A HEAP LEACH FEASIBILITY STUDY FOR MIMBULA REFRACTORY COPPER ORE OF NCHANGA MINE

By

Takunda Joseph Mhandu

A Dissertation submitted to the University of Zambia in partial fulfillment of the requirements of the degree of Master of Mineral Science in Metallurgy and Mineral Processing

The University of Zambia

April 2017

DECLARATION

I, Takunda Joseph Mhandu declare that I am the author of the dissertation submitted for the Master of Mineral Sciences Degree in Metallurgy and Mineral Processing, at the University of Zambia. This dissertation contains my own original work and has not been previously submitted to any other institution of higher learning. I further declare that all sources cited or quoted are indicated and acknowledged by means of a comprehensive list of references.

Signed……………………… …   Date……………………………

ABSTRACT

Studies done on Mimbula ore have been on agitation leaching, high temperature and pressure leaching and treatment of refractory ore (TORCO) process. However these methods attracted significantly high capital investment leaving heap leaching as a more economically attractive method to be investigated.

This research was aimed at assessing the amenability of Mimbula refractory ore to heap leaching and to understand the conditions under which copper dissolution can be enhanced. The specific objectives were to determine the optimum conditions for agglomeration, the optimum bed height that gave the best leaching efficiency and to establish the relationship between bed height and leaching efficiency.

Ore characterisation was done to establish the chemical and mineralogical compostion of the ore. Agglomeration and column leaching experiments were carried out in agglomeration drum and PVC pipes respectively. In order to validate the experiments, each experiment was repeated three times. The soak test, agglomerate size distribution and visual inspection were used as the tools for evaluating the agglomerates quality. For all the column leaching experiments, a chemical and mineralogical composition analyses of feed (ore) and leach residues was carried out in order to measure the extent of copper dissolution.

The optimized agglomeration parameters were 600 gpt binder dosage, 14 % moisture content of ore, 20 rpm agglomeration speed and 3 min residence time. Agglomeration greatly improved the leach efficiency and kinetics. The highest leach efficiency of 60 % was obtained with agglomerated ore in 12 days compared to 32 % for unagglomerated ore in 94 days. The mineralogical analysis showed that about 90 % of the cupriferous mica dissolved into solution whilst chrysocolla and pseudomalachite completely dissolved in solution. However, sulphides, especially chalcopyrite were difficult to leach under the conditions employed. It was concluded that Mimbula refractory copper ore is amenable to heap leaching. Further studies to improve dissolution of sulphide minerals using acidified ferric sulphate and/or bioleaching were recommended.

APPROVAL

The dissertation of Takunda Joseph Mhandu has been approved as partial fulfilment of the Master of Mineral Sciences Degree in Metallurgy and Mineral Processing of the University of Zambia by the following:

Supervisor	Signature	Date
Dr J Manchisi	…………………	…………………
Internal Examiner	Signature	Date
Dr Jere	……………………	…………………
Internal Examiner	Signature	Date
Dr Witika	…………………….	………………….
Internal Examiner	Signature	Date
Dr Siame	……………………..	…………………..

ACKNOWLEDGEMENTS

Firstly I would like to thank the Lord Almighty for His Love and Grace He provides in my life.

My sincere gratitude goes to the KCM management and Nchanga Tailing leach plant team for their support and giving me the opportunity to learn and undertake my research.

I would also like to thank my Supervisors Dr Manchisi and Mr. Kane for their unwavering support and guidance throughout the research.

Special mention goes to Mr. Trevor Chalwe for the support and technical guidance during the course of the research.

Many thanks go to my colleagues Mr. Tichaona Mukono and Mr. Tinashe Ndoro for their brotherly love and encouragement throughout the course of the studies.

Finally I would like to express my heartfelt gratitude to my wife for her support, love and encouragement throughout the course of my studies.

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

ACRONYMS

% ASCu	*Percent acid soluble copper*
% AICu	*Percent acid insoluble copper*
CRO	*Chingola Refractory Ore*
CCD	*Counter current decantation*
KCM	*Konkola Copper Mines*
tpa	*Tonnes per annum*
% TCu	*Percent total copper*
% TFe	*Percent total iron*
TORCO	*Treatment of refractory copper ore*
PLS	*Pregnant leach solution*
SX	*Solvent extraction*
Wt.	*Weight*
UNZA	*University of Zambia*

CHAPTER ONE: INTRODUCTION

1.1 BACKGROUND

Konkola Copper Mines (KCM) is situated on the Zambia copperbelt, which is a north-west trending zone 150km long by 50km wide and is one of the world's largest mining regions, characterized by size and quality of its copper and cobalt deposits (Phalwane, 2013). KCM produces its copper from the following sources; Nchanga Open Pit (NOP), Nchanga underground, Konkola underground, Chingola Refractory ore (CRO) and Chingola Open Pit.

The mine produces copper via pyrometallurgical and hydrometallurgical routes. In the pyrometallurgical route, the mine has a Direct Blister Flush Smelter (DBF) which produces blister copper, slag and other flue gases such as sulphur dioxide which is then used to produce sulphuric acid for hydrometallurgical processes. Hydrometallurgical processes are carried out in the Nchanga Tailings Leach Plant (TLP).

The mine produces electrowon copper cathodes at Nchanga Tailings Leach Plant (TLP) by leaching current concentrator tailings and reclaimed tailings with Sulphuric acid. Leaching is done in open air agitated vessels (Pachucas) at atmospheric temperature and pressure. The pregnant leach solution is purified and upgraded in the copper solvent extraction plant before being pumped to the tank house for electrowinning (Kane, 2016).

The TLP is currently producing an average of 50 000 tonnes per annum (tpa) of copper against the design of 95 000 tpa. Moreover the solvent extraction plant is currently running with one cell on standby leaving much room unutilized. The total integrated copper production for the financial year 2014-15 was 116,000 t. KCM plans to scale up the copper production from refractory copper ore via the proposed heap leaching. The heap leach feasibility tests work are underway on stockpile 12 of the Chingola refractory ore (SP12) but other ore reserves such as Mimbula, Kakosa ,ore are yet to be exploited.

Mimbula material is estimated to have a tonnage of 23 million. There has been flotation and agitation leaching tests done on Mimbula to assess its amenability to these copper recovery routes (Phalwane, 2013). From the studies it was concluded that it is feasible to extract copper from Mimbula using these methods but at remarkably high costs. There have not been heap leach feasibility tests done on Mimbula ore. Moreover Mimbula ore is slimy and highly friable, with 70 % of it being finely sized material of -2 mm. This leaves 30 % of this ore resource to be of suitable feed sizes for heap leaching. It is from this background that further research was proposed to assess its amenability to heap leaching and investigate on conditions under which copper dissolution can be understood and enhanced.

1.2 PROBLEM STATEMENT

The ore considered for heap leach is CRO which comprise of Mimbula, Kakosa, to mention but a few. Studies done on Mimbula ore have been on agitation leaching, high temperature and pressure leaching, treatment of refractory copper ore (TORCO) process. However, these leaching methods attracted high capital and operation costs, leaving heap leaching as a more economically attractive copper extraction route to be investigated. Therefore, this study aims at assessing the amenability of Mimbula ore to heap leaching and understanding the conditions under which copper dissolution from the Mimbula refractory ore can be enhanced.

1.3 JUSTIFICATION

The TLP processes only 50 000 dry metric tonnes of ore and any attempts made to increase the throughput pose challenges with regard to the design capacity of Pachucas and counter current decantation (CCD) circuit (Kane, 2016). However, the solvent extraction circuit is running with one train on standby and the tank house is producing 180 t/day of copper cathodes against the design capacity of 287 t/day. Heap leaching therefore provides a more attractive route to increase copper production by using existing infrastructure which is currently under-utilized.

1.4 AIM

- To determine the leaching kinetics and efficiency of copper dissolution under different experimental conditions.

1.5 SPECIFIC OBJECTIVES

- To optimize the agglomeration process and determine the effect of feed agglomerate size and binder dosage on leaching efficiency.
- To determine the effect of lixiviant flow rates and bed height on leaching efficiency.
- To determine the optimum bed height that yields the highest copper recovery.
- To determine the leaching mechanism of the micaceous copper ore.

1.6 RESEARCH QUESTIONS

- Is Mimbula ore amenable to heap leaching?
- What is the relationship between bed height and leaching efficiency?
- Under which experimental and solution conditions is dissolution of copper from refractory ores favorable?

1.7 EXPECTED OUTCOMES

- Generated experimental data will contribute to a better understanding of the heap leaching process of refractory copper ore.
- The research data may serve as a basis for the design and scale up of a commercial heap leach process.

CHAPTER TWO: LITERATURE REVIEW

2.1 INTRODUCTION

Mimbula ore is one of the Zambian Copperbelt ore deposits found in the south-western limit of the Nchanga mineralised arc. The pit was mined briefly in the 1970s, but abandoned because most of the ore was refractory i.e. cupriferous micas, hence not responsive to either flotation or leaching. However, recent explorations have found cores which indicated that there exists ore which has the conventional mineable copper mineralization i.e. chalcocite and malachite (Phalwane, 2013). This ore contains both acid soluble and acid insoluble copper with an average copper grade of 1 % Total copper (TCu). The ore consists of malachite ($Cu_2 (OH)_2 CO_3$); chrysocolla ($CuSiO_3.nH_2O$); pseudo malachite ($Cu_5(PO_4)(OH)_4$; cuprite (Cu_2O) and tenonite (CuO). In the cupriferous mica, the copper is found fixed in intersilicate layers of vermiculites. Table 1 below shows the mineralogy of Chingola refractory ore.

Table 1: Mineralogical composition of Chingola ore (Kane, 2016)

MINERALS	**Wt. (%)**
Chalcopyrite	0.181
Bornite	-
Chalcocite	0.145
Pyrite	0.215
Carrollite	0.020
Native Copper	-
Malachite	0.528
Pseudomalachite	0.458
Chrysocolla	0.032
Azurite	-
Cuprite	-
Cupriferous Mica	10.000
Gangue	88.422
Total	**100**

2.2 PREVIOUS WORK

The cupriferous mica is found in virtually all Zambian Copper belt oxide ore bodies and are low grade copper ores from which copper is difficult to extract under conventional or direct leaching methods (Notebaart, 1988). They exist in three forms which are phlogopite (no-copper bearing), chloride (no-copper bearing) and as interstratified vermiculites which are copper bearing. Owing to the presence of iron, magnesium, potassium, which compete with copper during acid leaching, a number of alternatives have been proposed and developed in order to extract copper from the refractory ore. These methods include the following:

a) TORCO (Treatment of Refractory Copper Ore) PROCESS

This method was developed specifically for the treatment of the Chingola Refractory copper ore (Notebaart, 1988). Pinkney (1970) reported that the process involved heating of the copper refractory ore in a fluidized bed reactor to around 800^{o}C and mixing it with coal and salt. The salt reacted with the gangue silicate and water formed hydrochloric acid (HCl), which chlorinated the copper. The copper chloride was reduced by hydrogen and metallic copper was deposited on the coal. The copper particles were seperated from the gangue by conventional flotation. However this process' success was hindered by high capital and operating costs as the process was energy intensive and also required high grade coal which was a challenge to constantly obtain.

b) Preconcentration

This method was developed mainly to minimize the gangue material prior to acid leaching, thus reducing the total acid consumption during leaching (Bolmes, 1972). It involved the removal of ore fines followed by concentrating the ore material via flotation. The major constraints of this method was that it resulted in low concentrate grade and ultimately low recoveries as most of the copper was lost in the fines (Notebaart, 1988).

c) Exfoliation prior to leaching

The high acid consumption in leaching refractory copper ore with cupriferous mica has been shown to be primarily for the need to break down mica to contact the copper with the leaching agent (Sulphuric acid). In the investigations carried out by Jebson (1970) on the occurrence of the micaceous copper materials, it was shown that copper bearing mica exfoliates on heating rapidly to 300-400 ^{o}C. Therefore from these findings, it was proposed that the copper ore material be heated for temperature up to 500 ^{o}C in oxidation pressure leaching vessels in order to extract the copper from the refracrory copper ores. However from the experiments carried out, it was observed that little change occurred in the leaching characteristics of the ore material. Moreover, the mineralogical examination showed that exfoliation up to 500 ^{o}C did not result in the breaking of copper-hydroxyl bond (Cu-OH). Therefore this method ultimately failed to produce favourable results.

In recent years, studies on the treatment on the Chingola refractory ores carried by Whyte et al (2001) concluded that hydrometallurgical extraction routes are favourable. Elevated temperature leaching and use of autoclaves was also recommended. However the recommended methods require high capital and operation cost.

2.3 EXTRACTIVE METALLURGY OF COPPER

Copper is produced through two main extractive routes which are pyrometallurgical and hydrometallurgical routes. About 20 % of the world's primary copper production comes from hydrometallurgical processing of mainly oxide copper and chalcocite (Hyvarinen, 2005). In 2010, about 4.5 million tonnes of copper was produced using hydrometallurgical processes (Hill, 2013).

2.3.1 HYDROMETALLURGICAL COPPER EXTRACTION

Hydrometallurgical flowsheets comprise of three main essential steps for recovery of copper which are; *leaching, solvent extraction* and *electrowinning.* ***Leaching*** involves the dissolution of copper bearing material by a leachant such as sulphuric acid to produce pregnant leach solution (PLS) which contains other impurities such as Fe, Al, Co, Mn, Zn, Mg, Ca etc. The PLS is fed into the ***solvent extraction*** circuit where the impure PLS is upgraded and purified to produce an electrolyte suitable for ***electrowinning*** of copper. The solvent extraction (SX) circuit has two steps which are extraction and stripping. In the extraction step copper is loaded into an organic solvent which contains an organic extractant which selectively reacts with copper over other metal cations in the PLS. The barren raffinate solution which is high in acid is recycled back into the leach circuit as lixiviant. The second step is the stripping stage where copper is stripped from loaded organic solvent into the electrolyte from which copper is electrowon (Davenport, 2002).

2.3.2 LEACHING METHODS

The choice of leaching methods depends on the mineralogy of ore, copper grade of the ore and the particle size of the ore (Sheffel , 2002). Figure 1 below shows the criteria used to choose the method of leaching.

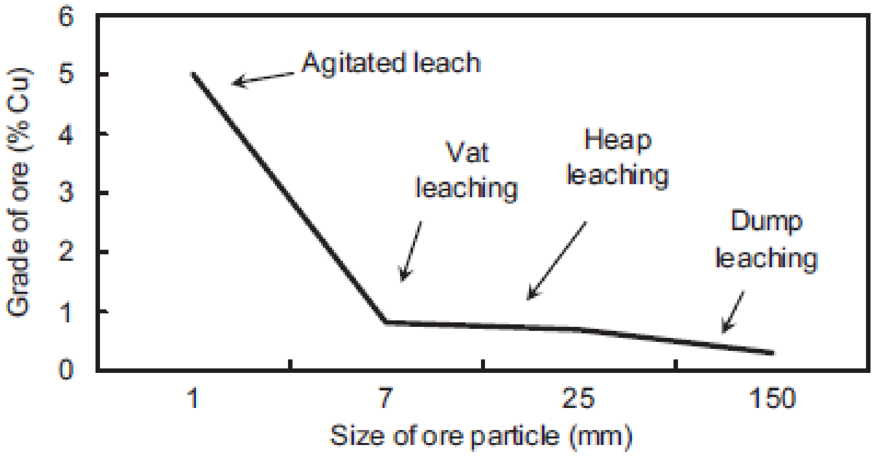


Figure 1: Criteria for selection of leaching method (Sheffel , 2002)

However, the topography of the mine site, geological location of the ore body, climatic conditions and prevailing economic conditions may also influence the choice of the leaching method. There are several leaching methods employed in hydrometallurgical extraction of copper. These include in-situ leaching, agitation leaching, vat leaching, dump and heap leaching.

a) **In-situ leaching**

This method is also referred to as ‘leaching in place’, and is mainly used in cases where the grade of the copper ores are too low to incur the expenses associated with mining and transportation. According to Habashi (1970), the ore is simply shattered and leached in its place of occurrence over long periods of time. The presence of sulphides allows for the formation of ferric sulphates under the controlled action of air and water over extended period of time. The oxidation reactions are exorthemic and the heat genearated facilitates more oxidation reactions to occur. The copper sulphate is collected through wells and pumped to the solvent extraction tanks for further processing.

b) Bacterial Leaching

This method is becoming more and more significant in the mining industry because of its environmentally friendliness and low costs. In bacterial leaching, the oxidant is Fe^{3+} which is generated by bacteria assisted reactions in the presence of atmospheric oxygen (Devenport, (2002). The reaction of Fe^{3+} and water from sulphide minerals produces Fe^{2+} and sulphuric acid which will be used to leach oxide minerals. In this way pyrite from leach dumps could become an important provider of sulphuric acid for the purpose of leaching. However the reaction of FeS_2 and O_2 to produce sulphuric acid is able to proceed without bacterial action but the rate of reaction is substantially increased by enzyme catalysts which the bacterial produces (Liu, 2008). Most sulphide waters naturally contain active bacteria and automatically provide bacterial for leaching. The major disadvantage of this method is that it is significantly slower than conventional copper processes such as smelting and very sensitive to pH change (Gupta, 2003).

c) Vat Leaching

It is a method similar to heap leaching but more contained (Mullar, 2008). It is essentially heap leaching contained within a tank, with four sides and a base, in contrast to heap leaching where the ore is placed on a base only. During vat leaching, a number of vats (compartments) can be connected in series by means of pipes and subsequantley the solutions pumped through the vats contents allow flow through the ore (Lowrie, 2002).

In comparison to the heap leach method, vat leaching attracts capital constraints (Mullar, 2008). However vat leaching has an advantage of not being affected by climate conditions if properly installed. It involves emersing the the ore in the lixiviant, thus providing better contact. It also has disadvantages associated with it such as long leaching periods (2-3 days). It is most applicable to coarse fast leaching ores, which are porous, sandy and not on material that tend to pack (for instance mica) which would cause channelling within the bed. There are also challenges encoutered due to loading and unloading of the spent ore (Lowrie, 2002).

d) **Agitation Leaching**

It involves mixing of finely ground ore and lixiviant and is agitated by pneumatic (Pachucas) or mechanical agitation means (Gupta, 2003). It is more aggressive as turbulent flow conditions are applicable and can be conducted on a large scale in continous operations by making use of reactors in series (Merwe, 2010). This method yields mostly high rates of dissolution due to large surface area generated by size reduction to fine particles. Agitation minimizes the gas-liquid interfacial area. Copper leaching rates can be effectively increased up to 60-70 ℃ and produce high copper recoveries within a few days (Davenport, 2002).

In the Zambian Copperbelt, agitation is mainly applied to leaching copper oxide (Merwe, 2010). However, Davenport (1980) pointed out that these ores are rare but leaching can occasionally be based for leaching roaster calcites. Leaching of oxides through agitation is rapid in strong sulphuric acid solution whereas, in other methods like in-situ, dump, heap leaching can take days, months and even years.

e) **Pressure Leaching**

This method involves dissolving oxygen into solution under pressure so that it oxidizes the sulphide minerals (Merwe, 2010). It can be carried out in two ways which are:

- In the absence of oxygen- ore and lixiviant are heated above the boiling point of solution.
- Oxygen present- in this case the rate of metal dissolution is dependent on the partial pressure maintained in the autoclave.

Refractory secondary copper ores are mainly dissolved using this method at eleveted temperature and pressure. Pressure leaching is more capital intensive of all the leaching methods (Gupta, 2003). The autoclaves used are constracted from very expensive material such as stainless steel, titanium or glass fibre and reinforced plastic (Merwe, 2010). However it is possible to offset these costs due to high copper recoveries which are greater than 90 % over a shorter contact times of (2-5 minutes) (Gupta, 2003).

2.3.2.1 Theory of Heap Leaching

Heap and dump leaching are very similar leaching methods in which in both cases the lixiviant trickles through a large dump or heap of ore under normal atmospheric conditions. They are all used for treating low grade ores. However, dump leaching is used for typical low grade run -of-ore (ROM) of less than 0.5 % Cu, with rocks ranging from 500 mm. Heap leaching is used for oxides and secondary sulphide ores, containing approximately 2 % Cu. The ore is crushed to uniform particle size, often agglomerated and stacked into heaps. The leaching rates of heap leaching varies with the type of ore composition. Chalcocite and oxide minerals have faster leaching rates whilst bornite and covellite have slower leaching kinetics. The heap leaching times range from 90 days – 3 years depending with the ore mineralogy. The long leaching times are due to changes in the diffusion rate caused by geo-technical characteristic of the ore, especially if the ore has clay material, micas and plate acicular particles and not intrinsic rate of leaching (Miller et al.,2008).

2.3.2.2 Industrial Leach Practice and Construction

In a typical industrial heap leach plant, the bed height is typically 7 m, having a surface area ranging from 0.01-1 km^2 (Davenport, 2002). The lixiviant is sprinkled from the top of the heap using pipes. The pregnant leach solution (PLS) is collected on a sloped impermeable surface beneath the heap and directed to the PLS pond. The heaps are in two forms which are multiple heap and dynamic on/off heap.

Multiple heaps consist of an initial lift built on an impermeable surface and when that lift has been leached, subsequent lifts are built on top of the first. Each lift is 5-8 m high and the total bed height can be as high as 200 m (Connelly, 2009). Dynamic heaps consists of a single lift usually 7-8 m high built on an impermeable surface. The leached material is renewed after leaching and replaced by a new lift of fresh ore. For ore with clay material, a dynamic leach pad is recommended (Davenport, 2002).

Advantages of Multiple Heaps

- The ore is only moved once during installation of the heaps.
- Lixiviant can flow through all lifts until leaching is moved to another area, permitting additional recovery of the valuable metals from slower leaching minerals (Mullar, 2008).

Disadvantages

- Require strong impermeable base as the heap lift can be as high as 200 m (Breitbech, 1999).
- There can be significant copper inventory locked up in the heap for extended periods of time.
- Control of internal solution head and geotechnical stability of the heap can also be challenging (Miller , 2003).

Advantages of the Dynamic Heap

- Simple to construct (Breitbech, 1999).
- Easier to control the leaching in the optimum conditions (Breitbech, 1999).
- Aeration and pregnant leach solution pipe lines can be maintained when ore is emptied from the pads (Davenport, 2002).

Disadvantage of the Dynamic Heap

- The ore must be moved twice (on and off the pad) and the associated costs of residue disposal (Davenport, 2002).

2.3.2.3 Heap leach pad base

Leach heaps are built on an impermeable base which permits complete collection of the leached copper in the pregnant leach solution and prevent penetration into the underlying environment and potential contamination of ground water. The base consists of a synthetic plastic liner or geo-membrane sheet such as high or low density polyethane (HDPE, LDPE), polyvinyl chloride PVC, 1-2 mm thick, with a large rolled clay or earth (0.1-0.5 m) beneath and 0.5 m of finely crushed and screened rock. PLS and aeration pipes are placed on top of this layer. The base is sloped to direct PLS to a collection pad. The slope should be less than 5 % (5 m drop in 100 m horizontal to avoid slippage of the heap on the plastic liner (Breitbech, 1999).

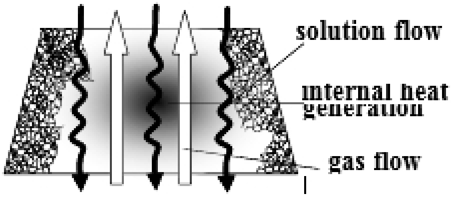
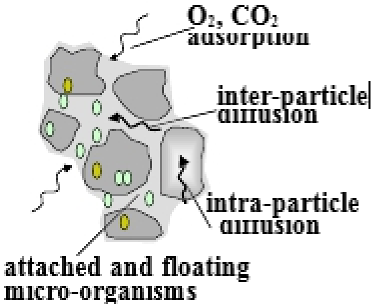
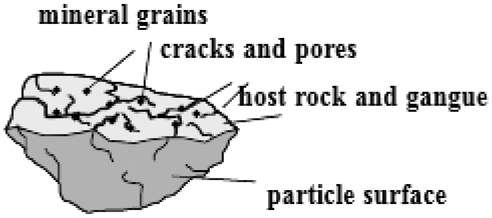
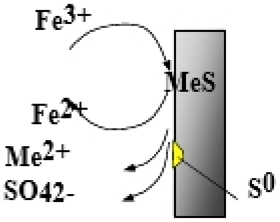
2.3.2.4 Chemistry of Heap Leaching

Heap leaching of refractory copper ore is a complex process that occurs in several sub-processes at different levels (Dixon, 2003). At macro-scale, the process is governed by mass and energy transport through the stacked mineral, including the flow of solution, gas and heat (Vilca, 2013).

At meso-scale the leaching process can be represented by particle clusters. There are three processes that will be taking place namely gaseous mass transfer which is a function of temperature, bacteria growth in the event of bio leaching and intra/inter particle diffusion. At individual particle scale, leaching is mainly governed by ore particle topology, valuable mineral distribution, host rock composition and pores.

In the case of low grade ores, the ore matrix interferes with the chemical and biological phenomena that in turn affect the leaching kinetics. Petersen (2007) observed negative effects on the kinetics of microbio oxidation of secondary copper sulphides due to high magnesium and aluminium ions in the leach solution. At micro scale level of a single grain, the metal dissolution is as a result of chemical and electrochemical reactions at grain surfaces. The table below summarises the heap leaching reaction mechanisms at various scales of the bed.

Table 2: Mechanism of Heap Leaching (Petersen, 2007)

Level	*Sub-process*	*Illustration*
Heap Scale	-Solution flow through packed bed -Gas advection -Water vapor transport -Heat Balance	
Cluster Scale	-Gas absorption -Particle diffusion -Microbial growth -Microbial attachment -Microbial oxidation	
Particle Scale	-Topological effect -Intra particle diffusion -Particle and grain size diffusion	
Grain Scale	-Ferric/ Ferrous diffusion -Mineral oxidation -Sulphur oxidation -Surface processes	

The following tables shows the various reactions of copper minerals and gangue with sulphuric acid.

Table 3: Chemical reactions of oxide copper ores and sulphuric acid (Walting, 2006)

Tenorite	$CuO + H_2SO_4 \rightarrow CuSO_4 + H_2O$
Cuprite	$Cu_2O + H_2SO_4 \rightarrow CuSO_4 + Cu + H_2O$
Copper	$Cu + Fe_2(SO_4)_3 \rightarrow CuSO_4 + 2FeSO_4$
Azurite	$Cu_2(CO_3)_2 \cdot Cu(OH)_2 + 3H_2SO_4 \rightarrow 3CuSO_4 + 2CO_2 + 4H_2O$
Malachite	$CuCO_3 \cdot Cu(OH)_2 + 2H_2SO_4 \rightarrow 2CuSO_4 + CO_2 + 3H_2O$
Chrysocolla	$CuSiO_3 \cdot 2H_2O + H_2SO_4 \rightarrow CuSO_4 + 2CuSO_4 + SiO_2 + 3H_2O$
Chalcanthine	$CuSO_4 \cdot 5H_2O \rightarrow CuSO_4 + 5H_2O$

Table 4: Chemical reactions of sulphide copper minerals with sulphuric acid (Walting, 2006)

Chalcocite	$5Cu_2S + 4Fe_2(SO_4)_3 \rightarrow 4CuSO_4 + 8FeSO_4 + Cu_6S_5$
Blaubleibender	$Cu_6S_5 + 6Fe_2(SO_4)_3 \rightarrow 6CuSO_4 + 12FeSO_4 + 5S^0$
Covellite	$CuS + Fe_2(SO_4)_3 \rightarrow CuSO_4 + 2FeSO_4 + S^0$
Chalcopyrite	$CuFeS_2 + 2Fe_2(SO_4)_3 \rightarrow CuSO_4 + 5FeSO_4 + 2S^0$

Table 5: Chemical reactions of gangue minerals with sulphuric acid (Walting, 2006)

Pyrite	$5FeS_2 + (1-6\beta)Fe_2(SO_4)_3 \rightarrow (3-12\beta)FeSO_4 + 8\beta H_2SO_4 + (2-2\beta)S^0$
Calcite	$CaCO_3 + H_2SO_4 \rightarrow CaSO_4 \cdot 2H_2O + CO_2$
Siderite	$FeCO_3 + H_2SO_4 \rightarrow FeSO_4 + CO_2 + H_2O$
Limonite	$Fe_2O_3 \cdot 3H_2O + 3H_2SO_4 \rightarrow Fe_2(SO_4)_3 + 6H_2O$

2.3.2.5 Oxidation-Reduction Potential (ORP)

When minerals and metals are contacted with an aqueous solution, they behave differently depending on the pH of solution and temperature. The redox potential is used to explain the stability of metal and other ionic species in aqueous solution by means of an Eh- pH diagram. Each line on the Eh-pH diagram represents the condition where the actual reactants and products of a given reaction are in equilibrium. Figure 2 below illustrates an Eh-pH diagram of typical copper leaching conditions (Davenport, 2002). The Eh-pH diagram also gives insight into the role of other metal elements competing with copper in a sulphate media. From Figure 2, copper can be dissolved at pH 4, in an oxidative environment. However, in the presence of iron, the pH is limited to above pH 2, iron will precipitate. Hence leaching copper beyond pH 2 results in the precipitation of ferric ions in solution forming jarosite (Walting, 2006).

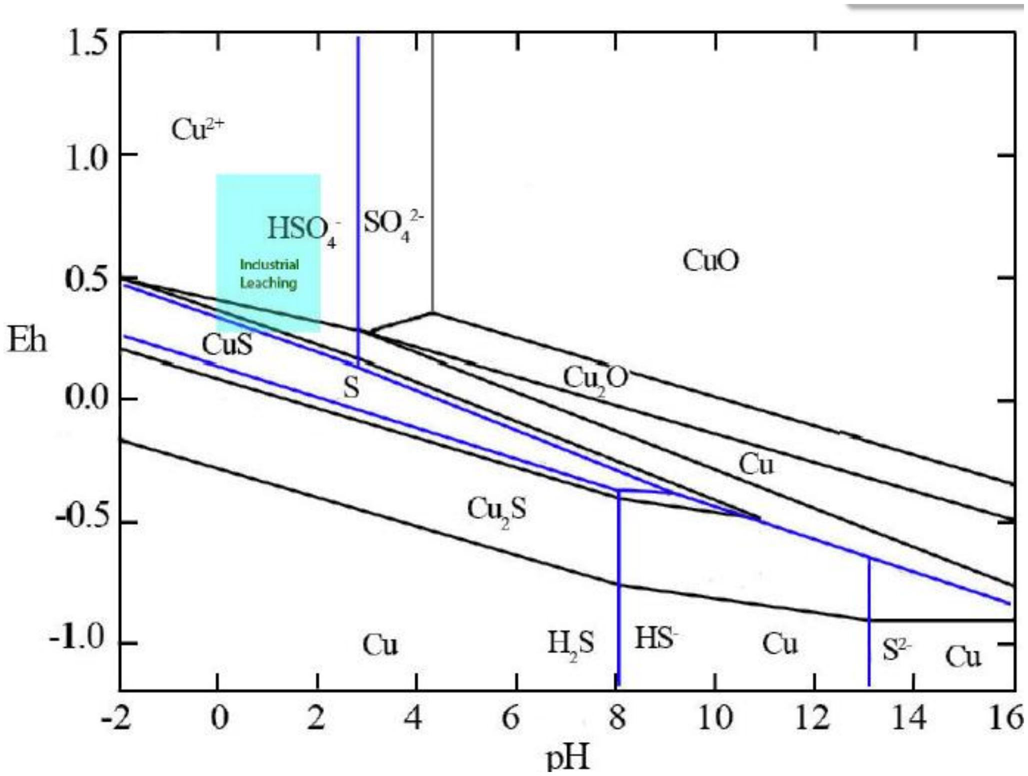


Figure 2: Pourbaix diagram for Cu-S system at 25 ℃ (Bolorundoro, 1990)

2.3.2.6 Heap leach solute transport phenomena

The solution flow through a porous media has been extensively studied over the years and many equations have been put forward to explain the movement of dissolved metal and fluids within the bed. However, these mathematical equations do not predict the permeability of a stacked ore bed but give the factors influencing the heap permeability (Vilca, 2013). Darcy represented the solution flow through a porous media by the following equation.

$$\boldsymbol{v_s = \frac{q}{A} = \frac{k}{\theta}\left(\frac{\Delta h}{\Delta l}\right)} \qquad (2.1)$$

Where:

$\boldsymbol{v_s}$ - Superficial velocity (ms^{-1})

$\Delta \boldsymbol{h}$- head pressure differential (m)

$\boldsymbol{\theta}$ - water content for unsaturated media

$\Delta \boldsymbol{l}$- heap height (m)

$\boldsymbol{k}$- specific permeability

$\boldsymbol{q/A}$-volumetric flow rate/ area unit (m)

Kozeny-Carman also came up with an equation that describes the flow of solution but on an assumption that it is laminar flow. The equation is given by:

$$\boldsymbol{v_s = \frac{q}{A} = \frac{1}{170}\frac{\Delta h}{\Delta L}\frac{\varepsilon^3}{(1-\varepsilon)^2}\frac{(\rho_s d_p)^2}{\mu}} \tag{2.2}$$

Where:

$\boldsymbol{\varepsilon}$ - superficial velocity(ms^{-1})

$\boldsymbol{\rho_s}$ -sphericity

$\boldsymbol{d_p}$ - particle size (m)

$\boldsymbol{k}$- the specific permeability is given by:

$$\boldsymbol{k \propto \frac{\varepsilon^2}{(1-\varepsilon)^2}(\rho_s d_p)^2} \tag{2.3}$$

This equation showed the importance of grain size and particle size distribution. If there are more fines in the heap bed, the bed permeability decreases due to the interstitial spaces which would have been filled by fine particles. Hence, minimizing fine particles will increase porosity of the bed. Thus to reduce the amount of fines, agglomeration is recommended. The research done by Yin (2015) showed that during heap leaching, pore size and path within the bed may change with time due to chemical precipitation and dissolution of minerals as well as weathering of particles.

2.3.2.7 Variables affecting Copper dissolution in Heap leaching

a) Mineral Liberation and Size Reduction

Generally, in every mineral extraction set up, crushing and grinding is done to liberate the valuable minerals from the gangue matrix. This adversely affects leaching as it is dependent on the interaction of the solid, which contains the targeted metal, the associated gangue species, the leach solution as well as the gas phase. In the case of heap leaching, the particles have a certain maximum size but are very small compared to dump leaching. The consequence of working with small particle size is the low permeability of the bed. The optimum particle size is determined experimentally from column tests. The particle sizes ranges from 10-40 mm (Vilca, 2013). The permeability of the bed is a critical factor for leach solution mobility. The permeability depends on particle size, grain size distribution and stacking methods. However, the permeability of the heap changes during the leaching period depending on the operating condition such as flow rate and irrigation method being employed.

b) Reagent Consumption

The main reagent consumed in copper heap leaching is sulphuric acid. According to Baum (1999), acid consumption represents 10-25 % of the operating costs of the heap leach process. Acid consumption is considered as the main controlling economic factor in the heap leaching of oxide ores (Rich, 2008). Acid consumption during copper leaching is a function of acid consuming minerals in the ore and their leaching properties as well as their association with the host rock (Plumb, 2007). In all ore leaching processes the gangue (undesired) minerals also consume lixiviant along with copper. Certain gangue minerals like calcite are very reactive to acid and will dissolve completely even in very dilute sulphuric acid solutions. Ores that contain a significant proportion of such minerals will be clearly unsuitable for acidic heap leaching (Ghorbani, 2015). However, more commonly silicate minerals constitute the most significant acid consuming component of the ore. The silicate minerals react incompletely with sulphuric acid, and with several of them the extent of acid-gangue reaction is a function of the acid strength. The moderate reactivity to acid of these gangue minerals limits their acid consumption sufficiently such that acid heap leaching of the ore can be economically viable with a relatively low valuable metal grade

c) Heap Leaching Flow rate (Irrigation rate)

The optimum flow rate in heap leach operations depends on mineral type as well as the permeability of the heap. The flow rate is crucial in ensuring that the reagents are supplied to the targeted areas within such as the oxygen which in the case of sulphides is an oxidant. In a bid to facilitate efficient oxidative environments for heaps, several methods have been proposed to improve the bed aeration. These include low pressure air blowers in the bottom of the heap (Walting, 2006), while others use naturally occurring bacterial oxidation.

d) Heap bed height

Studies carried out by Lizama (2005) established an inverse relationship between heap height and the copper leaching efficiency. The work done by Padalli (2008) confirmed this relationship and it was observed that if the heap height of the heap is lowered, the leaching efficiency increases. In contrast, the concentration of the copper in the pregnant leach solution increases as the bed height is increases.

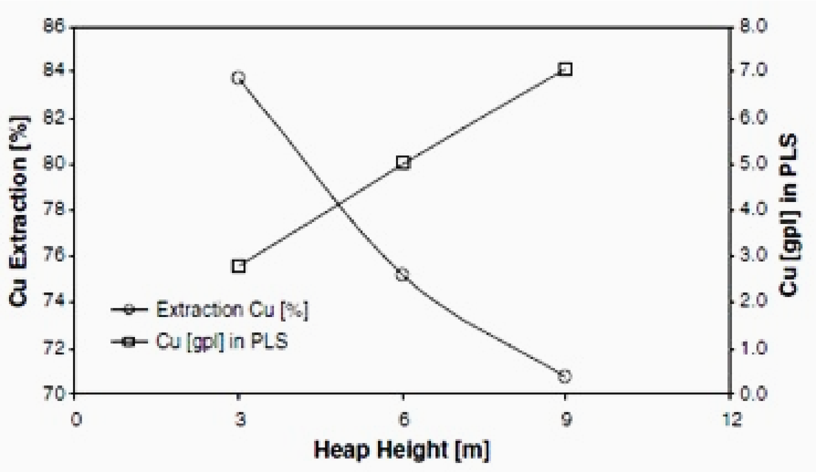


Figure 3: Leaching efficiency from the leaching stage and copper concentration in the PLS, as a function of the heap height (Padalli, 2008)

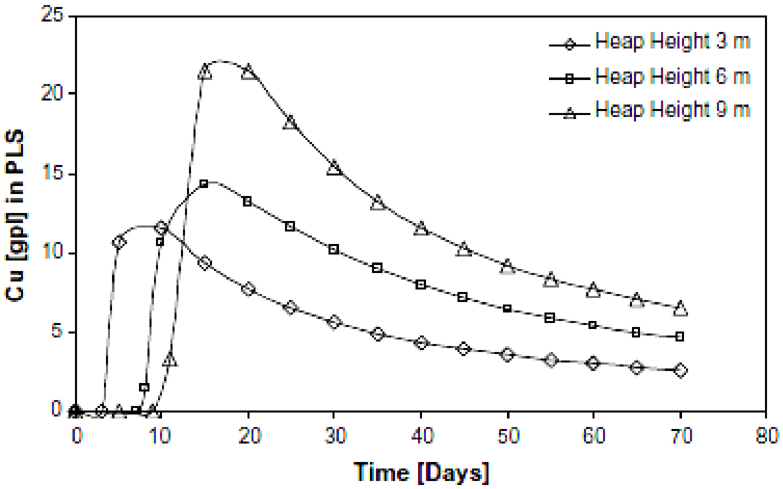


Figure 4: Concentration of copper in the PLS solution for different bed height as a function of time (Padalli, 2008)

Figure 3 and 4 show the results obtained by Padalli et al. (2008). The results showed a linear increase of maximum copper concentration with a linear increase in bed height. The leaching period was also observed to increase as the bed height increases. When designing the parameters for a heap leach operation, the heap height and time of leaching are variables which have great effects on determining the optimum leaching circuits (Padalli, 2008). However optimum conditions do not necessarily represent the maximum leaching efficiency, even observing that the copper concentration of the pregnant leach solution is not a good indicator of maximum copper extraction (Petersen, 2007).

2.3.3 Pretreatment of Ore for Heap Leaching

Ores used for heap leaching exhibit different characteristics. Heap leaching, being a process that is diffusion driven, the permeability of the heap is a crucial factor that guarantees good results. Therefore, the ore material to be subjected to heap leaching must be of the right particle size. From the work done by consultants (Heinen, 1979; Garcia and Jorgensen, 1997; Mc Clelland, 1986) revealed that for ore material to be subjected to heap leaching, the proportion of fines smaller than 74 µm, determines if agglomeration is needed. According to Garcia and Jorgensen (1997), the composition 74 µm of fines should be about 15 % of the ore composition. Therefore agglomeration of the ore before heap leaching will reduce fine particle migration and improves permeability in the heap.

In most heap leach operations, acid curing is also another ore pretreatment process that is carried out before subjecting the ore to heap leaching. Sulphiric acid curing involves addition of concentrated Sulphiric acid onto crushed ore material before heap leaching, usually added during agglomeration. The acid will react with the gangue and minerals to form sulphate crystals prior to irrigation of the heap, which easily dissolve in an acidic solution. Studies carried out by Johnson (1977) revealed that acid curing has numerous benefits which include; genaration of metal salts crystals that enhance bed permeability, crack generation which enhances diffusion of solutes within the ore particles, accelerate leaching ,dehydration and carbon dioxide evolution.

2.3.3.1 Agglomeration

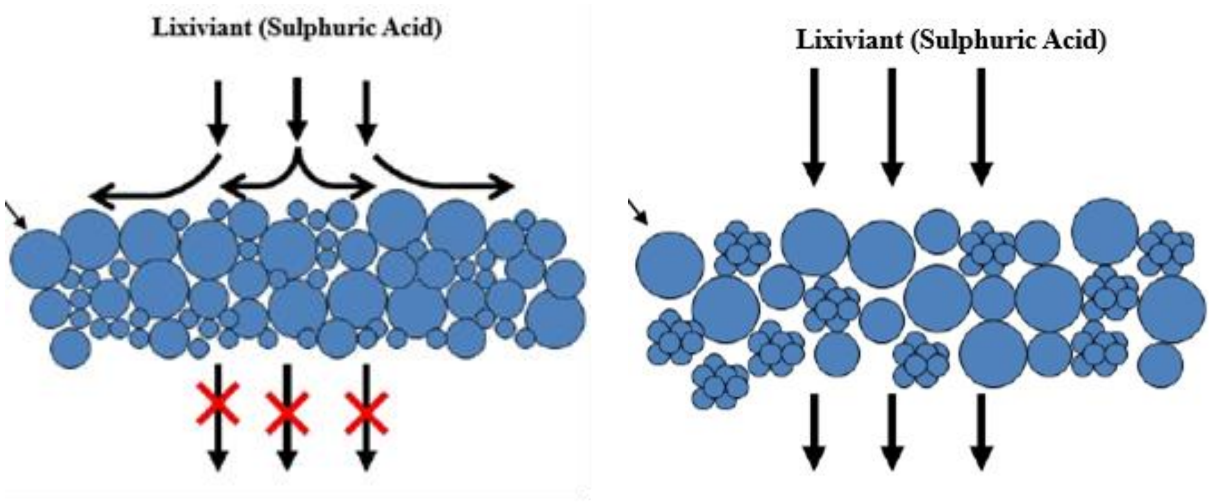
Agglomeration is a process in which fine particles are bound together to produce coarser particles. The particles get attached to each other due to capillary forces. It was started more than 100 years ago, used at 60 inclined chutes to agglomerate ore in 1905 (Boaffard, 2005). A lot of other literature has been produced and have proven the fundamental benefits of agglomeration to heap leaching. It is a method used to improve the percolation within the heap when ores contain a considerable amount of fine particles. The fine particles would migrate down during irrigation with raffinate and eventually block the pores within the heap. The benefits of agglomeration can be categorized into three groups which are;

- Improvement of physical structure (Hill, 2013).
- Improvements in leaching kinetics
- Reduced environmental impacts.

In terms of improvement of the heap physical structure, agglomeration reduces fine migration within the heap. It also improves the porosity of the heap leach (Phifer, 1988). In heap leaching, bulk density is used as an indicator of porosity. Miller (2012) reported that the bulk density of the non-agglomerated ore is often 1.15-1.3 t/m^3 at the surface to 2-2.1 t/m^3 at 4 m below. After agglomeration, it can be 0.88 t/m^3 (Butwell, 1990).

Agglomeration reduces the leaching residence time, improves the recovery and allows leaching of fines regarded as not economically viable to process using conventional methods. Studies by DeMull (1984) concluded that the leaching efficiency improved for agglomerated than for unagglomerated ore. Agglomerated ore had a higher recovery with the shortest possible time and lower reagent consumption as compared to un- agglomerated ore. The environmental benefits of agglomeration is that good agglomerates quality reduces dust emissions.

For agglomeration to yield the best results in heap leach technology, there are a lot of crucial parameters that need to be optimized in order to produce the best pellets. These parameters include moisture content, binder type, binder dosage, curing time, agglomeration equipment, agglomeration speed and residence time. Heinam (1979) carried out a comparative test of non-agglomerated ore and agglomerated ore and found that the percolation rate can be improved by a factor of 10-100 times by agglomeration (Mc Clelland et all,1986). This improvement results in 33 % reduction in leach time compared to un-agglomerated ore as well as reagent consumption. (Vethosodsakda, 2012). The results from a test work carried out by Chamberlin (1981) showed that agglomeration enhanced the percolation of lixiviant. The diagram below shows how unagglomerated ore impedes percolation and how agglomeration improves percolation.



a. Unagglomerated Ore **b. Agglomerated Ore**

Figure 5: Effects of agglomeration to lixiviant percolation (Vethosodsakda, 2012)

Figure 5a, shows how fine are blocking the lixiviant from percolating down the ore bed. In Figure 5b, there are more poles that enable better percolation of the within the ore bed. There have been studies carried out on the effects of various agglomeration parameters on the agglomeration process. Moisture content plays a critical in agglomerate formation and stability in the heap. Excessive moisture leads to unstable agglomerates and mud formation (Vethosodsakda, 2012).

2.3.3.2 Mechanism of Agglomeration

Agglomeration occurs in numerous mechanisms, Rumpf (1962), catigorised these mechanisms into six ways that is: solid bridges mechanism, adhersion and cohesion mechanism, interfacial forces and capillary pressure, attraction forces between solid particle (Van der Waals forces) and Interlocking bonds.

a) Solid Bridges

In this mechanism, molecules diffuse from ore particle to another at the contact point as temperature increases, resulting in solid bridges. Solid bridges can be formed by sintering, chemical reaction, hardening of binders, partial smelting, crystalisation of substances and deposition of colloidal particles (Adirek, 2012).

b) Adhersion and Cohesion Mechanism

In this mechanism, agglomeration takes place between the solid-liquid interfaces when highly viscous binder is applied. Finely divided particles attract free atoms or molecules from the neighbouring atmosphere to their surface, forming a thin adsorption layer. The thin adsorption layer formed is not freely movable, however, they contact and penetrate each other (Peitsch, 1997). For layers less than 3 µm, the molecular forces are high enough to deform at the contact point, leading to large contact areas and higher strength of bonds between the two particles.

c) Interfacial Forces and Capillary Pressure

This mechanism is most common in wet agglomeration process (Adirek, 2012). In this mechanism, agglomerates are bonded by liquid bridges at the coordinated points between particles. These liquid bridges can be developed by free water or by capillary condensation. If the entire process between the particles is filled with liquid at the pore ends on the surface of the systems , a negative capillary pressure develop in the interior, causing agglomeration. As the liquid consumes more pore space between the particle systems, the contrubution of interfacial forces to the total strength of the system decrease with the capillary pressure contribution increasing.

d) Attraction Forces Between Solid Particles

This mechanism is attributed to the Van Der Waals forces. The forces are higher as the proximity between particles increases and in smaller particles it is higher as compared to larger particles due to surface roughness of the particles. The Van der Waals forces is attractive or repulsive forces between molecures bonded by interparticle friction (Sherrington, 1981).

e) Interlocking Bonds

It is as a result of particles interlocking with each other in the disperse system due to their shape. For this to occur, a compression or shear force must always act on the system to enhance the interlocking. The competence or strength of these agglomerates is dependent upon the type of interaction and material characteristics.

2.3.3.3 Equipment for Agglomeration

There are various techniques and equipment used in the agglomeration process , some of which are; belt agglomeration technique, drum agglomeration, disc agglomeration and stockpile agglomeration.

a) Belt Agglomeration

This technique is suitable for ore of less than 15 wt % of 104 µm fines (Boaffard, 2005). The binder is added to the ore whilst transfering in a series of inclined (about 15 ℃) ,conveyer belts. The moisture is added at transfer points and agglomeration occurs during the handling associated with the transfer of ore from one conveyor to another. The typical belt speed is 1.25-1.5 ms^{-1} (Chamberlin, 1981). The number of transfer points depends on the amount of fines in the ore. The amount of moisture added is crucial as excess leads to spillages and belt damage. Inadequate moisture leads to dusting and inadequate agglomeration. Agglomeration occurs when ore drops to high velocity conveyer belt. The high angle forces the agglomerates to roll to the bottom transfer belt while the fines and bed agglomerates move up to the top due to forward momentum of the belt.

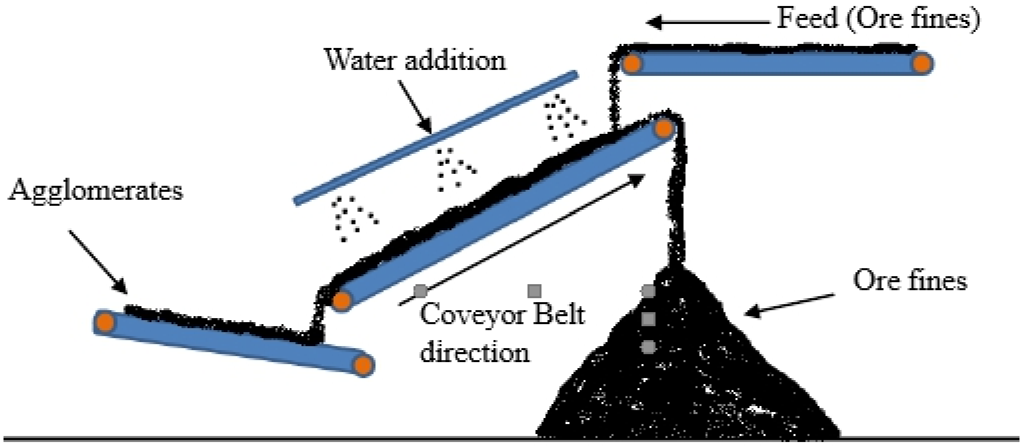


Figure 6: Belt agglomeration technique (Vethosodsakda, 2012)

b) Drum Agglomeration

This equipment is suitable for ore containing 15 wt % clay or fines. The ore and binder are fed through an inclined rotating drum and moisture is added through nozzles located at the first two thirds of the drum length. The interior walls of the drum are covered with rubber to minimize corrosion and prevent sticking of ore to the drum's walls. Agglomeration is achieved through cascading of particles inside the drum. For good pellets to be formed, there are parameters which must be critically monitored that is moisture content, drum rotation speed and residence time. Moisture in the agglomeration enables formation of pellets and excess moisture leads to mudding formation in the drum. Low moisture content results in formation poor agglomerate size and quality due to inadequate moisture for particle adhersion. The drum speed plays a crucial role in the growth of pellets. At low agglomeration speeds, agglomerates start to roll instead of cascading thus leading to fragmentation which impedes the growth of agglomerates. Excess speed leads to sticking of particles to the drum walls due to centrifugal forces. The typical agglomeration speeds used range from 30-50 % of the critical speed. The critical speed is the minimum speed required to keep a particle on the drum's walls during rotation.. The critical speed (Vethosodsakda, 2012) is given by:

$$N_c = \sqrt{\frac{g\sin\theta}{2\pi D}} \approx \frac{42.3}{\sqrt{D}} \qquad (2.4)$$

Where:

N_c – critical speed (rpm)

D – drum diameter (m)

g - acceleration due to gravity (ms^{-2})

θ – angle of the drum from the vertical (usually 80-90^{o})

The residence time is essential to the process in the sense that longer residence time leads to breakages of the agglomerates, wasted time and energy. The following diagram shows a drum agglomerator and its interior.

Figure 7: Agglomeration drum (Picture taken at Nchanga Mine Tailings Leach Plant)

c) Disk Agglomeration

This type of equpiment is used mainly for agriculture and chemical industries. Disk agglomerators have low throughputs thus unsuitable for heap leaching. Agglomeration is achieved when solids and moisture are continuously added to rotating, tilted disks or pans with rims. The feed rate affects the rotation of the agglomeration disk and the nozzle location have a significant effect on agglomerate quality (Holley, 1979). The rotation speed of the disk is usually 30-50 rpm depending on the disk diameter and slop. Retention time is between 60-120 s (Boaffard, 2005). This can be increased or lowered by the solid feed rate or disk angle. Disc agglomerators produce more agglomerates of unifom size as compared to other techniques. The diagram below, Figure 8 shows a disk pelletiser/ agglomerator.

Figure 8: Disk agglomerator (Mullar, 2008)

d) Stockpile Replacement

Mainly applied to ores with fairly course particles with little clay or fines. In this technique, ore with binder is transferred onto a conveyer belt and dropped down to a stockpile. At drop point moisture is added to the ores and agglomeration occurs due to the cascading of the material down the slope. A dozer is used to move the agglomerates up and down to help particle agglomeration.

2.3.3.4 Pellets/ Agglomerates Growth Stages

Bouffard (2005), categorized the stages of pellets growth into four stages, these are:

a) ***Wetting Stages***- When moisture is added to the ore, the coated particles start to act as the nuclei for agglomerates.

b) ***Growth Stage***- The liquid film forms a liquid bridge with particles and leads to attachment of new particle layers. Therefore insufficient moisture content inhibits the growth stage.

c) ***Consolidation Stage***- The agglomerates become more packed and porosity is reduced due to the compaction pressure and agitation intensity applied to the pellets/agglomerates.

d) ***Breaking Stage***- Due to excessive compaction, the formed pellets start to break into fragments. The fragments can also grow into large pellets if there is adequate moisture. This results in formation of agglomerates of almost uniform size.

2.3.3.5 Agglomerates Quality Evaluation Techniques

The quality of agglomerates can be defined in terms of size distribution, moisture content, agglomerate strength on stability and agglomerate internal porosity (Boaffard, 2005). In order to produce quality, competent agglomerates, numerous techniques have been developed to evaluate agglomerates.

a) Glove Test

This test is also known as visual inspection and is commomly used by industrial operators. It involves visual inspection of the agglomerates by the operator. The effectiveness of this method is hinged on the experience of the operator. Therefore, it does not give comprehensive information pertaining to the quality of the agglomerates (Verlade, 2005).

b) **Agglomerate Size Distribution**

In this technique, both dry and moist agglomerates can be screened by sieving to obtain the agglomerate size distribution. It is carried out in a laboratory and during field operations, periodic samples are cut and taken for analysis. The required agglomerate size distrubution varies from ore to ore. However through research, there have been some recomendations that have been put forward. Lipiec (1998) suggested agglomeration should remove fines and produce agglomerates of uniform sizes. Chamberin (1988) also recommended that for effective agglomeration, particles smaller than 104 µm must not remain in the agglomerated material.

c) Attrition Test

This method was introduced by Southwood (1985) with the objective of simulating the abrasion occuring during agglomerate handling. The test involves feeding the agglomerates in a 200 mm diameter pan which is rigorously shaken for 20 mins. Particles smaller than 9.5 mm are collected after the test. The amount of the 9.5 mm particle as a percentage is used to compare other experimental trials.

d) **Drop Test**

This test was developed to simulate the dropping of agglomerates from a conveyer belt during stacking. The standard height used for this test is 1.83 m onto a steel plate. The number of drops taken for the agglomerate to break is recorded (Southwood, 1985). For this test to have valid results, the agglomerates used must be of uniform size. From the research done by Southwood (1985), it was found that the dropping numbers of the agglomerates have an inverse relationship to the size of agglomerates.

e) Compact Test

In this test, the agglomerates at different bed heights are subjected to compaction loading. The objective of this test is to investigate the ability of the agglomerates to withstand compaction forces when stacked in the heap. The minimum load the agglomerates can withstand can be calculated by:

$$M = \rho h A \tag{2.5}$$

Where:

M-Agglomerate minimum withstand load (N)

ρ- The bulk density of the heap (Kgm^{-3})

h- Bed height (m)

A- The cross sectional area of the agglomerate (m^2)

f) Soak Test

This is the most commonly method used to analyse agglomerates strength by submerging them into a solution usually water for a period of time. Chamberin (1988), suggested that good agglomerates are the ones that do not disintegrate for many hours when submerged in water. In addition Milligan and Engelhardt (1983) also measured the amount of fines generated when dipping the agglomerates in water for a period of time. Lewandoski et al (2009) carried the soak test in an acid media of 6 g/l of sulphuric acid for 30 minutes and measured the amount of fines generated. The amount of fines generated acts as a measure of the competence of the agglomerates. The fines migration was calculated as:

$$\text{Fines migration} = \frac{Weight\ of\ ore\ migrated\ out\ of\ the\ sample}{Total\ weight\ of-10mesh\ fines\ available\ in\ the\ sample} \qquad \textbf{(2.6)}$$

g) Electrical Conductivity

This method is mainly used to assess the moisture content of the agglomerates. It is based on the drastic changes of the electrical conductivity which will occur after the moisture is added to ore to produce a film around the particle. The moisture is key for particle agglomeration through liquid bridging which at the same time forms an electrical connection between the particles. Increase in moisture results in very small increase in the electrical conductivity values (Verlade, 2005).

2.3.4 Literature Review Summary

The cupriferous mica is found in virtually all Zambian Copper belt oxide ore bodies and are low grade copper ores from which copper is difficult to extract under conventional or direct leaching methods (Notebaart, 1988). Extensive work has been done on heap leaching of oxides and secondary copper sulphides. Several approaches to leaching have been suggested such as bio leaching for secondary copper sulphides (Murr, 1980).

Pinkley (1970) proposed the use of treatment of refractory copper ore (TORCO) process to leach the Chingola refractory copper ore. The TORCO process was developed specifically for the Chingola refractory ore (Notebaart, 1988). This method involved heating the copper refractory ore in a fluidized bed at about 800 °C and mixing it with coal and salt. The leaching efficiency attained using the TORCO process was in the range of 20-40 %. However the leaching efficiency attained could not offset the operational costs of the process. Whyte (2001) proposed the use of high pressure temperature leaching method (HPAL) for leaching of the Chingola refractory ore. The leaching efficiency obtained using this method in the experiments carried out was about 80.7 %. The conclusions drawn by Whyte et al (2001) were that leaching of the Chingola refractory ore using the HPAL was technically sound to KCM but came at a significantly high cost of investment. Merwe (2010) proposed the use of agitation leaching to extract copper from Chingola refractory ore in a two step mechanism. The first step involved the pre-oxidation of the insoluble minerals by enhanced absorption of oxygen into the slurry feed stream in a pre-oxidation chamber (POC). Consequantley, no acid was added to the POC and leaching of the oxidized minerals took place in the second stage of the process. The maximum leaching efficiency obtained using this method was 65 %. Moreover, Merwe (2010) reported high gangue acid consumption of about 30 kg/t. This has left heap leaching as a more economically attractive copper extraction route to be investigated. The amenability of Mimbula ore to heap leaching is yet to be investigated. Hence this research seeks to investigate the conditions under which copper dissolution can be enhanced by employing heap leaching.

CHAPTER THREE: MATERIALS AND METHODS

3.1 Introduction

This chapter describes the experimental procedure and apparatus employed in the research. It also gives the source and types of mineral samples used, leaching experiments and chemical analyses that were carried out.

3.2 Materials

About 1.2 tonnes of Mimbula copper refractory ore was sampled from stockpile belonging to KCM using the grid method and screened on a 25 mm sieve. The +25 mm ore material was crushed to -25 mm and then blended with the initial -25 mm heap of ore. After blending, homogenization of the material was done.

For all the column leaching experiments carried out, the raffinate drawn from the KCM solvent extraction plant was used as the lixiviant. Concentrated sulphuric acid (98 %) was used for agglomeration experiments to cure the ore. *Kempel 8201* binder was employed as a binder.

3.3 Experimental Procedure

3.3.1 Feed Characterization

Using coning and quartering sampling technique, a representative sample of 20 kg was sampled from the bulk ore sample for the following: *Mineralogical composition, chemical composition and particle size distribution*. The chemical analysis was carried out for the following; total copper *(% TCu*), acid soluble copper (*% AsCu*), silica (*% SiO_2*) and iron (*% TFe)* .Using a Riffle splitter the 20kg sample was cut into two portions of which the other half was subjected to particle size analysis. The screen sizes used for particle size analysis were as follows: *25.4 mm; 19 mm; 12.5 mm; 4.75 mm; 2 mm; 300 µm; 150 µm and 75µm*. 10kg sample of ore was then tested for moisture content after which representative samples were drawn for chemical and mineralogical analysis.

For moisture content determination, a 10 kg sample was dried for 24 hours at 200 ^{o}C, the mass after drying was measured and the moisture content for the ore was calculated as follows:

$$\textbf{Moisture content (\%)} = \left(\frac{Mass(wet)-Mass(dry)}{Mass(dry)}\right) 100 \qquad (3.1)$$

Where:

$Mass_{(wet)}$ - Mass of sample before drying (Kg)

$Mass_{(dry)}$ - Mass of sample after drying (Kg)

3.3.2 Agglomeration Experiments

Agglomeration tests were carried out in a 300 mm diameter by 910 mm length agglomeration drum. The calculated critical speed for the drum was found to be 72.2 rpm, calculated using equation *(2.3)* as shown in the calculation below. Agglomeration was carried out at various conditions to determine the optimum moisture content, binder dosage, residence time and agglomeration speed.

***Calculation of agglomeration drum critical speed*.**

From; $$Nc = \sqrt{\frac{g \sin\theta}{2\pi D}} \approx \frac{42.3}{\sqrt{D}}$$

Where: *D- 0.3m*

$$\Rightarrow \quad Nc = \frac{42.3}{\sqrt{0.3}}$$

$$\Rightarrow \quad Nc = 72.2 rpm$$

3.3.2.1 General Agglomeration Procedure

About 60 kg ore sample mass was sampled from the 1.2 tonnes homogenized ore using corning and quartering sampling technique, this small fraction of ore was then split into thirty 2 kg lots using the grid method. To each 2 kg sample charged into the agglomerator, water was added first to facilitate the formation of pellets, then the binder was added for growth and stability of pellets and finally acid was added for curing purpose. The formed agglomerates were allowed to dry in open air for 24 hours before being subjected for agglomerate quality assessment tests. The following are the initial agglomeration conditions used:

- 2 kg ore /charge (Drum pelletizer)
- Terminal moisture of 14 % (polymer binder and water) into drum pelletizer.
- Residence time of agglomeration of three minutes.
- Acid curing at 30 kg/t (concentrated sulphuric acid) dosage.

Table 6 below shows the experimental conditions used for investigating the effects of moisture content, binder dosage, residence time and agglomeration speed on the strength of the agglomerates. For every test carried out, three trials were done in order to minimize experimental errors and to validate the data.

Table 6: Agglomeration experiments conditions

Test number	**Parameter**	**Variations**	**Fixed Parameters**
1	Moisture Content	12, 13, 14, 15, 16 (%)	-Binder dosage -Residence time -Agglomeration speed
2	Binder Dosage	0,100,200,300,400,500,600 and 800 (gpt)	-Moisture Content -Residence time -Agglomeration speed
3	Agglomeration residence time	2,3, 4, 5 and 6 (minutes)	-Moisture Content -Binder dosage -Agglomerating speed
4	Agglomeration Speed	10, 20, 30, 40 and 50(rpm)	-Moisture Content -Residence time -Binder dosage

3.3.2.2 Agglomerates quality tests

For all the agglomeration experiments carried out visual inspection/glove test, agglomeration size distribution and soak tests were performed to assess the quality of agglomerates at various experimental conditions. 1 kg of agglomerates sample was cut for every test, subjected to sieving for three minutes in a sieve shaker. The screen sizes used for agglomerates particle size distribution were: *25.4 mm; 12.5 mm; 4.75 mm; 2 mm; 212 µm and 75 µm.*

3.3.2.3 Soak Test Procedure

1. Using the grid method cut a sample of agglomerates and weigh.
2. Place the sample on a 2 mm sieve screen over a dish and allow the fines (-2 mm) to collect in the dish.
3. Discard the -2 mm fines from the dish and place the +2 mm agglomerates in the dish.
4. Add water to the agglomerates until they are completely immersed.
5. Record the time of soaking.
6. After thirty minutes, remove the sieve gently making sure that soaked pellets do not break in the process.
7. Filter the residual solution in the dish and dry the cake residue for 2 hours at about 200 ^{o}C.
8. Measure the mass of the cake after drying and calculate the fines migration using equation ***(3.1)***.

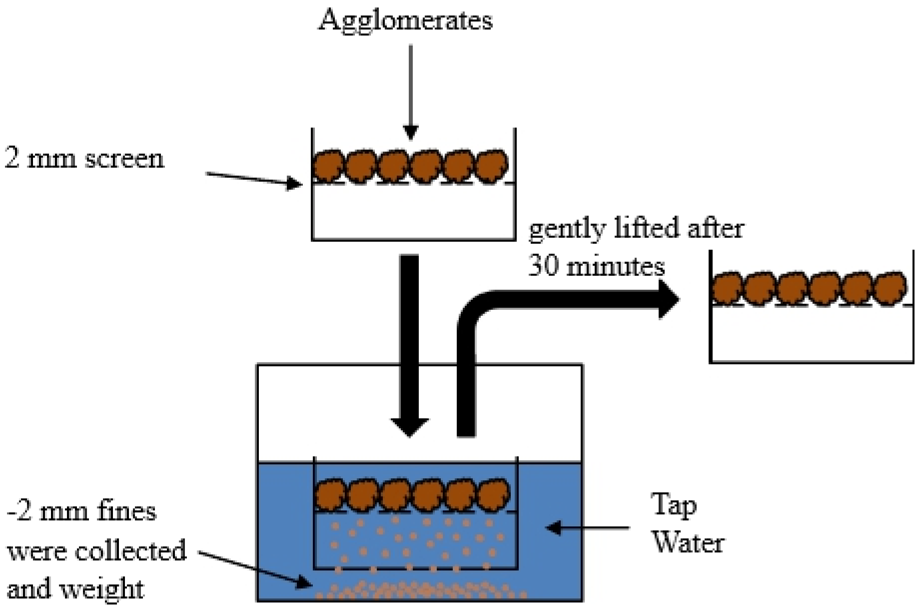


Figure 9: Schematic diagram of a Soak test

3.3.4 Column Leaching Experiments

3.3.4.1 Column Leach Set Up and Design

Four PVC leach columns *(0.150 m diameter and 6 m length)* were set up; each had a pump on top to transport a constant flow rate of lixiviant, sprinklers to evenly distribute the lixiviant over the column diameter. From the Rule of thumb, the minimum diameter of the column must be 6 times the size of the largest particles size and the length must be 6 times the diameter of the column (Chamberlin, 1981). The columns had a bucket at the bottom to collect the pregnant leach solution (PLS) which was collected daily and sent for assaying. The base of the columns had a perforated steel base and above it, a drainage layer of stones *(0.25 m)* was installed before loading the agglomerates. On top of the agglomerates were multiple layers of cotton *(0.050 m thick)* installed to evenly distribute the lixiviant over the agglomerates. Figure 10 below shows the column leaching set up.

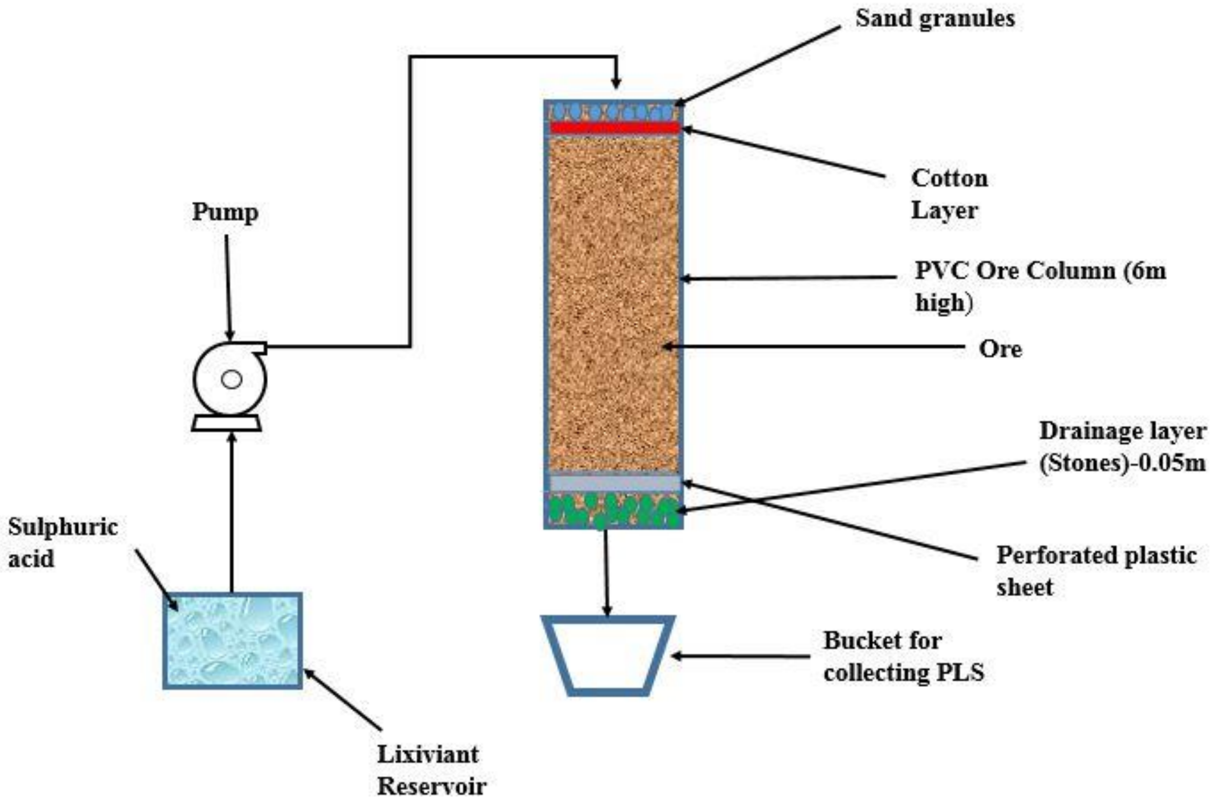


Figure 10: Schematic diagram for column leaching experiments (Lizama, 2005)

3.3.4.2 Column leaching termination procedure

Acid cured agglomerates were loaded into the columns and the bulk density of the bed before leaching commences was calculated by Equation ***(3.2)*****.** The time for commencing irrigation with raffinate was recorded as well as the time the first solution, pregnant leach solution (*PLS)*, exists the column from the bottom, known as *breakthrough period*. Samples of PLS were cut on a daily basis and assayed for copper in solution and pH. The height of the agglomerates in the column was also measured on a daily basis. When the daily copper extraction was below 0.5 g/l and constant, irrigation with raffinate was stopped. The PLS was allowed to drain for 24 hours before resuming irrigation with deionized water, collecting samples for analysis of pH. When the pH was above 6, irrigation with water was stopped and the column offloaded. Representative leach residue samples were cut from the top, middle and bottom of the column for assaying and mineralogical analysis.

Table 7: Column leaching experiments conditions

Parameter	**Column 1**	**Column 2**	**Column 3**	**Column 4**
Height (meters)	3	2	4	6
Raffinate acid (gpl)	21.5	21.5	21.5	21.5
pH	1.77	1.77	1.77	1.77
Flow rate (l/hr.m^2)	6	6	6	6
Ore type	Unagglomerated	Agglomerated	Agglomerated	Agglomerated
Column dimensions	0.15 m diameter and 6 m height			

The bulk density for the bed was calculated as follows:

$$\textbf{\textit{Bulk density (kg/l)}} = \frac{\boldsymbol{Mass\ of\ agglomerates}(\boldsymbol{kg})}{\boldsymbol{Volume\ of\ agglomerates\ in\ the\ column}} \qquad \textbf{\textit{(3.2)}}$$

CHAPTER FOUR: RESULTS AND DISCUSSION

4.1 Introduction

This chapter discusses the effects of agglomeration parameters on the agglomerates quality and the leaching behavior of Mimbula copper ore. Detailed leaching data and profile are given in Appendix C_{1-4}.

4.2 Mimbula Copper Ore Characteristics

4.2.1 Chemical Composition

The chemical analysis was carried out at Nchanga mine Analytical Services Department (ASD) and the following results were obtained. The overall feed grade was found to be 1.2 % TCu, ASCu was 0.4% and AICu was 0.8% by weight.

Table 8: Mimbula ore Feed head grade assays

SAMPLE	**ASSAYS (%)**		
	TCu	**ASCu**	**AICu**
FEED TO HEAP LEACHING	1.2	0.4	0.8

4.2.2 Mineralogical Composition

The mineralogical analysis was carried out at Nchanga mine Mineralogical Department and the following results were attained. The results are shown in Table 9 below

Table 9: Mineralogical Composition of Mimbula ore feed

Mineral	**Chalcopyrite**	**Bornite**	**Chalcocite**	**Pyrite**	**Malachite**	**Pseudo-malachite**	**Chrysocolla**	**Cupriferous Mica**
W t (%)	***0.024***	***0.346***	***0.161***	***0.026***	***0.322***	***0.095***	***0.015***	***15***
TCu (%)	***0.008***	***0.219***	***0.129***	***-***	***0.186***	***0.053***	***0.006***	***0.6***

From the mineralogical analysis, *see Table 9*, approximately *34 %* of the **TCu** is from copper sulphides, *23 %* of the **TCu** is from the copper oxides and *43 %* is from the cupriferous mica. The ASCu derived from copper oxides and micas was found to be *0.245 %***ASCu** and *0.15 %* **ASCu** which is *60 %* and *39 %* of the **ASCu** respectively.

4.2.3 Particle Size Distribution

The particle size distribution of the ore was done to ascertain the need for the agglomeration stage prior to agglomeration. From the work done by Heinen (1979), Garcia and Jorgensen (1997) and Mc Clelland (1986) concurred that for ore material to be subjected for heap leaching, the proportion of fines smaller than 74 µm, determines the need for agglomeration. According to Garcia and Jorgensen (1997), if the composition of fines of 74 µm is greater than 15 % of the ore composition, agglomeration of the ore before heap leaching is imperative. From Figure 11, approximately 22 % passes through the 75 µm sieve size and 63 % is passing the 2 mm sieve size indicating that Mimbula ore has more fine particles. Therefore agglomeration of the material before heap leaching is imperative for good bed permeability. Figure 11 below shows the feed particle size distribution of Mimbula ore.

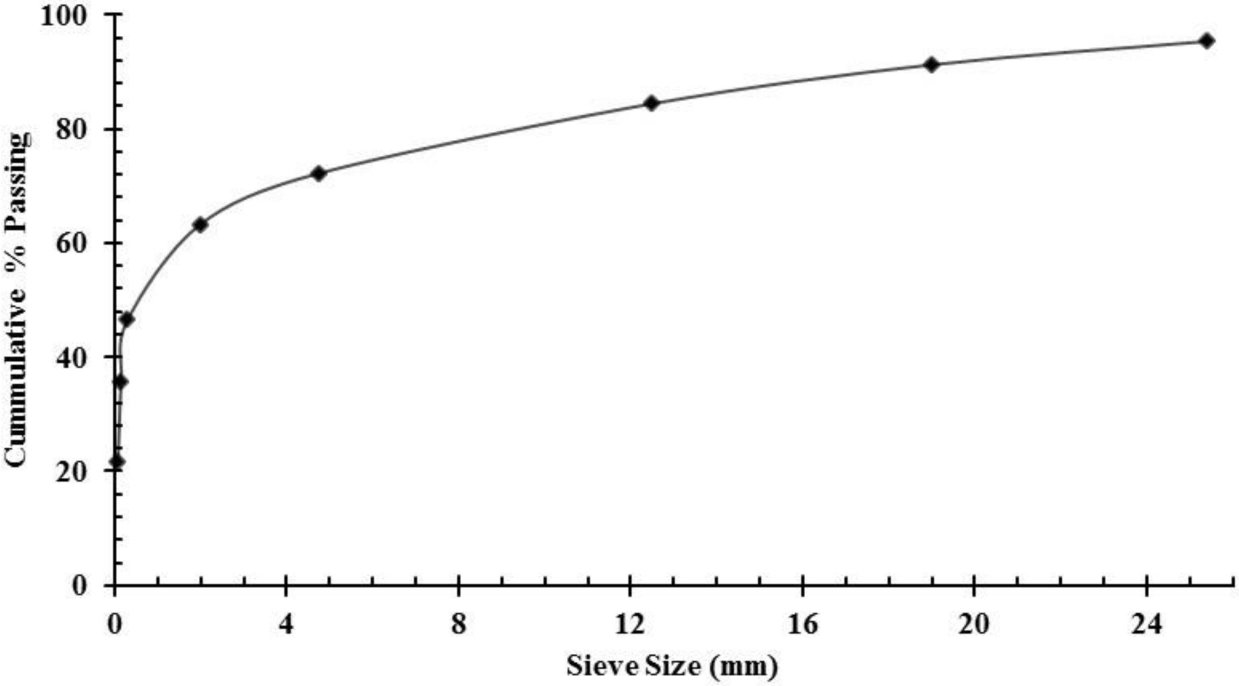


Figure 11: Particle Size distribution of Mimbula ore

4.3 Agglomerates product quality

The primary objective for agglomeration prior to heap leaching is to reduce the amount of fines. Reducing the fines enhances the bed permeability, reduces channeling and increases leaching kinetics (Boaffard, 2005). The experiments conducted in this research were to observe the effects of moisture content, binder dosage, residence time and agglomeration speed on agglomerates quality and determine the optimum agglomeration conditions for Mimbula ore. The agglomerate quality was assessed by visual inspection, agglomerate particle size distribution and soak test.

4.3.1 Effect of moisture content on agglomerate quality

The soak test was carried out, the effects of moisture content on agglomerates prepared at 12 %, 13 %, 14 %, 15 % and 16 % moisture content. A particles size analysis was also carried out to measure the extent of fine removal using the 2 mm sieve size as the benchmark. Material below 2 mm sieve size was considered to be fine. The following results were observed.

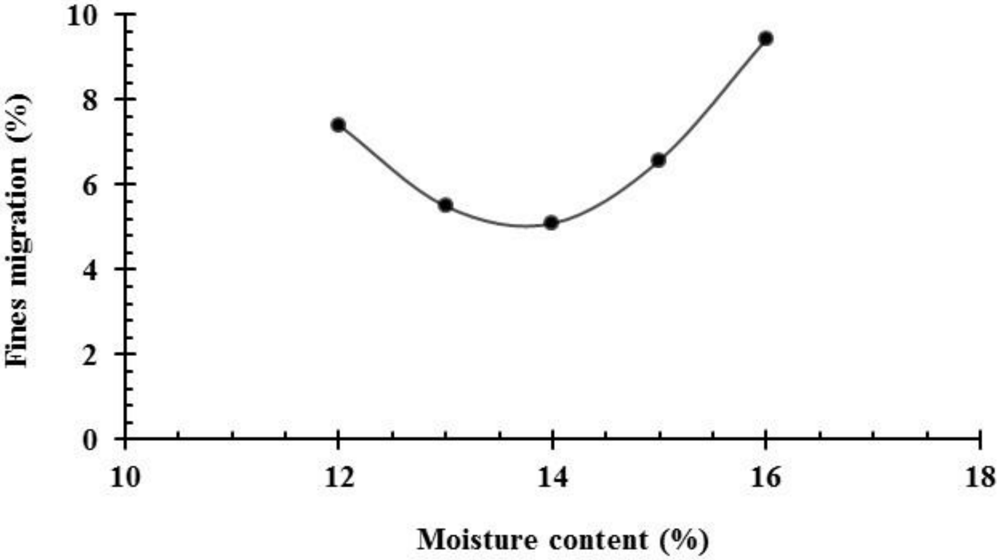


Figure 12: Effect of moisture composition on agglomerate quality

The fine migration was observed to decrease as the moisture increases up to a point where the fine migration starts to increase *(see Fig. 12)***.** From 12-13 % moisture, the high fine migration is due to insufficient moisture which results in unstable agglomerates. Above 14 %, the fine migration starts to significantly increase due to excess moisture which led to the formation of bigger and unstable pellets which are liable to deformation due to their plasticity. Figure 13 shows agglomerates prepared at 12 % and 14 % moisture content respectively.

12 % Moisture Content *14 % Moisture Content*

Figure 13: Agglomerates at 12 % and 14 % Moisture content

Figure 14 below shows the agglomerate particle size distribution of the agglomerated material at different moisture content.

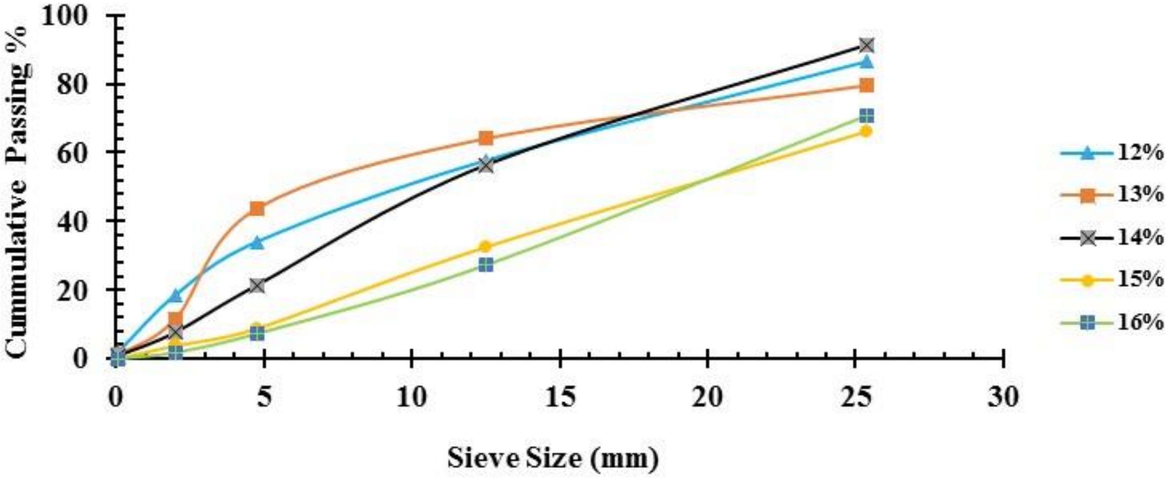


Figure 14: Agglomerates particle size distribution at different moisture content

In Figure 14 above, the size of agglomerates formed were observed to become coarser as the moisture content increases. The implication to that was as moisture content increases, the volume of the liquid in the agglomerate matrix increases. Hence an increase in the liquid composition increases the ability of liquid bridges formation and consequently increases the agglomerate size. By visual inspection and soak test results, agglomerates at 14 % were well formed and had the least fine migration indicating that they are more competent. Therefore 14 % was taken to be the optimum moisture content for agglomeration.

4.3.2 Effect of binder dosage on agglomerates quality.

The soak test provided more information on the stability. Fines migration decreased steadily with increase in binder addition until 600 gpt (see *Fig. 15*), beyond this point the decrease in fines migration was minimal. From 400 to 600 gpt fines migration reduced by about 6.0 %. However, increasing binder from 600 to 800 gpt decreased fines migration by only 0.67 %. Thus, a reasonable amount of binder is very effective, but additional binder above a useful level is not effective in reducing fines migration at various binder dosage

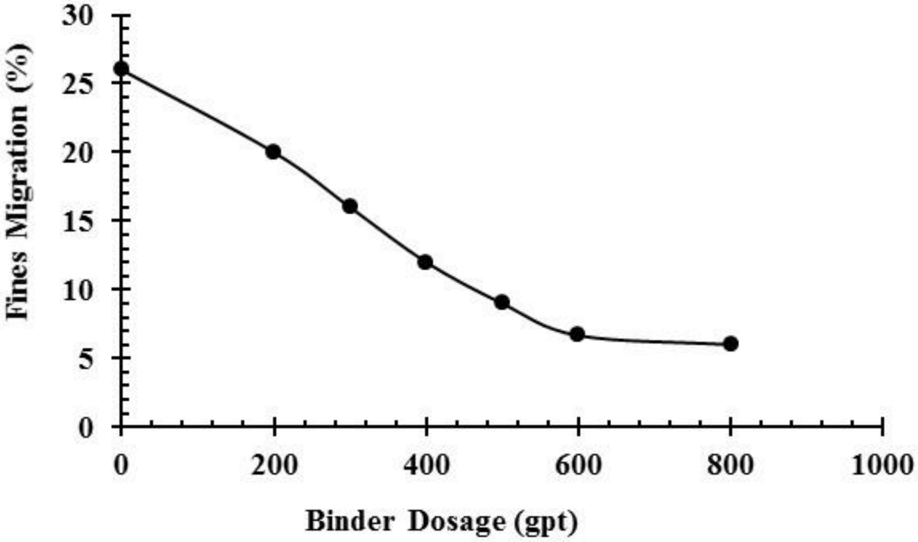


Figure 15: Effect of binder dosage on agglomerates quality (Soak test)

No significant effects of the Kempel binder on the particle size distribution of the agglomerates as the binder mainly affects the stability of agglomerates. Figure 15 shows percent cumulating passing 2 mm sieve against the binder dosage. From 0-200 gpt there was a drastic increase in agglomerate size due to the effectiveness of the binder. Beyond 200 gpt, increase in binder dosage does not have significant increase in particle size. From the results obtained, 600 gpt binder dosage was adopted as the optimum binder dosage as beyond 600 gpt, there was no significant effect on agglomerate strength, dosing beyond that would just be an additional operating cost on the process.

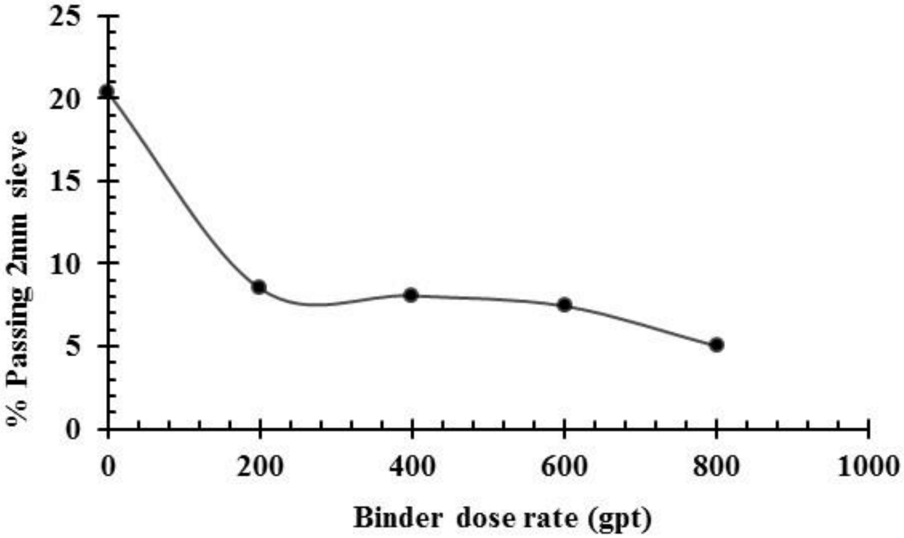


Figure 16: Effect of binder dosage on agglomerate size distribution

4.3.3 Effect of residence time on agglomerates quality

The soak test indicate that the fine migration decreased as the agglomeration time is increase and the fine migration reaches a turning point as the time exceeds 3 minutes of agglomeration (*see Fi**g.** 17*). Fines migration decreased as agglomeration time increased, however the fine migration reaches a plateau as time exceeds 3 minutes. The agglomerates gained strength after 4 minutes and this was indicated by the decrease in fines migration due to sufficient mixing of moisture, binder and ore.

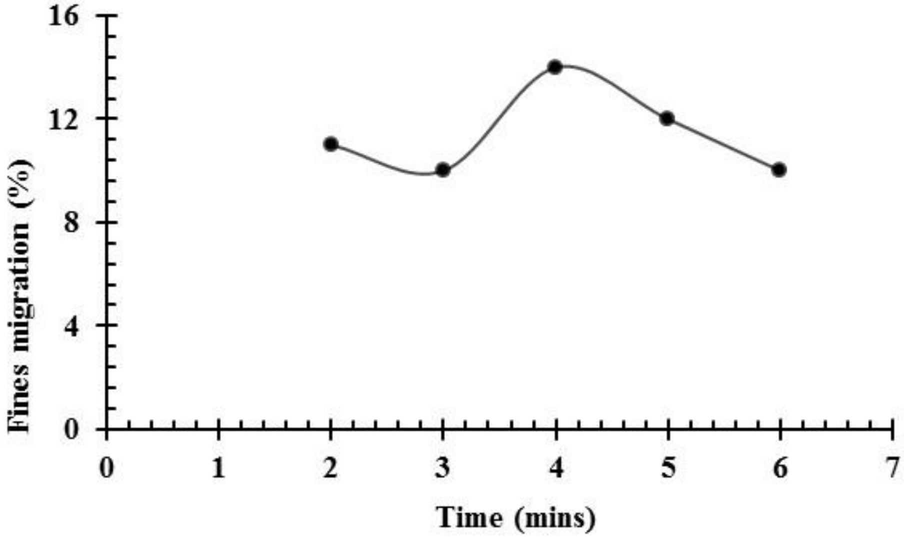


Figure 17: Effect of residence time on agglomerates quality (Soak test)

Figure 18 shows that there was an increase in particle size from 2 to 3 minutes. However, particle size decrease steadily was observed beyond 3 minutes due to attrition.

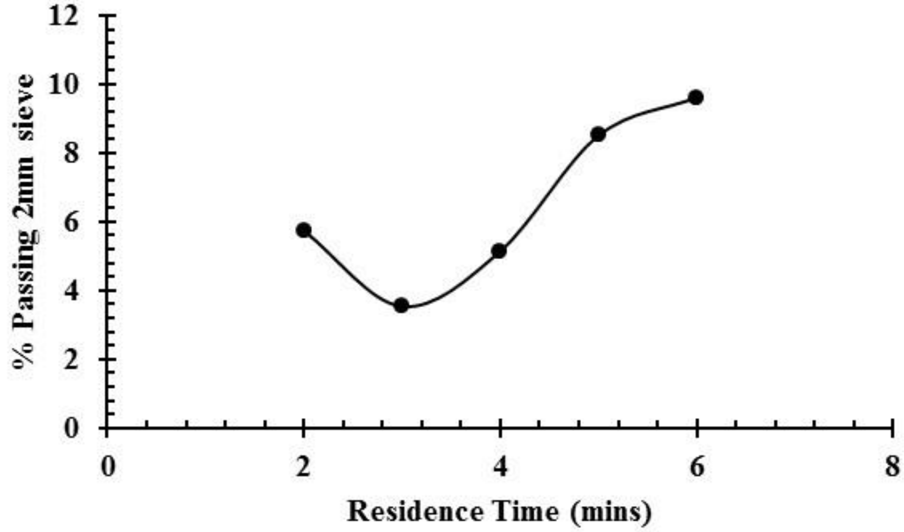


Figure 18: Particle size distribution of agglomerates at different residence times

4.3.4 Effect of agglomeration speed on agglomerates quality

The agglomeration drum speed tests were performed at 10 rpm interval from 10 to 60 rpm. The agglomeration drum speed plays a pivortal role in the growth of pellets. At low agglomeration speeds, agglomerates start to roll instead of cascading thus leading to fragmentation which impedes the growth of agglomerates. Excess speed leads to sticking of particles to the drum walls due to centrifugal forces (Vethosodsakda, 2012). Figure 19 below shows the agglomerates particle distrubution of agglomerates prepared at different agglomeration speeds.

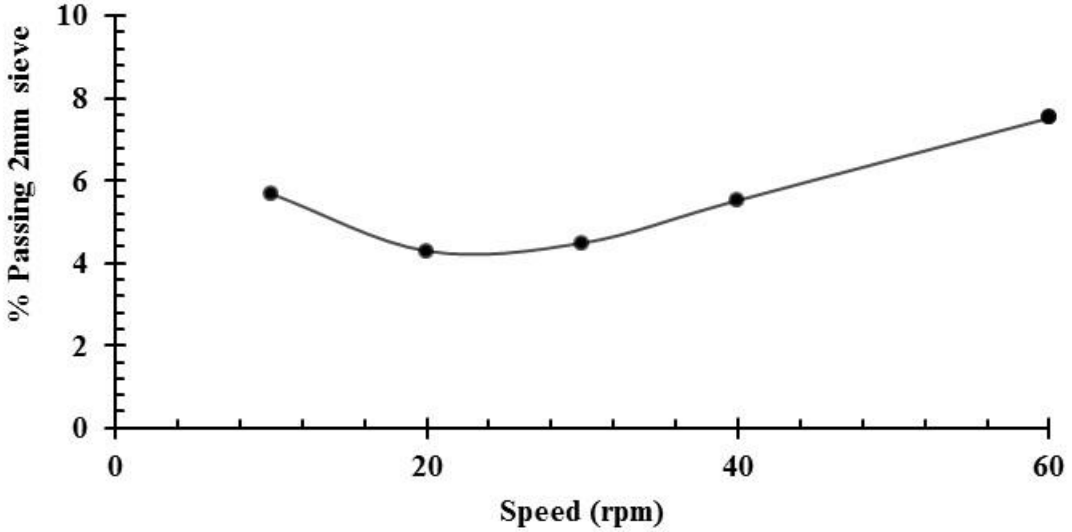


Figure 19: Effect of agglomeration speed on particle size distribution

At 20 rpm more coarse agglomerates were produced than at the other speeds as indicated by the size distribution information in Figure 19. Below 20 rpm agglomerates were fine due to low speed which resulted to insufficiently mix the ore, moisture and binder for agglomerates formation and growth. Beyond 20 rpm, the particle size started to become fine due to agglomerate particle bumping into each other resulting in breakage and simultaneously formation of smaller fine agglomerates.

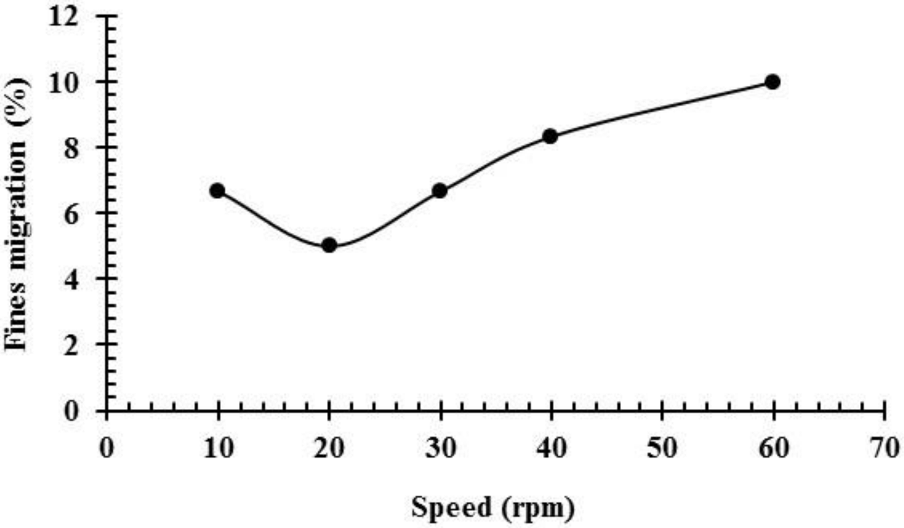


Figure 20: Effect of agglomeration speed on agglomerates quality (Soak test)

Fine migration data presented in Figure 20 show that fines are reduced to low levels at 20 rpm relative to either higher or lower drum speeds indicating that at that agglomeration speed, more stable agglomerates were formed. Thus 20 rpm was adopted as the optimum agglomeration drum speed.

4.3.5 Agglomeration tests summary

From the results obtained on the agglomeration test work, the following agglomeration parameters were adopted as the optimum agglomeration parameters and column leaching feed was prepared under those condition.

Table 10: Optimum agglomeration parameters for Mimbula ore

Parameter	***Specification***
Moisture Content (%)	*14*
Binder Dosage (gpt)	*600*
Agglomeration Residence Time (minutes)	*3*
Agglomerator Rotation Speed (rpm)	*20*

4.4 Column Leaching Characteristics for Agglomerated Copper Ore.

Column leaching tests were designed to establish the effect of bed height on copper recovery, heap leaching kinetics, acid consumption and measure the extent of dissolution of the copper mineral constituents within the ore. For all the column leaching tests carried out, the lixiviant flow rate was fixed at 6 l/hr.m^2 and the irrigation was carried out on an ON/OFF basis.

4.4.1 Effect of bed height on total copper leaching efficiency.

Connelly and West (2009) found that heap leaching at a lower bed height can require greater capital due to the greater area under leach, increased pumping requirements from more irrigation, extra drainage piping, and a lower PLS grade (due to greater solution requirements). On the other hand, a lower heap height can produce better overall copper leaching efficiency (Diaz et al., 2010), may not require forced aeration, and is unlikely to suffer permeability problems from compaction at the base of the heap.

The results obtained concur with the work done by Connelly and West (2009) and Diaz et al (2010). From Figure 21 below, the highest leaching efficiency of 59.32 % was recorded at a bed height of 4 m after leaching for 48 days. At 2 m bed height, the overall copper leaching efficiency was 58.32 % which was attained in 12 days of leaching, which is better than at 4 m bed height when comparing the leaching kinetics. This was due to better permeability as observed by having the lowest break through period of 72 hours, whilst for unagglomerated ore, 4 m and 6 m bed height it was observed to be 14, 19 and 25 days respectively. Poor permeability was due to compaction as the weight of the agglomerates in column increased as bed height increased.

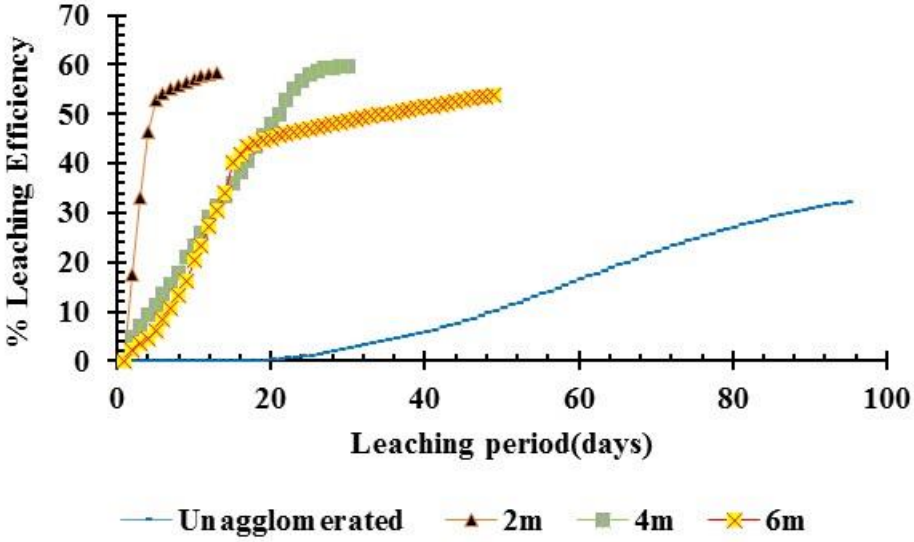


Figure 21: Effect of bed height on leaching efficiency

The copper leaching efficiency was calculated as follows:

$$\boldsymbol{Leaching\ Efficiency}\ (\%) = \left(\frac{\boldsymbol{Copper\ in\ solution}}{\boldsymbol{Total\ copper\ in\ ore}}\right) \times \mathbf{100}$$

$$\boldsymbol{Copper\ in\ Solution} = \boldsymbol{Copper\ assay\ value}\ (\boldsymbol{gpl}) \times \boldsymbol{Volumeof\ PLS}(\boldsymbol{l})$$

$$\boldsymbol{Total\ copper\ in\ ore}\ (\boldsymbol{kg}) = \frac{\boldsymbol{Ore\ grade} \times \boldsymbol{Mass\ of\ ore}}{\mathbf{100}}$$

<u>Example Calculation</u>

Taking the data recorded for column 2, on the first day of leaching, the leaching efficiency is given by:

$$\boldsymbol{Leaching\ Efficiency}\ (\%) = \left(\frac{\boldsymbol{Copper\ in\ solution}}{\boldsymbol{Total\ copper\ in\ ore}}\right) \times \mathbf{100}$$

$$\boldsymbol{Copper\ in\ Solution} = \mathbf{67.5}(\boldsymbol{gpl}) \times \mathbf{1.5}\ (\boldsymbol{l})$$

$$\Rightarrow \boldsymbol{Total\ copper\ in\ ore}\ (\boldsymbol{kg}) = \frac{\mathbf{1.2} \times \mathbf{48}}{\mathbf{100}}$$

$$\Rightarrow\ \boldsymbol{Total\ copper\ in\ ore}\ (\boldsymbol{kg}) = \mathbf{0.576}$$

$$\Rightarrow \boldsymbol{Leaching\ Efficiency}\ (\%) = \left(\frac{\mathbf{0.10125}}{\mathbf{0.576}}\right) \times \mathbf{100}$$

$$\underline{\therefore\ \boldsymbol{Leaching\ Efficiency}\ (\%) = \mathbf{17.58}}$$

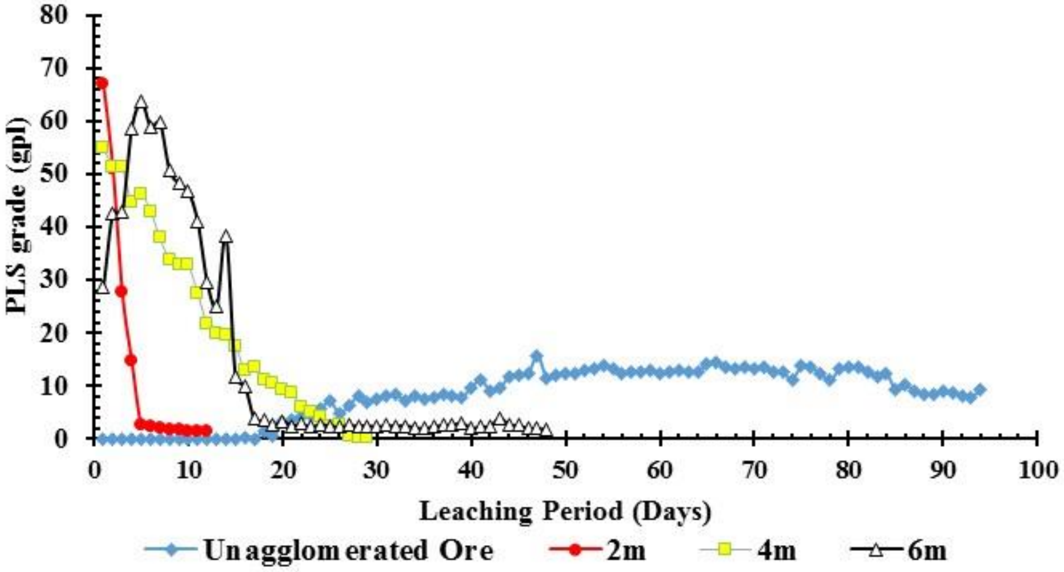


Figure 22: Effect of bed height on PLS grade profile

From Figure 22, the highest PLS grade was observed at 2 m bed height after leaching for 3 days which indicates high copper dissolution within the bed, followed by 4 m and 6 m bed height, and unagglomerated ore stacked at 3 m bed height. The reason for the trend was due to poor permeability resulting in insufficient distribution of the lixiviant to the targeted ore material for copper dissolution to occur efficiently.

4.4.3 Effect of bed height on heap leach kinetics

Column leach tests were undertaken to also assess the heap leaching kinetics of the ore material as a function of the bed height. Salomon-de-Friedberg (2000) established that heap leach kinetics are a function of bed permeability. Thus the more permeable the bed, the greater the leaching efficiencies.

The leaching kinetics were measured on the basis of the break through period and overall leaching efficiency for the leaching cycle. Table 12 below summarizes the leaching kinetics observed for Mimbula ore. At 2 m bed height, the break through period was 3 days and yielding total leaching efficiency of 58.35 % in 12 days. The breakthrough period for 4m and 6 m bed height was 19 and 27 days and leaching efficiencies of 59.62 and 58.35 % respectively. Hence it was observed that as the bed height increased, the permeability of the bed also decreases and ultimately increase the leaching period.

Table 11: Summary of column leaching tests

Column	Leaching Efficiency (%)	Leaching Period (Days)	Break through Period	Total Acid Consumption (Kg/t)	GAC (Kg/t)
Column 1 **(3 m)** **(**Unagglomerated ore)	32.2	94	14	24.96	19.43
Column 2 **(2 m)**	58.35	12	3	34.30	23.62
Column 3 **(4 m)**	59.62	27	19	32.89	21.86
Column 4 **(6 m)**	53.75	48	25	33.49	23.73

4.4.4 Effect of agglomerates feed size on copper leaching efficiency

The particles size leaching efficiency test was carried in order to find the agglomerate feed size that produced the highest leaching efficiency. Figure 23 gives a summary of the leaching efficiency against the feed size. A more detailed table of the results is given in Appendix D.

The feed material was analyzed for the copper contained before leaching and the copper contained after leaching at the targeted feed sizes (*25.4 mm, 12.5 mm, 4.75 mm, 2 mm, 0.212 mm, 0.15 mm, 0.075 mm).* The highest leaching efficiency of 96 % was observed at 4.75 mm agglomerate feed size, the leaching efficiency decreased as the particle size increased. Fine particle size exposes the mineral to leach solution (Vethosodsakda, 2012). However, in heap leaching fine particle size attracts permeability problems to the bed which will in turn lead to poor copper leaching efficiencies (Hill, 2013). From the results obtained, 4.75 mm was taken to be the optimum particle size.

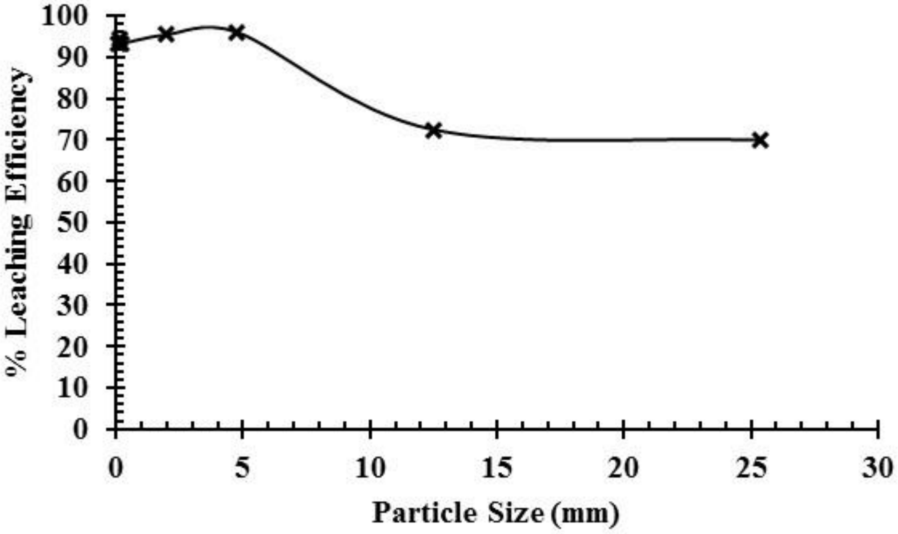


Figure 23: Effect of feed size on leaching efficiency

4.4.5 Effect of bed height on acid consumption

Column test were carried out to approximate acid consumption that can be used at different bed height. Refractory copper ores exhibit high acid consumption due to the presence of gangue materials which compete with copper for sulphuric acid (Phalwane, 2013). Acid consumption is a function of rock type and alteration, acid consuming minerals and particle size (Baum, 1999).

The cumulative acid consumption was calculated on the basis of volume of sulphuric acid in the form of raffinate and the acid added during curing. The trend of the acid consumption for the column tests is shown in Figure 24 below. It was observed that the highest acid consumption was recorded at 2 m bed height. This was due to good bed permeability which ensured sufficient lixiviant supply to the ore material.

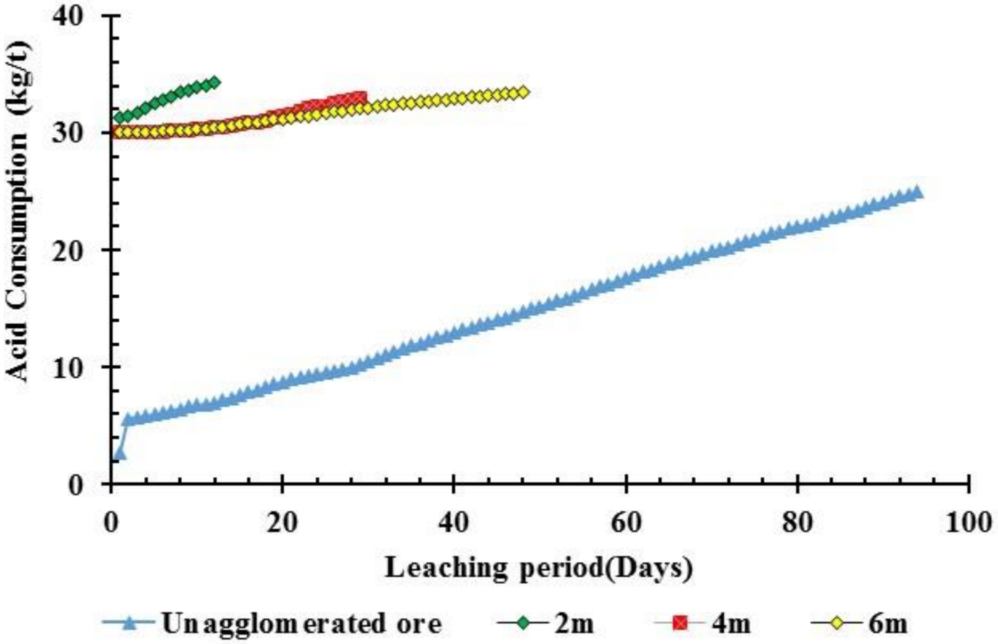


Figure 24: Acid consumption profile

4.4.6.1 Mineralogical analysis

The following table, gives a summary of the mineralogical results of Mimbula ore for feed and leach residue.

Table 12: Mineralogical analysis of the feed and leach residue

Mineral	**Chalcopyrite**	**Bornite**	**Chalcocite**	**Pyrite**	**Malachite**	**Pseudo-malachite**	**Chrysocolla**	**Cupriferous Mica**
Feed Wt. (%)	0.024	0.346	0.161	0.026	0.322	0.095	0.015	15
TCu Feed (%)	**0.008**	**0.219**	**0.129**		**0.186**	**0.053**	**0.006**	**0.6**
Leach Residue Wt. (%)	0.079	0.166	0.127	0.014	0.072	-	-	1.5
T Cu Leach Residue Wt. (%)	**-**	**0.105**	**0.101**	**-**	**0.041**	**-**	**-**	**0.06**

From Table 12, the extent of dissolution was as follows:

- 90 % cupriferous mica.
- 100 % pseudomalachite.
- 78 % malachite
- 0 % chalcopyrite

Acid curing employed in this study successfully leached cupriferous micas which are reported to be difficult to leach (Notebaart, 1988). However, sulphides, especially chalcopyrite were difficult to leach under the conditions employed.

CHAPTER FIVE: CONCLUSIONS AND RECOMMENDATIONS

5.1 CONCLUSIONS

The agglomeration and heap leaching of Mimbula ore was investigated and the following conclusions were reached:

- The optimized agglomeration parameters were 600 gpt binder dosage, 14 % moisture content of ore, 20 rpm agglomeration speed & 3 min residence time.
- Up to 60% copper dissolution is achievable from CRO. This suggests that Mimbula refractory ore is amenable to heap leaching and has great potential for the leach efficiencies to be improved even further.
- The mineralogical analysis of the leach residues indicated that 90 % of the cupriferous mica dissolved and there were no traces of pseudomalachite and chrysocolla in the leach residue due to acid curing. Hence this hybrid leaching method employed in this study has successfully leached cupriferous micas which are reported to be difficult to leach in the literature. However, sulphides, especially chalcopyrite are difficult to leach under the employed conditions.
- Agglomeration greatly improved leach efficiency and kinetics. The highest leach efficiency of 60 % was obtained with agglomerated ore in 12 days compared to 32 % for unagglomerated ore in 94 days.
- The optimum bed height for heap leaching was taken to be 4 m.

5.2 RECOMMENDATIONS

The study carried out revealed the metallurgical response of Mimbula ore to heap leaching and also forms the basis for further work to be done to assess other heap leach parameters. Hence recommendation on specific areas not covered were recommended and are as follows:

- Further studies to improve dissolution of sulphide minerals using acidified ferric sulphates and/or bioleaching should be done.
- The effects of lixiviant flow rate, pH and concentration should be investigated.

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APPENDICES

Appendix A: Ore Characterisation

Table 13: Mimbula ore feed particle size distribution

Particle Size	Retained Wt (Kg)	% Retained	%Cumulative Retained	%Cumulative passing
25.4 mm	0.48	4.76	4.76	95.24
19 mm	0.42	4.19	8.95	91.05
12.5mm	0.70	6.99	15.94	84.06
4.75mm	1.25	12.48	28.42	71.58
2mm	0.91	9.08	37.51	62.49
300µm	1.72	17.17	54.68	45.32
150µm	1.11	11.08	65.76	34.24
75µm	1.45	14.48	80.23	19.77
-75µm	1.98	19.77	100.00	0.00
Total	**10.02**			

Appendix B: Agglomeration Experiments

Figure 25: Soak test-agglomerates soaked for 30mins

Appendix C1: Column Leaching experiments Data for Column 1

Table 14: Column 1 leaching data

Days	Pregnant Liqour							Free acid in Raff		Remnant Acid		Cumulative						Total Acid Cons
	Volume	Irrigation Rate	Cu	FAC	pH	Cont'd Cu	Cum Cu	Acid used	Cum H2SO4	Cont'd H2SO4	Cum H2SO4	Copper Extraction				GAC	Acid Cons	Mineral + Gangue Acid Cons (kg/t)
	(Litres)	(Litres/hr.m^2)	(gpl)	(gpl)		g	g	g			g	%TCu	%Cum	%AsCu	%Cum	(Kg/tonne)	(t/tCu)	
1	2.30	1.5	0.09	0.20	5.20	0.21	0.21	972.90	972.90	0.46	0.46	0.01	0.01	0.01	0.01	2.78	4697.78	2.78
2	2.16	1.5	0.06	0.20	5.40	0.13	0.34	969.68	1942.58	0.43	0.89	0.00	0.01	0.01	0.02	5.55	5768.53	5.55
3	2.10	1.5	0.04	0.10	6.10	0.08	0.42	48.30	1990.88	0.21	1.10	0.00	0.01	0.01	0.03	5.68	4730.81	5.69
4	2.19	1.5	0.05	0.10	5.60	0.11	0.53	50.37	2041.25	0.22	1.32	0.00	0.01	0.01	0.04	5.83	3848.20	5.83
5	2.44	1.5	0.02	0.20	6.80	0.05	0.58	56.12	2097.37	0.49	1.81	0.00	0.01	0.00	0.04	5.98	3619.90	5.99
6	2.50	1.5	0.02	0.20	6.90	0.05	0.63	57.50	2154.87	0.50	2.31	0.00	0.02	0.00	0.04	6.15	3422.74	6.15
7	2.50	1.5	0.01	0.10	6.00	0.03	0.65	57.50	2212.37	0.25	2.56	0.00	0.02	0.00	0.04	6.31	3379.43	6.31
8	2.32	1.5	0.01	0.10	6.00	0.03	0.68	53.36	2265.73	0.23	2.79	0.00	0.02	0.00	0.05	6.46	3308.10	6.47
9	2.40	1.5	0.01	0.10	6.00	0.02	0.71	55.20	2320.93	0.24	3.03	0.00	0.02	0.00	0.05	6.62	3273.59	6.62
10	2.16	1.5	0.02	0.20	6.10	0.04	0.75	49.68	2370.61	0.43	3.46	0.00	0.02	0.00	0.05	6.76	3150.90	6.76
11	1.52	1.5	0.01	0.10	8.60	0.02	0.77	34.96	2405.57	0.15	3.62	0.00	0.02	0.00	0.05	6.86	3133.83	6.86
12	1.72	1.5	0.02	0.10	7.70	0.03	0.80	39.56	2445.13	0.17	3.79	0.00	0.02	0.00	0.05	6.97	3048.40	6.98
13	1.82	1.5	0.02	0.10	6.50	0.04	0.84	82.81	2527.94	0.18	3.97	0.00	0.02	0.00	0.06	7.21	3014.56	7.21
14	1.52	1.5	0.03	0.10	7.00	0.05	0.88	69.16	2597.10	0.15	4.12	0.00	0.02	0.00	0.06	7.40	2937.02	7.41
15	1.88	1.5	0.03	0.10	6.90	0.06	0.94	85.54	2682.64	0.19	4.31	0.00	0.02	0.00	0.06	7.65	2851.53	7.65
16	1.50	1.5	0.19	0.10	5.10	0.29	1.22	68.25	2750.89	0.15	4.46	0.01	0.03	0.02	0.08	7.84	2243.34	7.85
17	1.80	1.5	0.07	0.20	4.90	0.13	1.35	81.90	2832.79	0.36	4.82	0.00	0.03	0.01	0.09	8.07	2094.39	8.08
18	1.82	1.5	1.46	0.20	4.30	2.66	4.01	82.81	2915.60	0.36	5.18	0.07	0.10	0.18	0.27	8.30	726.25	8.32
19	1.72	1.5	0.66	0.20	4.90	1.14	5.14	78.26	2993.86	0.34	5.53	0.03	0.13	0.08	0.34	8.52	581.09	8.54
20	1.70	1.5	3.18	0.10	5.00	5.41	10.55	77.35	3071.21	0.17	5.70	0.14	0.27	0.36	0.70	8.71	290.61	8.76
21	1.90	1.5	3.70	0.10	3.80	7.03	17.58	63.46	3134.67	0.19	5.89	0.18	0.45	0.47	1.17	8.86	177.99	8.94
22	1.72	1.5	4.10	0.20	3.10	7.05	24.63	57.45	3192.12	0.34	6.23	0.18	0.63	0.47	1.64	8.99	129.35	9.10
23	1.42	1.5	4.62	0.20	4.40	6.56	31.19	47.43	3239.55	0.28	6.52	0.17	0.80	0.44	2.08	9.10	103.65	9.24
24	1.60	1.5	5.65	0.20	4.10	9.04	40.23	53.44	3292.99	0.32	6.84	0.23	1.03	0.60	2.68	9.21	81.68	9.39
25	1.62	1.5	7.18	0.10	3.80	11.63	51.86	54.11	3347.09	0.16	7.00	0.30	1.33	0.78	3.46	9.31	64.40	9.54
26	1.62	1.5	4.91	0.10	4.20	7.95	59.82	54.11	3401.20	0.16	7.16	0.20	1.54	0.53	3.99	9.43	56.74	9.70
27	1.60	1.5	6.26	0.10	4.30	10.02	69.83	53.44	3454.64	0.16	7.32	0.26	1.79	0.67	4.66	9.54	49.37	9.85
28	1.65	1.5	8.02	0.10	3.70	13.23	83.07	44.55	3499.19	0.17	7.48	0.34	2.13	0.88	5.54	9.61	42.04	9.98
29	1.66	1.5	6.93	0.30	3.20	11.50	94.57	96.28	3595.47	0.50	7.98	0.30	2.43	0.77	6.31	9.83	37.93	10.25
30	1.70	1.5	7.61	0.10	3.90	12.94	107.51	98.60	3694.07	0.17	8.15	0.33	2.76	0.86	7.17	10.06	34.29	10.53
31	1.60	1.5	8.28	0.10	3.50	13.25	120.75	92.80	3786.87	0.16	8.31	0.34	3.10	0.88	8.06	10.26	31.29	10.80
32	1.52	1.5	8.32	0.10	3.70	12.65	133.40	88.16	3875.03	0.15	8.46	0.32	3.42	0.84	8.90	10.46	28.98	11.05
33	1.80	1.5	7.36	0.10	3.70	13.25	146.65	104.40	3979.43	0.18	8.64	0.34	3.76	0.88	9.79	10.70	27.08	11.35
34	1.62	1.5	8.02	0.30	2.40	12.99	159.64	93.96	4073.39	0.49	9.13	0.33	4.10	0.87	10.65	10.91	25.46	11.61
35	1.64	1.5	7.68	0.50	4.00	12.60	172.24	78.39	4151.78	0.82	9.95	0.32	4.42	0.84	11.49	11.08	24.05	11.83
36	1.60	1.5	7.98	0.40	3.80	12.77	185.00	76.48	4228.26	0.64	10.59	0.33	4.75	0.85	12.34	11.24	22.80	12.05
37	1.60	1.5	8.60	0.20	4.30	13.76	198.76	76.48	4304.74	0.32	10.91	0.35	5.10	0.92	13.26	11.39	21.60	12.27
38	1.94	1.5	8.02	0.20	3.70	15.56	214.32	92.73	4397.48	0.39	11.30	0.40	5.50	1.04	14.30	11.59	20.47	12.53
39	1.70	1.5	7.92	0.20	4.50	13.46	227.79	81.26	4478.74	0.34	11.64	0.35	5.85	0.90	15.20	11.76	19.61	12.76
40	1.64	1.5	9.76	0.20	4.00	16.01	243.79	78.39	4557.13	0.33	11.97	0.41	6.26	1.07	16.27	11.91	18.64	12.99
41	1.48	1.5	11.20	0.10	4.30	16.58	260.37	70.74	4627.87	0.15	12.11	0.43	6.68	1.11	17.37	12.04	17.73	13.19
42	1.46	1.5	9.14	0.10	3.80	13.34	273.71	69.79	4697.66	0.15	12.26	0.34	7.02	0.89	18.26	12.18	17.12	13.39
43	1.82	1.5	9.52	0.10	4.00	17.33	291.04	87.00	4784.66	0.18	12.44	0.44	7.47	1.16	19.42	12.35	16.40	13.63
44	1.50	1.5	11.90	0.10	3.60	17.85	308.89	71.70	4856.36	0.15	12.59	0.46	7.93	1.19	20.61	12.48	15.68	13.84
45	1.60	1.5	12.20	0.10	3.90	19.52	328.41	76.48	4932.84	0.16	12.75	0.50	8.43	1.30	21.91	12.61	14.98	14.06
46	1.50	1.5	12.25	0.10	3.60	18.38	346.79	71.70	5004.54	0.15	12.90	0.47	8.90	1.23	23.14	12.74	14.39	14.26
47	1.70	1.5	15.80	0.20	3.30	26.86	373.65	81.26	5085.80	0.34	13.24	0.69	9.59	1.79	24.93	12.85	13.58	14.49
48	1.80	1.5	11.50	0.20	3.60	20.70	394.35	86.40	5172.20	0.36	13.60	0.53	10.12	1.38	26.31	13.00	13.08	14.74
49	1.88	1.5	12.03	0.10	4.30	22.62	416.96	90.24	5262.44	0.19	13.79	0.58	10.70	1.51	27.82	13.16	12.59	15.00
50	1.50	1.5	12.50	0.10	4.00	18.75	435.71	72.00	5334.44	0.15	13.94	0.48	11.18	1.25	29.07	13.28	12.21	15.20
51	1.72	1.5	12.33	0.10	3.60	21.21	456.92	82.56	5417.00	0.17	14.11	0.54	11.73	1.42	30.49	13.43	11.82	15.44
52	1.65	1.5	13.00	0.10	4.00	21.45	478.37	79.20	5496.20	0.17	14.28	0.55	12.28	1.43	31.92	13.56	11.46	15.66
53	1.60	1.5	13.25	0.10	3.30	21.20	499.57	76.80	5573.00	0.16	14.44	0.54	12.82	1.41	33.34	13.68	11.13	15.88
54	1.70	1.5	13.80	0.10	4.10	23.46	523.03	81.60	5654.60	0.17	14.61	0.60	13.42	1.57	34.90	13.81	10.78	16.11
55	1.52	1.5	13.15	0.10	4.40	19.99	543.02	72.96	5727.56	0.15	14.76	0.51	13.94	1.33	36.23	13.93	10.52	16.32
56	2.00	1.5	12.40	0.10	4.30	24.80	567.82	96.00	5823.56	0.20	14.96	0.64	14.57	1.65	37.89	14.10	10.23	16.60
57	2.14	1.5	12.55	0.10	4.40	26.86	594.67	102.72	5926.28	0.21	15.17	0.69	15.26	1.79	39.68	14.27	9.94	16.89
58	1.80	1.5	12.60	0.10	3.60	22.68	617.35	86.40	6012.68	0.18	15.35	0.58	15.84	1.51	41.19	14.42	9.71	17.14
59	1.70	1.5	13.05	0.10	4.20	22.19	639.54	81.60	6094.28	0.17	15.52	0.57	16.41	1.48	42.68	14.55	9.50	17.37
60	1.82	1.5	12.30	0.10	3.80	22.39	661.93	87.36	6181.64	0.18	15.71	0.57	16.99	1.49	44.17	14.70	9.32	17.62
61	1.90	1.5	12.65	0.10	4.10	24.04	685.96	91.20	6272.84	0.19	15.90	0.62	17.60	1.60	45.77	14.86	9.12	17.88
62	1.70	1.5	12.95	0.10	4.10	22.02	707.98	81.60	6354.44	0.17	16.07	0.57	18.17	1.47	47.24	14.99	8.95	18.11
63	1.68	1.5	12.55	0.10	3.80	21.08	729.06	80.64	6435.08	0.17	16.23	0.54	18.71	1.41	48.65	15.13	8.80	18.34
64	1.70	1.5	12.70	0.10	3.90	21.59	750.65	81.60	6516.68	0.17	16.40	0.55	19.27	1.44	50.09	15.27	8.66	18.57
65	1.72	1.5	14.20	0.10	5.50	24.42	775.07	82.56	6599.24	0.17	16.58	0.63	19.89	1.63	51.72	15.40	8.49	18.81
66	1.54	1.5	14.50	0.10	5.50	22.33	797.40	73.92	6673.16	0.15	16.73	0.57	20.47	1.49	53.21	15.51	8.35	19.02
67	1.62	1.5	13.45	0.10	5.10	21.79	819.19	77.76	6750.92	0.16	16.89	0.56	21.02	1.45	54.66	15.64	8.22	19.24
68	1.54	1.5	13.30	0.10	5.60	20.48	839.67	73.92	6824.84	0.15	17.05	0.53	21.55	1.37	56.03	15.76	8.11	19.45
69	1.56	1.5	13.50	0.10	5.30	21.06	860.73	74.88	6899.72	0.16	17.20	0.54	22.09	1.41	57.44	15.88	8.00	19.66
70	1.58	1.5	13.15	0.10	5.20	20.78	881.51	75.84	6975.56	0.16	17.36	0.53	22.62	1.39	58.82	16.00	7.89	19.88
71	1.50	1.5	13.60	0.10	4.90	20.40	901.91	72.00	7047.56	0.15	17.51	0.52	23.15	1.36	60.18	16.12	7.79	20.09
72	1.46	1.5	12.75	0.10	3.70	18.62	920.53	70.08	7117.64	0.15	17.66	0.48	23.62	1.24	61.42	16.24	7.71	20.29
73	1.52	1.5	12.65	0.10	3.80	19.23	939.75	72.96	7190.60	0.15	17.81	0.49	24.12	1.28	62.71	16.36	7.63	20.49
74	1.58	1.5	11.28	0.10	4.00	17.82	957.58	75.84	7266.44	0.16	17.97	0.46	24.58	1.19	63.90	16.50	7.57	20.71
75	1.54	1.5	13.85	0.20	4.10	21.33	978.91	80.08	7346.52	0.31	18.27	0.55	25.12	1.42	65.32	16.63	7.49	20.94
76	1.52	1.5	13.55	0.10	4.20	20.60	999.50	79.04	7425.56	0.15	18.43	0.53	25.65	1.37	66.69	16.77	7.41	21.16
77	1.50	1.5	12.40	0.10	3.90	18.60	1018.10	78.00	7503.56	0.15	18.58	0.48	26.13	1.24	67.94	16.91	7.35	21.39
78	1.50	1.5	11.10	0.20	4.00	16.65	1034.75	78.00	7581.56	0.30	18.88	0.43	26.56	1.11	69.05	17.05	7.31	21.61
79	1.32	1.5	13.15	0.20	4.10	17.36	1052.11	68.64	7650.20	0.26	19.14	0.45	27.00	1.16	70.21	17.17	7.25	21.80
80	1.23	1.5	13.50	0.10	4.00	16.61	1068.72	63.96	7714.16	0.12	19.26	0.43	27.43	1.11	71.31	17.28	7.20	21.99
81	1.14	1.5	13.70	0.10	4.10	15.62	1084.33	59.28	7773.44	0.11	19.38	0.40	27.83	1.04	72.36	17.38	7.15	22.15
82	1.00	1.5	12.60	0.10	4.10	12.60	1096.93	52.00	7825.44	0.10	19.48	0.32	28.15	0.84	73.20	17.48	7.12	22.30
83	1.50	1.5	11.90	0.20	4.10	17.85	1114.78	78.00	7903.44	0.30	19.78	0.46	28.61	1.19	74.39	17.62	7.07	22.52
84	1.48	1.5	12.4	0.10	3.80	18.35	1133.14	82.88	7986.32	0.15	19.92	0.47	29.08	1.22	75.61	17.78	7.03	22.76
85	1.32	1.5	9.32	0.20	4.10	12.30	1145.44	73.92	8060.24	0.26	20.19	0.32	29.40	0.82	76.43	17.93	7.02	22.97
86	1.30	1.5	10.25	0.10	3.80	13.33	1158.76	72.80	8133.04	0.13	20.32	0.34	29.74	0.89	77.32	18.08	7.00	23.18
87	1.51	1.5	9.00	0.10	3.70	13.59	1172.35	84.56	8217.60	0.15	20.47	0.35	30.09	0.91	78.23	18.26	6.99	23.42
88	1.46	1.5	8.50	0.10	4.10	12.41	1184.76	81.76	8299.36	0.15	20.62	0.32	30.41	0.83	79.06	18.44	6.99	23.65
89	1.42	1.5	8.55	0.20	3.70	12.14	1196.90	79.52	8378.88	0.28	20.90	0.31	30.72	0.81	79.87	18.61	6.98	23.88
90	1.40	1.5	8.95	0.20	3.80	12.53	1209.43	78.40	8457.28	0.28	21.18	0.32	31.04	0.84	80.70	18.78	6.98	24.10
91	1.38	1.5	8.75	0.20	3.50	12.08	1221.51	77.28	8534.56	0.28	21.46	0.31	31.35	0.81	81.51	18.95	6.97	24.32
92	1.32	1.5	8.17	0.10	4.20	10.78	1232.29	73.92	8608.48	0.13	21.59	0.28	31.63	0.72	82.23	19.11	6.97	24.53
93	1.32	1.5	7.85	0.10	4.20	10.36	1242.65	73.92	8682.40	0.13	21.72	0.27	31.89	0.69	82.92	19.28	6.97	24.74
94	1.32	1.5	9.50	0.10	4.30	12.54	1255.19	73.92	8756.32	0.13	21.85	0.32	32.21	0.84	83.76	19.43	6.96	24.96

Appendix C2: Leaching Data for Column 2

Table 15: Column 2 leaching data

Pregnant Liqour								**Free acid in Raff**		**Reminant Acid**		**Cumulative**						Total Acid Cons
Days	Volume	Irrigation Rate	Cu	FAC	pH	Cont'd Cu	Cum Cu	Acid used	Cum H2SO4	Cont'd H2SO	Cum H2SO4	Copper Extraction				GAC	Acid Cons	Mineral + Gangue Acid
	(Litres)	(Litres/hr.m^2)	(gpl)	(gpl)		g	g	g			g	%TCu	%Cum	%AsCu	%Cum	(Kg/tonne)	(t/tCu)	
		6																
1	1.50	6-On & Off	67.50	6.00	2.80	100.55	100.55	11.40	11.40	9.00	9.00	17.61	17.61	52.84	52.84	28.07	14.94	31.30
2	1.74	6-On & Off	51.50	4.20	2.40	88.79	189.34	13.22	24.62	7.31	16.31	15.55	33.17	46.66	99.50	25.35	7.97	31.42
3	2.72	6	28.10	2.60	3.30	75.15	264.49	20.67	45.30	7.07	23.38	13.16	46.33	39.49	138.99	23.22	5.75	31.71
4	2.56	6	15.15	0.20	4.00	37.58	302.07	19.46	64.75	0.51	23.89	6.58	52.91	19.75	158.74	22.41	5.10	32.10
5	2.42	6	3.15	0.10	4.30	6.51	308.58	17.64	82.39	0.24	24.13	1.14	54.05	3.42	162.16	22.56	5.05	32.46
6	2.34	6	2.85	0.20	3.70	5.62	314.20	16.45	98.84	0.47	24.60	0.98	55.04	2.95	165.11	22.72	5.01	32.80
7	2.28	6-On & Off	2.48	0.10	3.90	4.65	318.85	15.50	114.35	0.23	24.83	0.81	55.85	2.44	167.56	22.89	4.99	33.11
8	2.20	6-On & Off	2.10	0.10	4.00	3.67	322.52	14.54	128.89	0.22	25.05	0.64	56.50	1.93	169.49	23.07	4.97	33.41
9	1.82	6-On & Off	2.08	0.30	4.10	3.02	325.54	11.78	140.67	0.55	25.60	0.53	57.03	1.59	171.08	23.20	4.96	33.65
10	1.84	6-On & Off	1.98	0.30	3.80	2.93	328.47	11.90	152.57	0.55	26.15	0.51	57.54	1.54	172.61	23.35	4.95	33.88
11	1.50	6-On & Off	1.90	0.20	4.10	2.31	330.78	9.71	162.28	0.30	26.45	0.40	57.94	1.21	173.83	23.47	4.95	34.08
12	1.72	6-On & Off	1.68	0.20	4.00	2.32	333.10	11.13	173.40	0.34	26.79	0.41	58.35	1.22	175.05		4.94	34.30

Appendix C3: Leaching Data for Column 3

Table 16: Leaching data for Column 3

Pregnant Liqour								Free acid in Raff		Remnant Acid		Cumulative						Total Acid Cons
Days	Volume	Irrigation Rate	Cu	FAC	pH	Cont'd Cu	Cum Cu	Acid used	Cum H2SO4	Cont'd H2SO4	Cum H2SO4	Copper Extraction				GAC	Acid Cons	Mineral + Gangue Acid Cons (kg/t)
	(Litres)	(Litres/hr.m^2)	(gpl)	(gpl)		g	g	g			g	%TCu	%Cum	%AsCu	%Cum	(Kg/tonne)	(t/tCu)	
1	1.02	6-On & Off	55.25	4.00	2.90	56.02	56.02	6.60	6.60	4.08	4.08	4.91	4.91	14.72	14.72	29.12	0.04	30.03
2	0.48	6-On & Off	51.78	4.50	3.30	24.70	80.71	3.11	9.71	2.16	6.24	2.16	7.07	6.49	21.21	28.73	0.04	30.04
3	0.52	6-On & Off	51.50	4.30	3.10	26.61	107.32	3.36	13.07	2.24	8.48	2.33	9.40	6.99	28.20	28.31	0.04	30.05
4	0.50	6-On & Off	45.00	3.90	3.20	22.34	129.66	3.24	16.30	1.95	10.43	1.96	11.36	5.87	34.07	27.96	0.05	30.06
5	0.52	6-On & Off	46.50	3.00	3.50	24.01	153.67	3.36	19.67	1.56	11.99	2.10	13.46	6.31	40.38	27.59	0.05	30.08
6	0.52	6-On & Off	43.25	2.00	3.60	22.32	175.98	3.36	23.03	1.04	13.03	1.95	15.41	5.86	46.24	27.25	0.06	30.11
7	0.76	6-On & Off	38.25	1.50	3.50	28.82	204.80	4.92	27.95	1.14	14.17	2.52	17.94	7.57	53.81	26.83	0.07	30.15
8	1.00	6-On & Off	34.25	2.00	3.40	33.92	238.72	6.47	34.42	2.00	16.17	2.97	20.91	8.91	62.73	26.32	0.08	30.19
9	0.86	6-On & Off	33.25	1.80	3.60	28.31	267.04	5.56	39.98	1.55	17.71	2.48	23.39	7.44	70.16	25.91	0.08	30.23
10	0.84	6-On & Off	33.10	0.40	3.50	27.53	294.56	5.43	45.42	0.34	18.05	2.41	25.80	7.23	77.40	25.51	0.09	30.29
11	1.40	6-On & Off	27.75	0.30	3.60	38.39	332.95	9.06	54.48	0.42	18.47	3.36	29.16	10.09	87.48	24.98	0.11	30.38
12	1.10	6-On & Off	22.00	2.00	3.30	23.84	356.79	7.12	61.59	2.20	20.67	2.09	31.25	6.26	93.75	24.65	0.11	30.43
13	1.30	6-On & Off	20.15	1.50	3.60	25.77	382.55	8.41	70.01	1.95	22.62	2.26	33.51	6.77	100.52	24.30	0.12	30.50
14	1.38	6-On & Off	19.90	0.30	3.60	27.01	409.56	8.93	78.93	0.41	23.03	2.37	35.87	7.10	107.61	23.95	0.14	30.59
15	1.48	6-On & Off	17.80	0.10	3.60	25.86	435.42	9.58	88.51	0.15	23.18	2.26	38.14	6.79	114.41	23.63	0.15	30.69
16	2.14	6-On & Off	13.35	0.50	3.60	27.86	463.28	13.85	102.36	1.07	24.25	2.44	40.58	7.32	121.73	23.31	0.17	30.82
17	2.36	6-On & Off	14.00	2.50	3.10	32.26	495.54	15.27	117.62	5.90	30.15	2.83	43.40	8.48	130.20	22.89	0.18	30.92
18	2.34	6-On & Off	11.55	0.30	3.70	26.25	521.79	15.14	132.76	0.70	30.85	2.30	45.70	6.90	137.10	22.61	0.20	31.07
19	2.52	6-On & Off	10.73	0.50	3.20	26.21	548.00	16.30	149.07	1.26	32.11	2.30	48.00	6.89	143.99	22.35	0.21	31.23
20	2.36	6-On & Off	9.73	0.20	3.60	22.18	570.19	15.27	164.34	0.47	32.59	1.94	49.94	5.83	149.82	22.14	0.23	31.39
21	3.62	6-On & Off	9.05	1.50	3.50	31.57	601.75	23.42	187.76	5.43	38.02	2.76	52.70	8.29	158.11	21.82	0.25	31.58
22	4.42	6-On & Off	6.25	0.50	3.20	26.17	627.92	28.60	216.36	2.21	40.23	2.29	55.00	6.88	164.99	21.68	0.28	31.85
23	3.64	6-On & Off	5.50	0.40	3.30	18.82	646.74	23.55	239.91	1.46	41.68	1.65	56.64	4.94	169.93	21.60	0.31	32.09
24	3.58	6-On & Off	4.60	0.50	3.00	15.29	662.02	23.16	263.07	1.79	43.47	1.34	57.98	4.02	173.95	21.58	0.33	32.31
25	3.24	6-On & Off	3.22	7.20	3.40	7.29	669.31	20.96	284.03	23.33	66.80	0.64	58.62	1.92	175.86	21.44	0.32	32.29
26	2.92	6-On & Off	3.60	0.60	3.30	7.68	676.99	18.89	302.93	1.75	68.55	0.67	59.29	2.02	177.88	21.49	0.35	32.47
27	2.62	6-On & Off	1.66	1.00	3.20	1.81	678.80	16.95	319.88	2.62	71.17	0.16	59.45	0.48	178.36	21.61	0.37	32.62
28	2.42	6-On & Off	1.36	0.50	3.30	0.94	679.75	15.66	335.53	1.21	72.38	0.08	59.54	0.25	178.61	21.75	0.39	32.77
29	2.38	6-On & Off	1.34	1.50	3.70	0.88	680.63	15.40	350.93	3.57	75.95	0.08	59.61	0.23	178.84	21.86	0.40	32.89

Appendix C4: Leaching Data for Column 4

Table 17: Leaching data for Column 4

	Pregnant Liqour							Free acid in Raff		Remnant Acid		Cumulative						Total Acid Cons
Days	Volume	Irrigation Rate	Cu	FAC	pH	Cont'd Cu	Cum Cu	Acid used	Cum H2SO4	Cont'd H2SO4	Cum H2SO4	Copper Extraction				GAC	Acid Cons	Mineral + Gangue
	(Litres)	(Litres/hr.m^2)	(gpl)	(gpl)		g	g	g			g	%TCu	%Cum	%AsCu	%Cum	(Kg/tonne)	(t/tCu)	Acid Cons (kg/t)
1	1.25	6-On & Off	28.90	0.20	2.50	35.75	35.75	8.09	8.09	0.25	0.25	2.20	2.20	6.59	6.59	29.66	0.22	30.06
2	0.44	6-On & Off	43.00	4.00	2.30	18.79	54.54	2.85	10.93	1.76	2.01	1.15	3.35	3.46	10.06	29.46	0.16	30.06
3	0.44	6-On & Off	43.25	2.00	2.70	18.90	73.44	2.85	13.78	0.88	2.89	1.16	4.51	3.48	13.54	29.26	0.15	30.08
4	0.44	6-On & Off	59.00	2.00	2.40	25.83	99.26	2.85	16.63	0.88	3.77	1.59	6.10	4.76	18.30	28.99	0.13	30.09
5	0.60	6-On & Off	64.00	2.30	2.70	38.22	137.48	3.88	20.51	1.38	5.15	2.35	8.45	7.05	25.35	28.58	0.11	30.11
6	0.61	6-On & Off	59.25	0.20	2.80	35.96	173.44	3.95	24.46	0.12	5.27	2.21	10.66	6.63	31.98	28.20	0.11	30.14
7	0.74	6-On & Off	60.00	0.50	2.60	44.18	217.62	4.79	29.24	0.37	5.64	2.72	13.38	8.15	40.13	27.74	0.11	30.17
8	0.86	6-On & Off	50.91	0.50	2.40	43.52	261.15	5.56	34.81	0.43	6.07	2.68	16.05	8.03	48.15	27.29	0.11	30.21
9	0.78	6-On & Off	91.50	0.30	3.10	71.14	332.28	5.05	39.86	0.23	6.31	4.37	20.42	13.12	61.27	26.54	0.10	30.24
10	1.00	6-On & Off	47.00	2.00	3.30	46.70	378.98	6.47	46.33	2.00	8.31	2.87	23.29	8.61	69.88	26.05	0.10	30.28
11	1.50	6-On & Off	41.25	0.2	3.0	61.43	440.41	9.71	56.03	0.30	8.61	3.78	27.07	11.33	81.21	25.43	0.11	30.34
12	1.80	6-On & Off	29.90	0.2	3.8	53.28	493.69	11.65	67.68	0.36	8.97	3.27	30.34	9.82	91.03	24.92	0.12	30.43
13	2.42	6-On & Off	25.30	0.1	3.6	60.50	554.19	15.66	83.33	0.24	9.21	3.72	34.06	11.16	102.19	24.35	0.13	30.54
14	2.54	6-On & Off	38.48	0.1	3.4	96.98	651.16	16.43	99.77	0.25	9.46	5.96	40.02	17.88	120.07	23.39	0.14	30.65
15	2.62	6-On & Off	12.05	1.0	4.1	30.79	681.95	16.95	116.72	2.62	12.08	1.89	41.91	5.68	125.74	23.15	0.15	30.76
16	2.54	6-On & Off	10.20	2.0	4.2	25.15	707.10	16.43	133.15	5.08	17.16	1.55	43.46	4.64	130.38	22.95	0.16	30.84
17	2.62	6-On & Off	4.10	2.6	4.1	9.96	717.05	16.95	150.10	6.81	23.97	0.61	44.07	1.84	132.22	22.91	0.18	30.91
18	2.42	6-On & Off	3.80	0.4	3.7	8.47	725.52	15.66	165.76	0.97	24.94	0.52	44.59	1.56	133.78	22.92	0.19	31.02
19	2.64	6-On & Off	2.90	2.1	4.0	6.86	732.39	17.08	182.84	5.54	30.49	0.42	45.01	1.27	135.04	22.93	0.21	31.10
20	2.64	6-On & Off	3.45	2.00	3.50	9.03	741.41	17.08	199.92	5.28	35.77	0.55	45.57	1.66	136.71	22.92	0.22	31.19
21	2.10	6-On & Off	2.54	0.10	4.20	5.27	746.69	13.59	213.51	0.21	35.98	0.32	45.89	0.97	137.68	22.95	0.24	31.29
22	2.20	6-On & Off	2.92	0.10	4.00	6.36	753.04	14.23	227.74	0.22	36.20	0.39	46.28	1.17	138.85	22.98	0.25	31.39
23	2.10	6-On & Off	2.32	0.10	3.80	4.81	757.85	13.59	241.33	0.21	36.41	0.30	46.58	0.89	139.74	23.03	0.27	31.48
24	2.26	6-On & Off	2.79	0.10	4.10	6.24	764.09	14.62	255.95	0.23	36.63	0.38	46.96	1.15	140.89	23.06	0.29	31.59
25	2.12	6-On & Off	2.56	0.20	3.70	5.36	769.45	13.72	269.67	0.42	37.06	0.33	47.29	0.99	141.88	23.10	0.30	31.69
26	2.10	6-On & Off	2.30	0.20	4.30	4.77	774.22	13.59	283.26	0.42	37.48	0.29	47.59	0.88	142.76	23.14	0.32	31.78
27	1.88	6-On & Off	2.6	0.80	3.90	4.83	779.05	12.16	295.42	1.50	38.98	0.30	47.88	0.89	143.65	23.16	0.33	31.86
28	2.20	6-On & Off	2.48	0.20	4.10	5.39	784.44	14.23	309.65	0.44	39.42	0.33	48.21	0.99	144.64	23.20	0.34	31.96
29	2.10	6-On & Off	2.56	0.30	3.90	5.31	789.75	13.59	323.24	0.63	40.05	0.33	48.54	0.98	145.62	23.24	0.36	32.05
30	1.88	6-On & Off	2.30	0.20	4.30	4.27	794.02	12.16	335.40	0.38	40.43	0.26	48.80	0.79	146.41	23.28	0.37	32.14
31	2.20	6-On & Off	2.60	0.80	3.90	5.65	799.68	14.23	349.64	1.76	42.19	0.35	49.15	1.04	147.45	23.30	0.38	32.23
32	2.10	6-On & Off	2.48	0.20	4.10	5.15	804.82	13.59	363.23	0.42	42.61	0.32	49.47	0.95	148.40	23.34	0.40	32.32
33	1.80	6-On & Off	2.56	0.30	3.90	4.55	809.38	11.65	374.87	0.54	43.15	0.28	49.75	0.84	149.24	23.37	0.41	32.40
34	1.60	6-On & Off	2.26	0.40	3.80	3.57	812.94	10.35	385.22	0.64	43.79	0.22	49.97	0.66	149.90	23.40	0.42	32.47
35	1.54	6-On & Off	1.98	0.30	4.10	3.00	815.95	9.96	395.19	0.46	44.25	0.18	50.15	0.55	150.45	23.44	0.43	32.54
36	2.32	6-On & Off	2.40	3.40	3.70	5.50	821.44	15.01	410.20	7.89	52.14	0.34	50.49	1.01	151.47	23.43	0.44	32.59
37	2.40	6-On & Off	2.62	0.60	4.00	6.22	827.66	15.53	425.73	1.44	53.58	0.38	50.87	1.15	152.61	23.46	0.45	32.70
38	2.00	6-On & Off	2.70	0.30	4.30	5.34	833.00	12.94	438.67	0.60	54.18	0.33	51.20	0.98	153.60	23.49	0.46	32.79
39	1.50	6-On & Off	3.06	0.50	4.10	4.55	837.55	9.71	448.37	0.75	54.93	0.28	51.48	0.84	154.43	23.50	0.47	32.85
40	1.36	6-On & Off	2.09	0.60	2.70	2.80	840.35	8.80	457.17	0.82	55.74	0.17	51.65	0.52	154.95	23.53	0.48	32.91
41	1.42	6-On & Off	2.36	0.40	2.50	3.31	843.66	9.19	466.36	0.57	56.31	0.20	51.85	0.61	155.56	23.56	0.49	32.97
42	1.46	6-On & Off	2.40	1.10	2.60	3.46	847.12	9.45	475.80	1.61	57.92	0.21	52.07	0.64	156.20	23.57	0.49	33.03
43	2.00	6-On & Off	4.02	0.80	3.20	7.98	855.10	12.94	488.74	1.60	59.52	0.49	52.56	1.47	157.67	23.57	0.50	33.11
44	1.52	6-On & Off	2.71	0.60	2.80	4.07	859.17	9.83	498.58	0.91	60.43	0.25	52.81	0.75	158.42	23.59	0.51	33.18
45	1.44	6-On & Off	2.58	0.60	2.60	3.67	862.84	9.32	507.90	0.86	61.29	0.23	53.03	0.68	159.10	23.61	0.52	33.24
46	2.30	6-On & Off	2.10	0.50	3.40	4.76	867.60	14.88	522.78	1.15	62.44	0.29	53.33	0.88	159.98	23.65	0.53	33.34
47	1.50	6-On & Off	2.00	0.90	3.00	2.96	870.56	9.71	532.48	1.35	63.79	0.18	53.51	0.54	160.52	23.68	0.54	33.40
48	2.10	6-On & Off	1.89	0.40	3.10	3.91	874.46	13.59	546.07	0.84	64.63	0.24	53.75	0.72	161.24	23.73	0.55	33.49

Appendix D: Effect of feed size of leaching efficiency data

Table 18: Effect of feed size on leaching efficiency-Feed assays

Particle Size	Retained Wt (Kg)	% Retained	%Cumulative Retained	%Cumulative passing	%TCu	TCont.(g)	%AsCu	ASCont(g)	%AICu	AICont(g)
25.4 mm	0.56	8.84	8.84	91.16	0.58	3.24	0.22	1.23	0.36	2.01
12.5mm	0.57	9.00	17.85	82.15	1.12	6.37	0.68	3.87	0.44	2.50
4.75mm	0.55	8.69	26.53	73.47	1.48	8.13	0.97	5.33	0.51	2.80
2mm	0.33	5.21	31.74	68.26	1.36	4.47	0.85	2.80	0.51	1.68
212µm	0.94	14.86	46.60	53.40	0.82	7.70	0.55	5.16	0.27	2.54
150µm	0.33	5.21	51.80	48.20	0.64	2.11	0.38	1.25	0.26	0.86
75µm	0.93	14.70	66.50	33.50	1.05	9.75	0.54	5.02	0.51	4.74
-75µm	2.12	33.50	100.00	0.00	1.61	34.08	0.99	20.96	0.62	13.13
Total	**6.32**				**1.20**	75.86	**0.72**	45.61	**0.48**	30.25

Table 19: Effect of particle size on leaching efficiency-Leach Residue assays

Particle Size	Retained Wt (Kg)	% Retained	%Cumulative Retained	%Cumulative passing	%TCu	TCont.(g)	%AsCu	ASCont(g)	%AICu	AICont(g)
25.4 mm	0.22	11.06	11.06	88.94	0.34	0.75	0.25	0.55	0.09	0.20
12.5mm	0.22	11.06	22.11	77.89	0.87	1.91	0.16	0.35	0.71	1.56
4.75mm	0.16	8.04	30.15	69.85	0.2	0.32	0.07	0.11	0.13	0.21
2mm	0.12	6.03	36.18	63.82	0.17	0.20	0.10	0.12	0.07	0.08
212µm	0.32	16.08	52.26	47.74	0.16	0.51	0.05	0.16	0.11	0.35
150µm	0.13	6.53	58.79	41.21	0.09	0.12	0.02	0.03	0.07	0.09
75µm	0.35	17.59	76.38	23.62	0.19	0.67	0.04	0.14	0.15	0.53
-75µm	0.47	23.62	100.00	0.00	0.24	1.13	0.05	0.24	0.19	0.89
Total	**1.99**				**0.28**	5.61	**0.09**	1.70	**0.20**	3.91