing and high percentage recovery. The easy of dissolution of the material is generally closely related to its chemical and mineralogical constitution, and in the case of an ore, to its geological form. Thus among the copper mineral malachite ($\text{CuCO}_3 \cdot \text{Cu}(\text{OH})_2$) Chrysocolla ($\text{CuSiO}_3 \cdot 2\text{H}_2\text{O}$) (and azurite ($2\text{CuCO}_3 \cdot \text{CuCO}_3$) dissolve readily in dilute sulphuric acid whereas covellite (CuS), chalcocite (Cu_2S) and chalcopyrite (CuFeS_2) require an oxidizing agent and dissolve less readily particularly chalcopyrite which is very refractory.

There are several leaching methods and include the following:

- Agitation leaching,
- Percolation or vat leaching,
- Dump and heap leaching,
- In-situ leaching or solution mining,
- Counter current decantation leaching,
- Pressure leaching, and
- Bacteria aided leaching.

The operations of these methods of leaching have been described in books like Extractive Copper Metallurgy by Biwas and Davenport and in Hydrometallurgical Extraction and Reclamation by Eric Jackson.

This project was based on agitation leaching and hence our attention will be on the agitation method.

Agitation leaching

Agitation leaching is carried out on finely ground ore, on melts and granulated alloy scrap, and on alloy turnings and borings. It provides mostly high rates of dissolution because of the large surface area involved. The agitation minimizes the diffusion-layer thickness and, when gaseous reactants are involved, maximizes the gas-liquid interface area. Agitation leaching at normal atmospheric pressure is extensively used. The reaction vessel may be equipped with mechanical impellers or may utilize air agitation.

Agitation leaching may be carried out as a batch process or as steady-state, continuous-flow process. The former tends to be adopted for higher-value, low-volume processes, whereas the latter is favoured for large-scale operations that lend themselves to automation. The degree of agitation in the latter situation is generally high enough to approximate to a continuous-flow, stirred tank reactor. Several reactors in series are needed to ensure the appropriate residence time at the given conditions. When the leaching has been completed, separation of the solution from the insoluble material is generally necessary. This is mostly done through counter-current decantation which also incorporates washing of the residues and is carried out in thickeners. Flocculant help the solids to settle and advantage has been taken of this to modify the design of the thickener to give beneficial space saving. Slurry agitation is important in leaching vessels to ensure that the slurry is well mixed and to maintain maximum contact between the leachant and the milled ore.

CHAPTER THREE

3 EXPERIMENTAL PROCEDURE

3.1 MATERIALS / REAGENTS

- six different ore types
- concentrated sulphuric acid
- high grade raffinate (15gpl H₂SO₄ and 0.36 gpl Cu)
- water
- flocculant (N100)

3.2 APPARATUS USED

- electric balance
- Four 5000ml beakers
- Four 1000ml beakers
- Four rotating impellers
- pH metre
- a 500ml sampling can (beaker)
- Filter papers
- Pressure filter
- Syringe
- Drying oven
- Twelve 100ml bottles

3.3 MATERIAL PREPARATION

For each ore type, five 2 kg ore samples were weighed. Each 2 Kg ore sample was ground for twenty minutes to 80% passing 150 μm using a laboratory rod mill.

3.3.1 PULP FORMATION

For each milled ore type, four 1.5kg ore samples were weighed and packed in plastics labelled 1, 2, 3 and 4 e.g. (for high grade blue the plastics were labelled HGB 1, 2, 3 and 4)

Thereafter four clean 5000 ml beakers labelled 1, 2, 3, and 4 were mounted with baffles inside.

Rotating impellers were set in the centre of each beaker with the impeller blades placed at a convenient height from the bottom to provide good agitation of the slurry.

1225ml of water was poured into each beaker and the rotating impellers in each beaker were switched on.

The 1.5Kg of milled ore in plastics 1, 2, 3 and 4 were then poured in beakers 1, 2, 3 and 4 respectively to form slurry of 55% solids in each beaker.

Finally 2500ml of high grade raffinate containing 15 gpl H_2SO_4 and 0.3 gpl Cu was added to the slurry in each beaker to form slurry of about 28% solids and leaching begins.

3.3.2 LEACHING PROCESS

Immediately after the addition of raffinate the stop watch was started and the pH of the slurry was read from the pH meter placed in the slurry.

Leaching was done at the pH of 1.6.

When the pH rose above 1.64 an appropriate amount of sulphuric acid was added using a syringe to the mixture in each of the four the beakers and the amount of acid added was recorded. This addition of acid was repeated to keep the pH between 1.56 and 1.64 during the entire leaching process.

3.3.3 SAMPLING

After 2 hours of leaching

After 2 hours of leaching 500ml of slurry was removed from beakers 1, 2 and 3. Immediately thereafter the same volume of slurry was as that removed was added to beakers 1, 2, and 3 by using slurry from beaker 4. The aim of replenishing the slurry was to have constant volume in the three beakers during the entire leaching process. This was followed by filtration of the slurry from the three beakers.

The filtrate from the three beakers was analysed for gpl Cu. The residue was dried and then analysed for %TCu and %AS.Cu.

The same procedure for two hours of leaching was followed with the three hours of leaching. After four hours of leaching the same procedure was followed as for the two and three hours of leaching time except that there was no replenishment of the slurry. The filtrates from the three beakers were analysed gpl Cu and gpl Fe. The residue was dried and then analysed for %TCu and %AS.Cu.

3.3.4 WASHING TESTS

After 20 minutes of settling the clear solution was siphoned out of each beaker.

2000ml low grade raffinate was added to the thickened slurry in the beaker and the impellers were switched on.

After a period of 1 hour 500ml slurry samples were removed from beaker 1, 2 and 3. The slurry was filtered.

The filtrate from the three beakers was analysed gpl Cu and gpl Fe.

The residue was dried and analysed for %TCu and %AS.Cu.

This procedure was repeated on all the six ore types.

CHAPTER FOUR

4 RESULTS AND DISCUSSION

4.1 MINERALOGICAL ANALYSIS OF THE ORE TYPES

From the mineralogical analysis of the ores, it was observed that the chemical compositions of the ores were very similar and that the ores only differed in the relative abundance of the different chemical species as can be seen from Tables 1, 2, and 3 below. The chemical species in the ores that are of interest in leaching (i.e. acid soluble species) were: malachite, pseudomalachite, and Chrysocolla. These species were all 100% liberated from the gangue material (i.e. the ratio F:L were 100:0) after milling. No native copper was traced.

The acid insoluble copper species that were traced in the ore samples were: chalcopyrite, bornite, chalcocite, and some carrollite. These are basically sulphide ores and were found to comprise of portion which were fully liberated from and some portion partially locked in the gangue as can be seen from the F:L ratios in Tables 1, 2, and 3. These would be problematic in flotation where they would not float but report to the tails as middlings.

Table 1: Ore Mineral Data for Kansanshi High and Low Grade Green Samples.

MINERALS	HIGH GRADE GREEN				LOW GRADE GREEN			
	Wt%	%TCu	%ASCu	F:L	Wt%	%TCu	%ASCu	F:L
Chalcopyrite	0.2	0.07	-	67:33	-	-	-	-
Bornite	0.2	0.13	<<0.01	100:0	-	-	-	-
Chalcocite	-	-	-	-	0.1	0.08	<<0.01	100:0
Pyrite	4.9	-	-	92:8	0.2	-	-	92:8
Carrollite	-	-	-	-	-	-	-	-
Pyrrhotite	-	-	-	-	-	-	-	-
Native Copper	-	-	-	-	-	-	-	-
Malachite	1.7	0.98	0.98	98:2	0.7	0.40	0.40	89:11
Pseudomalachite	0.5	0.28	0.28	100:0	0.2	0.11	0.11	100:0
Chrysocolla	0.2	0.07	0.07	100:0	0.8	0.29	0.29	100:0
Cupriferous Mica	0.3 - 0.5	0.02	<0.01	-	0.3 - 0.5	0.02	<0.01	-
Gangue	91.9	-	-	-	97.6	-	-	-
TOTAL	100.0	1.55	1.33	-	100.0	0.90	0.80	-
		%HFCu = 0.01				%HFCu = 0.20		

Table 2: Ore Mineral Data for Kansanshi High and Low Grade Blue Samples

MINERALS	HIGH GRADE BLUE				LOW GRADE BLUE			
	Wt%	%TCu	%ASCu	F:L	Wt%	%TCu	%ASCu	F:L
Chalcopyrite	0.5	0.17	-	73:27	0.3	0.10	-	70:30
Bornite	<<0.1	<<0.1	<<0.01	100:0	<<0.1	<<0.01	<<0.01	100:0
Chalcocite	0.5	0.40	0.01	58:42	0.5	0.40	0.01	69:31
Pyrite	<<0.1	-	-	100:0	<<0.1	-	-	90:10
Carrollite	<<0.1	<<0.01	-	27:73	-	-	-	-
Pyrrhotite	<<0.1	-	-	100:0	-	-	-	-
Native Copper	Trace	-	-	-	-	-	-	-
Malachite	2.8	1.61	1.61	100:0	4.6	2.65	2.65	100:0
Pseudomalachite	0.4	0.22	0.22	100:0	0.1	0.06	0.06	100:0
Chrysocolla	1.6	0.58	0.58	100:0	0.6	0.22	0.22	100:0
Cupriferous Mica	0.2-0.4	0.01	<0.01	-	-	-	-	-
Gangue	93.9	-	-	-	93.9	-	-	-
TOTAL	100.0	2.99	2.42	-	100.0	3.43	2.94	-
		%HFCu = 0.08				%HFCu = 0.07		

Table 3: Ore Mineral Data for Kansanshi High and Low Grade Red Samples.

MINERALS	HIGH GRADE RED				LOW GRADE RED			
	Wt%	%TCu	%ASCu	F:L	Wt%	%TCu	%ASCu	F:L
Chalcopyrite	1.0	0.34	-	82:18	0.1	0.03	-	67:33
Bornite	-	-	-	-	-	-	-	-
Chalcocite	0.9	0.72	0.01	95.5	0.1	0.08	<<0.01	47:53
Pyrite	1.1	-	-	92:8	<<0.1	-	-	50:50
Carrollite	<<0.1	<<0.01	-	100:0	-	-	-	-
Pyrrhotite	-	-	-	-	-	-	-	-
Native Copper	-	-	-	-	-	-	-	-
Malachite	2.2	1.27	1.27	100:0	0.6	0.34	0.34	94:4
Pseudomalachite	0.5	0.28	0.28	100:0	<<0.1	0.01	0.01	100:0
Chrysocolla	0.1	0.04	0.04	100:0	0.4	0.14	0.14	100:0
Cupriferous Mica	-	-	-	-	-	-	-	-
Gangue	94.2	-	-	-	98.8	-	-	-
TOTAL	100.0	2.65	1.59	-	100.0	0.60	0.49	-
		%HFCu = 0.07				%HFCu = 0.13		

GPL Cu, Recovery versus Leaching Time graphs for the different ore types

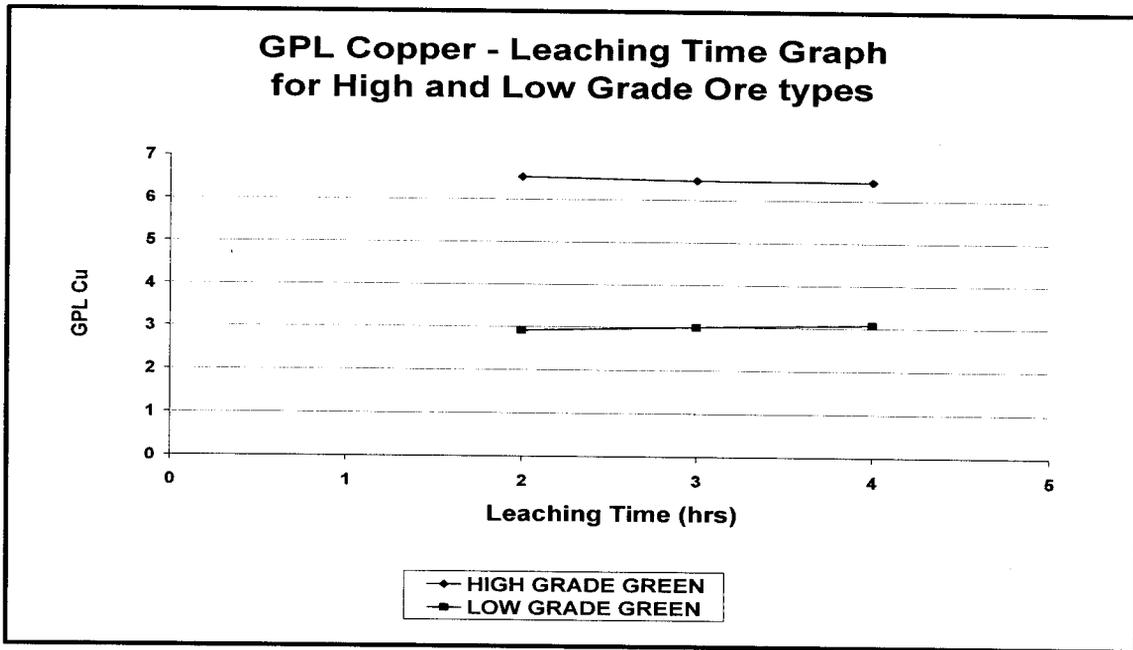


Figure 4: gpl Cu versus leaching time graph for HGG and LGG ore types

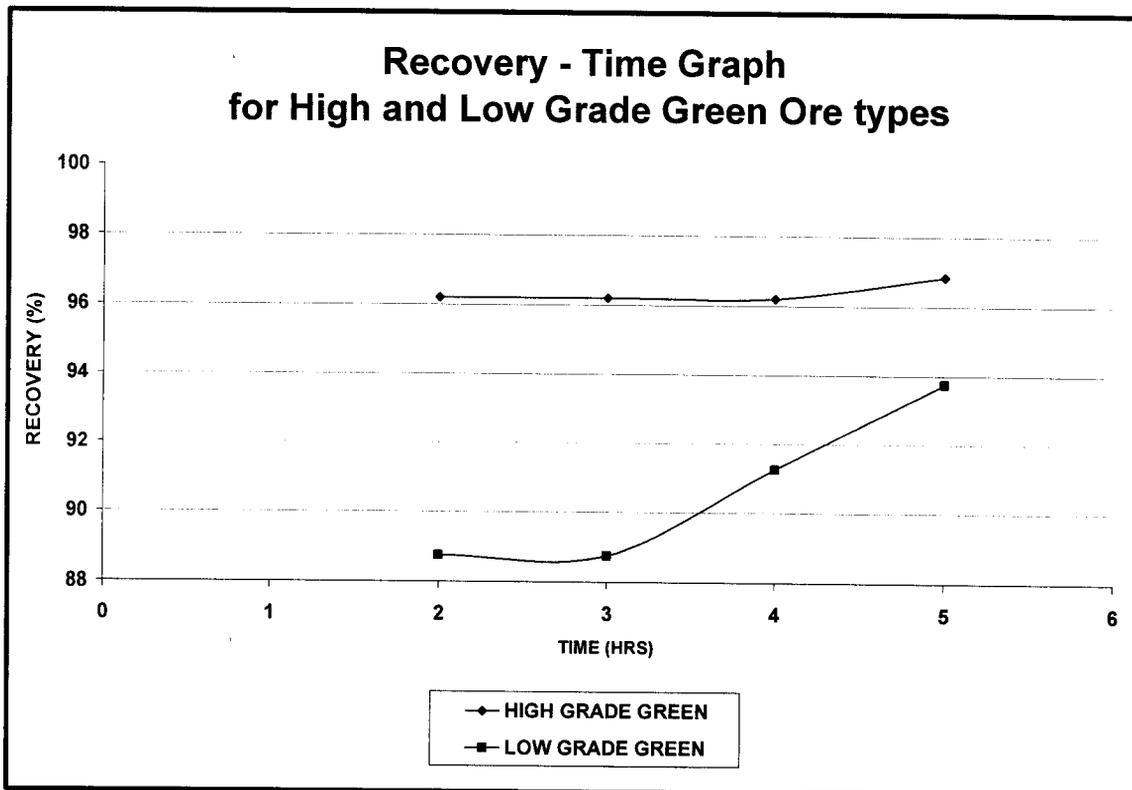


Figure 5: Recovery versus leaching time graph for HGG and LGG ore types

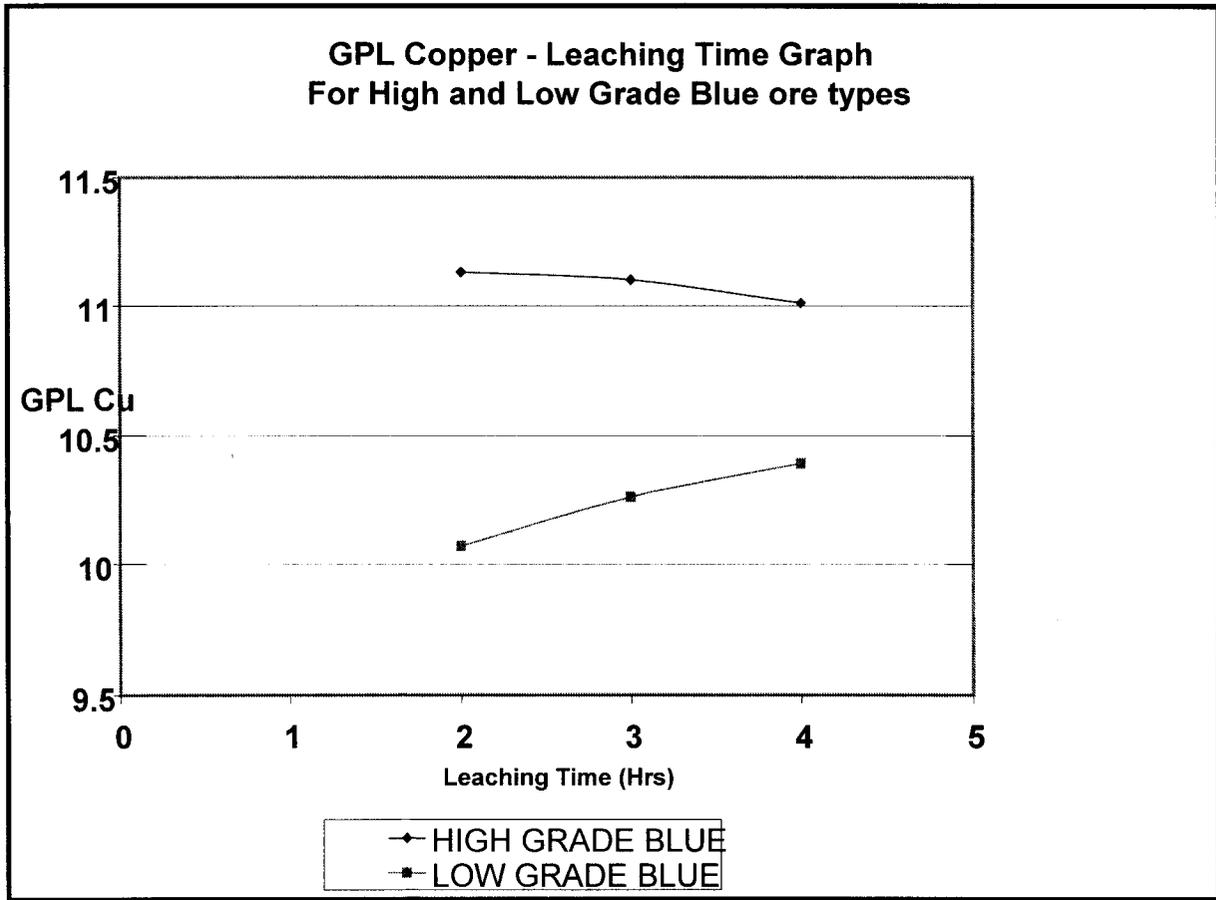


Figure 6: gpl Cu versus leaching time graph for HGB and LGB ore types

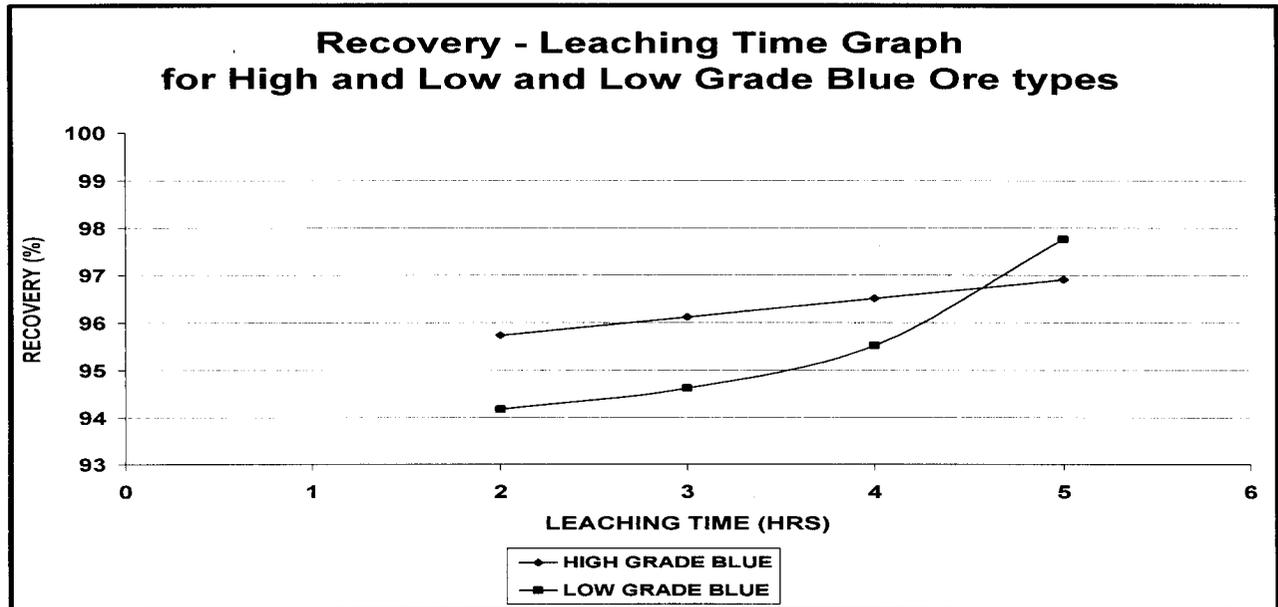


Figure 7: gpl Cu versus leaching time graph for HGB and LGB ore types

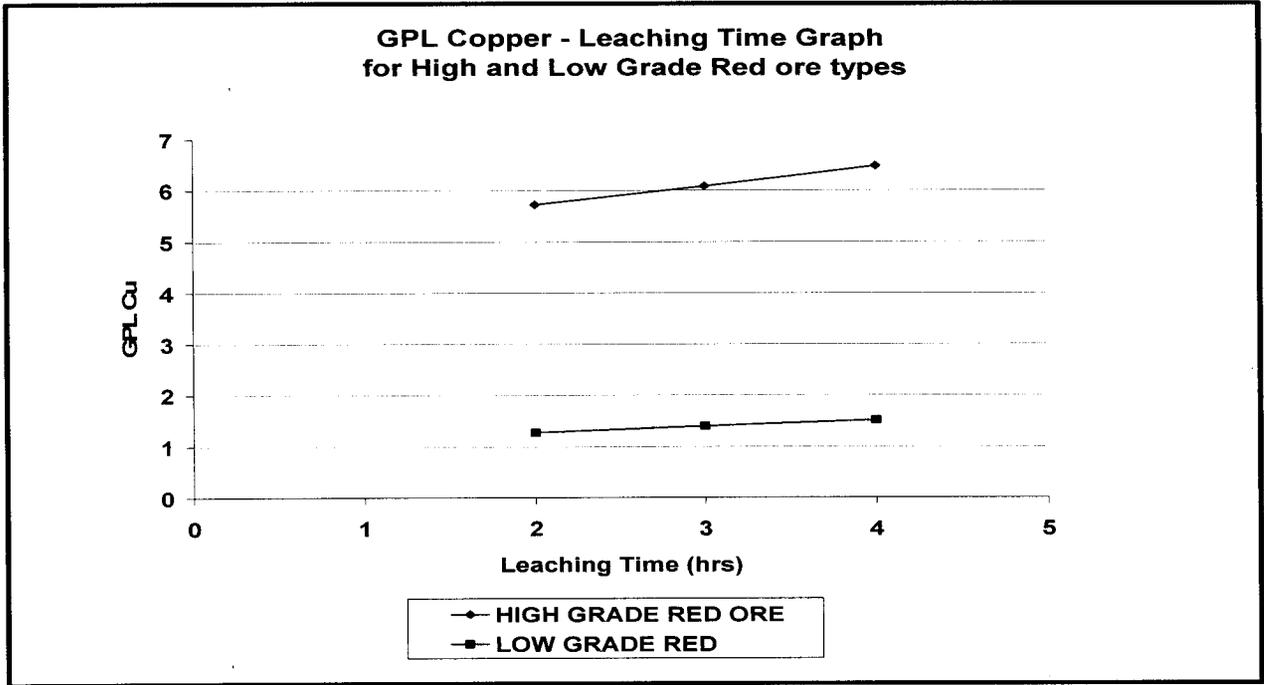


Figure 9: gpl Cu versus leaching time graph for HGR and LGR ore types

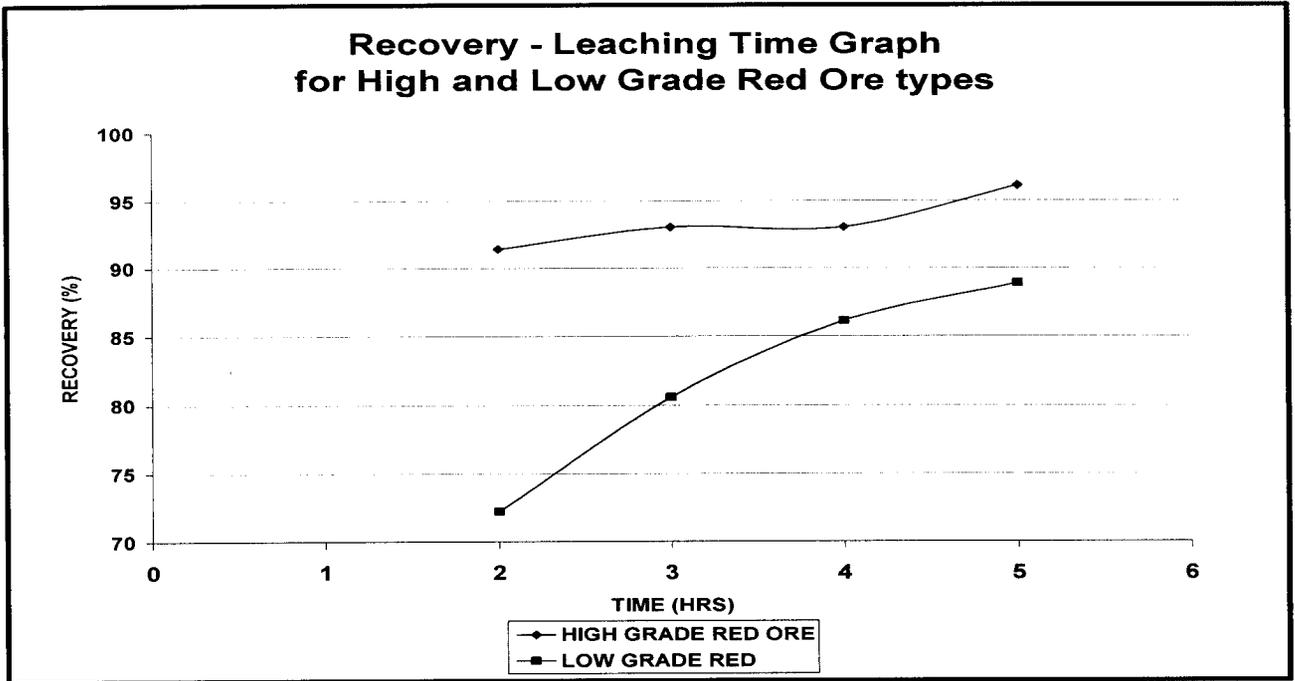


Figure 8: Recovery versus leaching time graph for HGR and LGR ore type

4.2 OPTIMISATION OF RESIDENCE TIME

The experiments were successfully done despite irregularities in the values of gpl Cu for HGB ore as can be seen from figure 6. The values were supposed to remain constant or else increase from 2 to 3 and 4 hours of leaching. This can be attributed to errors made during the analysis for gpl Cu. It could also be due to impurities on the walls of the container of the solution which would enhance the precipitation of the copper sulphate crystals. Alternatively, it could be explained that the solution became saturated and hence precipitation of the copper as copper sulphate.

The recoveries at 2, 3, and 4 hours of leaching and then the overall recovery after 4 hours of leaching and 1 hour of washing are shown in figures 5, 7 and 9 the following was observations were made;

- From figure 5, the recoveries of AS.Cu after 2, 3 and 4 hours of leaching HGG ore were the same, i.e. 96.2%. After 4hrs of leaching and 1hr wash the total recovery of %AS.Cu was 96.84%.
- From figure 5, the recoveries of AS.Cu after 2, 3 and 4 hours of leaching LGG were, 88.75%, 88.75% and 91.25% respectively. After 4 hrs of leaching and 1hr wash the total recovery of %AS.Cu was 93.75%.
- From figure 7, the recoveries of AS.Cu after 2, 3 and 4 hours of leaching HGB were, 95.74, 96.12% and 96.51% respectively. After 4 hrs of leaching and 1 hr wash the total recovery of %AS.Cu was 96.90%.
- From figure 7, the recoveries of AS.Cu after 2, 3 and 4 hours of leaching LGB ore were, 94.17%, 94.62% and 95.52% respectively. After 4 hrs of leaching and 1 hr wash the total recovery of %AS.Cu was 97.76%.
- From figure 9, the recoveries of AS.Cu after 2, 3 and 4 hours of leaching HGR ore were, 91.47%, 93.02% % and 93.02% respectively. After 4 hrs of leaching and 1 hr wash the total recovery of %AS.Cu was 96.12%.
- From figure 5, the recoveries of AS.Cu after 2, 3 and 4 hours of leaching LGR ore were, 72.22%, 80.56% % and 86.11% respectively. After 4 hrs of leaching and 1 hr wash the total recovery of %AS.Cu was 88.89%.

ACID CONSUMPTION CHART FOR THE ORES

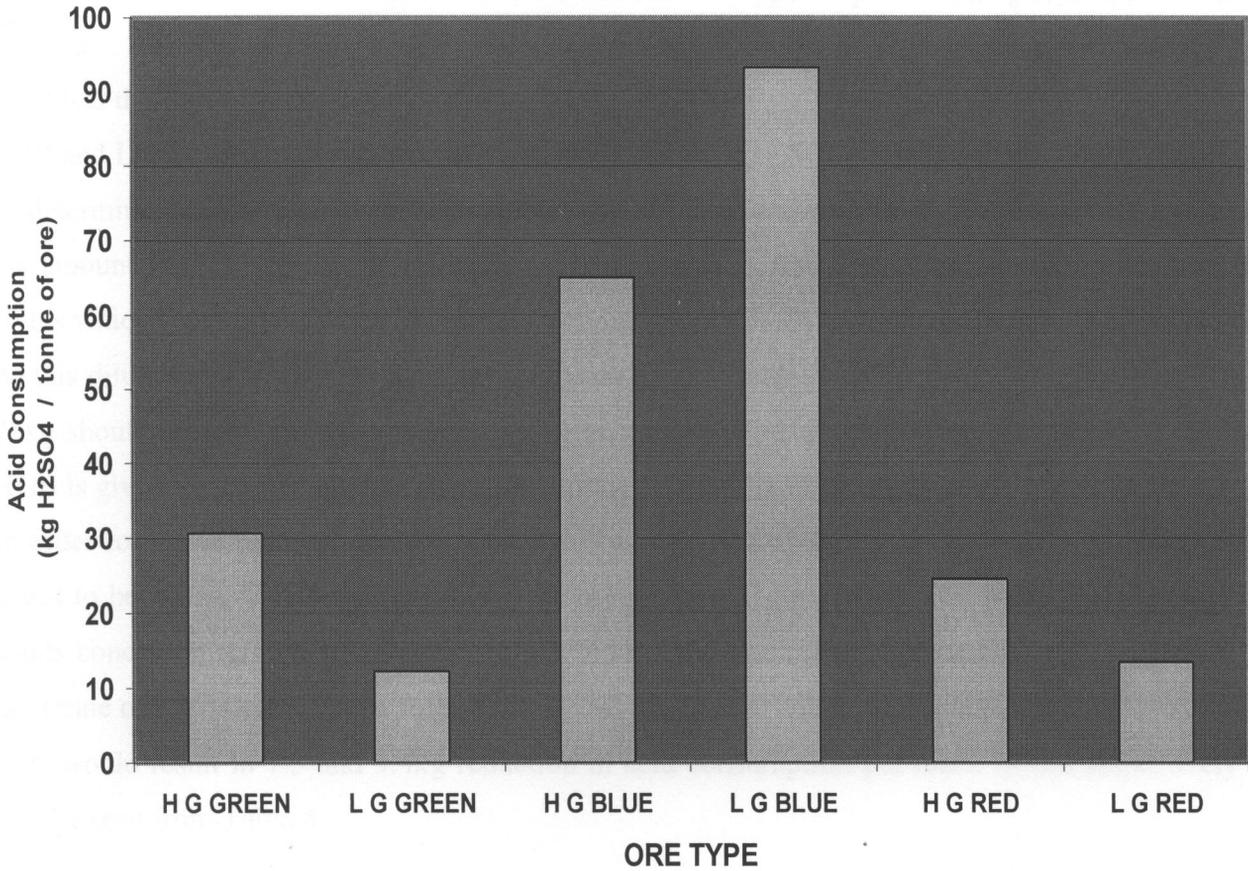


Figure 10: Acid Consumption for the different ore types

By using slurry of higher solid concentration from the preleach discharge as feed to the repulp tank, the consumption of concentrated acid could be lowered as shown in Table 1 below.

Table 4: acid served on different solids concentration

% solids of the preleach discharge	GPL H ₂ SO ₄ IN RAFF	Vol (l) of Raff used for leaching	Mass of H ₂ SO ₄ (g) in Raff	H ₂ SO ₄ from Raff per tonne ore leached	Acid (kg) served per tonne of ore leached
55	15	1.67	25.0	25.0	0.0
65	15	1.95	29.2	29.2	4.2
75	15	2.15	32.3	32.3	7.3
85	15	2.30	34.4	34.4	9.4

4.3 ACID CONSUMPTION

Figure 10 shows the acid consumption of the different ore types expressed in kg H₂SO₄ per tonne of ore leached.

LGB had the highest concentrated acid consumption 93 kg H₂SO₄ per tonne of ore.

LGR and LGG had the lowest consumption of 13 and 12kg H₂SO₄ per tonne of ore.

In determining the amount of acid consumed for leaching a tonne of ore, it is important to account for the amount acid present in the raffinate used for leaching. At 55% solids, to dilute the slurry to 28% solids which is the optimum solids' concentration, 25kg of H₂SO₄ will be present in the raffinate used to do this dilution. To determine the total acid used for leaching at 55% solids feed to the leaching circuit 25kg should be added to the concentrated acid used. For instance the total acid required for leaching HGB is given as (25+93)=118kg H₂SO₄ per tonne of ore. The 25kg originates from the raffinate.

In order to reduce the acid consumption, the concentration of solids in the feed to the leaching circuit needs to be raised. This would allow for the use of more H₂SO₄ present in the raffinate. Increasing the solids concentration in the feed from 55% to 65% solids would reduce the acid consumption by 4.2kg per tonne of ore as can be seen from Table 4. An increase of solids concentration from 55% to 75% and 85% would result in 7.3 and 9.4kg reduction in acid consumption per tonne of ore respectively as can also be seen from Table 4.

CHAPTER FIVE

5 CONCLUSION

From the results and analysis, it can be concluded that:

- From figures 4 and 5, the optimum residence time for **HGG** was 2 hours. 0.493 kg of Fe per tonne ore was co-leached.
- From figures 4 and 5, the optimum residence time for **LGG** was 4 hours.
- From figures 6 and 7, the optimum residence time for **HGB** and **LGB** was 2 hours. 0.853 kg of Fe per tonne ore was co-leached.
- LGB 0.70 kg of Fe per tonne ore was co-leached.
- From figures 8 and 9, the optimum residence time for **HGR** and **LGR** was 2 hours, the acid consumption was kg H₂SO₄ per tonne of ore.
- **LGR** the kg.
- From table 4, it can be concluded that the increasing the concentration of solids in the feed to the leaching circuit would lower the consumption of concentrated sulphuric acid. Increasing the concentration of solids from 55% to 65% solids in the feed, the consumption would reduce by 4.2kg H₂SO₄/ tonne ore. From 55% to 75% and 85% solids the consumption would reduce by 7.3and 9.4 kg / tonne of ore respectively.
- From figure 10, the acid consumption for HGG, LGG, HGB, LGB, HGR and LGR are 12.4, 31, 64, 93.1, 24.53 and 13.49 kg H₂SO₄ per tonne of ore respectively.

CHAPTER SIX

6 RECOMMENDATIONS

From the experimental results, the following recommendations can be made;

- a) For HGG, HGB and LGB ore types, the residence time in the leach circuit should be two hours and thereafter followed by one hour of washing / leaching of the leach residues in the CCDs. This would increase the working capacity of the leaching circuit. It would also reduce the amount of impurities that would otherwise be co-leached together with the AS.Cu copper.
- b) From the AS.Cu assays of the leach residues at a particle size distribution of 80% passing 150 μ m, the minimum % AS.Cu should be 0.04%. At this assay of the solids leaching should be assumed to have completed. This is because from experimental result the AS.Cu assays never went less than 0.04% AS.Cu in the tails.
- c) The acid consumption per tonne of acid soluble copper (AS.Cu) extracted for HGB and LGB ore types was very high. This can be reduced by increasing %solids of the slurry from the preleach thickener to the repulp tank from 55% to 65% or more. This would mean that more raffinate would be required to be added to the repulp tank in order to dilute the slurry to 28% solids. This would increase the utilization of the free acid in raffinate for leaching and thus reducing the amount of concentrated acid required.

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APPENDIX 1

Leach Residue assays and solution Concentrations

Table 5: Leach Residues assay and leach Solution concentration for High Grade Green ore

Leaching Time (hrs)	HIGH GRADE GREEN					
	Sample Identity	% TCu	% AS.Cu	GPL Cu	mls H ₂ SO ₄	Acid Consumption
2	Lot 1	0.44	0.06	7.25		
	Lot 2	0.42	0.07	6.10		
	Lot 3	0.39	0.06	6.20		
	average	0.42	0.06	6.52		
3	Lot 1	0.40	0.07	6.85		
	Lot 2	0.38	0.06			
	Lot 3	0.41	0.05	6.05		
	average	0.40	0.06	6.45		
4	Lot 1	0.40		7.05		
	Lot 2		0.07	6.10		
	Lot 3	0.40	0.05	6.15		
	average	0.40	0.06	6.43		
After 1hr of washing	Lot 1	0.37	0.04	3.10		
	Lot 2	0.36	0.05	2.95		
	Lot 3	0.34	0.06	3.05		
	average	0.36	0.05	3.03		

Table 6: Leach Residues assay and leach Solution concentration for Low Grade Green ore

Leaching Time (hrs)	LOW GRADE GREEN					
	Sample Identity	% TCu	% AS.Cu	GPL Cu	mls H ₂ SO ₄	Acid Consumption
2	Lot 1	0.38	0.10	3.00		
	Lot 2	0.35	0.09	2.88		
	Lot 3	0.33	0.09	2.88		
	average	0.35	0.09	2.92		
3	Lot 1	0.34	0.08	3.13		
	Lot 2	0.34	0.09	2.88		
	Lot 3	0.37	0.09	3.00		
	average	0.35	0.09	3.00		
4	Lot 1	0.28	0.06	3.18		
	Lot 2	0.33	0.07	3.00		
	Lot 3	0.31	0.07	3.03		
	average	0.31	0.07	3.07		
After 1hr of washing	Lot 1	0.26	0.05	1.58		
	Lot 2	0.28	0.05	1.50		
	Lot 3	0.28	0.05	1.48		
	average	0.27	0.05	1.52		

Table 7: Leach Residues assay and leach Solution concentration for High Blue Grade ore

Leaching Time (hrs)	HIGH GRADE BLUE						
	Sample Identity	% TCu	% AS.Cu	GPL Cu	mls H ₂ SO ₄	Acid Consumption	
2	Lot 1	0.54	0.11	11.05			
	Lot 2	0.61	0.10	11.13			
	Lot 3	0.62	0.11	11.20			
	average	0.59	0.11	11.13			
3	Lot 1	0.56	0.10	11.20			
	Lot 2	0.59					
	Lot 3	0.53	0.09	11.00			
	average	0.56	0.10	11.10			
4	Lot 1	0.48	0.08	10.90			
	Lot 2	0.57	0.09	11.13			
	Lot 3	0.55	0.09	11.00			
	average	0.53	0.09	11.01			54
After 1hr of washing	Lot 1	0.51	0.09	5.70			
	Lot 2	0.54	0.07	5.35			
	Lot 3	0.50	0.07	5.48			
	average	0.52	0.08	5.51			

Table8: Leach Residues assay and leach Solution concentration for Low Blue Grade ore

Leaching Time (hrs)	LOW GRADE BLUE						
	Sample Identity	% TCu	% AS.Cu	GPL Cu	mls H ₂ SO ₄	Acid Consumption	
2	Lot 1	0.72	0.12	9.95			
	Lot 2	0.78	0.13	10.20			
	Lot 3	0.74	0.11	10.05			
	average	0.75	0.12	10.07			
3	Lot 1	0.73	0.13	10.15			
	Lot 2	0.77	0.13	10.50			
	Lot 3	0.74	0.11	10.12			
	average	0.75	0.12	10.26			
4	Lot 1	0.70	0.10	10.22			
	Lot 2	0.74	0.09	10.65			
	Lot 3	0.67	0.10	10.30			
	average	0.70	0.10	10.39			76
After 1hr of washing	Lot 1	0.59	0.05	5.25			
	Lot 2	0.66	0.05	5.20			
	Lot 3	0.62	0.06	4.85			
	average	0.62	0.05	5.10			

Table9: Leach Residues assay and leach Solution concentration for High Grade Red ore

Leaching Time (hrs)	HIGH GRADE RED						
	Sample Identity	% Tcu	% AS.Cu	GPL Cu	mls H ₂ SO ₄	Acid Consumption	
2	Lot 1	1.33	0.11	5.75			
	Lot 2	1.28	0.10	6.35			
	Lot 3	1.28	0.11	6.15			
	average	1.30	0.11	6.08			
3	Lot 1	1.27	0.09	5.75			
	Lot 2	1.28	0.08	5.95			
	Lot 3	1.19	0.09	5.45			
	average	1.25	0.09	5.72			
4	Lot 1	1.56	0.09	6.65			
	Lot 2	1.18	0.10	6.45			
	Lot 3	1.32	0.07	6.30			
	average	1.35	0.09	6.47			20
After 1hr of washing	Lot 1	1.12	0.05	3.00			
	Lot 2	1.14	0.04	2.90			
	Lot 3	1.03	0.05	3.10			
	average	1.10	0.05	3.00			



Table10: Leach Residues assay and leach Solution concentration for Low Grade Red ore

Leaching Time (hrs)	LOW GRADE RED						
	Sample Identity	% Tcu	% AS.Cu	GPL Cu	mls H ₂ SO ₄	Acid Consumption	
2	Lot 1	0.39	0.10	1.33			
	Lot 2	0.36	0.09	1.39			
	Lot 3	0.33	0.10	1.45			
	average	0.36	0.10	1.39			
3	Lot 1	0.31	0.07	1.06			
	Lot 2	0.31	0.06	1.32			
	Lot 3	0.33	0.08	1.42			
	average	0.32	0.07	1.27			
4	Lot 1	0.30	0.06	1.40			
	Lot 2	0.32	0.05	1.55			
	Lot 3	0.31	0.05	1.58			
	average	0.31	0.05	1.51			11
After 1hr of washing	Lot 1	0.29	0.04				
	Lot 2	0.27	0.04				
	Lot 3	0.28	0.05				
	average	0.28	0.04				