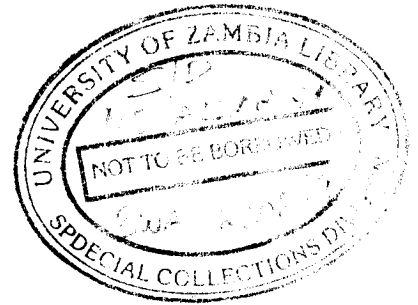# THE UNIVERSITY OF ZAMBIA

SCHOOL OF MINES

DEPARTMENT OF METALLURGY AND MINERAL PROCESSING

MM590 PROJECT REPORT

OPTIMISATION OF ACID AND LIME CONSUMPTION AT NKANA OXIDE LEACH PLANT

AUTHOR   : BWALYA CHISHIMBA S.

COMP. #   : 21027668

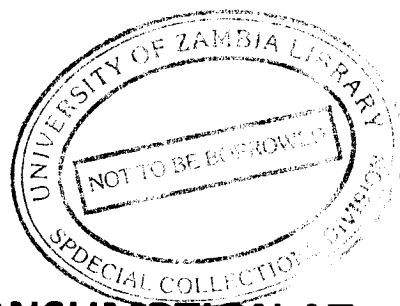SUPERVISOR   : DR J.H. MASINJA

PROJECT SITE   : MOPANI COPPER MINES

                  NKANA MINE LEACH PLANT

                  2008/2009

THE UNIVERSITY OF ZAMBIA

SCHOOL OF MINES

DEPARTMENT POF METALLURGY AND MINERAL PROCESSING

OPTIMISATION OF ACID AND LIME CONSUMPTION AT NKANA OXIDE LEACH PLANT

REPORT SUBMITTED TO DERPARTMENT OF METALURGY AND MINERAL PROCESSING OF THE SCHOOL OF MINES, IN PARTIAL FULFILLMENT OF THE REQUIREMENTS OF THE AWARD OF THE BACHELOR OF MINERAL SCIENCE DEGREE PROGRAMME (B.Mn.Sc.)
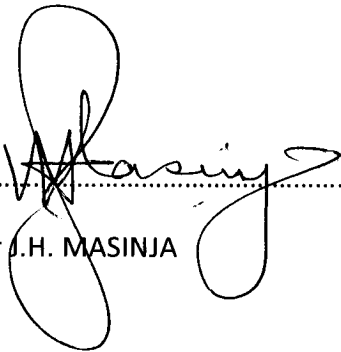
GREAT EAST ROAD CAMPUS

LASAKA; 2008/2009

I)   EXAMINERS PAGE
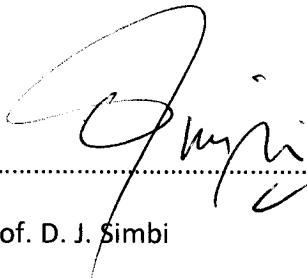
INTERNAL EXAMINER...........

Dr J.H. MASINJA
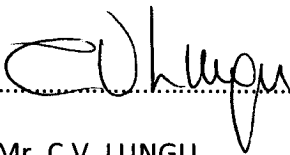
INTERNAL EXAMINER.........................

Prof. D. J. Simbi

HEAD OF DEPARTMENT............

Mr. C.V. LUNGU

II) DEDICATION

This report is dedicated to my parents and all my brothers and sisters for their encouragement and especially to my nephew Kangwa.

III) ACKNOWLEGEMENTS

Optimisations of acid and lime consumption at Nkana leach plant

Table of contents

LIST OF TABLES AND FIGURES

IV) ABBREVIATIONS

NLP: Nkana leach plant

SX: solvent extraction

PLS: pregnant leach solution

CCD: counter current decantation

Gpl: grams per liter

CV: conveyor belt

PLT: pre-leach thickener

HG PLS: high grade pregnant leach solution

LG PLS: low grade pregnant leach solution

RTD: resident time distributions

CSTR: continuous series time recondings

V)   ABSTRACT

Nkana Leach Plant (NLP) of the Mopani Copper Mines Plc Nkana mine site treats ore from Mindola open pit mine. Leaching is employed in the treatment of the oxide ore which is followed by solvent extraction (SX). However the leach circuit was characterized by low metal recoveries in the pregnant leach solution (PLS) ,while acid consumed was not equivalent to the amount of metal recovered. In an effort to lower acid consumption at the leach tanks test works were conducted on laboratory scale on the feed material to the ball mill. Sampling was done from the stock pile which feeds the belt to the sag and ball mills respectively. Parameter considered for study in leach experiments were, densities, Particle size and its distribution, Optimal pH for leeching, Rate of agitation. In order to see what effect these parameters have on acid consumption, laboratory tests were conducted on some of these parameters, the parameters selected were, densities and pH which showed significant response to acid consumption. Leaching experiments were done in a baffled tank and an agitator was used to agitate the slurry, a pH meter was used monitor the pH. Results obtained showed that an increase in the pH from 1.8-1.9 show a reduction in acid consumption from 87ml-75ml with a deference in the efficiency variations averaging 67.38%-48.79 respectively. The densities show significant change with reduction in densities, a change from 24%-27% solids showed acid consumption changes from 360ml-377ml. the metal recoveries in both cases averaged 4-6gpl Cu in solution. The ore was averaging 1.0-1.89% total copper and 0.8-1.2% acid soluble Cu, this was the ore used in the experiments. The plant maximum efficiency of about 90% on average is usually obtained, with acid consumptions of about 200tonnes per day on average. Lime was used for nuetralisation of tails after the leaching process; the two types of lime which were in use are hydrated lime and quick lime. Experiments were conducted using the two types of lime in a baffled tank and an agitator, without an agitator and using water when and during addition to the tailing. It was discovered that with quick lime, if an agitator is used the consumption averaged 23.21g in 200ml volume of tails and hydrated lime showed 25.48g in 200ml while in non agitated systems it was shown that quick lime and hydrated lime showed increases of about 26g and 30g respectively. And reaction kinetics is faster in agitated systems as opposed to non agitated systems as shown in the tables of the appendix. It was concluded from experiments that to have a good balance between acid consumed and good leach efficiency the pH of between 1.9 and 2.0 should be used with densities of below 27% solids. Tail tanks should be agitated to allow for faster kinetics and low lime consumption as experiments show that lime consumption is low when agitated tanks were used. Experiments also showed that when water is added to the lime consumption is greatly reduced hence the need to add water when adding to the tails.

CHAPTER 1:

1 INTRODUCTION

Nkana Leach Plant (NLP) is a conventional leach/ (solvent extraction) SX plant and treats oxide ores from Mindola Open Pit mine. The ore is predominantly malachite which is delivered at the site by tipper tracks continuously from mindola open pit.

The overall plant efficiencies are function of recoveries and wash efficiencies. Currently the plant leach efficiencies are between (80-91%) and acid consumptions is relatively high currently at (200 tonnes), the target leaching efficiency is about 95%. This efficiency target is not reached due to several factors yet acid consumption remains a challenge. Factors such as, degree of agitation, acid concentration, resident time, densities, particle size and particle size distribution needed to be revisited. It is the objective of this project to investigate some of these factors/parameters. Optimal usage of reagents in both leaching and tails nuetralisation is important to every plant especially that these reagents are expensive. Acid is currently costing between $250-$350 per tone at the current plant consumption of 200 tonnes per day this translates into $50000-$70000 per day clearly acid is a big cost to the plant operations and Mopani Copper Mines. Ways and means to optimize acid consumption had to be found by working around the parameters used during leaching. The most important parameters used during leaching at Nkana Leach Plant were, Agitation speed, Densities, pH, Particle size and its distribution. One of the other factors though it was not handled in this study but of very huge importance is gauge acid consumption this is simply how much acid is going into leaching other elements apart from copper. This is done by looking at the various elements contained in the ore and look at how much acid goes to leach these other elements like, Fe, Mn, Mg, S, SiO_2, etc.

Lime at the leach plant is used to neutralize the tails from counter current decantation tank # 5 (CCD5) underflow which is the last thickener in the counter current decantation (CCD) wash operation; this underflow is sent to the tailings tank where the tails are neutralized using lime. Currently NLP acquires the lime from Ndola Lime Company. The lime is received as hydrated lime; NLP also buys quick lime from South Africa. The dosage of lime at the tailing tank was of concern because there was a lot of high consumption of lime at the tailing tank. Because lime is acquired both locally and abroad it was prudent to make sure that the consumption is optimized as much as possible. The study aimed as much as possible on studying the consumption at the tailing tanks by looking at the various ways in which lime is added to the tanks. These ways had a direct bearing on how quickly the lime reacts with the tailings before reaching the pH at which the tailings can be disposed off. According to laboratory analysis of quality tests undertaken on the lime at, it was found that the best lime should be at least over

80% excellent. Addition of this lime to the tails is important dry addition or wet (addition meaning adding water before administering to the tailings), hence the aim of the project is to study the consumption of lime and look at ways to optimize it's consumption at Nkana oxide leach plant tailings tank.

CHAPTER 2:

2 BRIEF DESCRIPTION OF THE PLANT

2.1 OVERVIEW

Copper ore on the Copperbelt occurs in large deposits of extensive layers of sulphide ores with oxide, out crops of oxide ores as well. During extraction of these ores sulphide and oxide ores are processed separately. Usually sulphides are treated through the conventional pyrometallurgical process route while the oxides which are usually of leaner copper levels are treated through hydrometallurgical process routes which include leaching, solvent extraction and finally electrowining and electrorefining.

Lime occurs as lime stone in the Copperbelt Province of Zambia and is mined and sold by a company called Ndola Lime Company. This lime is mostly sold as hydrated lime. One of the objectives of this study is to review reaction kinetics of this lime in the context of the NLP operations. Nkana Leach Plant was commissioned in 2007 to recover copper from oxide ores mined from Mindola Open Pit. The ores are predominantly malachite (copper carbonates). Nkana Leach Plant treats an average of, 1000 tonnes of copper oxide ore per day. The ore is delivered as run off mine ore from Mindola open pit mine which is located a few kilometers from the leach plant mine site. Malachite is the chief consistituent of the ore which is delivered at the leach plant milling section and ore pretreatment starts from there, before subsequent leach and solvent extraction processes takes place.

Malachite sample was fed to the primary crusher (jaw crusher) of minimum gape of 105mm after which the ore was conveyed by CV 01 (conveyor belt one) to CV 02 which fed CV 03 which was subsequently fed to the secondary crusher, and from there, the feed went to the stock pile via the apron feeder which in turn fed CV 04 to the surge mill where the feed goes into the surge mill discharge box. When the feed is pumped to the surge mill cyclone, the over flow is then fed to the ball mill which feeds the ball mill discharge box and then feed is pumped to the

ball mill cyclone and the cyclone overflow is pumped to the pre-leach thickener (PLT) for pre leach thickening.

2.1.1 Grinding and Classification Function

The basic function of the Grinding and Classification circuit at NLP is to reduce the size of the crushed ore from the crushing circuit and to then thicken the resultant slurry prior to leaching of the oxide ore. The Grinding area of the process plant reduces the size of the feed rock from the crushed size of 75% passing 200mesh. Size reduction (comminution) in this circuit is achieved through the use of a Sag mill and Ball mill. As the study is concerned with the particle size and its distribution the milling section which is the size reduction area is important to look at and mention in our study of the problem at hand. The milling section is what feeds to the leach circuit for the start of the leaching process. The feed moves as follows:

The feed to the pre leach thickener is pumped at densities of 15-20% solids into the mixer vessel and then pumped to the thickener, where a flocculants A110 is added to build up solids to 50-55% solids before pumping to the leach tanks. The buildup is achieved by flocculation which is the process that cause particles suspended in water to aggregate into clumps or masses that then sink or can be removed by filtering, or aggregate in this way (Encarta, premium 2008), flocculation causes the suspended solids to coalesce. The thickener diameter and height are 20m and 4m respectively.

There are about four leach tanks in the leach circuit. From pre leach thickener the feed is pumped to the leach tanks (figure 2). In (leach tank 1) LT 1 the solids are diluted from 50-55% solids to 30% solids and leaching starts in LT 1. The feed moves to LT 2 due to the differences in solids concentration. This is enabled by a mass transfer process called diffusion which utilizes the differences in the tank concentrations, LT 2 solids concentration is at 28%. From LT 2 the feed is moved to LT 3 which is at 26% solids where leaching is continued via LT 4 at 24% solids and tank number 4 is the last leach tank in the series. In all the leach tanks the resident time was 1hr 30min giving a total of 6hrs leaching time this is illustrated in figure 2.

From leach tank four (LT4) the leach solution is pumped to counter current decantation operation one (CCD 1) mixer vessel where an acidic flocculant N100 is added to help settle the solids. The CCD 1 underflow is fed to CCD 2 while the overflow is sent to the high grade pregnant leach solution (HG PLS) storage tank, from where it is pumped to the pin bed clarifier before finally being pumped to the HG PLS pond (figure 2). From the pond the solution is sent via the high grade route to the SX plant for extraction and striping making the advance solution which is sent to the tank house for electrowining and electrorefining. The copper grade in the high grade PLS is between 4-10gpl copper. On the other hand CCD 2 is the (low grade pregnant

leach solution) LG PLS thickener. In this thickener wash operation continues as the underflow goes to thickener number three (CCD 3); whose overflow goes back to CCD 2; as LG PLS and the under flow proceeded to CCD 4 whose overflow goes back to CCD 3 then back to CCD 2, and whose under flow goes to the last thickener in the circuit CCD 5 whose overflow goes back to CCD 4 then back to CCD 3 and finally to CCD 2 to complete the LG PLS circuit (figure 2). The under flow from CCD 5 is sent to the tailings neutralisation tanks before being send to the disposal dams, kilometers away from the plant. The disposal of tailings is a major environmental problem becoming more serious with the increasing exploration for metals and the working of low-grade deposits.

Lime is traditionally associated with the construction industry and its application in building mortars. However, the scope of usage of lime products is considerably greater. Apart from the construction industry and the building materials industry, lime products are irreplaceable in – among others – the metallurgical industry, various sectors of the chemical industry, the glass industry, agriculture, and in the manufacture of cosmetics. Recently, it is being used more and more frequently in the protection of the natural environment. Lime is a product commonly occurring in the natural environment, it is cheap in comparison with other chemical compounds, and it facilitates the subsequent management of end products in an environmentally friendly manner. Lime products – both burnt and unburnt – are widely used in air, water and soil protection, i.e.:

For the purification of flue gases of both sulphur dioxide and other contaminants generated during the combustion of solid fuels in various industrial processes,

• For treating drinking and industrial water in a manner that prevents the corrosion of waterpipe networks and lowering water hardness,

• For treating municipal and industrial sewage in a manner enabling the nuetralisation of wastes prior to their discharge to sewage systems.

• For neutralising sludge and wastes in a manner enables their subsequent useful management.

The usage of burnt or hydrated lime for neutralising sludge not only improves its sanitary condition, but is also conducive to sludge stabilization and improves its dehydration properties. In consequence, one may obtain an environmentally friendly product that can be put to practical use. (Dr Eng. Bozena Sroda 2000). Environmental Council of Zambia has been mandated under the Environment Act to monitor the mine effluents and to make sure that these effluents are environmental friendly. In the leaching circuit the waste tailings which are a

product of leach operation are acidic in nature. To protect the environment it is required that these tailings are neutralised before disposal. On the other hand Mopani as a company has a social responsibility to protect the environment in which they operate; this includes disposing off neutral effluents. Therefore by law and obligation the Company has to neutralize the tailings before disposal. The Company in this regard wants to make sure that there is minimal lime consumption as this is a cost not only to NLP but also Mopani. This project seeks to find the best way to maximize lime usage at the tailings tanks.

The over flow of CCD 5 is sent back to CCD 2 via CCD 4, CCD 3, respectively where it ends up in the LG PLS storage tank before being pumped to the pin bed clarifier and finally pumped to the LG PLS pond (figure 2).

The resultant solution is pumped to the SX low grade circuit for extraction and striping to make the advance solution. In the SX plant the line from the PLS pond feeds the mixing tanks. The liquor is concentrated in three parallel streams known as mixer settlers E_1, E_2, and E_3 respectively. After the extraction stages follow the stripping stages which consist of two settlers S_1 and S_2 respectively. Figure 1 explains further the process as described.

Cu L/SX/EW Circuit
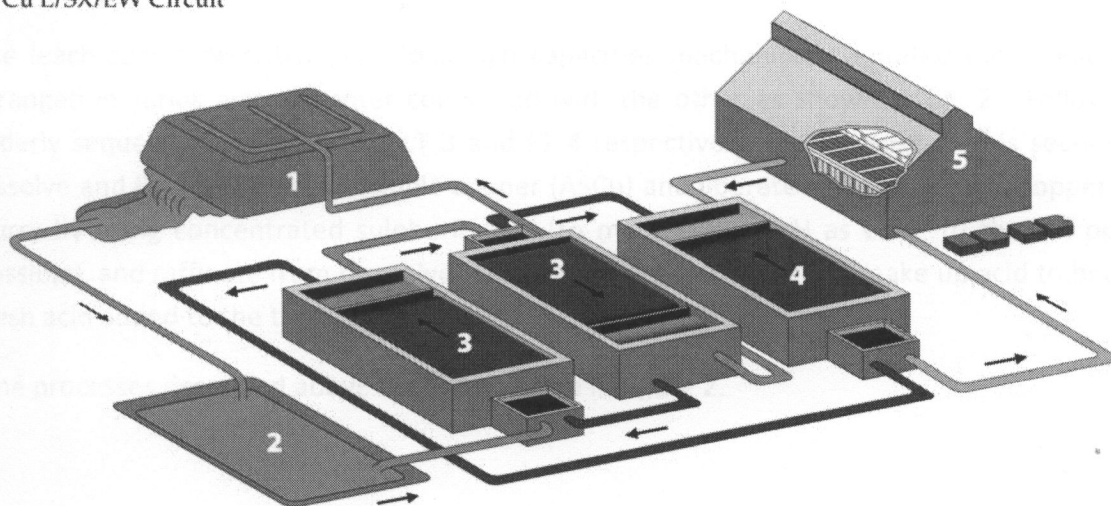
Fig. 1 Simplified leach, solvent extraction, electrowining circuit. The numbers 2, 3, 4 and 5 represent (pregnant leach solution) pls pond, mixer settlers and tank house respectively.

At the extraction stage, the pregnant leach solution is contacted with the organic. The extractant is selective at the set point of copper, this means that extractant used must be as

selective as possible to extract only copper. This is one of the requirements of a good extractant it should be as selective to the metal of interest as possible. The reaction is as follows.

$$2RHorg + Cu2^+ \longrightarrow R_2Cu_{org} + 2H^+_{(aq)}$$

(Organic extractant)(Loaded organic)(Acid generated)
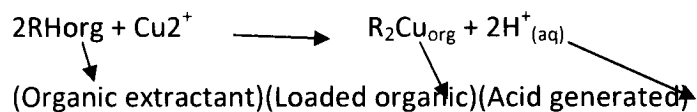
At the stripping stages, the high acidity of the spent electrolyte which is contacted with the loaded organic leads to the reaction being reversed;

$$R_2Cu_{org} + 2H^+_{(aq)} \longrightarrow 2RH_{org} + Cu^{2+}_{(aq)}$$
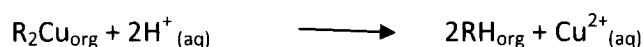
In the equation above, R_2Cu_{org} is the loaded organic; $2H^+_{(aq)}$ is the spent electrolyte and $Cu^{2+}_{(aq)}$ is the advance. The advance which is the product of the reaction above is filtered and pumped to the tank house at the cobalt plant for electrowining.

2.2 LEACH CIRCIUT

The leach circuit described, has four high capacities mechanically agitated tanks (Pachuca's) arranged in series and one inter connected with the other as shown in fig. 2. Following an orderly sequence i.e. LT 1, LT 2, LT 3 and LT 4 respectively, the purpose of this section is to dissolve and liberate the acid soluble copper (ASCu) and liberate the acid soluble copper in the slurry by using concentrated sulphuric acid (to maintain the pH as close to the set point as possible), and raffinate from the solvent extraction process is used as make up acid to boast the fresh acid added to the tanks.

The processes described above are summarised in figure 2.
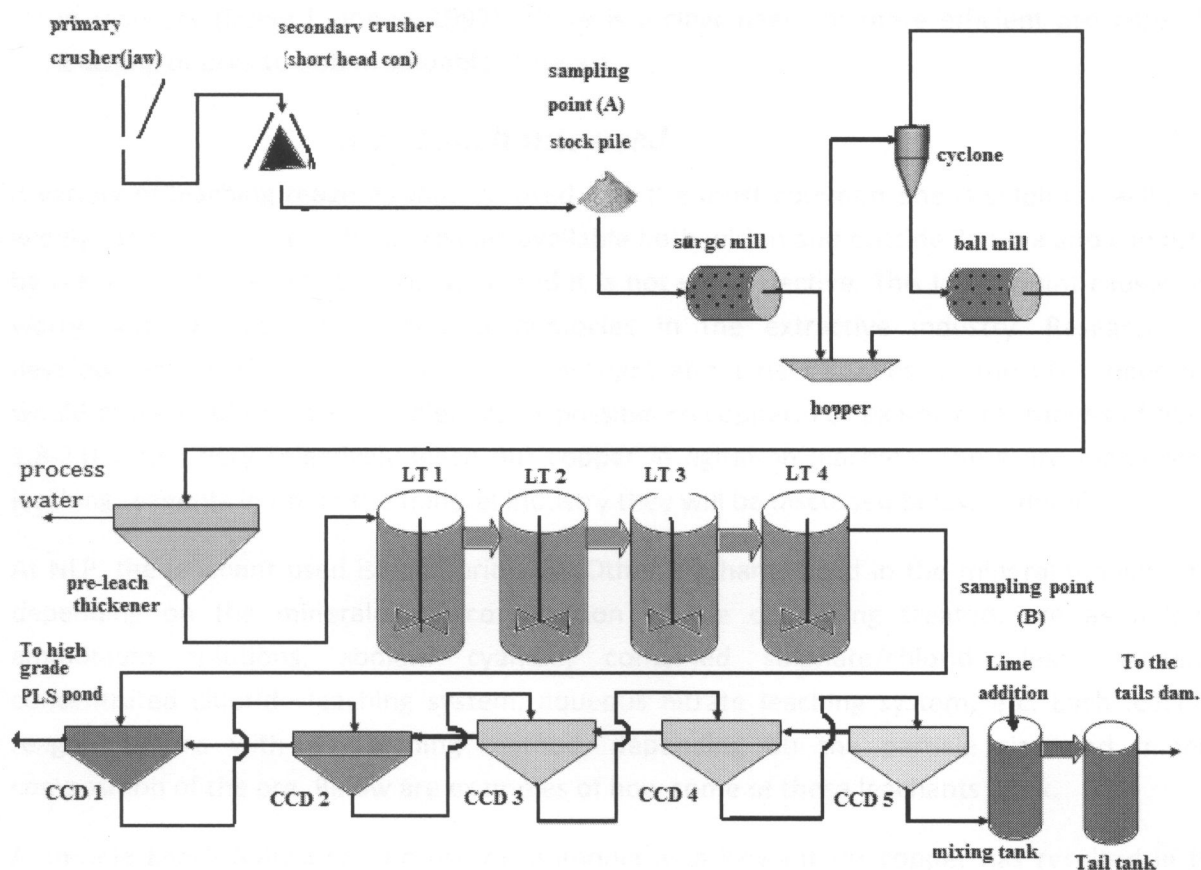
Fig. 2 Flow sheet of Nkana Leach Plant operations.

3 LITERATURE REVIEW

3.1 Definition

Leaching is the selective dissolution of one or more components from a metal bearing solid material due to the action of a leachant. A leachant on the other hand is the liquid reagent (often aqueous), which contains the active cation or anion causing a specified solid phase to dissolve in a hydrometallurgical process (Dr R L NYIRENDA's 2007). 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 milling and other pre leaching treatments such as roasting or smelting, and the easy of dissolution (Erick Jackson, 1986). The preferred method is likely to be the one with the lowest capital and operating costs, to commensurate with both speed of processing and high percentage

of recovery (David L. Jones, 1997). There is a clear need for more efficient processes for leaching of ores to obtain valuable minerals.

3.2 Leachants used

A variety of leaching reagents may be used, but the most common one is sulphuric acid. It is widely used because it is cheap, readily available both within and outside Zambia and can often be regenerated. Though it is corrosive and it is not very selective. This latter point causes less worry with the advent of new technologies in the extractive industry. Research and development on the subject matter has brought about new studies on the pH ranges that would make sulphuric acid as selective as possible to copper. For example pH ranges of about 1.8-2.0 would very selectively leach out copper in agitation leaching. There are many other leaching reagents in use in the mineral industry they will be discussed below in detail.

At NLP, the leachant used is sulphuric acid. Other leachants used in the mineral industry are, depending on the mineralogical composition of the ore being treated, are as follows: ammonium solutions, sodium cyanide, combined sulphate/chloride leach systems, concentrated chloride leaching system, aqueous nitrate leaching system, etc. Each leaching reagent comes with its leaching method, depending on the particle size and mineral composition of the ore. Below are examples of how some of these leachants work.

**Ammonia Leach Solutions**: the use of ammonia as a lixiviant for copper has resulted in the development of several copper recovery processes that incorporate solvent extraction; the imperial smelting process for leaching lead dross , the arbiter processes for leaching of copper concentrates, copper recovery from scrap metal and/or precipitate copper. Even though a number of processes have been developed, the tonnage of copper recovered using ammonia leaching followed by SX is quite modest.

Copper extraction from ammonia solution is faster and easier than it is from sulphuric acid leach solutions. A simplified equation describing the chemistry of extraction is shown below:

$$M(NH_3)_n^{2+}{}_{(aq.)} + 2RH_{(org.)} \rightleftharpoons R_2M_{(org.)} + 2RH_4^+{}_{(aq.)} + (n-2)NH_{3(aq.)}$$
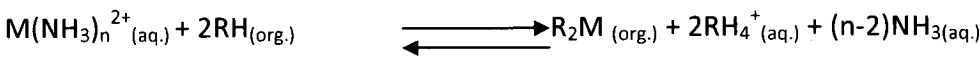
Where n is usually 4.

(Copper Solvent Extraction: The State of the Art, Gary A. Kordosky 1992)

Concentrated Chloride Leach Solutions: Chloride leaching, known for about 100 years, is a highly efficient leaching system for copper concentrates, perhaps too much so. One of the major problems associated with leaching technology is the production of high purity copper. In spite of the considerable efforts put forth by Duval Corporation, Cyprus Metallurgical

Corporation, and others to commercialise this technology, there are no large-scale commercial copper plants which use concentrated chloride leaching technology. In order to produce high-purity copper, several groups have incorporated SX into the chloride leaching system using conventional extractants, and a combination of both. Of particular interest is the extractant chemistry of the novel extractant Acorga CLX 50, not because it has resulted in a commercial application, but because it represents new chemistry in copper SX. Using Acorga LX 50, extraction is favoured at high chloride ion concentrations while stripping favored at low chloride ion concentrations. In addition, extraction is independent of pH. The equation below describes the extraction/stripping of this reagent.

$$Cu^{2+}{}_{(aq.)} + 2Cl^-{}_{(aq.)} + 2L_{(org.)} \rightleftharpoons L_2CuCl_{2(org.)}$$

Even though this new copper extraction chemistry has not yet been commercialised on large scale, it does provide an elegant example of coordination chemistry applied to hydrometallurgy, and for this reason alone it represents a true advance in the science and technology of copper SX. (Gary A. Kordosky, 1992)

Combined Sulphate Chloride Leaching System: Some of the plants where the technology is in use include; Associated Smelters Pty. ltd. (BHAS) plant for the treatment of copper matte arising from a lead blast furnace blast furnace, Lince Project in Chile and a plant at Cyprus Sieritta, a dump leach of low-grade chalcopyrite. In the first two cases, the chloride ion concentration of the leaching liquor will equilibrate at about one molar or slightly less; at Cyprus Sieritta, it has equilibrated at approximately 8 gpl. All three plant operations recover copper by SX-EW. For solutio9ns of this type, the oxime-extractants are suitable reagents for copper recovery. The SX circuit differs from those used to recover copper from conventional sulphates leach liquors in that at least one efficient wash stage is required on the loaded organic stream to minimize transfer of chloride ions to the strip electrolyte cannot be overemphasized. Copper leach solutions of this type could become more common if the BHAS is applied to complex sulphide ore concentrates or if more projects are developed in parts of the world lacking fresh water (Gary A. Kordosky, 1992).

Aqueous Nitrate Leaching systems: There is reported example of copper recovery from a silver nitrate plating bath using ketoxime-based copper extraction reagent. Even though this is a small application, it is interesting because the practitioners found that the extractant was prone to reactions with the acid nitrates solution, which converted the ketoxime to an extractant that could no longer be stripped of its copper. The reaction with nitric acid, called nitration, is well known in organic chemistry. The nitration rate increases with an increase in the nitrate ion concentration and a decrease in pH. This has implications in areas where the ore and/or the available water contain nitrate ions, most notably the Atacama Desert of Chile. It may also be a

concern where concentrates are leached with sulphuric acid containing some nitric acid. If the copper leach solution contains nitrate ions, two things are important.

1) During copper extraction, the pH will be maintained above some value that will depend on the nitrate ion concentration of the leach liquor.

2) A wash stage on the loaded organic will be required in most cases so that nitrate ions are not transferred to the pregnant strip solution.

Companies considering copper SX with an acid containing nitrate ions are should be prepared to pilot the process over some reasonable length of time. Fortunately, the nitration reaction is easy to monitor, so a pilot program has an excellent chance of successfully defining the parameters under which the SX circuit must operate to control nitration of the extractant. (Gary A. Kordosky 1992).

Cyanidation using (Cyanide): Cyanide is the most commonly used leachant for gold. Two molecules of cyanide complex with every molecule of gold. Copper also complexes with cyanide, but it take 4 molecules of cyanide for every copper molecule. Copper is often present in copper gold ores in the one tenth to one percent range. Gold in these ores is in the one to 10 parts per million ranges. The copper consumes so much cyanide it needs to be recovered by hydrogen cyanide distillation, an expensive and dangerous operation. Systems have been proposed where sulfuric acid is used to leach the copper first. Then, the heap or batch is neutralized. Cyanide can then be used to leach the gold. The problem, of course, is the expense of neutralization. In heap operations, the additional worry of incomplete neutralization is present. (David L. Jones et al, 1999).

Cyanidation leaching of gold; one of the earliest devised leaching processes in which oxygen participates via the atmosphere involves the dissolution of gold by alkaline cyanide solutions, the overall reaction which may be represented as?

$$4Au + 8CN^- + O_2 + 2H_2O = 4Au (CN)_2^- + 4OH^-$$
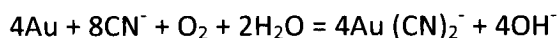
Similar reaction occurs with silver.

These may be regarded as accelerated corrosion processes with the gold dissolving anodically

$$4Au + 8CN^- = 4Au (CN)_2^- + 4e$$
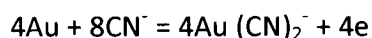
Supported by the oxygen reduction as the cathodic reaction

$$O_2 + 2H_2O + 4e = 4OH^- \text{ (Erick Jackson, 1986)}$$

Sulphuric Acid Leach Solution: By far the widely used leachining reagent is sulphuric acid. The active leaching agent for oxide ores and, to some extent, certain sulphide ores is sulphuric acid. An oxidation process is needed to leach sulphuric ores completely. The active leaching agents are ferric ions and certain naturally occurring bacteria, most commonly Thiobacillus ferrooxidans and Thiobacillus thiooxidans, which though a series of complicated mechanism use oxygen from the air as the consumable oxidant. The table below summarizes the range of solutions produced by the various leaching methods as they are used in industry today. The contents of these solutions set the metallurgical parameter under which the copper extractant must be able to efficiently and selectively extract copper. Of note are the wide ranges of copper concentrations and pH that occur in these solutions.

<u>Table of typical Sulphuric Acid Leach Solution.</u>

<u>Type</u>	<u>copper (gpl)*</u>	<u>pH</u>
Dump	< 1-3	1.2-2.2
In Situ	<1-3	1.8-2.1
Heap	3-6	1.5-2.2
Thin layer	3-10	1.7-2.0
Vat	3-50	1.6-2.0
Agitation	1-6	1.8-2.0

*other metals that may be found include Fe, Mn, Al, Mg, and Mo.

In most cases, final copper recovery is by electrowining (EW). The extraction/striping reaction of copper in a sulphate solution with chelating copper extractants can be described by the simplified equation shown below:

$$[2RH]_{(org.)} + [CuSO_4]_{(aq.)} \rightleftharpoons [R_2Cu]_{(org.)} + [2H^+SO_4]_{(aq.)}$$
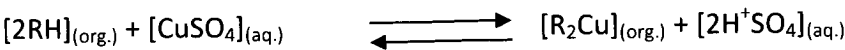
Where RH represents the extractant, the extractants works on a hydrogen cycle, generating acid as it extracts copper and consuming acid as copper is stripped. The copper SX is obviously most efficient if the extraction lies far to the right when copper barren organic is mixed with the copper leach solution and far to the left when the copper-loaded extractant is mixed with the tank house electrolyte solution. (Gary A. Kordosky 1992).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 milling and other pre

leaching treatments such as roasting or smelting, and the easy of dissolution (Erick Jackson 1986).

3.2.1 Factors Influencing Choice of Leachant

-Technical (thermodynamics and kinetics)

-Economical aspect

-Environmental aspects

-Safety and Health aspects

i) Selectivity is one of the factors influencing choice of leachant.

ii) Adequacy of leaching

iii) Must allow solid-liquid separation

iv) Must allow co-leached impurities to be removed at a later stage.

v) Leachant should preferably be regenerated

vi) Leachant should preferably leach at low stoichiometric requirement.

vii) Must be Affordable

viii) Leachant must not be too corrosive.

ix) Must not be toxic, if it is there must be a deliberate move to eliminate risk to personnel.

x) Must not be very volatile, If it is there must be use of a sealed environment to contain the volatility.

(Dr. L. N. Nyirenda's 2007)

On the other hand there are two types of lime commonly in used in nuetralisation reactions depending on the chemistry of the lime i.e. Quick lime and hydrated lime. Both types can be acquired both within and outside Zambia. The two types mentioned above are commonly used forms of lime at the tailings tank and are used as either dissolved in water to prevent dust loses or just as powder form in which it normally comes. LIME MEETS THE ENVIRONMENTAL CHALLENGE: In today's industrialized society, many processes create acidic waste streams. Waste tails discharge permits generally require that acidic waste be neutralized to the range of

pH 6.0 - 9.0. In the future, many facilities may be forced by changing permit requirements to treat and reuse all wastewaters. To achieve these goals, lime neutralization is the best solution.

LIME IS THE NEUTRALIZING MATERIAL OF CHOICE: Neutralization of acidic waste streams involves raising the pH, usually by adding an alkaline chemical, such as lime. There are several other chemicals that can also be used, but due to cost, handling problems, high total dissolved solids in the effluent, or mobility of heavy metals in the sludge, leach waste tails, lime is the neutralizing material of choice. As highlighted in the table below, when considering the cost of neutralizing agents and alkali requirements, lime is the most cost-effective option to neutralize acids.

Comparison of the two lime types used in this experiment is shown in the table1. TABLE I

Product	Lime (Calcium Oxide or Hydroxide)
Form of Material	Solid - CaO Powder-Ca(OH)$_2$ Slurry 35%-Ca(OH)$_2$
Alkali Requirement Per Ton H$_2$SO$_4$ Per Ton HCl	As CaO: 1.240 lbs. 1.670 lbs.
Cost Per Ton of Neutralizing Agent (on a dry basis)	CaO - $60 Ca(OH)$_2$ - $80 Slurry Ca(OH)$_2$ - $100 Cost Stable
Cost to Neutralize 1 Ton of H$_2$SO$_4$	CaO - $37 Ca(OH)$_2$ - $66 Slurry Ca(OH)$_2$ - $82
Maximum pH @ 25° C	12.45
Sludge Profile	Heavy, Low Volume, but easy handling, even if heavy metals present
Salts	Insoluble calcium metal hydroxyl salts
TDS (total dissolved solids)	Low
Reaction Time	Moderately fast-acting to complete neutralization

Unit's conversion: 1240 lbs = 562.464kg, 1670 lbs = 757.512kg

Factors to consider when choosing a neutralizing material include the cost of the material, the amount of tails produced, the handling characteristics of the tails, the leaching characteristics of the leaching process, and the resulting quality of the effluent.[*] Lime is most economical, but also offers other advantages. Lime neutralizes more quickly and completely, which can reduce the size of equipment. (Dr Eng. Bozena Sroda, 2000)

The costs given in the table were taken from a report compiled in September 2000 it is just used as an illustration to show how the two lime products compare against each other. The data was got from web site called (http://www.lime.org).

Leaching also may be characterized by the type of solution being used to leach the ore and recover the target metal. Acid Leaching, Certain target metals are particularly receptive to leaching by acidic solutions. Copper, for example, is leached by a sulphuric acid solution. Cyanide Leaching, Sodium cyanide has been used extensively to recover gold from low-grade ores as discussed above. Continued improvements in cyanidation technology have allowed increasingly lower grade gold ores to be mined economicall. Dissolution, Water is used to separate certain water-soluble compounds, such as sodium, boron, potassium, and certain salts (some that may be formed by roasting). The compounds are dissolved, purified using basic water chemistry and filtration, then recrystallised. Recovery Processes, The values contained in the pregnant leach solution are recovered via other processes. (David L. Jones, 1999)

3.3 Thermodynamic Description

Leaching is the process of extracting a soluble metallic compound from an ore by selectively dissolving it in a suitable solvent, such as water, sulphuric acid, or sodium cyanide solution. The target metal is then removed from the "pregnant" leach solution by one of several electrochemical or chemical means. (Note that digestion, where the ore concentrate is digested completely or significantly by strong liquor, is not considered leaching (Erick Jackson, 1986).

In a discussion of solution-solid equilibrium, it is particularly appropriate to use Eh-pH (pourbaix) diagrams. These diagrams primarily describe the solution composition or solid phase stability as a function of two variables; single electrode (reduction) potential and pH. This thermodynamic data may be used to predict the general conditions likely to be favourable to the dissolution of a mineral. Appropriate thermodynamic data are succinctly presented in the form of potential/pH diagrams and these can be found in various literatures, textbooks of hydrometallurgy (Erick Jackson, 1986)

3.3.1 Eh-pH Diagrams

Thermodynamics aim is to present an intercomparison of thermodynamic databases by means of Eh-pH diagrams, which are practical and useful for understanding geochemical behavior of elements. An Eh-pH diagram depicts the dominant aqueous species and stable solid phases on a plane defined by the Eh and pH axes. Eh-pH diagrams are thus essential to understanding solute and radionuclide transport in groundwater. The most well-known studies on comprehensive Eh-pH diagrams are those of Pourbaix (1966) and Brookins (1988). The former discussed corrosion, passivation and immunity of materials, while the latter reported geochemical aspects of the geological disposal of radioactive wastes. These studies have been followed by many thermodynamic databases (Naoto Takeno, 2005). Illustrations of Eh-pH diagrams are shown in figure 3 these are oxide copper ore dissolution.
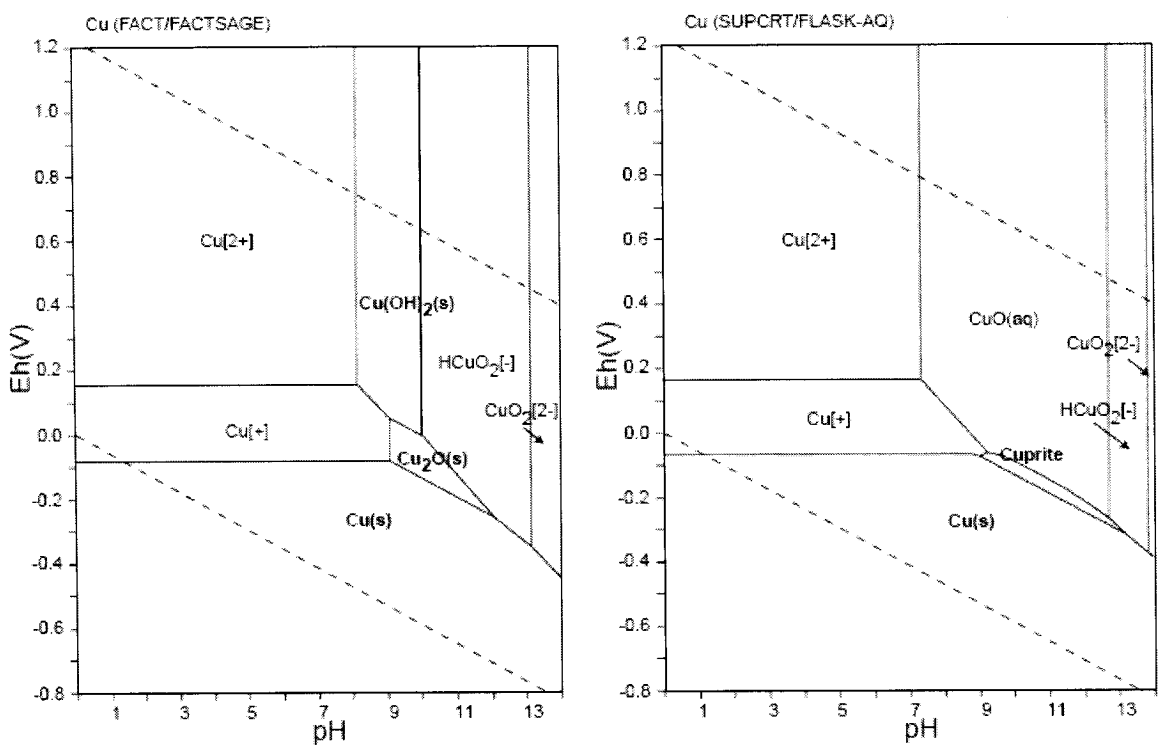
Figure 3. Eh-pH diagrams

Thermodynamic data maybe used to predict the general condition likely to be favourable to the dissolution of a mineral. Appropriate thermodynamic data are succinctly presented in the form of Eh-pH diagrams as shown in figure 3. The metal-water systems yield the simplest diagrams and are constructed from data at 25°C. In determining the boundary corresponding to a particular equilibrium, a specific activity or concentration must be assumed for the soluble

metal ionic species. Where diagrams are to be utilized for corrosion applications, this specific activity is usually a molal activity of 10^{-6} but for hydrometallurgical extraction like in our case it is 10^{-2} or 10^{-3} is more appropriate because the requirement is for a significant metal ion concentration in the aqueous phase. Establishment of the boundaries in an Eh-pH diagram results in the formation of domains, in which a particular species is thermodynamically stable.

3.4 Chemistry of dissolution

The chemistry involved in the dissolution of copper minerals is essentially simple decomposition in the case of oxide minerals. The leaching of oxide copper in agitated systems has also been found to be diffusion controlled. However, there are significant differences in the control mechanisms. Initially the leaching control is the supply of acid to the mineral surface. This is limited by the diffusion through the surface boundary layer and is thus subject to the level of agitation. Higher agitation provides thinner boundary layers and faster diffusion rates. Similarly, higher grade material creates a thicker concentration boundary layer and relatively slower leaching (Erick Jackson, 1986). Acid consumption in agitated leach systems is also a first-order system. However, the interpretation of the RTD is easier from batch testing to continuous multiple CSTR series plant operations. Altering the method of acid delivery can significantly reduce acid consumption rates. Predilution of the acid minimises the local acid concentration near the addition point and can reduce the consumption by up to 50 per cent.

The type of heterogeneous reaction that will be considered here is one in which the solid reacts with the reagent in solution and dissolves. The overall reaction process is broken down into the following.

i)   Transport of reactants to the solid-liquid interface.

ii)   Adsorption of reactants on the surface.

iii)   Reaction at the surface.

iv)   Desorption of soluble products of the reaction.

v)   Transport of soluble products away from the solid-liquid interface.
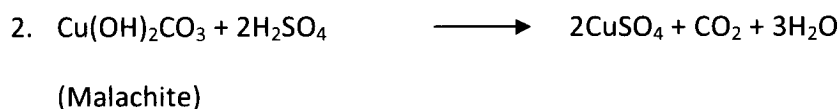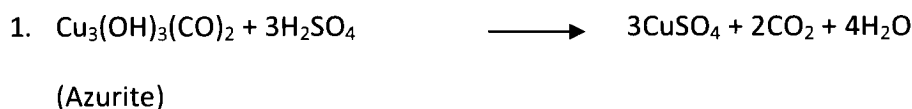
Stages (I) and (v) are controlled by the rate of diffusion of the solute species and in an agitated system. Stages (ii), (iii), and (IV) are considered to be chemically controlled process. According to Nernst, the chemical processes occurring at the interface are much faster than at least one of the transport processes so that the reaction rate is transport controlled.

The oxide mineral decomposition reactions products are much water soluble or soluble in excess solvents i.e. base metal oxides are soluble in mineral acids. While base metal sulphides are generally insoluble in the acids and require strong oxidizing agents for their decomposition (Erick Jackson, 1986).

Various oxides, silicates, carbonates and sulphides in the gangue may undergo similar reactions to form soluble salts. The amount of these dissolved impurities will depend on the nature of the gangue and may result in excessive consumption of solvent as well as contaminating the resultant copper-bearing solution. The dissolution of oxide copper minerals can be represented by the following chemical reactions.

Azurite and malachite in which the latter is the subject ore dissolve quite readily in dilute sulphuric acid according to the reactions:

1. $Cu_3(OH)_3(CO)_2 + 3H_2SO_4 \longrightarrow 3CuSO_4 + 2CO_2 + 4H_2O$

   (Azurite)

2. $Cu(OH)_2CO_3 + 2H_2SO_4 \longrightarrow 2CuSO_4 + CO_2 + 3H_2O$

   (Malachite)

The above two reactions are accompanied by an evolution of carbon dioxide. Which is one of the products as per the two reactions above?

Other oxides react as follows;

3. $CuO + H_2SO_4 \longrightarrow CuSO_4 + H_2O$

   (Tenorite)

4. $Cu_2O + H_2SO_4 \longrightarrow Cu + CuSO_4 + H_2O$
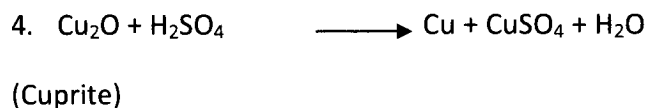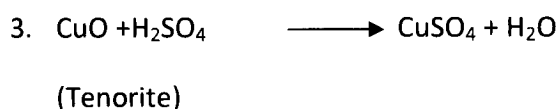
   (Cuprite)

About half of the copper in the dissolution of cuprite goes into the residue in the form of elemental copper. To recover this elemental copper from the residue, an oxidizing compound such as ferric sulphate may be used (F.D.Mendes; A.H.Martins, 2003).

$Cu + Fe_2(SO)_4 \longrightarrow CuSO_4 + 2FeSO_4$

Industrially, the oxidation process can be affected by aeration as the copper is in a finely divided state.

Chrysocolla ($CuSiO_3.2H_2O$) is readily soluble in dilute acids. It must be noted here, however, that term Chrysocolla is used to include many copper silicates of varying complexity, some of which do not dissolve so readily in acid. Dioptase ($CuSiO_3.2H_2O$) for example, is acid soluble but dissolves more slowly than Chrysocolla. Due to the difficulty of obtaining pure samples of Chrysocolla for study, the chemistry of dissolution of many copper silicates have not been fully established.

The general reaction however is,

5. $5CuSiO_3.7H_2O + 2H_2SO_4 \longrightarrow 5CuSO_4 + 5SiO_2 + 9H_2$

The silicate produced precipitates as a silicate gel rather than a granular product. The copper minerals so far discussed constitute the many oxide minerals found in Zambian copper oxide deposits.

Brochantite ($Cu_4(OH)_6SO_4$ and Chalcantite ($CuSO_4.5H_2O$) which are assumed to be present in very small proportions dissolve according to the reactions;

6. $Cu_4(OH)_6SO_4 + 3H_2SO_4 \longrightarrow 4CuSO_4 + 6H_2O$
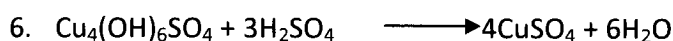
    (Brochantite)

Chalcantite is a water-soluble mineral. It can be formed by natural oxidation through the combined action of air, water and iron salts on exposed copper sulphide minerals (Erick Jackson, 1986).

3.5 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, to commensurate with both speed of processing and high percentage recovery. The methods used in leaching may for convenience of discussion, be divided into four principal groups. These are;

1) In-situ or in place leaching

2) Heap leaching

3) Percolation or vat leaching (sand leaching)

4) Agitation leaching-slimes leaching.

Leaching Operations; leaching operations may be categorized both by the type of leachant used as well as the physical design of the operations.

Physical Design. Several types of leaching operations are used, typically dependant on the ore-grade, the leachant, and the target material.

Dump Leaching. Piles of low-grade ore are often placed directly on the ground, leachant added by a spray or drip system, and leachant containing the solubilized target metal collected from underneath the dump over a period of months or years. The dumps are dedicated, that is they are designed to leave the ore in place after leaching operations are complete. Dump leach operations designed to recover gold more often are being designed with a plastic liner prior to placing the ore in order to facilitate recovery of pregnant solution as well as to minimize release to the environment of the cyanide leachant.

Heap Leaching. In heap leach operations the ore is placed on lined pads in engineered lifts or piles. The pad may be constructed such that heavy machinery may be used to off load the leached ore for disposal prior to placing new ore on the pad but more commonly the heap remains in place when leaching ends. As with dump leaching the leachant may be applied by spray or portable drip units; recovery is from beneath the ore on the impermeable pad (typically designed with a slight grade and a collection system).

Tank Leaching. In vat or tank leaching the milled ore is placed in a container equipped for agitation, heating, aeration, pressurization, and/or other means of facilitating the leaching of the target mineral.

In all three cases a solution management system is required, either in surface impoundments or tanks. Some operations use ponds that were designed with a compacted earthen liner (e. g., clay), but most copper and all gold operations use synthetic liners with leachant collection systems.

It is evident that the grade of the material and the association of the mineral and the association of the mineral with the gangue will be limiting factors determining the choice of the leaching method. Ore too low in grade to justify mining expenses might very well be shattered in place and leached over long periods of time with a cheap lixiviant. The other extreme situation is one in which ore of such grade and nature as to justify fine grinding and leaching by agitation. The leaching time involved and thus the inventory of the metal values during the leaching process will be decreased. With reference to this project, there is need to discuss the subject of agitation leaching even further;

3.6 Agitation Leaching

Agitation leaching or slimes leaching as it is often called, is limited to those ores or to that portion of the ore, that, because of particle size do not permit free passage of leach solution through the interstices between the ore particles and thus cannot be effectively treated by percolation methods i.e. agitation leaching is carried out on finely ground ore, on granulated alloy scrap, and on scrap alloy turnings and borings. It provides mostly high rates of dissolution because of the large surface area involved. In agitation leaching, dissolution is effected by keeping the finely ground ore particles in suspension in the solvent. Agitation may be imparted either mechanically or pneumatically (air agitation), but in either case, must be turbulent enough to prevent settling in the leaching tanks. Instead of effecting dissolution in a single tank, the solution mixture passes from tank to tank in series until optimal extraction is obtained. Pachuca tanks with airlifts are convenient for this type of agitation. The separation of residue gangue and leach solution presents a somewhat different problem than is encountered in percolation leaching, where simple draining suffices. Because of the nature and tonnage of the residual gangue, filtration is out of question. Adequate washing is a very necessary consideration in agitation leaching processes since a high percentage of occluded particles in the solution will be associated with the residual gangue. (Erick Jackson, 1986)

Thickeners carry out the separation of occluded solution from the residual gangue. Thickeners by definition take a dilute pulp and resolve it into two products-a clear solution and a thickened pulp. When used in series, thickened pulp from one thickener may be agitated with overflow wash solution from a precede thickener. This procedure facilitates the removal of occluded solution and wash solution before passing the pulp to the thickener in series (B.A.Wills 1986).

3.7 Optimal leaching

Obtaining optimal leaching results require a consideration of the following aspects; it requires the knowledge of: (I) ore or material characteristics and the pre-treatment of the material (II)

proper selection of leachant to use (III) proper choice of leaching conditions (IV) selection of leaching method.

Ore or material characteristics and pre-treatment of the material.

-Comminution and classification, chemical composition and mineralogical analysis.

Comminution and classification enables getting of particle size and particle size distribution. Particle size has a linkage with liberation, during communition cracks in particles (porosity) starts to appear. Charge bed permeability is also very important for proper leachant percolation. Though not very important to this project discussion is; Concentration which is upgrading the metal values in the ore so as to raise chemical composition.

CHAPTER 3:

4 TESTWORK

4.1 GERNERAL

This project was conducted on a laboratory scale. This involved the cutting and preparation of samples from the plant, and conducting analysis and experiments at analytical services department of Mopani Copper Mines. The solid samples after cutting were analysed for total copper %TCu and %ASCu. After leaching the liquors were analysed for grams per litre copper (gpl Cu) and other elements like Mn, Fe, Mg, Zn, all these elements were analysed at analytical services. And the CCD5 samples were neutralized at analytical services using lime to a required pH.

4.2 SAMPLE PREPARATION

Sampling is one of the most important components of any test work being conducted sampling is the means where by a small amount of material is taken from the main bulk in such a manner that it is representative of the larger amount (B.A Wills, 2003). Sampling methods depends on what is required to be looked into and gives an outlook of properties of the bulk material. Hence the main purpose of sampling is to draw a portion of a large entity from which measurement can be taken in order to infer the characteristics of the large entity called a population which is simply the aggregate from which a sample is taken and it contains all possible outcomes. It is important to distinguish between target and sample population. Target

population will describe the available material in say a concentrator, whilst sampled population is the portion drawn from the target population. In sampling, any portion taken for examination is referred to as a sample. Sample in general may be of any of the following type.

a) Representative sample –sample taken in such manner that measurement taken on it can be assumed to produce some answers as would be obtained if measurements were made on the aggregate.

b) Systemic sample – samples taken according to a fixed cyclic procedure in time and space.

c) Simple random sample – sample taken in such a way as to give all possible values of properties to have an equal chance of acceptance.

d) Stratified samples - samples taken in aggregates which are sub divided in smaller portions called strata which are then sampled independently but making sure that the strata do not overlap but homogeneous (L.K. Witika, PhD, 2003).

Sampling is dependent on probability, and the more frequently the incremental sample is taken the more accurate the final sample will be. The sampling method devised by Gy is often used to calculate the size of sample necessary to give the required degree accuracy. The method takes into account the particles size of the material, the content and degree of liberation of the minerals, and the particle shape (B.A Wills, 2003).

4.2.1 Sampling Gy's theory.

Assumptions

1) Normal distribution: it is assumed that if a large number of similar samples were taken from the bulk, then the frequency distribution of the assays would follow a normal curve.

2) Equal chance of acceptance: it is assumed that every separate particle and type of component has an equal chance of being selected.

3) Freedom from bias: it is assumed that the sampling technique and measurements are free from bias or systemic errors. In the case of gold, diamond and some sedimentary deposits the assay frequency distribution is likely to depart from normal and use should be of the central limit theory to take samples in such a way that the grouped mean grade will follow a normal distribution. Central limit theorem states that if random samples of size n are taken from any distribution the sampling distribution X will be

approximately normal with mean μ and standard deviation of $\sigma/\sqrt{n}$. approximately improve with n.

4.2.2 Sample preparations:

i)   Crushing

ii)   milling

iii)   screen analysis

The ore was subjected to the above preleach methods at laboratory level and; In accordance with the subject invention; the efficiency of the leaching process can be improved by grinding the ore prior to treatment. In a preferred embodiment, the ore is ground so that it can pass through a 200 mesh sieve. To facilitate rapid leaching, the feed can be ground into smaller particles. It is preferred that the feed be subjected to fine grinding. Preferably, the majority of particles in the feed are capable of passing through 75 micron sieve. Typically, at least 75% of the particles in the feed are of a size that can pass through 75 micron sieve openings (David L Jones, 1999).

Malachite reacts with sulphuric acid according to the equation

$$CU_2(OH)_2CO_3 + 2\ H_2SO_4 \rightarrow 2\ CUSO_4 +$$

$$+\ CO_2 + 3\ H_2O \tag{1}$$
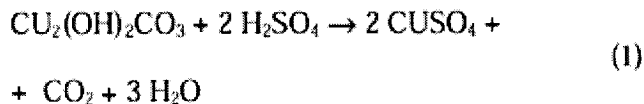
This work was aimed at evaluating the experimental conditions for maximum copper removal by sulphuric acid leaching of Nkana Leach Plant copper ore. Five parameters showing a big influence on the maximization of copper extraction were studied. These parameters were selected from the literature dedicated to the leaching of copper ores (Habashi, 1993; 1997; Gupta and Mukherjee, 1990) and preliminary tests performed on a laboratory scale. Due to the large number of tests for evaluation of the influence of so many variables on the experimental response the approach used was borrowed from previous leach laboratory tests conducted on the ore.

Parameters were:

i)   particle size and particle size distribution

This performance may be explained by the fact that when particle size is increased, it causes a decrease superficial area available, in limiting contact between the phases during the reaction. However, raising temperature in temperature operated systems pulps with coarser particles

overcomes this difficulty, since this causes an increase in the vibration of molecules inside the structure until they rupture and are easily attracted by the solvent particles, which results in an increase in copper extraction (F.D.Mendes; A.H.Martins, 2003). Particle size distribution is mostly concerned with charge bed permeability which is important for leachant percolation.

ii)   densities

In one embodiment, the leach solution in this step has solids content of up to about 55% by weight. Preferably, the leach solution will have solids content of from between24% to about 40%.

iii)   rate of agitation

The simultaneous increase in particle size and stirring speed values results in an increase in copper extraction. This means that in these small reactors, even if the particles are coarser, an increase in stirring speed makes suspension of the particles easier and causes a continuous renovation of sulfuric acid consumed on the particle surface, improving the external mass transfer characteristics (F.D.Mendes; A.H.Martins, 2003)

iv)   Optimum pH at which to leach

In agitation leaching the pH at which to leach is normally between 1.8—2.0 at this pH range, in agitated systems sulphuric is selective to the set point of copper (Gary A. Kordosky, 1992).

4.2.3 Leach procedure:

Cu oxide ore leaching is an acid-base reaction: mineral ores studied in this case was from the Mindola open pit most of which is the copper oxide ore malachite.

The sample was weighed and an amount in grams of sample is taken out of the bulk

Mineralogical analysis on the sample is done i.e. assay for the metal in the sample, TCu and ASCu.

After which a small amount was weighed to make a pulp required in the experiment.

A required milliliter of fixed volume of water is added. The prepared sample was put in a leaching vessel which contained baffles. The mixture was agitated with an agitator which was provided after which a mineralogical analysis for %TCu, %AsCu, %TCo was carried out on the solution as prepared, before any acid was added to the mixture to start the leaching process. A measured volume of concentrated Sulphuric acid was put in a measuring cylinder and poured in a beaker. And out of the concentrated acid, dilute acid was made of between 200gpl to 500gpl

acid by adding a volume of water to dilute the acid to the required dilution. Slowly the acid was added as the pulp was being agitated. The experiments were done over a period of 8hrs for the first set of experiments and the rest over a period of 6hrs. Every one hour a sample was taken for various required analysis i.e. assay for metal of interest in both the wash liquor and the pregnant leach liquor.

Leach liquor analysis gpl Cu, gpl Co.

Last two samples Analysis (liquors) gpl Cu, gpl; Co, Mg, Mn, Ca, Fe, Ni, Al, S, Zn, Pb, i.e. 7th and 8th hr. And 5th and 6th hr in the case were the experiment was done over a six hour period. Meaning for the last two hours of the experiments the above elements were analysed for. Initially they were to be analysed for even in the first hours of the experiments but the lab is overwhelmed with samples to analyse hence the analysis of just the last two hours only.

Total analysis of solids was done for: This is the filter cake left after filtration which was ground and taken to the lab for analysis

%TCu, %ASCu, %TCo, %ASCo, Ca, Mg, Mn, Zn, Ni, S, Al, Fe, Pb, GAC.

After the above analysis the pregnant solution contains more metal of interest than the wash liquor. The results are shown in the appendix tables 7A to 7F

4.2.4 Nuetralisation experiments using lime

Lime is the neutralizing agent of choice in almost all applications, and this project was not an exception. We therefore look at how the nuetralisation experiments were conducted.

The samples were cut from the Counter Current Decantation operation tank number five (CCD5).

Three (3) volume samples from CCD 5 were cut at different times.

Leach tails Analysis;

The tails were poured into a beaker

-and a measured weight of lime was added to the tails slowly while stirring with an agitator provided in a baffled tank to speed up the reaction.

-A pH probe was used to continue reading the pH and lime being consumed to attain the required pH.

Frequency of experiment per day:

At least six neutralization experiments were conducted per day.

At least 4 times per week; the sample results and tests conducted are shown below. Only one experiment is shown below.

Experiment 1

1. Initial volume of tails : 200ml

2. Initial mass of lime: 100g mixed in 300ml water

3. Agitator speed : 450 rpm

4. Amount consumed : 70 ml

5. Time at which pH reaches max: 49 seconds

6. Final mass of lime interms of volume: 230 ml

7. Initial pH 1.72

8. Final pH 8.87

5 RESULTS /DATA ANALYSIS AND DISCUSSION

In the leach experiments the leach efficiency was calculated using the formula, which was based on the liquors.

% leaching efficiency =

Volume of preg × metal in preg + Volume of wash liquor × metal in wash liquor

Metal in sample × weight of sample

From the formula above the efficiency will be calculated based on the results from liquor analysis. These liquors include the pregnant leach solution and the wash liquor.

Sample calculation

% leaching efficiency=

$$\frac{(0.195) \times (3.5) + (0.115) \times (0.5)}{(0.0196 \times 600g)} \times 100 \quad = 62.925\ \%$$

With the PLS at 195ml and the wash liquor at 115ml respectively the PLS deposited in the two liquors were 3.5gpl and 0.5gpl Cu respectively, with the 600g sample of about 1.96% Cu

The cost analysis of the acid: the cost of acid at the time of the experiment was in the range of $250-$350 US per tone.

Therefore

$350/1000 = $0.35/kg

We know that   D = M/V where M=mass and V= volume

V = M/D the specific gravity of acid is 1.84 for sulphuric Acid

V = 0.35/1.84 = $0.190/L

In the first expt the amount of acid consumed was 360ml this translates to 0.360L

As a sample calculation we have.

Cost of acid consumed will be $0.190/L × 0.360L = $0.0684

$0.0684 × 100 = 6.84c US worthy of acid was consumed in one experiment.

At this cost of acid the trends in the efficiencies are shown in the graphs plotted for %eff vs. leaching time
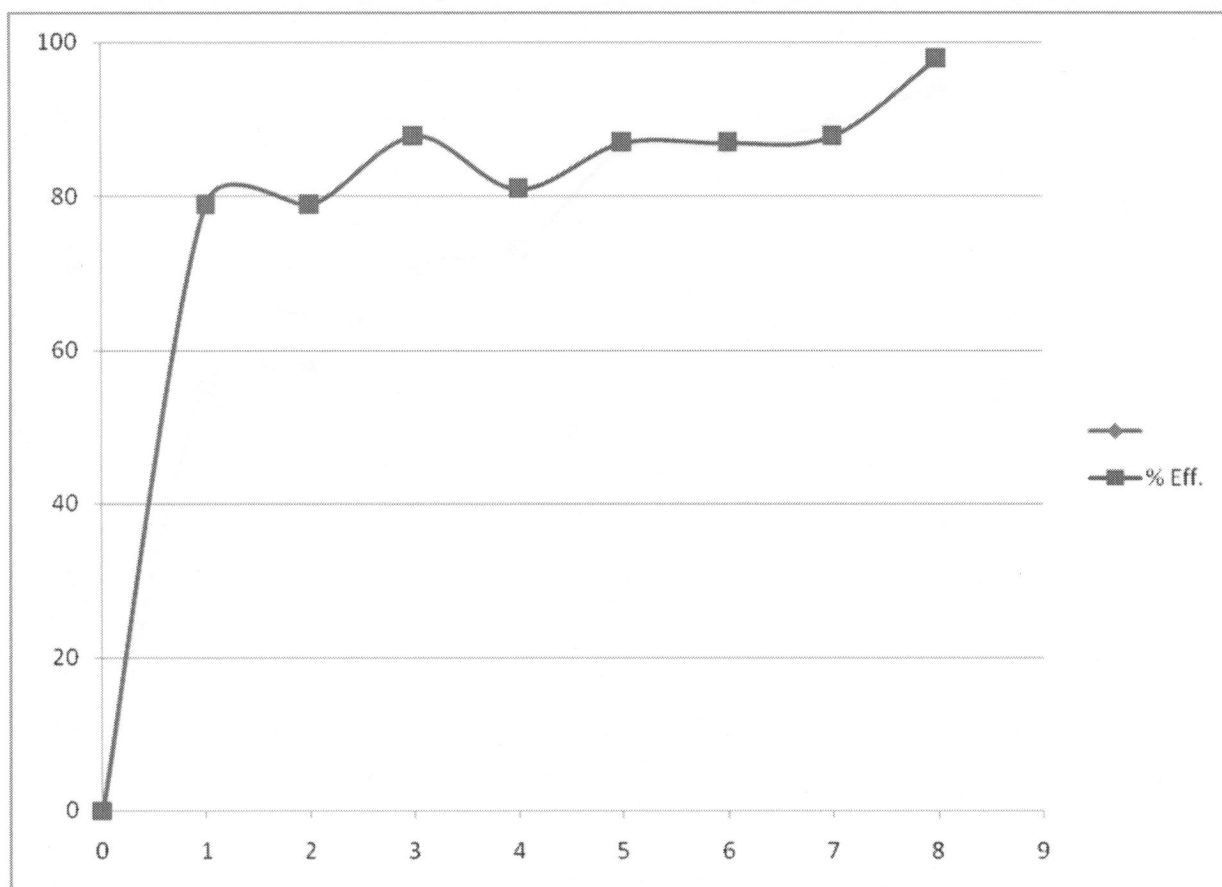
Figure 1 a.Graph of Variation of efficiency with time at pH 1.9 and 24% solids
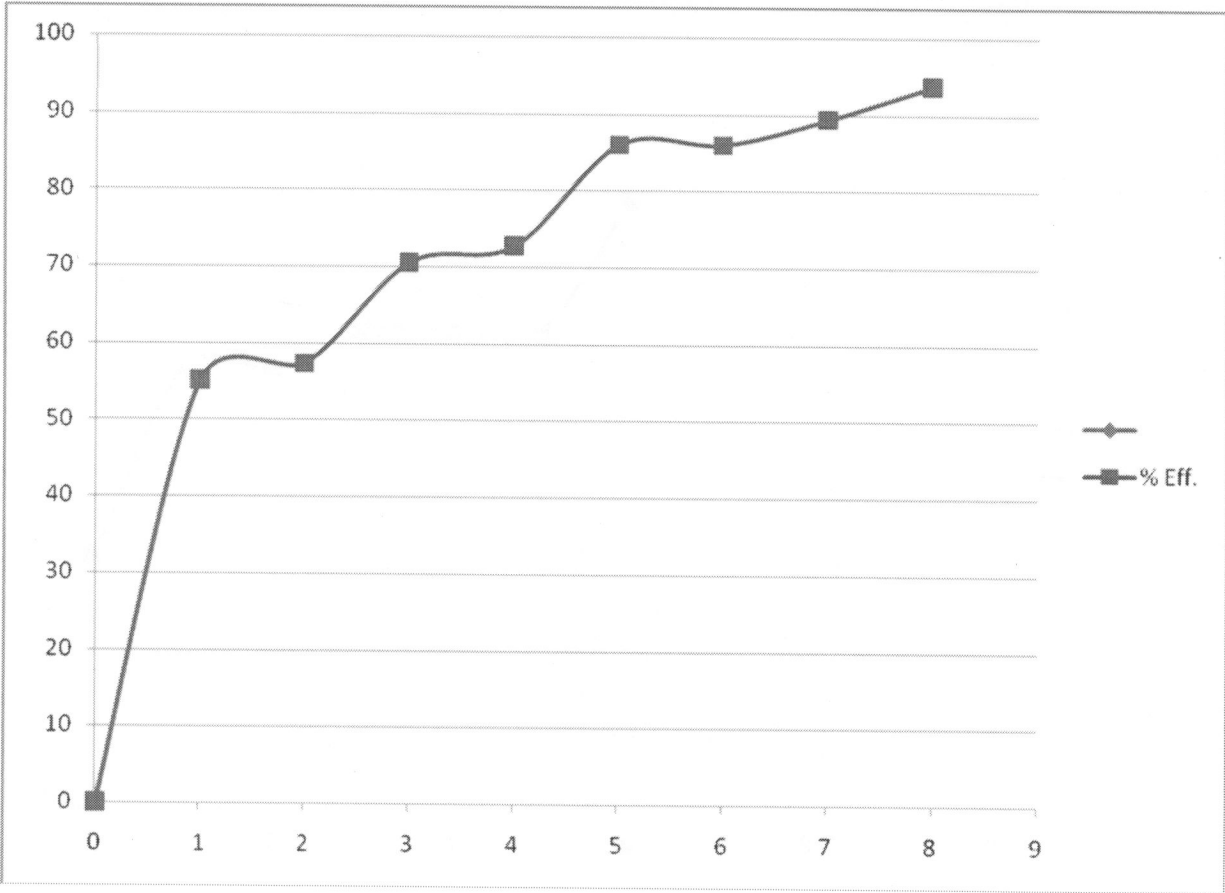
Figure 1b. variation of leach efficiency with time at pH 2.0 and 24 % solids

In figures 1a and b the leaching efficiency increased with increase in the time of leaching showing a possitive increase of over 90% leach efficiency as shown in the two graphs. After six hours of leaching there is a small increase in the leach efficiency showing that amost all the acid soluble copper has gone into solution at this time. Our concern is acid consumed in the experiments; in figure 1a there was slightly consumption of about 75 ml while in figure 1b the consumption was 87ml showing a defference of about 12ml acid. The general trend is that pH of 2.0 favours efficiency while pH 1.9 favours consumption meaning at least leaching should be done between the two pH ranges to achieve good efficiency and good lime consumption.
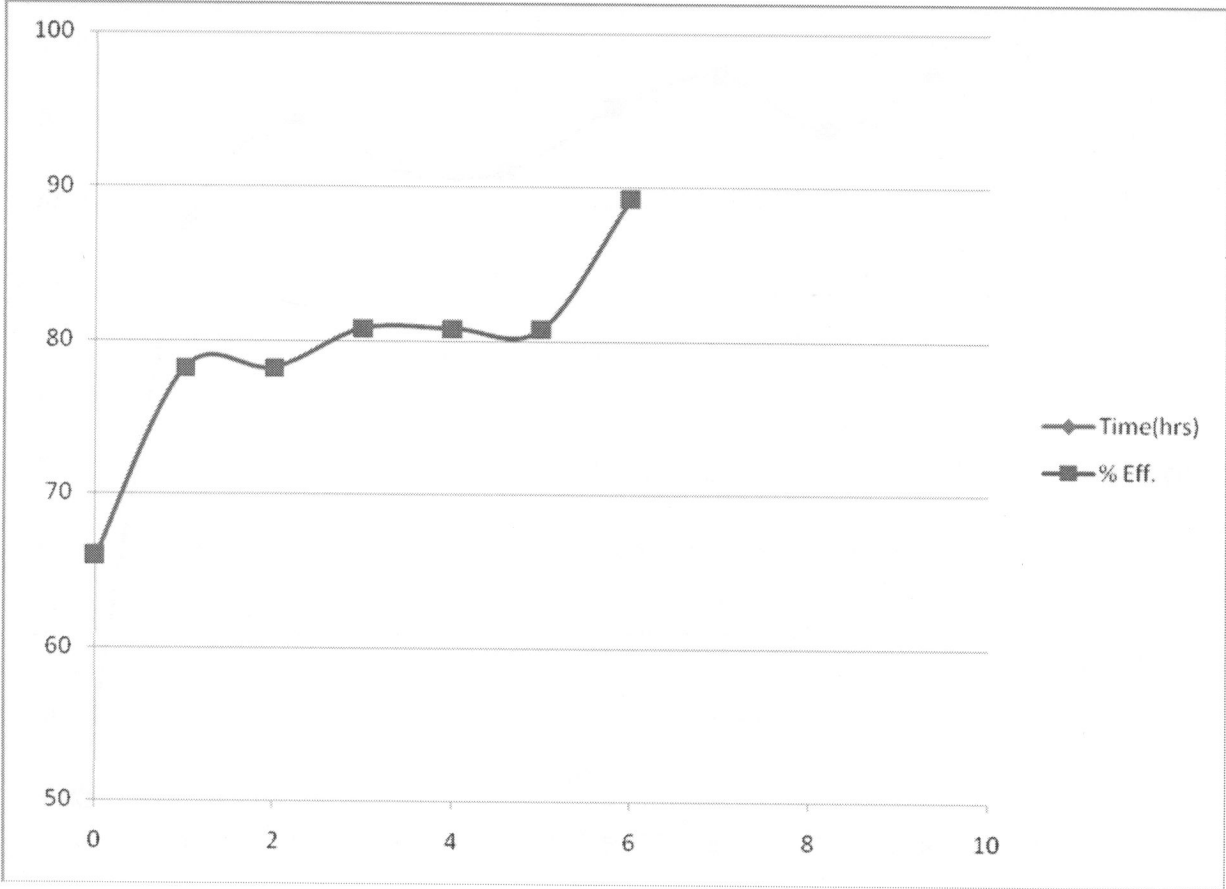
Figure 1c. Variation of leach efficiency with time at pH 2.0 and 26% solids

Figure 1d. variation of leach efficiency with time at pH 2.0 and 27% solids

In figure 1c and d the trend is that there is a reduction in the leach efficiency with time as the densties are increased from 26%-27% solids. The efficiency in both figures increase with increase in time though there is a small increase from the sixith hour and the eighth hour showing that the acid soluble Cu would have gone into solution. The acid consumed in this case increased from 360ml to 377ml showing an increase of 17ml. The solids concentration at 26% favours both consumption and good leach effiency approaching 90% efficiency as seen from fig 1c.
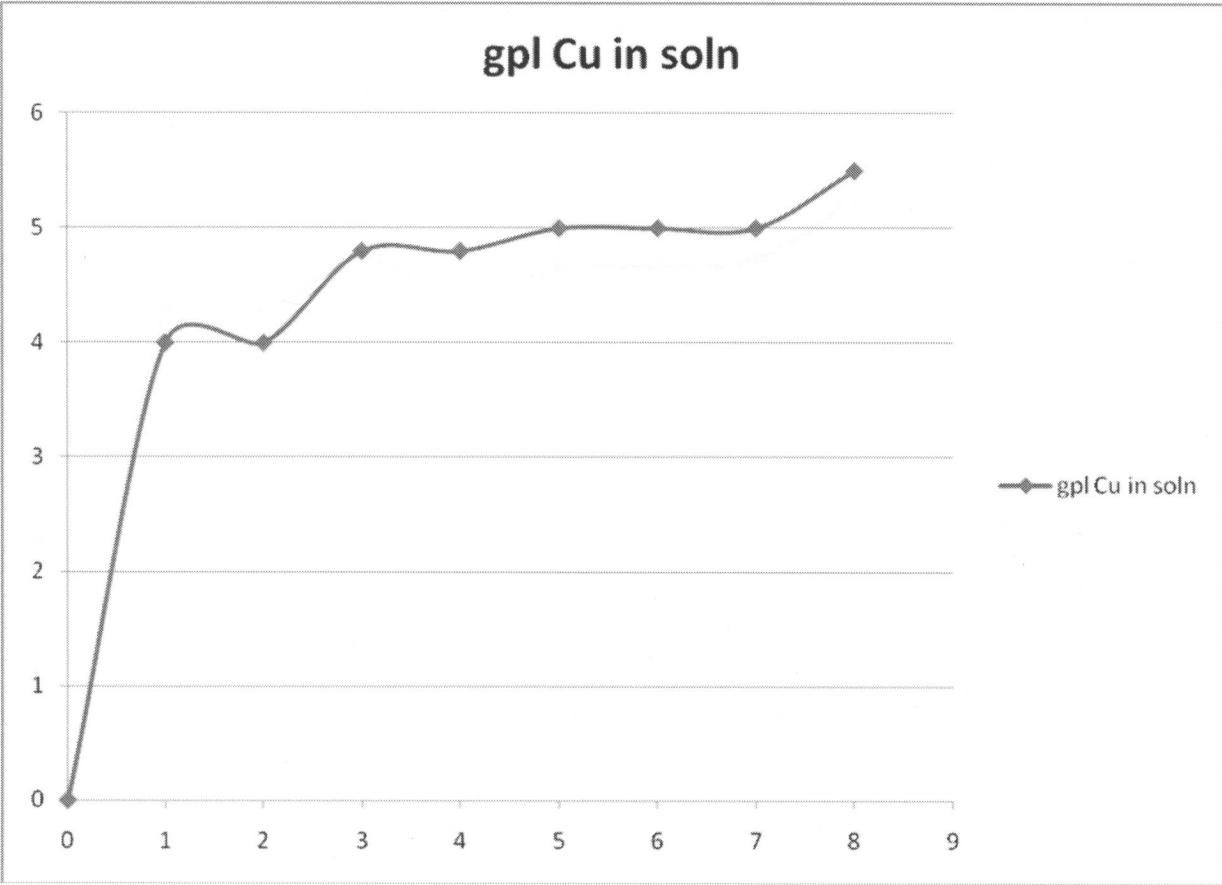
Figure 1e. gpl Cu in solution vs time at pH 2.0 and 27% solids
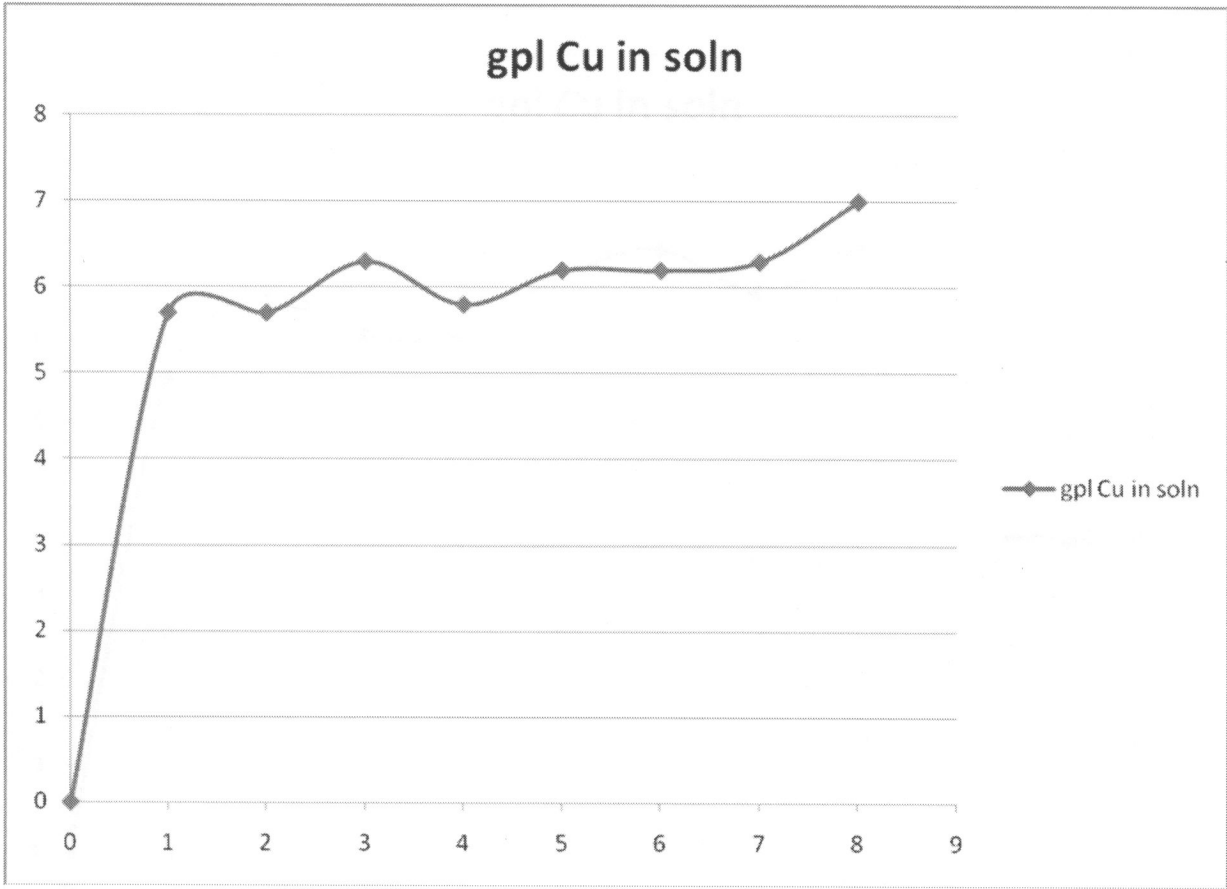
Figure 1h. gpl Cu in solution vs time at pH 1.9 and 24% solids
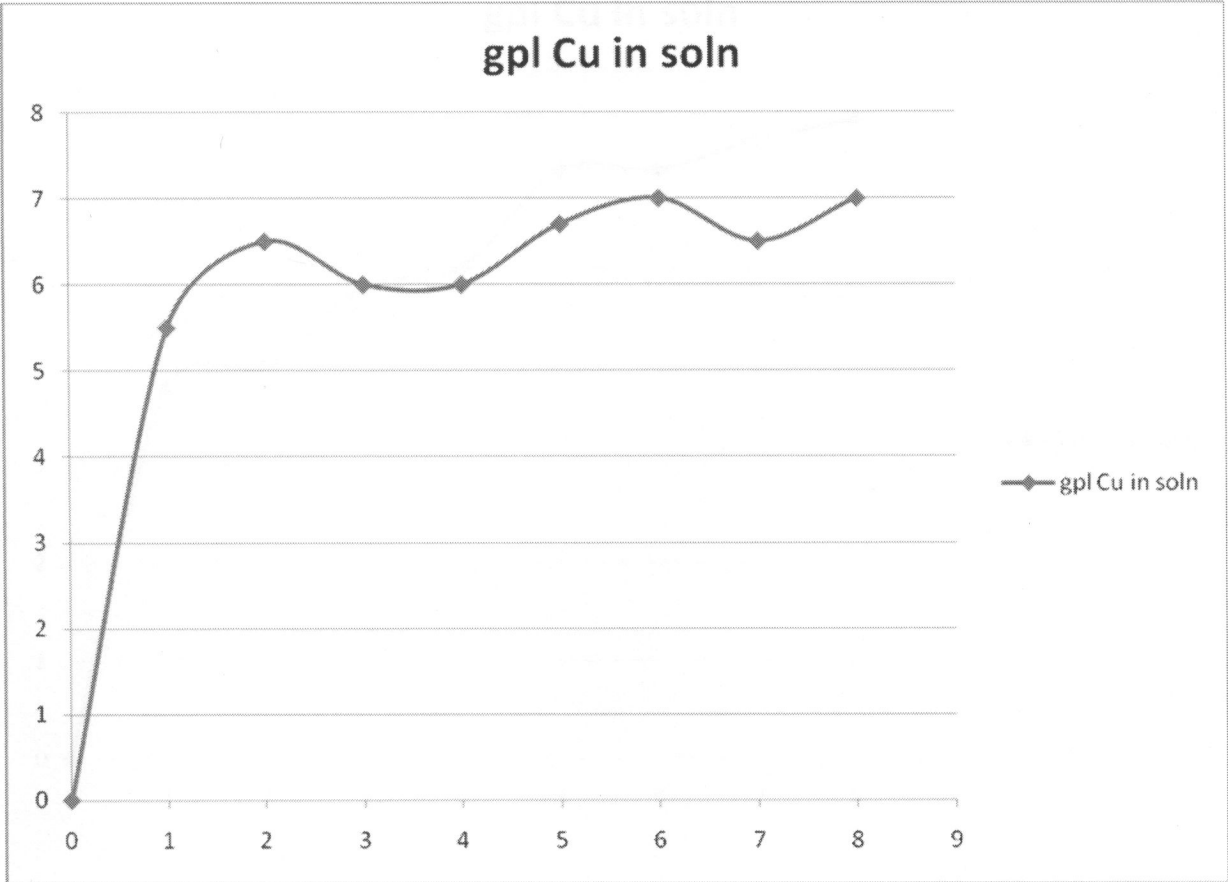
gpl Cu in soln

Figure 1f. gpl Cu in solution vs time at pH 2.0 and 24% solids
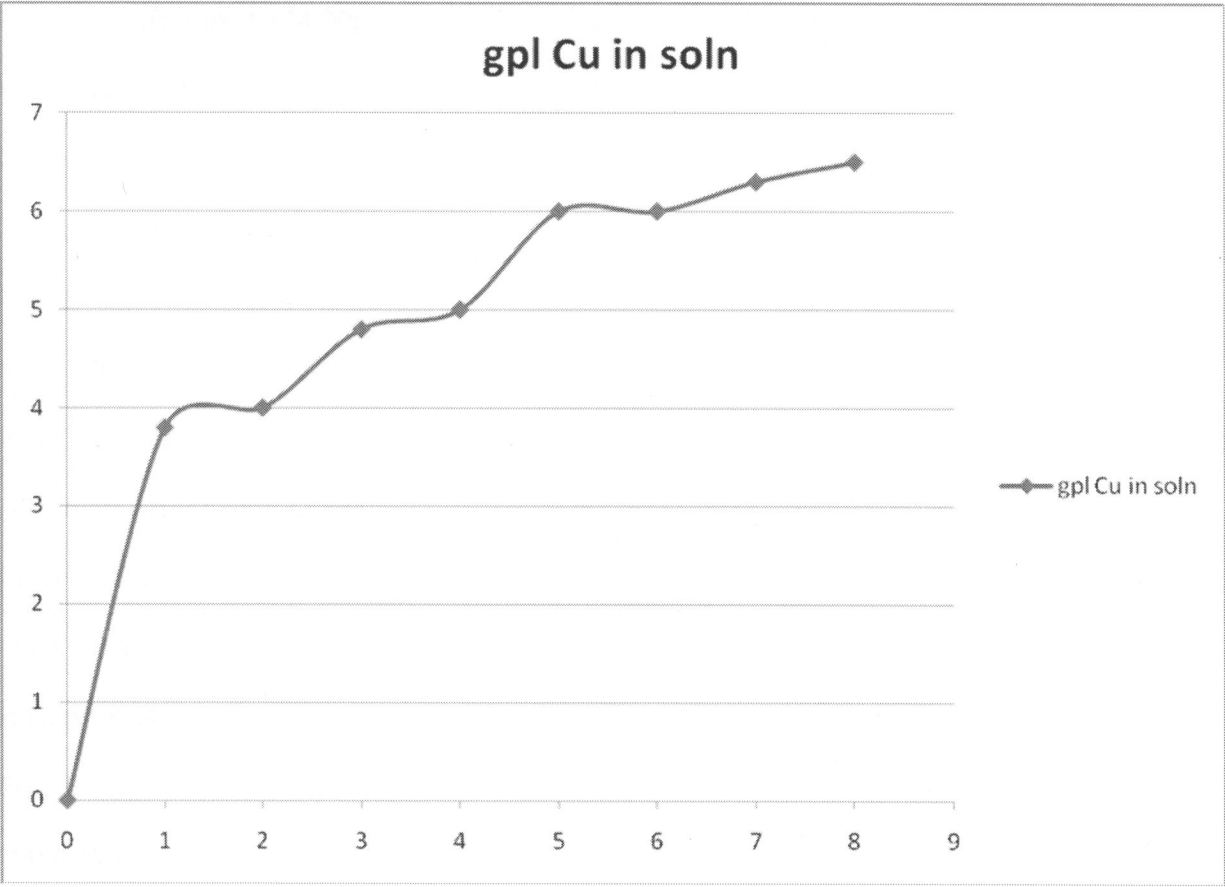
Figure 1g. gpl Cu in solution vs time at pH 2.0 and 26% solids

In the graphs shown above the plot of gpl Cu in solution is plotted against time trends in all the graphs shows an increase of Cu in solution with time. These graphs have been ploted to show that after the fifth hour the gpl Cu in the solution is almost constant meaning that almost all the Cu will have gone into solution. To save some significant amount of acid the leaching time must be kept to at least I five hours leaching time.

LIME CONSUMPTION EXPERIMENTS

Results were as follows: weight of lime 169.10g, no water addition

Expt 1

Initial 200ml

Initial 169.10g

Final mass after expt 144.00g

Mass used 25.1g

Initial pH 1.60

Final pH 8.82

Expt 2

Initial 200ml

Initial 144.00g

Final mass after expt 122.41g

Mass used 21.59g

Initial pH 1.60

Final pH 8.34

Expt 3

Initial 200ml

Initial 122.41g

Final mass after expt 93.75g

Mass used 28.66g

Initial pH 1.59

Final pH 7.82

The rest of the results will be reflected in the appendix.

Calculations for the estimated cost of lime consumed in the experiments. Sample calculation for experiment 1.

$$\frac{\$100}{1000kg} = \frac{\$0.10}{kg}, \$0.10 \times 0.0251 = \$0.00251$$

$$\$0.00251 \times 100c/\$1 = 0.251c$$

With these consumptions and with those shown in the tables in the appendix It was seen that consumption of lime was high when it was just added dry and that it was lower when added with water. The reaction kinetics were faster when lime was added with water.

5.1 Conclusion

The graphs plotted from the data collected shows a steady increase of efficiency. The graphs show how the leach system responds to changes in the parameters like pH over a period of leaching time.

As can be seen there was need to check the behavior of the system within the first 1 hr as this is the region where there seem to be a sharp rise in terms of the observable leach efficiency and amount of gpl Cu deposited into the solution. Clearly there was need to cut samples at least every 15min in order to clearly observe what was happening within the first one hour. From these observable trends we can conclude that within the first hour during leaching there is rapid deposition of Cu into solution and after which there is an almost steady deposition of Cu. The trends are shown in the graphs in figures 1a to 1d of the discussion.

In some cases the jump is as high as from 0 to 65.9% shows that there was quite a rapid reaction within the first hour that needed to be observed.

The other graphs are for gpl Cu deposited in the solution, the trend were quite consistent with observed increase as the leaching time increases, though there is no considerable changes in the gpl Cu deposited in most cases after the fifth hour. The gpl Cu deposited remained almost constant.

The leaching of 8hrs gives efficiencies of between 80% - 90% efficiencies with increases in the consumed acid as the densities increases from 24 – 27% of 17ml. while varying the pH between pH 1.9 and pH 2.0 shows a reduction of 12ml in acid consumed.

The lime neutralization experiments showed that addition of lime as slurry was observed to consume less lime as opposed to adding it without making a pulp. The mass of lime consumed when added in its dry form was slightly more compared with the amounts in slurry form. The two types of lime used at the plant differ in the kinetics of the reaction when added to tails for neutralization quick lime reacts faster than slacked lime and conversely its consumption is relatively low.

5.1.1 RECOMMENDATIONS

Nkana Leach Plant must adopt a good pH control system where there is a constant check on the variations in the pH as the pH has a great effect on the leaching efficiency. Because pH is a very

important parameter in leaching, the recommended pH should be in the range of 1.8-2.0, in order to take as much of the acid soluble copper into solution as possible and keep acid consumption low.

The pulp density should also should be in the properly maintained as this had a very great effect on the acid consumption densities of 24-26% should be utilized as this gave a good acid consumption with a good leach efficiency.

Lime dosage at the tail tanks should be fed wet meaning it should be fed mixed with water so that there is minimal loses as dust. This not only helps keep the working environment conducive but it also makes helps with the reaction kinetics. The reactions are faster the nuetralisation takes place at a faster rate.

APPENDIX

Basic formula on sampling

$$(\sigma/a)^2 = (1/p' - 1/p) \, fgmld^3$$

Or

$$(\sigma/a)^2 = (1/p' - 1/p) \, Cd^3 \quad \text{where } C = fglm$$

Where

a= mean metal content of the lot

σ^2 = variance of the grade distribution

P'= weight of the sample in (kg)

P = weight of the lot to be tested (kg)

f= shape factor

g= size factor

m= mineralogical index (kg/m^3)

d= size of the largest piece in the lot. That is the sieve aperture which retains between 5 to 10% of the original.

L= liberation index

L= (a'-a) / (α-a)

The samples were cut from the stock pile from the stock pile. Given that the material was Malachite and ult6ilising the theory above from the data collected the limit of error can be calculated as follows from the sample collected. Blending was done were necessary especially when an ore samples is suspected to have some sulphides contained.

Example of limit of error calculation, from the plant sample that was collected.

From the data collected: a= 1.96% $\approx$ 2% Cu & 0.020 Co taking f=0.5, g=0.25 a'=10% and d= (10/100)×105=10.5/1000=10.5×10^{-3} $\rho_{cu(OH)2CO3}$=4.0 σ=? L=? m=? Cu sample weight=3kg

We let the wt of Cu(OH)$_2$CO$_3$= 2/100×1kg=0.02kg

Molecular weight of $Cu(OH)_2CO_3$ = $(63.5+(16+1)\times2+(12+(16\times3)))$ =157.5

Cu contained in $Cu(OH)_2CO_3$ = $63.5/157.5 \times 0.02 = 8.06\times10^{-3}$

i) α = $8.06\times10^{-3}/0.02 \times 100$ = 40.3% Cu

ii) m = $(1-a/\alpha \ / \ a/\alpha)^2\rho_v + (1-a/\alpha)\rho_g$ where ρ_v = density of variable component ρ_g=density of gangue a/α= ratio of the average grade to metal assay of pure mineral.

Therefore a/α = 2/40.3= 0.0496 since the ratio $a/\alpha < 0.1$, $m = \rho_v / a/\alpha \approx$ low grade

m= $4.0\times1000kg/m^3/0.0496$ = 80645.16 kg/m^3

iii) l= $(a'-a)/(\alpha-a)$= $(10-2)/(40.3-2)$= 0.2089

Looking at the bulk from which the sample was taken it is too large compared to the sample taken. Hence as $p \longrightarrow \infty$ and therefore $1/p \longrightarrow 0$

σ^2 = $fglmd^3a^2/p'$= $0.5\times0.25\times80645.16\times0.2089\times(10.5\times10^{-3})^3 \times 2^2/ 3$

σ=$\sqrt{(0.015559)}$ = 0.125

σ=0.125

Limit of error =$\pm Z\sigma/\sqrt{n}$

At α = 99% in other words at 99% confidence

β= 100 - (100-99 / 2) = 99.5%

Therefore $Z_{0.995}$= 2.57 from the tables

Limit of error = $\pm 2.57\times (0.125/1)$ =± 0.320%

± 0.320 % as seen from the calculation and the best sampling method would have given a smaller marginal error had the Gy's formula been applied before sampling was done.

The error though reduces as the number of samples increases i.e. for the two samples cut the limit of error was;

Limit of error = $\pm Z\sigma/\sqrt{n}$

=$\pm 2.57\times (0.1.25/\sqrt{2})$ = ± 0.227

Optimisations of acid and lime consumption at Nkana leach plant

The Gy's formula is helpful in determining the quantity of sample that would give as minimal limit of error as possible at the required confidence level.

Tables of results and graphs

TABLE 1

Experiment	Initial volume of tails	Initial mass of lime	Final mass of lime	Time at which pH reaches max	Final mass of lime	Initial pH	Final pH
1	200ml	169.10g	144.00g	45seconds	25g	1.60	8.82
2	200ml	144.00g	122.41g	46seconds	21.59g	1.63	8.24
3	200ml	122.41g	93.75g	48seconds	28.66g	1.60	8.34
4	200ml	93.75g	67.09g	49seconds	26.66g	1.59	7.16

Hydrated lime results with Agitated with an impeller.

Optimisations of acid and lime consumption at Nkana leach plant

TABLE 2

Experiment	Initial volume of tails	Initial mass of lime	Final mass after expt	Mass and time used taken	Initial pH	Final pH
1	200ml	62.33g	38.43g	23.9 g 6.23 seconds	1.51	7.50
2	200ml	80.00g	54.40g	25.6g 7.23 seconds	1.50	7.87
3	200ml	67.23g	45.82g	21.41g 7.05 seconds	1.53	7.98
4	200ml	45.82g	22.72g	23.10g 7.10 seconds	1.51	8.01
5	200ml	22.72g	0.66g	22.06g 7.12 seconds	1.52	8.06

Quicklime dry analysis agitated with an agitator at the speed of 400.10 RPM (revolutions per minute)
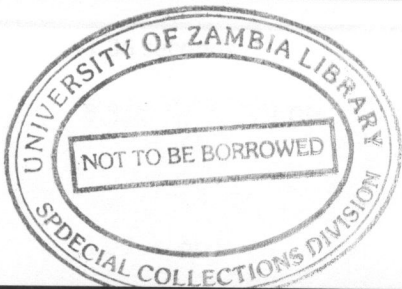
TABLE 3

Experiment	Initial volume of tails	Initial mass of lime	Final mass after expt	Mass used and time taken	Initial pH	Final pH
1	200ml	100g	90.72g	9.28g 1.10 seconds	1.05	9.10
2	200ml	90.72g	82.86g	7.86g 1.20 seconds	1.05	9.18
3	200ml	82.86g	74.37g	8.49g 1.26 seconds	1.06	9.30
4	200ml	74.37g	66.86g	7.51g 1.49 seconds	1.13	9.31
5	200ml	66.86g	55.20g	11.66g 1.31 seconds	1.10	9.06
6	200ml	55.20g	46.78g	8.45g	1.10	9.40

				1.35 seconds		
7	200ml	46.78g	37.95g	8.83g	1.09	8.99
				1.06 seconds		
8	200ml	37.95g	26.65g	11.3g	1.05	8.55
				1.30 seconds		
9	200ml	26.65g	16.53g	10.12g	1.10	7.55
				1.53 seconds		

Quicklime lime experiments. Agitator speed is 512.8 RPM (revolutions per minutes)

TABLE 4

Experiment	Initial volume of tails	Initial of volume lime	Final volume of lime	Volume of lime used and time of experiment	Initial pH	Final pH
1	200ml	300ml	130ml	170ml	1.74	8.87
				49 seconds		
2	200ml	230ml	200ml	30ml	1.84	8.20
				30 seconds		

3	200ml	200ml	190ml	10ml	1.84	8.80
				32 seconds		
4	200ml	190ml	178ml	12ml	1.84	8.60
				34 seconds		
5	200ml	160ml	135ml	25ml	1.94	7.89
				49 seconds		
6	200ml	135ml	115ml	20ml	1.19	8.50
				33 seconds		
7	200ml	115ml	90ml	25ml	2.18	8.80
				28 seconds		
8	200ml	90ml	60ml	30ml	2.17	8.03
				42 seconds		

Hydrated lime experiments done by mixing dry lime in water, the speed of the Agitator was 459.4 RPM (revolutions per minute)

TABLES FOR LEACHING EXPERIMENTS

TABLBE 1A

Time (hrs)		% Eff.		gpl Cu in soln	
0		0.003		0.003	
1		65.9		4	
		.			
2		65.9		4	
3		78.2		4.8	
4		78.2		4.8	
5		80.78		5	
6		80.78		5	
7		80.78		5	
8		89.29		5.5	

TABLE 2 B

Time (hrs)		% Eff.		gpl Cu in soln	
0		0.005		0	
1		67.52		5.5	
2		80.2		6.5	
3		73.89		6	
4		73.89		6	
5		81.5		6.7	
6		85.33		7	
7		78.95		6.5	
8		85.33		7	

TABLE 3 C

Time (hrs)		% Eff.		gpl Cu in soln	
0		0		0.003	
1		55.08		3.8	
2		57.3		4	
3		70.48		4.8	
4		72.7		5	
5		85.96		6	
6		85.96		6	
7		89.31		6.3	
8		93.66		6.5	

TABLE 4 D

Time (hrs)		% Eff.		gpl Cu in soln	
0		0		0	
1		78.82		5.7	
2		78.82		5.7	
3		87.76		6.3	
4		80.95		5.8	
5		86.9		6.2	
6		86.9		6.2	
7		87.76		6.3	
8		97.78		7	

TABLE 5 E

Time(hrs)	% Eff.		gpl Cu in soln	
0	0		0.003	
1	65.9		4	
	.			
2	65.9		4	
3	78.2		4.8	
4	78.2		4.8	
5	80.78		5	
6	80.78		5	
7	80.78		5	
8	89.29		5.5	

TABLE 6 F

Time(hrs)	% Eff		gpl Cu in soln	
0	0		0.003	

1		30.85	2.2
2		34.04	2.5
3		41.22	3
4		41.22	3
5		58.77	4.3
6		51.59	3.8
7		62.76	4.5
8		69.94	5

TABLE 7A

time(hrs)	gpl Cu	ppm Co	gpl Mn	gpl Zn	gpl Fe
0	5(ppm)	1.5			
1	3.5	56			
	0.5	16			
2	3.5	58			

	0.5	16			
3	4	62			
	0.8	22			
4	4	65			
	0.8	27			
5	4	58			
	1	27			
6	4	59			
	1	25			
7	4	58			1.02
	1	22			0.38
8	4.5	65			1.05
	1	24			0.4

TABLE 7B

time(hrs)	gpl Cu	ppm Co	gpl Mn	gpl Zn	gpl Fe
0	3pmm	0.5			
1	3	46			
	0.8	12			
2	3	47			
	1	18			
3	4	47			
	0.8	13			

4	4	47			
	1	11			
5	4.5	47			
	1.5	19			
6	4.5	49			
	1.5	17			
7	4.5	50	900	7.8	0.7
	1.8	12	375	1.6	0.19
8	5	54	850	8.2	0.78
	1.5	16	350	2	0.19

TABLE 7 B

time(hrs)	gpl Cu	ppm Co	gpl Mn	gpl Zn	gpl Fe
0	3ppm	0.2			
1	2	30			
	0.2	7			
2	2	33			
	0.5	12			
3	2.5	35			
	0.5	13			
4	2.5	38			
	0.5	12			
5	3.5	37			

	0.8	12			
6	3	37			
	0.8	12			
7	4	55	375ppm	2.8ppm	1.35
	0.5	21	200ppm	1.2ppm	0.78
8	4.5	59	400ppm	3.0ppm	1.4
	0.5	8	150pmm	0.8ppm	0.45

TABLE 7 C

time(hrs)	gpl Cu	ppm Co	gpl Mn	gpl Zn	gpl Fe
0	3ppm	0.25			
1	2.5	28			
	0.2	8			
2	2.5	29			
	0.2	7			
3	3	30			
	0.5	9			
4	3	30			
	0.5	9			
5	3	30			
	0.5	12			
6	2.5	31			
	0.5	12			

7	2.5	31			
	0.5	12			
8	4.5	37	3.6pmm	675pmm	1.2
	1	15	1.4pmm	300ppm	0.55
9	5	39	3.7ppm	700ppm	1.4
	1	15	1.2ppm	275ppm	0.5

TABLE 7 D

time(hrs)	gpl Cu	ppm Co	gpl Mn	gpl Zn	gpl Fe
0	0	0			
1	3	23			
	0.5	4			
2	2.5	24			
	0.5	6			
3	3.5	25			
	0.8	7			
4	3	25			
	0.5	6			
5	3.5	26			
	0.5	7			
6	4	27			
	0.5	6			
7	4.5	30	350pmm	3.6ppm	1.2

	0.8	9	175ppm	1.8pmm	0.35
8	4	33	375pmm	3.2pmm	1.3
	0.8	10	200pmm	1.6ppm	0.4

TABLE 7 E

time(hrs)	gpl Cu	ppm Co	gpl Mn	gpl Zn	gpl Fe
0	0	0			
1	5.2	34			
	0.5	7			
2	5.2	35			
	0.5	7			
3	5.5	36			
	0.8	7			
4	5	37			
	0.8	10			
5	5.2	41			
	1	9			
6	5.2	41			
	1	13			
7	5.5	45	400pmm	3.2ppm	1.25
	0.8	12	225ppm	1.2ppm	0.38
8	6	47	375ppm	3.4ppm	1.15
	1	12	150pmm	1.0ppm	0.32

TABLE 7 F

time(hrs)	gpl Cu	ppm Co	gpl Mn	gpl Zn	gpl Fe
0	3ppm	0			
1	4.5	60			
	1	13			
2	5.5	63			
	1	18			
3	5	66			
	1	18			
4	5	66			
	1	21			
5	5.2	71			
	1.5	24			
6	5.5	73			
	1.5	24			
7	5	85			
	1.5	28			
8	5.5	85			
	1.5	18			

APPENDIX (A) GRAPHS
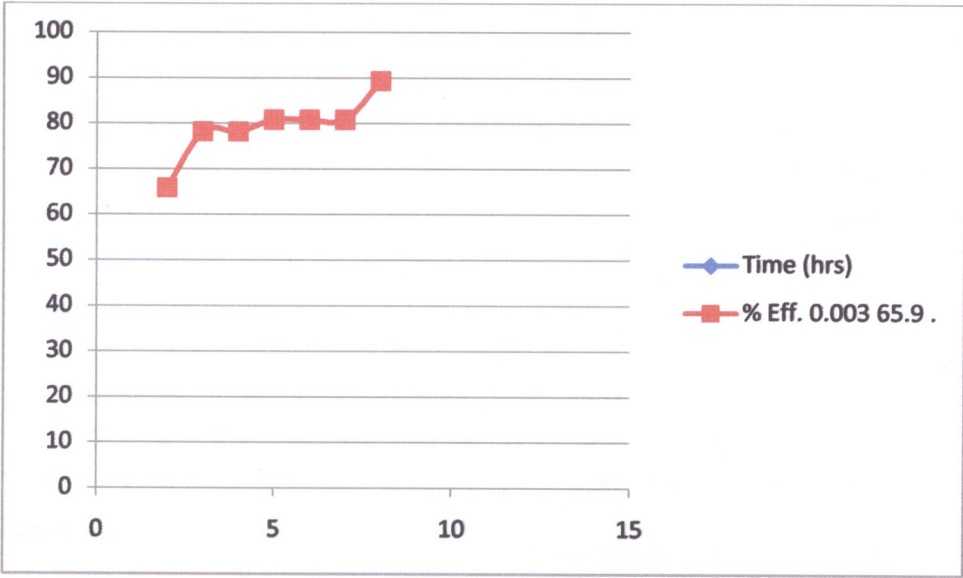
Figure 2.1 a
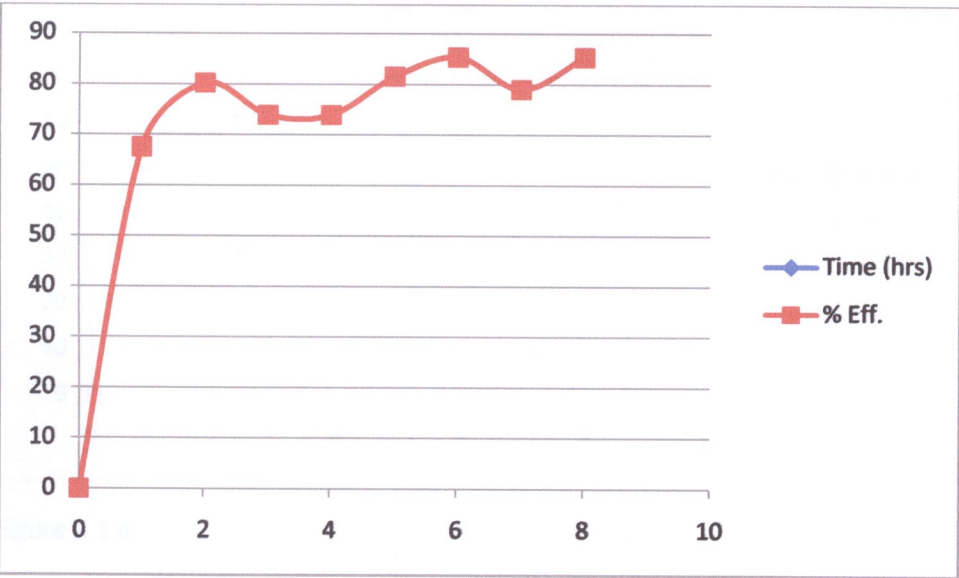
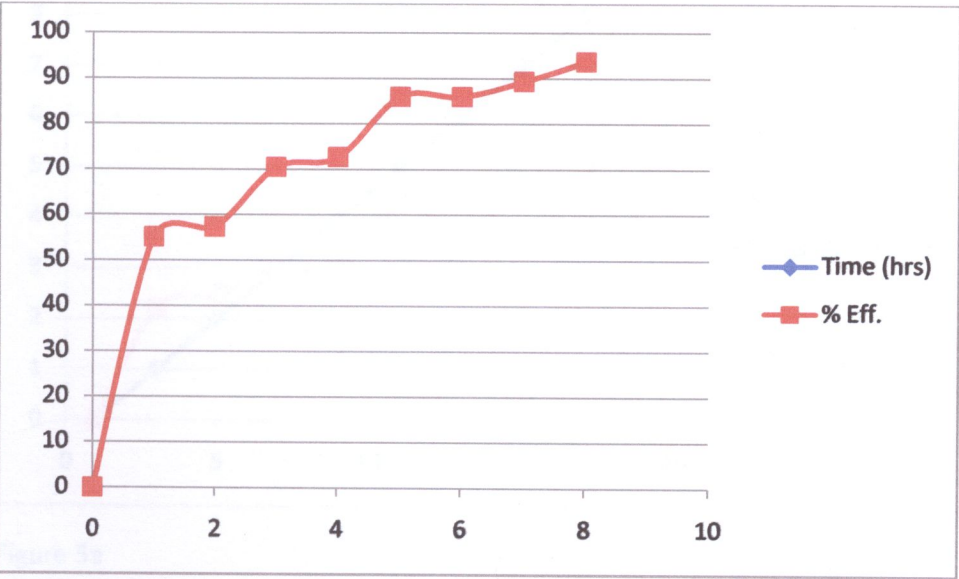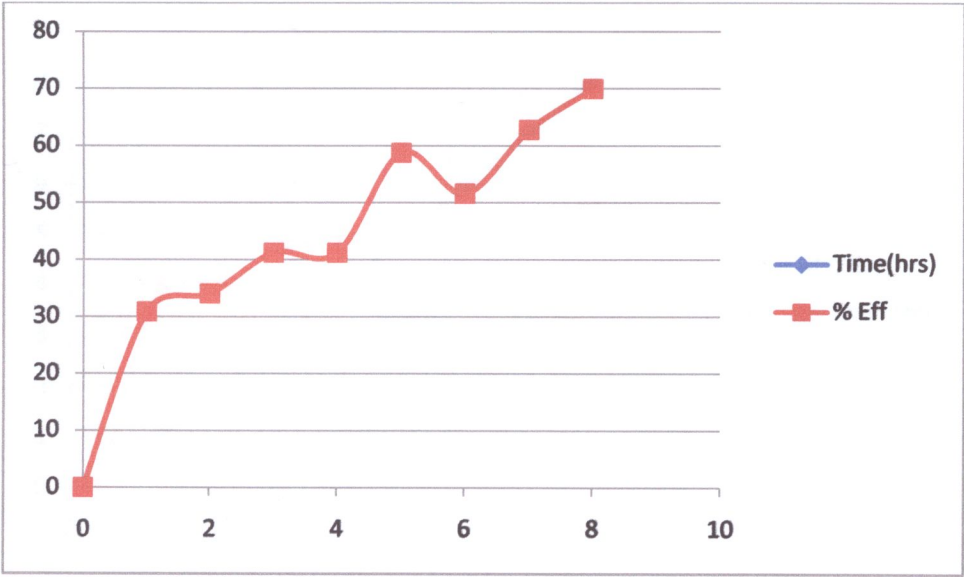
Figure 2.1b
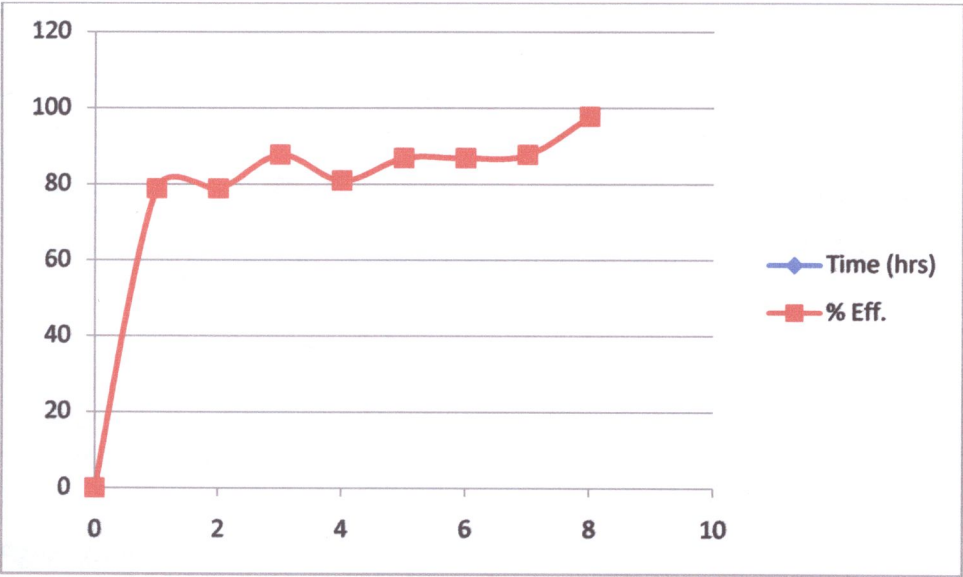
Figure 2.1c

Figure 2.1 d

Figure 2.1 e

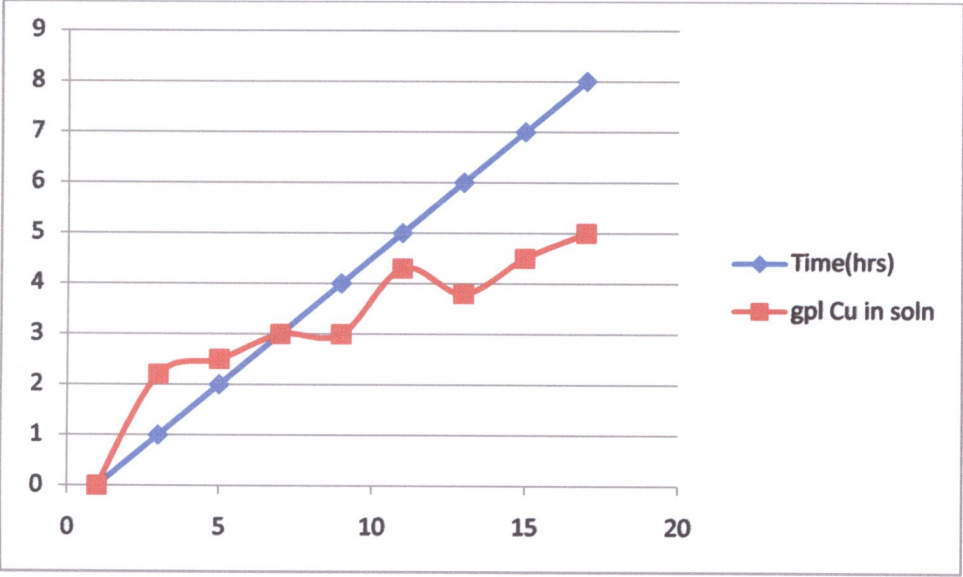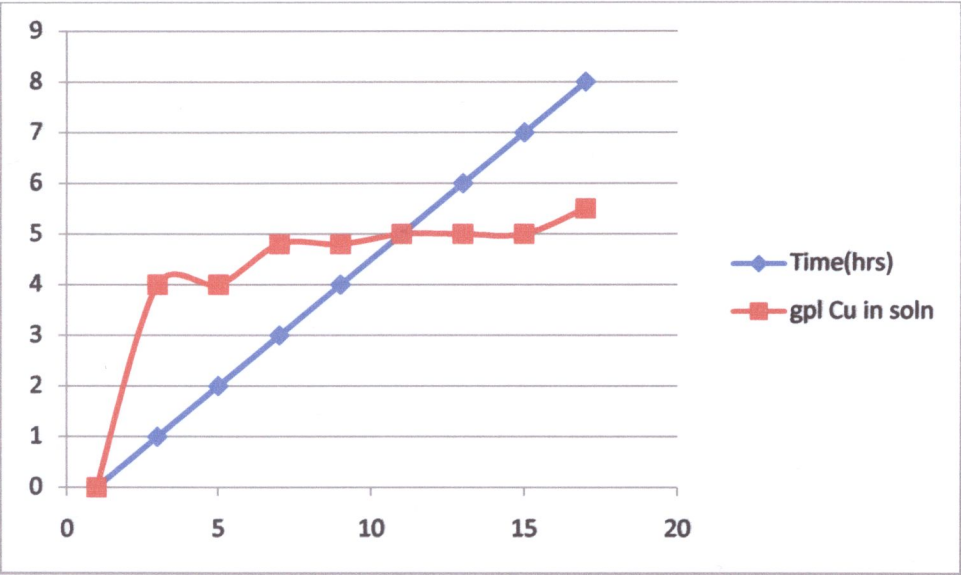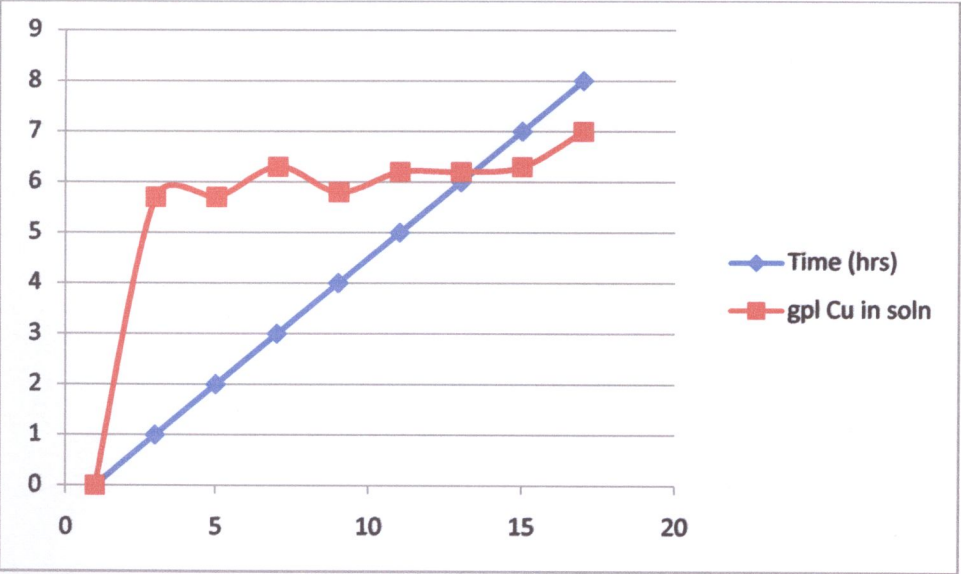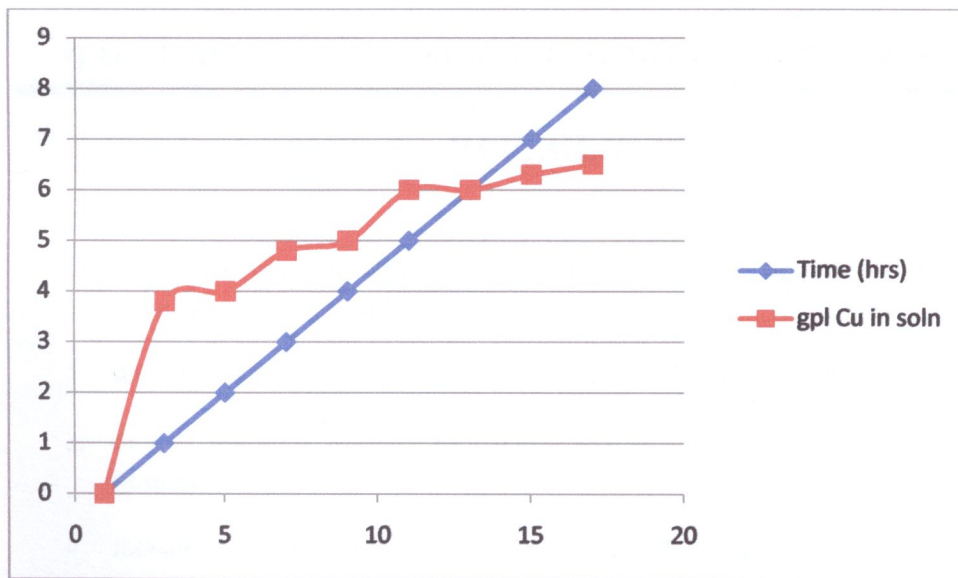GRAPHS: For gpl Cu plotted with efficiency

Figure 3a

Figure 3b

Figure 3c

Figure 3d

5. Acid [...]
   [...]

6. Nacid [...]
   Geologia [...]

7. David L. [...]
   5698170 Oct 15 [...]

8. Kordosky R. Y. [...]
   journal 1992 [...]

9. Hydrometallurgical [...]
   journal patent EP0831 [...]

10. F.D Mendes, R.A Martins, Select [...]
    approach to the experimental design [...]
    Braz. J. Chem. Eng. v 20 n3 São Paulo jul/sep [...]

11. Witika L. K & Mwaanga J.H. MM 515, lecture notes, 2011 [...]

REFERENCES

1. The Water Pollution control (Effluent and waste water) Regulation s of the Republic of Zambia, Third Schedule (Regulation 5(2)), 1993

2. Wills, B.A "Mineral Processing Technology", Third Edition, 1980. Pergamony Press

3. Dr. L. N. Nyirenda, MM442, fourth year lecture notes 2007/2008, UNZA, School of Mines. (Unpublished)

4. Jackson .E. "Hydrometallurgical Extraction and Reclamation", First Edition, 1986 Ellis Horwood Limited.

5. Acid Neutralizers Calm the Wastewaters. Hairston, Deborah W., Chemical Engineering, Dec. 1996.

6. Naoto TAKENO. Atlas of Eh-pH Diagrams, Intercomparison of thermodynamic databases; Geological Survey of Japan Open File Report No.419, 2005).

7. David L. Jones. Hydrometallurgical process for copper-containing materials, patent # 5698170 Dec 16 1998.

8. Kordosky A. G. Copper Solvent Extraction: The State of the Art, Mining and Metallurgy journal 1992

9. Hydrometallurgical process for copper containing minerals, Mining and Engineering journal patent #5698170; Dec 16, 1997.

10. F.D.Mendes; A.H.Martins Brazilian Journal of Chemical Engineering, A statistical approach to the experimental design of the sulfuric acid leaching of gold-copper ore. Braz. J. Chem. Eng. v.20 n.3 São Paulo jul./sep. 2003

11. Witika L. K.& Masinja J.H. MM 515, lecture notes 2008, UNZA, School of Mines