BINDURA UNIVERSITY OF SCIENCE EDUCATION

FACULTY OF SCIENCE AND ENGINEERING

DEPARTMENT OF CHEMISTRY

OPTIMIZATION STUDIES OF GOLD CYANIDATION LEACHING PROCESS FOR AN OXIDE GOLD ORE.

Submitted by:

RABSON CHIRIKUUTSI

(B192033B)

SUPERVISOR: Dr N. MUCHANYAREYI

A DISSERTATION SUBMITTED IN PARTIAL FULFILMENT OF THE REQUIREMENTS OF THE BACHELOR OF SCIENCE HONOURS DEGREE IN CHEMICAL TECHNOLOGY (HBScCHT)

APROVAL FORM

The undersigned certify that they have supervised, read and recommend to the Bindura University of Science Education for the acceptance of a research dissertation entitled:

Optimization studies of gold cyanidation leaching process for an oxide gold ore.

Submitted by Rabson Chirikuutsi (B192033B)

In the partial fulfilment of the requirements for the Bachelor of Science Education Honors Degree in Chemical Technology.

………………… ……………25/05/2023……..

(Signature of student) *Date*

……………………….. …………………. ………26/05/2023…..................

(Signature of Supervisor) *Date*

………… ………………………… ..………04/10/2023……………...

(Signature of the Chairperson) *Date*

DECLARATION FORM

I, Rabson Chirikuutsi do hereby declare to the best of my knowledge to Bindura University of Science Education that this dissertation is my original work and all materials and academic sources of information other have been duly acknowledged. I also declare that this current work has not been submitted to any other academic institution for the purposes of an academic merit.

Signed …………………………………... Date ….......................................

DEDICATION

L dedicate this thesis to my son Jahmiel Tafara Chirikuutsi

ACKNOWLEDGEMENTS

I invested time and energy in this project. However, without the generous assistance of numerous individuals, it would not have been feasible. L want to express my sincere gratitude to each and every one of them. L want to express my sincere appreciation to Dr N. Muchanyereyi for her exemplary oversight and wise counsel. My sincere thanks go to members of Sabi Gold Mine for their guidance and constant supervision as well as for providing necessary information and equipments regarding the project. My thanks and appreciation also go to my parents, wife, sister and brother who helped me physically and financially throughout the thesis.

Table of Contents

LIST OF TABLES

LIST OF FIGURES

Abbreviations

AC	Activated Carbon
ACF	Activated Carbon Fibers
AC-PA	Activated Carbon prepared by Phosphoric Acid activation
AC-PH	Activated Carbon prepared by Potassium Hydroxide activation
AC-ZC	Activated Carbon prepared by Zinc Chloride activation
AAS	Atomic Absorption Spectrometer
Ca	Calcium
Cu	Copper
GAC	Granular Activated CARBON
H	Hydrogen
O.S	Oxidation State
PAC	Powdered Activated Carbon
PSD	Pore Size Distribution
SEM	Scanning Electron Microscope
WHO	World Health Organization

ABSTRACT

Gold recovery has been a significant difficulty to metallurgist today with lot of gold lost in solution to slimes dam. There is need to investigate optimum conditions for gold leaching. The present study sought to establish the optimum conditions for gold leaching. The conditions employed during leaching, which includes, pH , cyanide concentration and leaching period were investigated. In this thesis the bottle roll process was employed as a substitute of leaching tanks used in a plant scale to study the optimal conditions. The main aim being to allow 80 % > gold dissolution and 90% > reagent use to prevent their effects during adsorption.

Optimum conditions of the dissolution process were determined through bottle roll experiments. Lime and caustic soda, were contrasted to choose the one with highest recovery. In this study pH, leaching time, cyanide concentration, grinding time and particle size proved to be important parameters in the gold recovery**.** Comparison between lime and caustic soda as pH modifier showed that lime worked better on this type of ore. Maximum gold dissolution occurred at pH 11.5**.** Optimum cyanide concentration being 65.00 ppm. The size of ore was also compared with the -75 µm being the most effective. An experiment to check the effectiveness of the above conditions with relationship to AC capacity to bind gold was designed and tested. The overall adsoption was 93.7 % showing improvement with less amount of gold lost to slimes dam.

CHAPTER 1: INTRODUCTION

1.1 BACKGROUND

The mining sector is a conservative and economy driver of a country and over the years it continues to encourage innovation and technological changes on the economy (Verbrugge et al.,2021; Bansah et al., 2016; Kansake et al., 2021). The action of mining entails recovering of economically valuable resources from the earth surface as suggested by Cort´es-McPherson (2019). Minerals mined in Zimbabwe include platinum, gold and iron. Towards the end of the Middle Stone Age and the onset of the Neolithic Age, Gold was the first element recognized by human kind (ca. 800 B.C). Several procedures have been employed in the recovering from the ore to gold, which includes amalgamation and cyanidation. Cyanidation involves the use of cyanide in removing locked minerals from the ore due to its ability to dissolute metals found in ore (Sitando et al., 2018). The cyanidation of gold, being a process of choice after substituting amalgamation which have a variety of health effect (Hilson and McQuilken, 2014), have major process which includes leaching/ gold dissolution from ore and gold adsorption using activated carbon (Habashi et al., 2016).

The gold concentrates are added cyanide and lime or caustic soda in an agitated leach tank. This process is known as leaching, in which gold dissolution occurs from the ore into a solution under optimal conditions. Floculants are also added to improve settling rates and the concentrate is agitated vigorously to allow enough contact of cyanide with gold particle which form a gold-aurocyanide $[Au(CN)_2^-]$ ion which will be adsorbed by activated carbon (Davidson and Sole, 2007). Leaching of gold using cyanide is affected by availability of other minerals in the ore such as copper and calcium which readily dissolve in cyanide solutions resulting in the consumption of cyanide being at high level leading to high operational costs and fouling of AC during adsorption.

(Habashi, 2016; Sitando et al., 2020). The method for the leaching of gold is illustrated by the equation 1.1 below;

$$Au + 2CN^- \rightarrow Au(CN)_2^- \qquad 1.1$$

In water the gold cyanide ion is neutral and the adverse charge will be counted for by any positively charged ions which includes CU, Ca, Mg, H. For calcium, it joins with the gold cyanide ion to form a calcium gold cyanide ion pair as shown by equation 1.2 (Sitando et al., 2015).

$$2Au(CN)^- + Ca^{2t} \Leftrightarrow Ca(Au(CN)_2{}^-)_2 \qquad 1.2$$

The ions of opposite charges joined together to form a neutral specie, the duo of calcium and gold cyanide ions. Due to a vast of enormous interior floor space, AC will adsorb the neutral ion pair. Since the reaction is an equilibrium reaction, the percentage of gold cyanide ion and the ion pair (calcium gold cyanide ion) may be equal. Consequently, the ion pair rather than the gold cyanide ion will be absorbed by due to competitive inhibition (Dunne et al., 2012; Snyders et al., 2017). Since it is a physical adsorption, gold is recovered using AC can be affected by the external factors.

Other effects arising during leaching may come from copper and sulfides that may be in the ore which form a surface coating on the gold preventing its dissolution and also consumes leaching reagents (Senanayake, 2015). Stewart and Kappes (2012) stated that there is more cyanide consumption of more than 2.3 kg for copper containing ore although research is underway for recycling of cyanide. Beside the presence of other minerals as a hinderance to gold dissolution, there are other factors which affect gold leaching (Cheng et al., 2021). Less emphasis has been employed on the effect of these conditions such as the concentration of cyanide and leaching time thus research must be conducted under those headings. Some researches such as have studies other effects in leaching (Bellec et al., 2011).

The carbon atom and nitrogen with different O.S combine to form a triple bonded molecule know as cyanide. Under optimum conditions cyanide is an effective substance for gold dissolution but above this, cyanide will have a negative effect of gold adsorption using activated carbon. It will form complexes that will bind on the surface of activated carbon preventing it from absorbing gold will result in overall low gold recoveries (Medina et al., 2020). Particles size of the ore also contribute to effective leaching. The larger the particles size is the lesser the ability of cyanide to interact with the valuable mineral thus an increased dissolution occurring at smaller particles (Singh, 2012, Kotake et al., 2010). This creates a high surface area for diffusion of cyanide with the ore liberating gold due to an improved structure and texture of the ore.

According to Marsden (2006) and (Sitando et al., 2018) the surrounding pH is the primary factor affecting dissolution of metals from the ore. During leaching pH is either controlled by addition of caustic soda or lime with the two having different strength on different types of ore. The primary role of lime in cyanidation is to stop cyanide from being lost through hydrolysis (Plessis et al., 2021). Lime (CaO) in comparison to caustic soda ($NaOH$) offers a variety of operational benefits including eliminating carbonate and Hydrogen carbonate ions as solids from the leach liquid. According to research, immersion in solutions with varying pH levels has a substantial impact on the amount of heavy metal dissolution from the ore (Bellec et al., 2011; Sitando et al 2018).

The time taken for the leaching process to be complete is termed residence time, it is also a period that decides how quickly cyanide and oxygen react on free particle surfaces. In some ore, the reaction reaches its peak during the first few hours (Brittan et al., 2015). Other researchers suggested that the there is more gold dissolution with an increase in residence time till a certain point when there is no more dissolution and the reaction remains constant. After complete gold dissolution from the ore, it is the recovered from the solution using activated carbon due to its aptitude for absorbing metals in different systems (Acharya et al., 2009)

Activated carbon is a black, high surface area material made from charcoal or carbon-based materials (Rogans et al., 2012). The most suitable type of carbon is made from coconut shells from the Philippines and Sri Lanka. Charcoal was introduced to recover gold from chlorine leachate. Not long after cyanide was discovered to be an excellent solvent for gold in 1890, a method of extracting gold from cyanide using charcoal was patented. Before techniques for removing gold and silver from loaded carbon, filtration and zinc cementation were more competitive than gold adsorption using. In addition to having distinctive structures and features that are not present in other organic material, high porosity and surface area characterize activated (Bandosz., 2006).

1.2 JUSTIFICATION OF THE STUDY

The major problem that have been faced for years by mineral processors has been the effective liberation of valuable mineral from the ore hence this prompted this research in-order to find ways to overcome the problem (Aylmore et al., 2014). The dream of every chemist/metallurgist at a mining company is to obtain maximum gold recovery with minimum reagent consumption. The method by which gold is adsorbing to activated carbon is not a straightforward due to a number of external factors that have a positive or negative influence. Overall this will lead to a decrease of the gold output and increase in loss of gold to slimes dam, thus there is need for a research that focuses on the study of the effects of the external factors and how they can be improved to suit adsorption. The factors that affect adsorption of gold include:

- Mixing efficiency
- Carbon activity
- Particle size of carbon
- Organic poisons
- Residence time
- Types of activated carbon
- Ionic strength
- Temperature
- pH
- Cyanide concentration (Rogans et al., 2012)

This study focuses on key effects including cyanide concentration, residence time, pH, carbon activity, and particle size analysis. Meanwhile, the budget consumption of sodium cyanide is gradually increasing, and the excess consumption exceeds 1.25 kg/t. From an economic point of view, this increased operating costs and consequently reduced company profits.

1.3 AIM

Determination optimum condition for gold dissolution

1.4 OBJECTIVES

- To investigate factors that affect leaching/dissolution rate of gold

- To determine the optimum pH, leaching time, cyanide concentration, and particle size for gold dissolution
- To determine effect of particle size on gold recovery

CHAPTER 2: LITERATURE REVIEW

2.1 INTRODUCTION

2.2.1 OPERATION OVERVIEW

The process of gold recovery has a number of stages that include comminution, milling, dewatering, leaching and smelting. Comminution is the reduction of run-off mine ore (ROM) particle size by the use of mechanical crushers.

2.2.2 CRUSHING

The initial stage of the communition procedure is crushing, whose main aim is extraction of precious minerals out of the ore (Hodouin et al., 2011). The crushing of ore before fine comminution decreases the energy consumption of mining operations (Rylander and Axelsson, 2013). The section is divided into three parts: Primary crushing is the first stage in the size reduction whose aim is to reduce ore particle size from 250 mm down to less than 65 mm (i.e. – 65 mm) and a jaw crusher is used. Secondary crushing involves close size monitoring since the product is now being prepared for milling, a cone crusher is used to meet the demands within the operation constraints (Quist, 2017). Tertiary crushing – at this stage ore particle size is reduced to 3 mm and barmac crusher is used.

2.2.2.1 OPEN CIRCUIT CRUSHING

In open circuit crushing, ore is screened only once and the oversize material is crushed. Undersized material combines with crusher product to give the final product

2.2.2.2 CLOSED CIRCUIT CRUSHING

The end product in closed circuit crushing is the undersized screen fragments. To ensure that any large material is recirculated, the crusher product is kept on the screen.

2.2.3 MILLING

The final step in the comminution process, grinding, reduces the size of the ore particles using a combination of impact and abrasion (Hulthen et al., 2010) The goal of milling is to reduce ore size to a level at which the valuable material can be economically liberated. Ball mills are volving, cylindrical steel tanks used for this. Two mills are used that is the primary and secondary mill (regrind), the pulp gravitates to the thickener. The most commonly used media for the ball mill are the steel and other metal balls ranging from 10 to 150 mm in diameter (Kuzev et al., 2009). The ball mills are also widely used in industry for blending, mixing and dispersing of materials and mechanical alloying (Kotake et al., 2010)

2.2.4 THICKENING

Thickening is a process primarily used to dewater pulp, thereby producing a solids-rich sludge for the mineral industry (Gladman et al., 2006). That is, this is said to increase the pulp density of the feed. Thickening is achieved through the use of substances called flocculants. The majority of the time, this is a natural material having active sites where tinny solid particles bond to one another. Agglomerated particles accumulate at the tank’s bottom.

2.2.5 PRECONDITIONING AND CARBON-IN- LEACH SECTION

The purpose of this step is to prepare the pulp for gold leaching. Pretreatment takes place in four adjacent cells, each of which is mechanically agitated and air blown from the bottom. The solution is richer in gold than the leachate (Xu et al., 2018). In the first cell, the pulp is mixed with caustic soda to raise the pH to 10-11. The pulp mixed with caustic soda in the first cell, from the second it overflows into the fourth cell, the pulp then passes through the Delkor linear screen and into the pretreatment sump. In the fourth cell, the pulp should have an oxygen level of 5 ppm and a pH of 10. This is ideal for gold dissolution and alkaline protection of CN^- that forms a very toxic gas HCN at pH less than 7 according to equation 2.2 (Marsden and House, 2006):

$$H_{(aq)} + CN^{-} \rightarrow HCN \qquad 2.2$$

Here leaching still continues with mechanical agitation and oxygen being added. These tanks are in series and are aligned in such a way that pulp flows by gravity from the leach tank to CIL tanks and then gold will be absorbed onto activated carbon using the CIP process (Zhang et al., 2004).

2.2.6 ELUTION

After the carbon has been transferred to the Carbon In Leach (CIL), it is then air lifted to the screen. This is a vibrating screen with a spray of water that washes the carbon. The carbon is then laundered under gravity to a measuring vessel whose volume is 2.53 m^3. A calibrated dip is used to check the volume of this vessel and when full it contains 1.26t of carbon loaded with gold. After that, an elution column is used to hold the loaded carbon, where the first phase of elution starts with temperatures being monitored (Oladele, 2015). The loaded carbon is then transferred to into an elution column where the first stage of elution begins.

2.2.7 ELECTRO-WINNING

This is a process that uses electrodes immersed in an electrolyte to reduce gold cyanide to solid gold (Lunga et al., 2006). This is a continuous process lasting 24 hours and performed in four identical cylindrical cells. However, physical factors have an impact on the and electrochemical parameters like temperature, current density and electrolyte flow rate. The anolyte is a 30 %

solution of sodium hydroxide and the catholyte is a highly doped gold solution. As the catholyte is pumped from the catholyte tank, it passes through a Scheibler filter that removes solids that have leaked through the screen. Catholyte from the cell discharge is pumped back into the catholyte tank. Electro-winning is the process of recovering metals that are separated from ores in solution and cannot be precipitated (Paul et al., 1983). At local demining plants, electrowinning is a batch process. In both cases electricity generation takes place from the catholyte tank (Lunga et al., 2006).

2.2.8 SMELTING

Before the wire wool melts, it is dried to remove moisture and volatile liquids that wet the wool. This is done in a retort of about 350 °C for 3 hours. Dry steel wool and flux are poured in alternating layers into an AC 60 carbon crucible until the crucible is full. The crucible is placed in a coke-burning oven with the fire already burning. The molten gold and slag are poured into a mold, cooled, the gold and slag separated outside the mold with a hammer, and the gold washed with dilute nitric acid. Once the bar is clean, it is weighed and found to be about 98 % pure with some silver (Styles et al., 2010).

2.2.9 FLOTATION

Flotation is a concentration process that separates valuable minerals from gangue by exploiting the physicochemical properties of various minerals exposed to specific reagents (Tschakert and Singh, 2007). During flotation enrichment, valuable minerals are usually transferred to the foam a fraction (direct flotation). However, valuable minerals may remain in the sink area (indirect flotation). The bottoms from adsorption tank (after cyanide decomposition) are sent to the holding tank where the pulp is conditioned and potassium amyl xanthate is added to the pulp. PAX is a collector and makes copper bearings hydrophobic by forming a thin oily, non-wetting film on the mineral surface. The concentrate from the final cell is sent to a sump that feeds the processing tanks of the concentrate leach plant and the tailings are disposed of at the tailings dam.

2.3 GOLD DISSOLUTION

Gold dissolution refers to a process in which gold is liberated from the ore into a solution. Gold dissolution occurs during laboratory fire assaying process and in plant during cyanidation/ leaching process.

2.3.1GOLD DISSOLUTION IN THE LABORATORY

Fire assaying is an established method used to liberate precious metals from the ore by using a combination of dry reagents which includes litharge, fluxes, soda ash and high temperatures using furnaces (Battaini et al., 2014).

2.3.1.1 CRUSHING AND PULVERIZING

Samples containing large particles are first crushed utilizing a small crusher in the laboratory. The jaw crusher is used to reduce samples with sizes +6 mm to -6 mm. After crushing a sample, compressed air is blown into the crusher and also the receiving tray to minimize contamination of samples that follow which in turn leads to false inaccurate results. After crushing, the samples are taken for pulverizing. A pulverizer uses an external force to reduce ore to a size of -75µm (Singh, 2012).

2.3.1.2 WEIGHING AND FLUXING

After pulverizing, the samples are weighed using an electronic balance, only 25g of each sample are used. During weighing 75g of flux is added into the crucible and 25g of sample added. Silver nitrate is also added this helps to trap the gold in the sample. A universal flux contains 6000g litharge (mainly lead- a heavy metal), 800 soda ash, 400g silica, 600g mealie-meal (a source of carbon) and 600g of Borax (Battaini et al., 2014). After weighing the remaining samples are stacked onto shelves for at least 3 months for references' sake.

2.3.1.3 FLUX PREPARATION

Flux is a mixture soda ash, borax, silica, litharge, mealie- meal. Table 2.3 shows how amounts of sample and flux used differ depending on samples origins (Balaram et al., 2022):

Table 2.3: Flux preparation

REAGENT	MASS(Kg)	ROLE
Litharge	18	Captures all heavy metals
Soda ash	24	Oxidizing agent
Borax	18	Adds fluidity to sample
Mealie meal	1.8	Source of carbon
Silica	1.8	Prevents corrosion of crucibles and cupels

2.3.1.4 FUSION AND CUPELLATION

After the weighing and fluxing comes the fusion and cupellation process (Battaini et al., 2014). The crucibles with the fluxed sample are placed in a fusion (furnace) heater at a maximum temperature of 1000 °C for 40 minutes. When the whole sample is molten, the crucibles are withdrawn from the furnace using tongs a nd poured into some iron pigs. When the slag has solidified the pigs are turned up-side-down, the slag will be at the bottom and the lead button at the top. Using a hammer, the slag is separated from the lead button. The lead button is then hammered flat and placed into cupels followed by placement into the furnace for cupellation procedure. This process takes place at a temperature between 890-1063°C for about 50 minutes (Battaini et al., 2014).

2.3.1.5 PARTING

When the cupels are removed from the furnace, only prills are left in the cupels. These prills are then hammered to increase their surface area, and then they are placed in porcelain crucibles. A parting acid 30% HNO_3 and 70% distilled water added to each crucible. The crucibles are then heated to increase rate of the reaction. When all the silver has been dissolved, the remaining gold fragment is washed at least twice with distilled water, with the latter being poured in a large container in order to recycle the silver. After washing, Aqua Regia added which is in the ratio 1:3 to HCL respectively. This combination then dissolves the gold into a solution later read by AAS machine. The final solution is poured into 100ml volumetric flasks and distilled water is added up to the 100ml point (Battaini et al., 2014).

2.3.1.6 ATOMIC ABSOPTION SPECTROPHOTOMETER

Varian 240FS AAS gives the grade of the solution, i.e. the amount of gold in the given sample in solution form in g/t. Acetylene is turned on and the pressure reads between 1.25 and 1.30Pa, atmospheric air is also turned on (Panova et al., 2013). The acetylene is ignited and burns with a yellow flame. A beam of light is projected from a light-emitting gold glow bulb. By means of a suction tube, the gold solution is sucked into the AAS machine. The solution is then burnt and dried. It is the glow bar that then gives a reading on the screen. Each mineral/precious metal has its own glow bulb. This then means that only gold can be detected when gold glow bulbs are inserted in the AAS (Panova et al., 2013).

2.4 GOLD DISSOLUTION IN THE PLANT

After the ore has passed the crushing and grinding stages, it undergoes the thickening which prepares the slurry for gold dissolution, then enters the leaching where dissolution occurs by addition of cyanide (cyanidation) and lime. However, the concentration of reagent varies with the type of ore, sulfide ores requires high cyanide and lime concentration (Muir, 2011) while some gold-coppers ores requires low cyanide concentration

Thickening also known as dewatering is a process of removal of excess water for the preparation of the feed for subsequent processes e.g. addition of reagents (Gladman et al., 2006). The process proceeds by sedimentation. When the solids and the liquid have a significant density difference, sedimentation is most effective. The majority of liquid is removed producing a thicker pulp with 55 – 65% solids by sedimentation. The principle of gravity sedimentation used by the thickener is called settling. Rapid solid-liquid settling results in a clear booze that requires filtration to remove water. Settling can be free or hindered. Hindered settling is a situation which leads to sliming (Kim et al., 2004). Sliming can be rectified by addition of dissolved flocculent on the sliming particles. Flocculent quickens the rate at which particles settle because it makes particle agglomerate (come together) quickly. Depending on the particles size, Stokes and Newton's law influences how these particle settle. Sliming can be rectified by addition of dissolved flocculent on the sliming particles. Flocculent quickens the rate at which particles settle because it makes particle agglomerate (come together) quickly. The following controlled operating parameters on table 2.4 apply during the function of the continuous thickener (Gladman et al 2006):

Table 1.4: Thickener reagents

thickener feed	10-15% solids
thickener overflow	clear solution
Thickener	45-50% solids
Pressure	80-100KPa
Lime	0.01- 0.015%

Thickener underflow at 45%-50% solids from the thickener is pumped by a warman pump into the leach tank past a trash screen for removal of wood-chips, wires, plastics and rubbers before the process of leaching. The pulp is fed into the leach tank through a pipe which extends right into the middle of the leach tank to about ½ the tank height. This is done so as to increase the residence time of the pulp in the tank before ejection. Leaching proceeds at different residence time with respect to the type of ore to achieve 92% leaching efficiency. Cyanide is added to facilitate gold dissolution from the pulp. Cyanided pulp enters the leach tank through a hollow pipe which fills the leach tank from the bottom.

2.4.1 THE CHEMISTRY OF GOLD DISSOLUTION

With the right oxidizing agent present (such as chlorine or ferric ion) gold dissolves easily in aqueous chloride solution to create tetrachloroauric ion and possibly the ion separates as a salt by evaporation (Oraby et al., 2014). Evaluation of cyanide-based leaching of gold is a very common methodology (Habashi et al., 2016). Provided that necessary precautions are put in place, the method has proved to be effective despite risks associated with cyanide. The cyanide concentration is frequently maintained at an extreme level because miners in small scale mining think it will benefit the procedure (Hilson and McQuilken, 2014). Investigations suggested that cyanidation of gold is a diffusion-controlled reaction. Air consumption rates are based on the need to maintain sufficient oxygen in the solution and in the stirring tank to ensure adequate agitation, and if the desired optimum rate at which oxygen is injected cannot be achieved, pure gold may be low due to the low dissolution as a result mining operations are well below potential (Sitando et al.,2015). According to Elsner's equation, the gold dissolution process proceeds as follows (Snyders et al., 2017; Muir, 2011):

$$4Au + 8KCN + 2H_2O + O_2 \rightarrow 4KAu(CN)_2 + 4HCN \qquad 2.3$$

Compressed air is blown to facilitate agitation. Residence time of particles during leaching is very crucial and is calculated as follows (Egan et al 2016). The time taken to fill the active leach tank volume determines the leaching period of particles during leaching but this does not mean that leaching ends here (Brittain et al., 2015). It continues through the adsorption tanks where leaching and adsorption takes place simultaneously. When the leach tank is full, because of agitation, pulp will start to flow to the first adsorption tank and the process continues through all the six adsorption tanks.

2.4.2 TITRATION PROCEDURE

This method is employed to determine cyanide free-ion content in the tanks (Dunne et al., 2012). The pulp sample is taken from the discharge point of the leach tank. It is then filtered using pressure filters and the filtrate collected for titration 25ml is pipetted into a beaker and start titration for cyanide by adding potassium iodide drop by drop in the beaker. Fill the burette with the titrant, $AgNO_3$. Observe and note down the burette reading. Drop by drop titration is done into the beaker, gently shaking the beaker, observing the color change. The color changes from colorless to a light greenish to indicate the end point of the reaction. The titration is stopped and noted down the final reading. Subtraction of the last reading from the first and dividing the result by 100 gives the cyanide concentration (Dunne et al 2012).

After the first titration, the contents of the burette are removed and the burette rinsed with distilled water. (Another burette may be used). The burette is filled with oxalic acid and addition of a drop of phenolphthalein indicator to the beaker and observing the color change from light green to pink. Titration is repeated as above and observe the color change from pink to creamy white. The percentage concentration of lime can be calculated using the equation 2.4.3 (Zanin et al., 2019). A pH meter can be used for determining the pH of the pulp by deeping the pH meter into the filtrate and reading the pH on the pH meter (Bellec et al., 2011; Muir, 2011).

2.5 GOLD ADSORPTION IN THE PLANT

After cyanidation the pulp will proceed to adsorption tanking which vary in number as per mine. Here is where activated carbon adsorb gold. Black, highly porous material called activated carbon is made from charcoal or carbon-based materials (Zhang et al., 2004). The most suitable type of carbon is made from coconut shells from the Philippines and Sri Lanka. Charcoal was introduced to recover gold from chlorine leachate. Not long after cyanide was discovered to be an excellent solvent for gold in 1890, a method of extracting gold from cyanide using charcoal was patented (Bandosz, 2006). High levels of oxygen and hydrogen are found in activated carbon, which also has special features that are not present in other natural compound.

2.5.1 TYPES OF CARBON MATERIAL

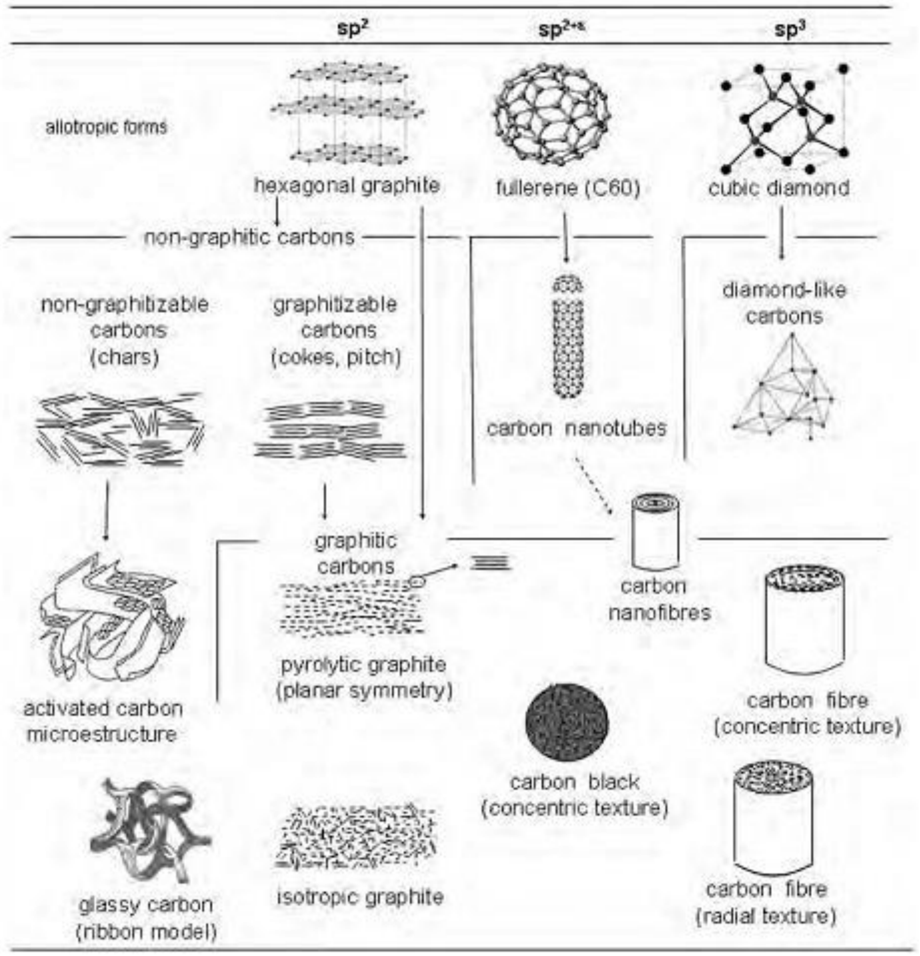


Figure 2.5: Major allotropic form of carbon (Bandosz., 2016)

2.5.2 USES OF ACTIVATION CARBON

In the industry, AC is probably amongst the most often utilized adsorbent in environmental application such as water purification, Pharmaceuticals, separation and gold recovery processes (Achayra et al., 2009; Adhoum, 2004). To eliminate natural and contaminants, AC is employed most frequently because of large internal structure and its unique appearance. Above all applications of AC is environmental friendly and pose less damage to the environment and it is cheap to acquire in bulk (Faria et al., 2004)

CHAPTER 3: MATERIALS AND METHODS

3.0. METHODS

To investigate the aim of research an array of tests was conducted in a laboratory and on a plant scale for the purpose of comparing and contrasting plant activities. The plant results present the actual operation on the ground in which both activated carbon gold solution is allowed to reach equilibrium over a number of days. In logical sequence to achieve or fulfill the goals of this project, the investigation was done in the following manner:

3.1 MATERIALS

The gold ore samples, apparatus and reagents used throughout the research originated from a local gold mine and the source of ore are disclosed due to commercial restrictions. Samples were taken through-out the month of January 2023 to conduct the experiment. Before being collected from the point of collection the ore undergoes crushing using jaw crushers, tailsmith and barmac crushers, milling using ball mill and dewatering to mention a few of the processes (Hulth'en et al., 2010). A daily concentrate leach plant feed of approximately 8 l were collected for a week and were treated as follows: the solid were cleaned using faucet water to remove any cyanide and residual dissolved gold. The solids were oven-dried at 200 ^{o}C. A weekly composite of concentrates leach plant feed were prepared after a week and then kept in storage room. Samples were collected continuously and after a month the weekly concentrates collected through-out the month were mixed and treated for different experiments. The experiments were compared to the month assayed gold for the plant given in an assay sheet. The particles used in leaching operations were typically 250 nm chosen with the view to stay consistent with small scale and independent mining (ASGM) practices (Zhang et al., 2004). All the experiments were done under room temperature and other conditions and reagents used differs as mentioned in each chapter.

3.2 REAGENTS

The reagents used throughout the project varies and are stated within the methods with the most important ones being:

- Cyanide
- Lime
- Caustic soda

3.3.1 DETERMINATION OF OPTIMUM ALKALINITY

In order to pinpoint optimal conditions, various conditions and ore amounts were used to determine ore susceptibility to cyanidation and leaching (Sitando et al., 2018). In this experiment, 5 kg was leached at 50 % solids with a pH gradient while the cyanide concentration (0.36 %) and residence time (24 hours) remained constant. The first test was to compare the effectiveness of lime or caustic soda as pH modifiers in this experiment. The amount of gold before the procedure was recorded as head solution and gold left undissolved labeled tail solid. The pH value for the first experiment was staged at 10. After finding the correct/ effective chemical for leaching in the second test pH was varied from 9 to 14 and samples were periodically taken for assaying from 30 minutes to 24 hrs. A graph of pH versus % dissolution of gold was plotted. As suggested by (Sitando et al., 2018), the bottle roll process was done for this process with the rate of bottle rolling kept constant at 30-32 rpm as required by the procedure.

3.3.2 DETERMINATION OF OPTIMUM RESIDENCE TIME/LEACHING TIME

In this experiment the bottle role method was used which is a well-established procedure for the basis assessment of the leachability of ore with respect to different conditions so as to minimizes gold loses (Brittan et al., 2015). The following apparatus and reagents were used: bottle roll machine, empty 2 l bottles, measuring cylinders, pH meter, test tubes, sample plates, sample bottle, sodium cyanide and lime. The monthly composite of leach feed was mixed thoroughly and split using a riffle splitter to reduce the size of the ore. A sample of 1kg each was weighed and placed into the sample plate. 0.36 % liquid cyanide was prepared and 1 g/l of lime was introduced into one 2 l measuring cylinder. A pH solution of 10.5 was prepared using lime. Small holes were opened on the bottle for oxygen supply and the sample was placed on the bottle role machine with the samples taken periodically for assaying 30 minutes, 1 hour, 2 hours, 4 hours 8 hours, 24 hours and after 48 hrs. The rate of bottle rolling was kept constant at 30-32 rpm as required by the procedure (Muir., 2011). Table 3.3 show some of the experimental conditions employed for this experiment.

Table 2.3: Determination of optimum residence time

PARAMETER	SPECIFICATION
Temperature	Room temperature
Cyanide concentration	0.36%
pH	10.5>
MASS OF ORE	1kg/sample

3.3.3 DETERMINATION OF OPTIMUM CYANIDE CONCENTRATION

In this experiment the concentration used was 5 g/l (Medina et al., 2020). For dilute systems in commercial applications, concentration of cyanide can vary from 0.1 g/l to 0.5 g/l (Hilson et al., 2007). Bottle roll test was employed for the establishment of ideal cyanide concentration for gold disintegration into the solution. In this experiment a 500 g sample was weighed using a balance from the monthly concentrates leach plant feed after they were mixed. The rate of bottle rolling was kept constant at 30-32 rpm as required by the procedure (Muir, 2011). The pregnant solution was sampled after certain period of time for gold analysis and plotting the gold extraction and cyanide consumption data yielded the leaching profile and the cyanide consumption tendency of ore. The Table 3.3 shows a description of the test that were employed in the course of this experiment

Table 3.3: Determination of optimum cyanide concentration

PARAMETER	SPECIFICATION
Temperature	Room temperature
Mass of ore used	500 g
Reagent concentration	5 g/l
pH	10.5>
Particle size	50 % solid by mass

Lime was added to maintain pH

3.3.4 GRIND SIZE EFFECT

After reaching the maximum conditions for gold adsorption, further tests were performed to identify the effect of milling. Impact of milling period and milling size on gold adsorption. The primary aim is to determine how grind size affects the dissolution of gold from ore within a specific size interval (Bellec et al., 2011). Samples were collected from the ball mill feed and used for analysis. A standardized ore (-12 mm) was used. The ore container was removed. This ore was used in all experimental work. Lab jaw crushers, grinders/pitch mills, sample sheets, spatulas, scales, sample bags, and micron sieves of: 75, 150, 300, 425, and 600 µm. 10 kg of standardized ore was crushed with a laboratory jaw crusher and crush the ore with a laboratory jaw crusher

3.3.4.1 ESTABLISHING A GOLD CONCENTRATION BAR GRAPH

10 kg crushed ore was pulverized randomly and then sieved using the sieves listed above. A sample was taken to the assay laboratory for average grade analysis. 1.150 kg screen undersize was collected from each sieve. These were labeled and packed in sample bags. Conning and quartering sampling method was used to collect a 150 g sample from each packet. Samples were fire assayed for gold analysis. The laboratory jaw crusher was used to reduce samples with sizes +6 mm to -6 mm and the sample was further reduced to -75 µm using a pulvarizer. After pulverizing, the samples are weighed using an electronic balance, only 25 g of each sample was used. During weighing 75 g of flux was added into the crucible and 25 g of sample added. Silver nitrate was added this helps to trap the gold in the sample. Flux contained litharge, silica, mealie meal, soda ash and borax. The crucible was placed into a fusion furnace and then cupellation furnace and after that parting was done using parting acid (30% HNO_3 and 70% distilled water) and washing using aqua regia solution. This combination dissolves gold into a solution which is then read using AAS.

3.3.4.2 ESTABLISHING A LABORATORY GRIND CURVE

A grinding curve was needed to establish the time required to reach an optimal screen grind of 80% passing through -75 µm. A 1 kg sample was taken from the crushed ore and 150 g of this sample was batch ground for 15, 30, 45, 60 and 75 seconds. Each crushed ore was screened using a -75 µm screen and a vibrating screen shaker. The undersize was weighed, the results recorded, and the polishing curve plotted. A 25 x 1.1 kg, -75 µm sample was bagged and stored for further testing.

3.3.5 ANALYSIS OF PARTICLE SIZE

The size of particle influences how ore leaching recovers gold with a reduction in the size of particle as shown by the experimental results studied using mono-size particles (Deschˆenes et al., 2011). The ore used in this test work was taken from the remains of grinding analysis. The samples from each of the packed 80% passing -75, -150, 300, 425 and 600 µm samples were sent for analysis at the mine laboratory for fire assaying (Egan, 2016). Five samples with the above size were treated in the following way: 1g lime was added, 943 ml water for pulping to achieve 47 % solids, 1.8 ml dissolved cyanide was added. Compressed air was blown to enhance agitation and samples were taken, filtered, collected for titration for cyanide and pH concentration (Egan, 2016). The table 3.3 shows a summary of the conditions used (Singh et al., 2012):

Table 3.3: Analysis of particle size

Tank no.	Ore Quantity (kg)	Amount of water (ml)	Grind: 80% passing (µm)	Lime (g)	Dissolved cyanide (ml)
1	1	943	-600	1.0	1.8
2	1	943	-425	1.0	1.8

3	1	943	-300	1.0	1.8
4	1	943	-150	1.0	1.8
5	1	943	-75	1.0	1.8

3.3.6 ACTIVATED CARBON ACTIVITY TEST

A laboratory pilot test was designed based on the carbon activity test. Activated carbon used during the experiment was provided commercially by the company (supplied by STC chemicals) and was eluted carbon from the plant. Lime was used as a pH modifier and greater than or equal to 11.5, particle size of the slurry was -75 µm, cyanide concentration set at 65.00 ppm and the beaker placed with a magnetic stirrer was placed on a magnetic stirrer for 48 hrs. After gold dissolution for 48 hours the sample was tested for gold content (head solution) and activated carbon was then added. Samples were taken after 5 minutes, 15 minutes and 30 minutes. Afterwards activated carbon was tested for fouling and other effects caused by excess cyanide remaining after leaching. Since eluted carbon was used in this experiment the activity was calculated using equation 3.3;

$$\frac{\text{Head solution - value of carbon used}}{\text{value of head solution - fresh carbon}} \times 100$$

3.3

$$= \text{Average activity of carbon}$$

CHAPTER 4: RESULTS AND DISCUSSION

4.1 OPTIMUM PH DETERMINATION

Table 4.1: Comparison between lime and caustic soda on effective gold dissolution

TIME	LIME	CAUSTIC
30minutes	1.42	0.87
1hr	1.54	0.84
2hrs	1.34	1.07
4hrs	2.23	1.01
8hrs	1.81	1.77
24hrs	1.27	1.27
HEAD solid	2.75	2.75
TAIL solid	1.43	1.60

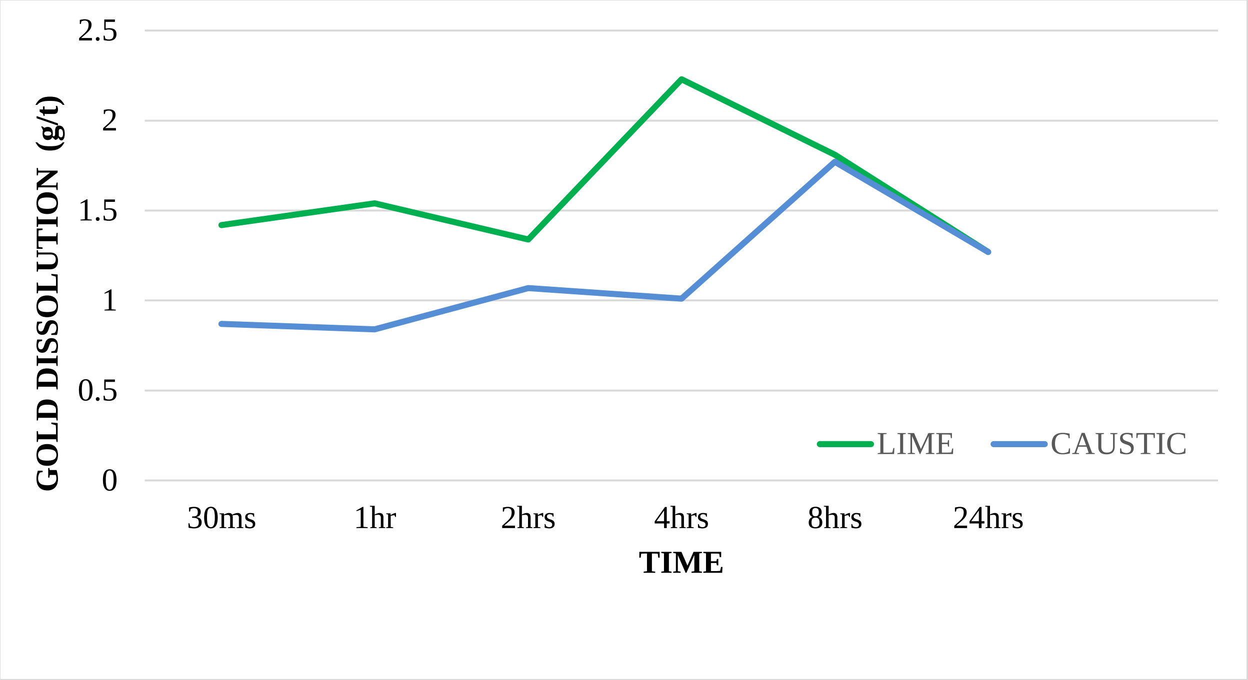


Figure 4.1: Gold dissolution using lime and caustic soda

$$\text{Leaching recovery} = \frac{\text{head - tail}}{\text{head}} \times 100 \qquad 4.1$$

The objective of this process was to compare the impact of using either lime or caustic soda on this type of ore using the leaching/ residence time of 24 hrs. From Figure 4.1 it can be observed that using lime at pH of 10.5 and a CN concentration of 0.36 %, leaching proved to be faster as compared to caustic leach. Lime proved to be the most effective in this cyanidation process by maintaining proper pH and its prevention of hydrogen cyanide gas formation thus preventing loss of cyanide by hydrolysis (Plessis et al., 2021). The graph showed that from 30 minutes to 8 hours, lime had a better gold dissolution compared to caustic and there was a rapid decrease from 8 hours to 24 hours. du Plessis (2021) suggested that this might be caused by the production of calcium hydroxide within the surface of gold. The results above also corresponded to (Oraby and

Eksteen, 2014) who suggested that in the beginning of cyanidation there is a slight increase in dissolution of gold and decreases as the leaching process advances. Equation 4.1 was used to calculate the leach recovery using lime and caustic with the recorded values of 48 % and 42% respectively. Lime also acted as a flocculant/coagulant, removal of $CO_3^- \ and\ HCO_3^-$ from the leach liquor and its ability to increase the settling rate of particles compared to caustic soda resulted in lime having a better leach recovery. Although caustic soda has various advantages compared to lime such as high solubility, high effectiveness in neutralization of weak and strong acid but on this type of ore it proved to be a set-back so lime was the main pH modifier throughout. Aside from their advantages, caustic soda (sodium hydroxide) is expensive and corrosive thus its use has proven to be a possible drawback to leaching (Medina et al., 2020). When caustic soda reacts it also forms a gelatinous precipitate which cannot be separated from the particles and these precipitates have the ability to passivate the mineral surface and hinder the leaching of gold (Medina et al., 2020). From the graph it can be observed that highest gold dissolution was at 4 hours and it decrease after 24 hours. This may have been caused by a decrease in gold content or other possible factors so the same recovery was compared to a 48hr leaching time to compare the recovery.

4.2 DETERMINATION OF ALKALINITY CONDITIONS (LIME VARIED)

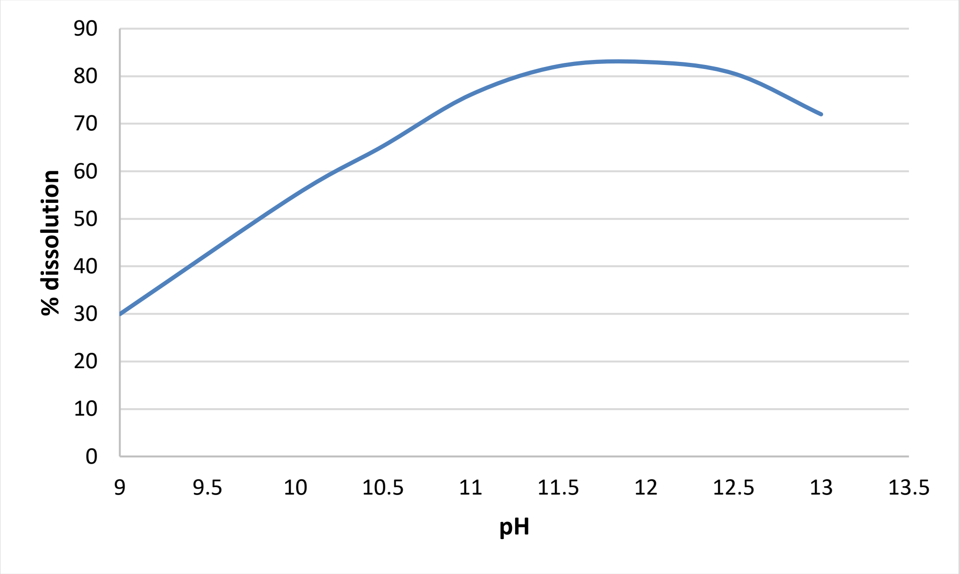


Figure 4.2: Determination of optimum pH

Having done the above trials based on ideal protective alkalinity conditions for gold dissolution while cyanide concentration (0.36 %) and residence time remain (24 hours) at constant the results yielded the above graph. A pH of 11.5 yielded the highest percentage dissolution of gold at 82 %. This can be due to the calcium ions in lime which are capable of bridging particles together thereby increasing the viscosity and settling velocity of particles thus high gold recovery (Zanin et al., 2019). The percentage dissolution increases from pH of 9 to 12 and starts to fall as reported by du Plessis (2021) that the speed in which gold dissolves is high within the first hours and then decreases with time. Thiousulfate degrading products such as tetrathionate has been studied as factors causing gold loss during leaching so leaching at higher pH values helps in hydrolysis of these products. Further rise in pH results in a reduction in percentage dissolution. This is due to the increase in ammonia and thiosulfate consumption. An increase in the rate of interfering reactions like sulphides, antimony and arsenic minerals also causes the decrease in %

dissolution showing that it has reached its optimum it cannot leach much further than this. Alkalinity should be maintained at the minimum level necessary to afford protection to the cyanide ion, thus optimum pH must be staged at 11.5.

4.3 DERTEMINATION OF OPTIMUM LEACHING TIME

Table 4.3: Determination of optimum leaching time

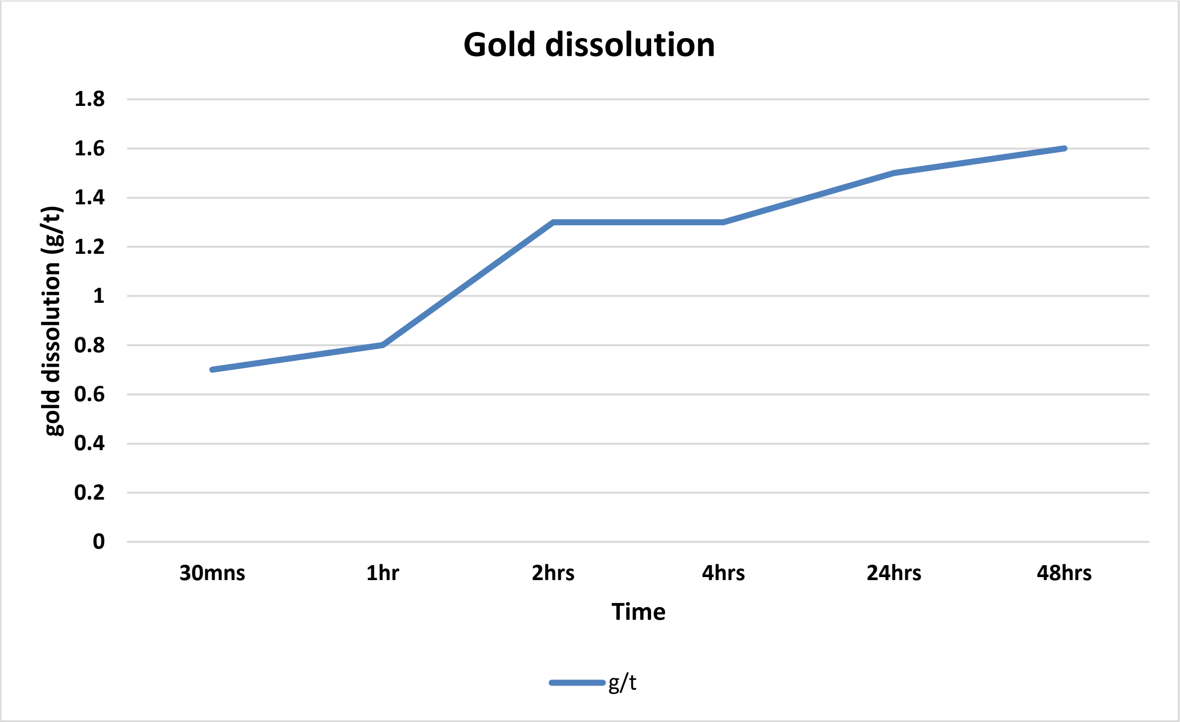
Time (hrs)	**EXPERIMENT**			**Standard deviation**
	A	**B**	**C**	
30 min	0.2	1.3	0.6	0.7 ± 0.56
1	0.3	1.1	0.8	0.7 ± 0.44
2	0.3	1.2	0.4	0.6 ± 0.49
4	0.6	1.2	2.1	1.3 ± 0.75
24	0.5	1.4	2.6	1.5 ± 1.05
48	0.7	1.5	2.6	1.6 ± 0.95

Figure 1.3: Determination of optimum leaching time

Head= 1.68

Tail= 0.2

From the last bottle roll the leaching recovery was 48 % after 24 hours. Residence time was then moved from 24 hours to 48 hours, equation 4.1 was used for calculations resulting in 88 % gold



recovery hence higher residence time results in high gold recovery. The above graph and calculations shows that as leaching period increases the recovery also becomes better. From the graph it can be seen that from 30 minutes to 1 hour there was a slow increase, from 1hour to 2 hours the leaching efficient increases with leaching time and from 2 -24 hours it is almost unchanged and then increases up to 48 hours. This may be explained from the following perspective. More gold can be leached from the gold particles the longer they remain in contact with cyanide in the leaching tank.

4.4 DERTEMINATION OF OPTIMUM CYANICE CONCENTRATION

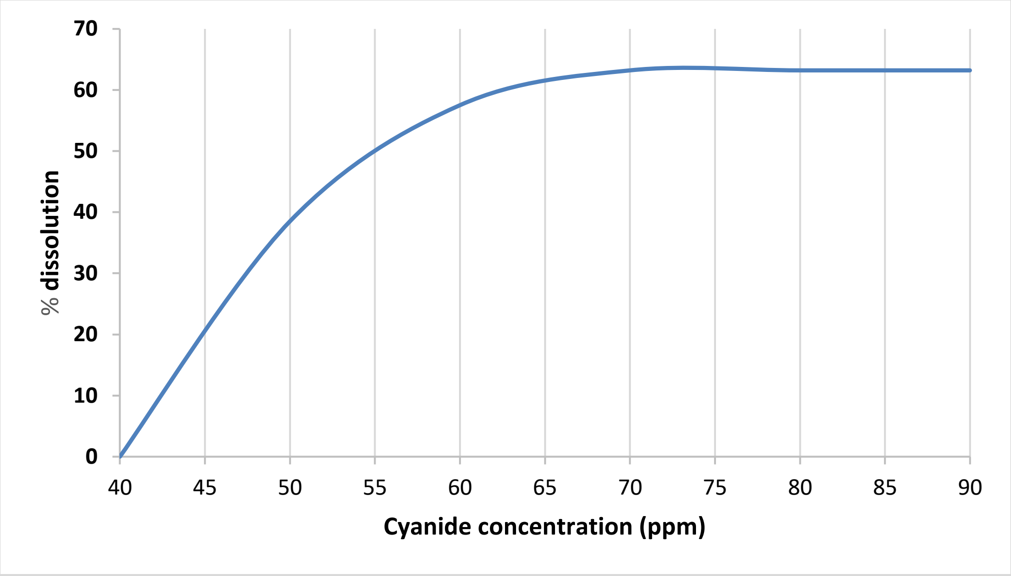


Figure 4.4: Determination of optimum cyanide

When the level of cyanide is elevated, the rate of gold dissolution typically rise until it reaches a maximum level at which adding more cyanide does not dissolve any additional gold. The above graph shows that the highest goal dissolution of 63.2 % occurs at 65.00 ppm of cyanide and dissolution after that does not reach any further increase. Cyanide might be used by side reaction which increases its consumption (Bakatula et al., 2016) Further increase of cyanide will however have side effects during gold adsorption by activated carbon by producing a auro-cyanide ion which will be a competitive inhibitor for absorption. Analyzing the above graph shows that operating at less-than ideal cyanide has detrimental effect on recovery of gold, it only increases the cyanide consumption which results in high operational costs.

4.4.1 LABORATORY GRINDING

The table below shows the results of laboratory grinding carried out to analyze gold concentration and determine the grinding time required to obtain a grind of 80 % passing -75 µm. The average grade of the ore used was 6.55 g/t. The table 4.4.1 shows the gold concentration results:

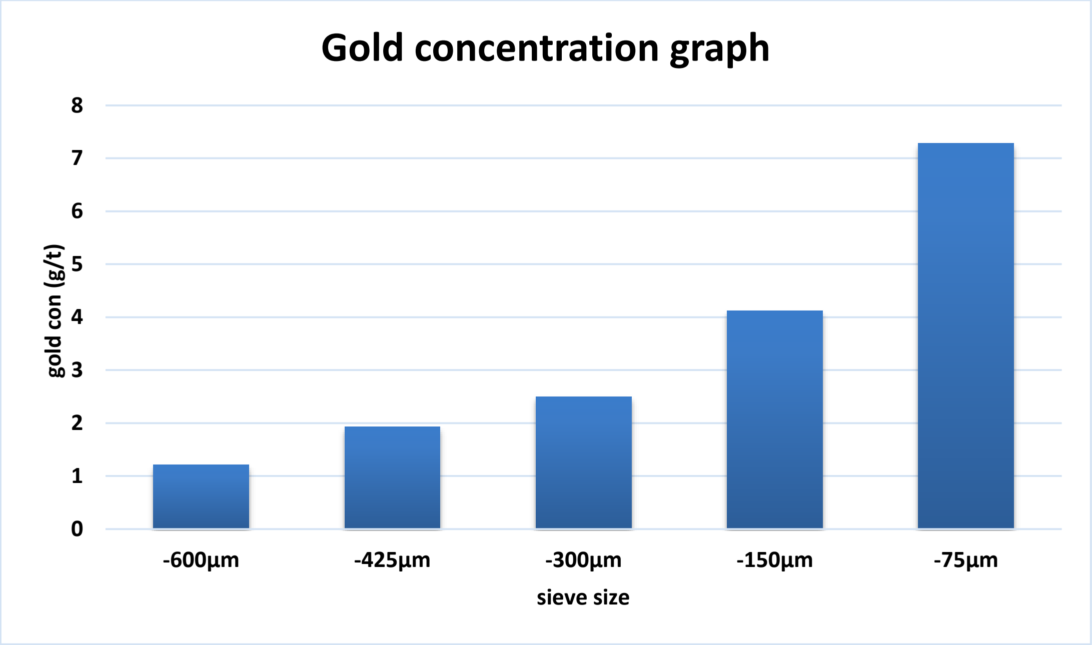


Figure 4.4: Gold concentration graph

According to the bar graph above with reduction in size of particles, the dissolution increases with the -75 µm sieve attaining the highest gold concentration. This observation is in line with earlier research by different authors. (Singh et al 2012; Kotake et al., 2010). This may be due to an increased contact surface area associated with finer particles which gives high dissolution. Smaller particles also prevented the passivation of the surface of gold grains. The next objective was to find the time required to find the grinding time needed to attain higher recovery.

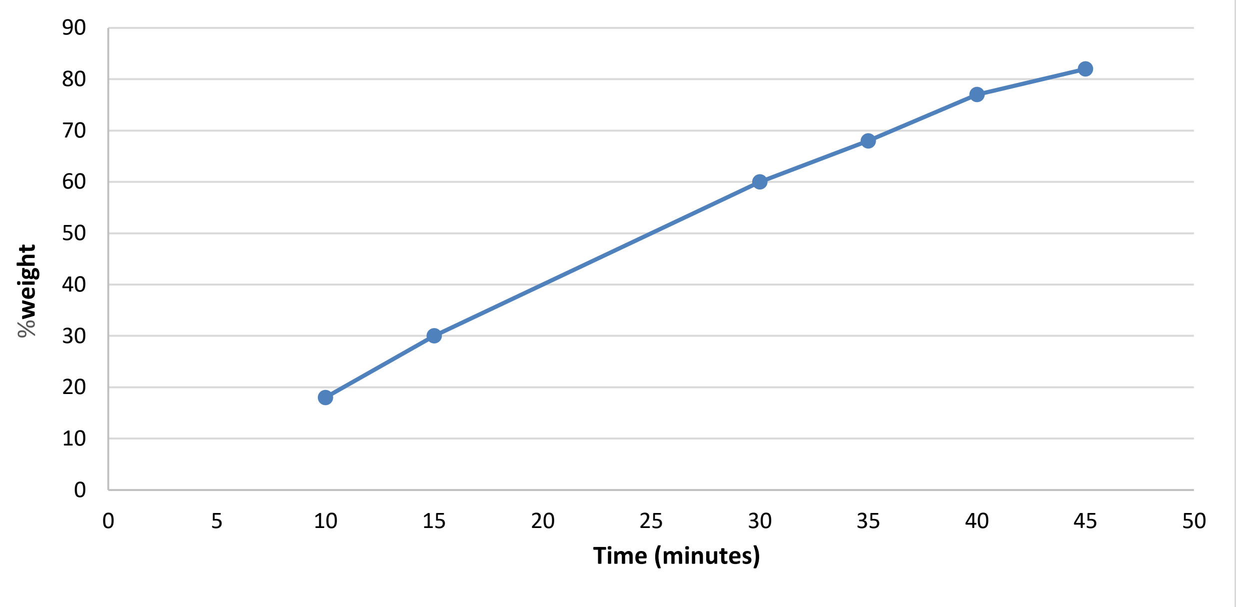


Figure 4.4: Grind analysis curve

Figure 4.4.2 shows that 45 minutes is required to achieve a grind of 80 % passing -75 µm. However, the results could not be counted on since the pulverizer was used for the grinding purpose and it took a lot of batch pulverizing to reach the results and the results also depended on the pulverizer grindability. This was a tiresome way due to hand feeding to the pulverizer. The ore used was then screened using a -75 µm screen to reach 80 % passing -75 µm. the screened ore was packed in 1.1 kg samples and used for test works.

4.5 PARTICLE SIZE ANALYSIS

Table 4.5 shows calculated recoveries from particle size analysis assay results.

Table 4.5: Particle analysis

PARTICLE		TIME/HRS							STANDARD DEVIATION.
SIZE/µm	3	6	9	12	15	18	21	24	
-600	16	19	23	25	27	31	33	34	±6.52
-425	18	22	25	34	38	40	45	46	±10.66
-300	38	45	50	53	59	61	65	70	±10.65
-150	43	51	52	54	58	60	61	70	±8.06
-75	54	66	72	79	82	85	89	92	±12.73

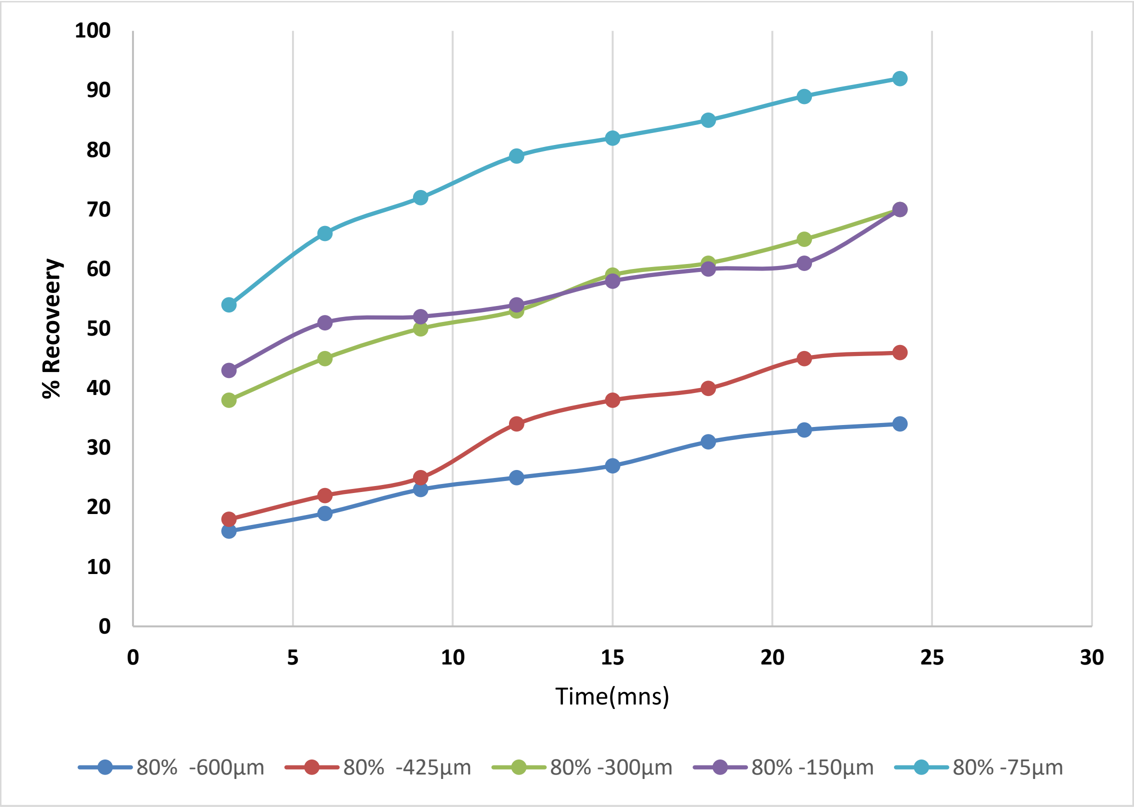


Figure 4.5: Particle analysis graph

Figure 4.5 shows that that highest leaching rates occur in the -75 µm leaching tank. Recovery in the -600 µm leaching pot is slightly above 30 %. There is no much leaching difference between -300 µm and -425 µm particle sizes. Since there is a relationship between particle size and grinding the above graphs supports the concept that the gold recover is higher when the particle size is smaller with the -75 µm having the highest recovery compared to larger particles.

4.6 CARBON ACTIVITY

Head solution (after 48hours) = 9.48 ppm
After addition of carbon:
after 5 minutes - 8.92 ppm
after 15 minutes - 6.07 ppm
after 30 minutes - 4.10 ppm

The bottle roll test with optimum conditions proved to be very effective with high gold dissolution. This was due to an increased duration of the interaction of the pulp and the leaching reagents (Rogans et al 2012). In-order to check for maximum regent consumption activated carbon was then added to adsorb gold and to calculate how much gold was left in solution. Equation 3.3 was used for the calculations of average carbon activity. Since after 30 minutes there was high gold absorption by carbon of 93.70 % with activity of 66.98 % this was a success. Although the carbon that was used in this experiment was eluted and once used in the plant, this showed that there was less hinderance from the leaching reagents such as cyanide which is a competitive inhibitor for absorption into the pores of activated carbon. This however at a plant scale would result in high recoveries and less gold lost as tails to slime dam.

CHAPTER 5: CONCLUSION AND RECOMMENDATIONS

5.1 SUMMARY

The purpose of this experimental study was to evaluate the optimum conditions required in the gold dissolution from the ore as well as gold recovery using activated carbon with more emphasis on pH, leaching time, cyanide concentration and the particle size of the ore. This was done to reduce cyanide consumption at the mine and to increase gold recovery thereby reducing the amount of gold lost as tails to the slimes dam.

5.2 CONCLUSION

On the basis of the outcome obtained the conclusion can be stated as follows:

- The first analysis of this thesis was to compare two pH regulators which were lime and caustic soda. The concentration of the two reagents were varied with concentration of gold leached recorded from 30 minutes up to a residence time of 24 hours. The cyanide concentration was kept constant at 0,36 %. Lime proved to be a better reagent in the gold dissolution with the amount of gold dissolved after 24 hours at 48% and caustic soda at 42 % from a head solid of 2.75 g. The amount of gold left in solid form (tail solid) for lime and caustic soda were 1.43 and 1,.60 respectively showing that caustic soda left a lot of gold undissolved. There was no a higher gold dissolution of above 80 % as required for an effective method due to the concentration of cyanide and optimum leaching time which were yet to be determined so at this point lime was concluded to be the best reagent for Ph.
- The next step was to determine the optimum pH using lime. A bottle roll method of analysis was employed with pH varied from 9 to 13.5. Cyanide concentration and leaching time were kept constant at 0.36 % and 24 hours respectively. There was a highest gold dissolution of 82 % at pH 11.5 with a further increase in pH resulting in no more dissolution. This might have been caused by an increase in interfering reaction such as sulfides and thiosulfate degrading products like tetrathionate which causes loss of gold during leaching, Thus the optimum pH was suggested to be 11.5.
- From the first bottle roll method the leaching time was 24hours which resulted in a 48 % gold dissolution. Another bottle roll was done with an increased leaching time at 48hours and cyanide concentration kept constant at 0.36 % and pH at 11.5. There is an improvement in gold dissolution with a rise in leaching period. Comparing with first bottle

roll this would have been caused by operating at optimum pH and an increased contact time between particles and leaching reagent. The leaching time was concluded to be 48 hours.

- The next bottle roll procedure was done to establish the optimum cyanide concentration. This was to prevent high cyanide consumption at the mine and prevention of its effects on activated carbon which hinder its ability to adsorb gold. A 500 g sample was treated with lime and pH staged above 11. A rise in cyanide level resulted in an improved disintergration of gold into solution with cyanide concentration of 65.00 ppm having highest gold dissolution of 63.2. Further increase in cyanide concentration did not increase further gold dissolution.
- The particle size of the ore was compared for gold dissolution. First the sample was obtained from the ball mill feed and sieving was done to separate ore into -75, -150, -300, -425, and -600 µm. The analysis showed that -75 µm had a higher gold dissolution thus the gold dissolution increased with a decrease in ore size. A comparison was made from plant result with the laboratory results. A laboratory jaw crusher and a pulverizer was used to reduce ore into the above sizes. A grind analysis curve was established to compare with plant grinding and it was almost similar. The particle size analysis corresponded to the laboratory analysis showing a high gold dissolution at -75 µm. A graph of particle size against % dissolution was established with -75 µm having above 90 % gold dissolution.
- After determination of optimum conditions, a final bottle roll was established with a slury with particle size -75 µm, cyanide concentration at 65.00 ppm, lime used as a pH regulator at 11.5. The leaching was done for an effective 48hrs with gold dissolution into solution recorded at 9.48 ppm. Activated carbon was then added with sample placed on a magnetic

stirrer. Samples were taken for gold adsorption test at 5, 15 and 30 minutes. Calculations showed that there was maximum gold adsorption just after 15 minutes of 93.7 % and the amount of gold in solution decreased with time showing an effective absorption by activated carbon. The result showed that all the cyanide was used up during leaching and had no further effect during gold adsoption.

- The results showed that with maximum reagent use there is high gold recovery using AC and less carbon fouling caused by excess cyanide in solution during adsorption.

5.3 RECOMENDATIONS

- L recommend use of modern automated bottle roll machines for better results in which optimum conditions can be set rather than manual control e.g controlling pH by manual addition of lime and testing using a pH meter
- A research focused on use of different activated carbon must be done comparing the god recoveries. Activated carbon is manufactured differently with some being physically or chemically activated and this results in a difference is their ability to adsorb goal. So l recommend a project to be done for certain type of ore to determine the activated carbon to use.
- Most small scale mines use caustic soda for gold recovery due to its cheaper prices but as shown from the result caustic soda some time have poor gold recoveries so l suggested use of lime instead. The high reagent costs will be compensated with high output.
- Due to electricity load shadings in the country, there is a setback in the mining industry when electricity is off the leach and adsorption tanks stops the agitation and the carbon and pulp will settle down below the tank which will require higher residence time so l recommend use of solar power, generators and probably research on use of wind as a source for agitation in the tanks to allow gold dissolution and adsorption through-out the processing period without shutdowns.

REFERENCES

Acharya, J., Sahu, J.N., Mohanty, C.R., Meikap, B.C. (2009) Removal of lead (II) from wastewater by activated carbon developed from tamarind wood by zinc chloride activation. Chemical Engineering Journal, 149. 249-262.

Acharya, J., Sahu, J.N., Sahoo, B.K., Mohanty, C.R., Meikap, B.C. (2009) Removal of chromium(VI) from wastewater by activated carbon developed from Tamarind wood activated with zinc chloride. Chemical Engineering Journal, 150 (1), 25-39.

Adhoum, N., Monser, L. (2004) Removal of phthalate on modified activated carbon: application to the treatment of industrial wastewater. Separation and Purification Technology, 3, 233- 239.

Aylmore, M.G., Muir, D.M., Staunton, W.P. (2014) Effect of minerals on the stability of gold in copper ammoniacal thiosulfate solutions – the role of copper, silver and polythionates. Hydrometallurgy 143, 12–22.

Bandosz, T.J. (2006) Activated Carbon Surfaces in Environmental Remediation. Academic Press, The Netherlands.

Bansah, K.J., Yalley, A.B., Dumakor-Dupey, N. (2016) The hazardous nature of smallscale underground mining in Ghana. J. Sustain. Min. 15, 8e25. https://doi. org/10.1016/j.jsm.2016.04.004.

Bakatula, E.N., Tutu, H. (2016) Characterization and speciation modelling of cyanide in effluent from an active slimes dam. S. Afr. J. Chem. 69, 140–147. https://doi.org/ 10.17159/0379-4350/2016/v69a17

Battaini P, Bemporad E, De Felicis D (2014). The fire assay reloaded. Gold Bull. 47:9-20.

Bellec, S.; Hodouin, D.; Bazin, C.; Khalesi, M.R.; Duchesne, C. (2011) Hydrocyclone classification modeling for gold ore grinding circuit simulation. In Proceedings of the 50th World Gold Conference, Montreal, QC, Canada.

Cheng, K.Y., Acuna, ˜ C.C.R., Boxall, N.J., Li, J., Collinson, D., Morris, C., du Plessis, C.A., Streltsova, N., Kaksonen, A.H. (2021). Effect of initial cell concentration on biooxidation of pyrite before gold cyanidation. Minerals 2021 (11), 834. https://doi. org/10.3390/min11080834.

Cort´es-McPherson, D. (2019) Expansion of small-scale gold mining in Madre de Dios: 'Capital interests' and the emergence of a new elite of entrepreneurs in the Peruvian Amazon. Extract. Indus. Soc. 6, 382–389. https://doi.org/10.1016/j. exis.2019.01.002.

Davidson, R.J., Sole, M.J. (2007) The major role played by calcium in gold plant circuits. J. South Afr. Inst. Min. Metall. 107, 463–468. https://www.saimm.co.za/Journal/ v107n07p463.pdf.

Deschˆenes, G., Pratt, A., Fulton, M., Guo, H. (2011) Kinetics and mechanism of leaching synthetic calaverite in cyanide solutions. Mining, Metallurgy & Exploration. 23, 133–138. https://doi.org/10.1007/BF03403200.

Deschˆenes, G., Pratt, A., Riveros, P, Fulton, M. (2011) Reactions of gold and sulfide minerals in cyanide media Minerals & Metallurgical Processing. 19, 4, 169-177. 10.1007/BF03403265.

Dunne, R., Buda, K., Hill, M., Staunton, W., Wardell-Johnson, G., Tjandrawan, V. (2012) Assessment of options for economic processing of preg-robbing gold ores. Mineral Processing and Extractive Metallurgy. 121 (4), 217–223. https://doi.org/10.1179/ 1743285512Y.0000000019.

du Plessis, C.A., Lambert, H., G¨ artner, R. (2020) Process for obtaining size-calibrated tricalcium aluminate particles and use of such particles in a process of alumina refining. Patent application WO2020011952A1. https://patentimages.storage. googleapis.com/71/1f/93/4cff13f371f303/WO2020011952A1.pdf.

du Plessis, C.A., Lambert, H., Hoummady, E., McDonald, R.G., Bedell, D. (2021) Lime properties and dose effects on causticisation of synthetic Bayer liquor. Miner. Eng. 160, 106664. https://doi.org/10.1016/j.mineng.2020.106664.

Egan, J.; Bazin, C, Hodouin, D. (2016) Effect of ore particle size on gold dissolution in a cyanide solution. In Proceedings of the SME Annual Meeting, Denver, CO, USA, 24–27

Faria, P.C.C., Orfao, J.J.M., Pereira, M.F.R. (2004) Adsorption of anionic and cationic dyes on activated carbons with different surface chemistries. Water Research, 38 (8), 2043-2052

Gladman, B., de Kretser, R.G., Rudman, M. and Scales, P.J. (2005) Effect of shear on particulate suspension dewatering Transactions Institute of Chemical Engineers 83, pp. 1-4.

Habashi, F. (2016) Chapter 1 - Gold – An Historical Introduction. Gold Ore Processing (Second Edition). Project Development and Operations. 2016, 1-20. 10.1016/B978- 0-444-63658-4.00001-3

Hilson, G.M. (2006) The Socio-Economic Impacts of Artisanal and Small-Scale Mining in Developing Countries. Taylor & Francis, 198 pp.

Hilson, G., McQuilken, J. (2014) Four decades of support for artisanal and small-scale mining in sub-Saharan Africa: a critical review. Extract. Indus. Soc. 1, 104–118.

Hilson, G., Hilson, C.J. and Pardie, S. (2007) 'Improving awareness of mercury pollution in small-scale gold mining communities: Challenges and ways forward in rural Ghana', Environmental Research, Vol. 103, No. 2, pp.275–287.

Hodouin, D. (2011) Methods for Automatic Control, Observation, and Optimization in Mineral Processing Plants. – Journal of Process Control, Vol. 21, 211-225. 27.

Hodouin, D., S-L. Ja sa-Joumela, M.T. Carvahlo, L. (2001) Bergh. State of the Art and Challenges in Mineral Processing Control. – Control Engineering Practice, Vol. 9, 995-1005. 28.

Hulth´en, E. (2010) Real-time optimization of cone crushers. Ph.D. Thesis Chalmers University of Technology.

Kansake, B.A., Kudzawu-D'Pherdd, R., Tetteh, H.N., Dagbui, S.P., Mends, E.A. (2021) Artisanal and small-scale gold mining in Ghana - Past, Present and Future. Mining Spectra. 2-20210328. ISSN2616-1532.

Kotake, N., K. Daubo, T. Yamamoto. (2004) Experimental Investigation on a Grinding Rate Constant of Solid Materials by a Ball Mill – Effect of Ball Diameter and Feed Size. – Powder Technology, Vol. 143/144, 196-203. 29.

Kotake, N., M. Kuboki, S. Kiya, Y. Kanda. (2010) Influence of Dry and Wet Grinding Conditions on Fineness and Shape of Particle Size Distribution of Product in a Ball Mill. – Advanced Powder Technology, 2010, doi:10.1016/j.apt.03.015. 30.

Kuzev, L., T. Penchev, D. Karastoyanov. (2009) New Shape Milling Bodies for Ball Mills. – Problems of Engineering Cybernetics and Robotics, Vol. 61, 11-19.

Lunga, A. L. (2006) Optimizing the operating conditions of gold elution and electrowinning for Tau Lekoa stream at Kopanang gold plant. University of the Witwatersrand

Marsden, J.O., House, C.I. (2006) The Chemistry of Gold Extraction, 2nd Ed. Society for Mining, Metallurgy, and Exploration, Inc.

Medina, D., Anderson, C.G. (2020) A Review of the Cyanidation Treatment of Copper Gold Ores and Concentrates. Metals. 10 (7), 897. https://doi.org/10.3390/ met10070897

Muir, D.M. (2011) A review of the selective leaching of gold from oxidised copper– gold ores with ammonia–cyanide and new insights for plant control and operation. Miner. Eng. 24 (6), 576–582

concentrate in glycine solutions. Hydrometallurgy 150, 14–19.

Oladele, T. P. (2015) Effect of temperature, contact time and agitation speed during pre-treatment on elution of gold. Stellenbosch University.

Oraby, E.A., Eksteen, J.J. (2014) The selective leaching of copper from a gold–copper.

Panova S., Kryazhov A. (2013). Aplication of the oktil-sulfid as extraction agent for determination gold in raw material, XIV International Scientific Conference “Chemistry and Chemical Engineering in XXI century” dedicated to Professor L.P. Kulyov. Tomsk. 1. p. 280-281

Quist, J. (2012) Cone Crusher Modelling and Simulation. Master’s Thesis Chalmers University of Technology.

Quist, J. (2017) DEM modelling and simulation of cone crushers and high pressure grinding rolls. Chalmers University of Technology.

Rylander, D., Axelsson, J. (2013) Lean Method to Identify Improvements for Operation Control at Quarry Sites. In: 30th International Symposium on Automation and Robotics in Construction (ISARC 2013). Montreal, Canada: International Association for Automation and Robotics in Construction (IAARC). doi: 10.22260/isarc2013/ 0010.

Rogans, G, Li, J., Dabrowski, B., Miller, J.D., Acar, S., Dietrich, M., LeVier, K.M., Wan, R.Y. (2012) Factors that affect adsorption on gold recovery by cyanidation. Minerals Engineering 19, 883–895

Sayiner, B. (2014) “Modeling of the carbon-in pulp (CIP) process in gold cyanide leaching plants using plant data,” Physicochemical Problems of Mineral Processing, 50(2), pp. 841–851. doi: 10.5277/ppmp140234.

Senanayake, G. (2005) Role of copper(II), carbonate and sulphite in gold leaching and thiosulphate degradation by oxygenated alkaline non-ammoniacal solutions. Miner. Eng. 18, 409–426

Singh, N. (2012) A rugged, precise and accurate new gravimetry method for the determination of gold: an alternative to fire assay method. SpringerPlus 1:1-6.

Snyders, C.A., Akdogan, G., Bradshaw, S.M., van Wyk, A.P. (2017) Gold CIP and CIL process optimization in a capital constraint environment. J. South Afr. Inst. Min. Metall. 117 (8), 819–828. https://doi.org/10.17159/2411-9717/2017/v117n8a13.

Stewart, M., Kappes, D. (2012) SART for copper control in cyanide heap leaching. J. S. Afr. Inst. Min. Metall. 112, 1037–1043.

Styles, M.T., Amankwah, R.K., Al Hassan, S. and Nartey, R.S. (2009) ‘The identification and testing of a method for mercury-free gold processing for artisanal and small-scale gold miners in Ghana’, International Journal of Environment and Pollution.

Sitando, O., Dai, X., Senanayake, G., Zhang, H., Nikoloski, A.N., Breuer, P. (2015) Gold dissolution in non-ammoniacal thiosulfate solutions. Comparison of fundamentals and leaching studies. In: The Proceedings of the World Gold Conference, Gauteng, South Africa. SAIMM, pp. 273–288.

Sitando, O., Senanayake, G., Dai, X., Nikoloski, A.N., Breuer, P. (2018) A review of factors affecting gold leaching in non-ammoniacal thiosulfate solutions including degradation and in-situ generation of thiosulfate. Hydrometallurgy 178, 151–175.

Tschakert, P. and Singha, K. (2007) 'Contaminated identities: Mercury and marginalization in Ghana's artisanal mining sector', Geoforum, Vol. 38, No. 6, pp.1304–1321.

V. Balaram, W. Rahaman, P. Roy. (2022) Recent advances in MC-ICP-MS applications in the earth, environmental sciences: challenges and solution, Geosyst. Geoenviron. 1.

Verbrugge, B., Geenen, S. (2018) The gold commodity frontier: a fresh perspectiveon change and diversity in the global gold mining economy. Extract. Indus. Soc. 6 (2019), 413–423. https://doi.org/10.1016/j.exis.2018.10.014.

Verbrugge, B., Lanzano, C., Libassi, M. (2021) The cyanide revolution: efficiency gains and exclusion in artisanal- and small-scale gold mining. Geoforum 126 (2021), 267–276. https://doi.org/10.1016/j.geoforum.2021.07.030

Xu, B., Li, K., Zhong, Q., Li, Q., Yang, Y., Jiang, T. (2018) Study on the oxygen pressure alkaline leaching of gold with generated thiosulfate from sulfur oxidation. Hydrometallurgy 177, 178–186.

Zanin, M., Lambert, H., du Plessis, C.A. (2019) Lime use and functionality in sulphide mineral flotation: A review. Miner. Eng. 143, 2019. https://doi.org/10.1016/j. mineng.2019.105922.

Zhang, X.M., Senanayake, G., Nicol, M.J. (2004) A study of the gold colloid dissolution kinetics in oxygenated ammoniacal thiosulfate solutions. Hydrometallurgy 74, 243–257

Zhang, H., Ritchie, I.M., La Brooy, S.R. (2004) The adsorption of gold thiourea complex onto activated carbon. Hydrometallurgy 72, 291– 301