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A THESIS REPORT TITLED

Optimisation of Biooxidation Nutrients for Enhanced Gold Recovery– A Case Study at Golden Star Bogoso/Prestea Limited

BY

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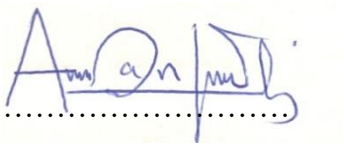
DECLARATION

I declare that this thesis is my own work. It is being submitted for the degree of Master of Science in Minerals Engineering at the University of Mines and Technology (UMaT), Tarkwa. It has not been submitted for any degree or examination in any other University.



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ABSTRACT

With growing interest in the development of novel technologies for the treatment of refractory gold deposits, the biological oxidation process has gained wide range acceptance due to its relatively lower operating cost and environmental friendliness. However, with the downward plunge in gold prices in recent years, the profitability of using this process is questionable and for Golden Star Resources, the mine under study, the Biological Oxidation (BIOX) Plant is under care and maintenance because the cost of production outperformed the sale of products. In an effort to reduce the operating cost further to make this process an economically viable option, a nutrient optimisation study was undertaken, which is the main objective of this work.

Chemical and mineralogical characterisation of flotation concentrate revealed average sulphide sulphur and organic carbon contents of 13.83% and 2.91% respectively, and a gold grade of 39.47 g/t.

Acid leaching of the flotation concentrate for a 5 day residence time revealed that, 0.362 kg/t nitrogen (N), 0.702 kg/t phosphorus (P) and 0.417 kg/t potassium (K) could be made bio-available in the process stream. Sulphide sulphur oxidation was conducted using varying percentages of NPK relative to the Genmin standard of 1.7:0.3:0.9 kg/t of NPK, after which the products were subjected to gold leaching by cyanidation. For no nutrient addition (0% Genmin standard NPK), 46% sulphur oxidation (SOX) was achieved, whereas 100% addition 95.2% sulphur oxidation. The corresponding gold recoveries were about 73.9% and 95.7% respectively. Various SOX ranging between 41.1% and 95.7% were obtained for the different dropwise reductions in NPK, and subsequent cyanidation gave recoveries between 55.7% and 97.7%. The highest gold recovery of 97.7% was recorded after the flotation concentrate was oxidised with 0.58:0.30:0.90 kg/t of NPK to obtain a SOX of 91.2%. This thesis contributes greatly to knowledge as the results obtained set a platform for deeper research into reduced processing cost based on the optimisation of BIOX nutrients.

DEDICATION

This research work is dedicated to my wife, Rebecca Pokuaah Amankona, and my son, Jayden Agyei Adjartey, for enduring a long period of my absence in order to get this dream come through.

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CHAPTER 1

INTRODUCTION

1.1 Background

From the metallurgical perspective, gold ores can be broadly classified into placer, free-milling and refractory ores. Typically, free-milling ores are defined as those from which over 95% of gold can be recovered by gravity concentration and conventional cyanide leaching. Generally, placer, quartz vein gold and oxidised ores are free-milling and the associated gold can also be recovered by gravity concentration and/or direct cyanide leaching (Joe Zhou *et al*., 2004).

Refractory ores on the other hand are those that give low gold recoveries (< 90%), in some cases much less than 50% or give acceptable gold recoveries only with the use of more significant reagents or more complex pre-treatment process. Refractoriness is caused by the presence of sulphides, tellurides, cyanicides and/or carbonaceous matter (Boyle, 1979; Guay, 1981). The presences of both sulphides and carbonaceous matter (CM), renders the ore double refractory (Nyavor and Egiebor, 1992).

Generally, pre-treatment processes for refractory ores are classified into pyrometallurgical methods such as roasting and hydrometallurgical processes like chlorine oxidation, pressure oxidation and bacteria oxidation. Due to environmental concerns like emission of acidic gases, carcinogenic substances and herbicides; and operational safety concerns like high temperature and pressure, most of the pre-treatment processes pose techno-economic challenges and make the bacteria oxidation process the preferred process (Marsden and House, 2006; Ofori-Sarpong et al., 2011;2013).

Bacteria Oxidation (BIOX®) process, makes use of chemolithotropic or 'autotrophic bacteria". Autotrophic bacteria, the most common mineral-oxidising bacteria, derive their energy from oxidation of inorganic compounds (Madigan and Martinko, 2006). These bacteria oxidise ions such as ferrous and sulphur. Hence, gold locked up in iron and sulphur

containing minerals can be liberated when subjected to bacteria oxidation, as a result of the feeding habit of these microbes.

1.2 Statement of the Problem

The Golden Star Bogoso/Prestea Limited (GSBPL), BIOX® plant was designed to treat 781 tonnes per day of pyrite/arsenopyrite concentrate with a grade of 21.6% sulphide sulphur and 2.9% arsenic. The main sulphide minerals in the concentrate are pyrite (38%) and arsenopyrite (6%). In these minerals, gold occurs as sub-microscopic grains which render the concentrate highly refractory, in other words, direct cyanidation of the concentrate yields very low recoveries.

In order to liberate the gold particles prior to cyanidation, the sulphide mineral matrix must first be destroyed. This can be done by roasting, pressure oxidation, or as in the case of GSBPL, by the BIOX® process. The BIOX® process utilises a naturally occurring mixed bacteria population consisting of *Acidithiobacillus ferrooxidans*, *Acidithiobacillus thiooxidans* and *Leptospirillum ferrooxidans*, a culture that was adapted from the Anglo-Gold Obuasi Mine, Sansu BIOX® Plant strain (Marsden and House, 2006).

Acidithiobacillus ferrooxidans and *Leptospirillum ferrooxidans* are chemolithotrophs and they obtain their cell carbon from dissolved carbon dioxide (Madigan and Martinko, 2006). Their energy comes from ferrous ions and/or sulphur compounds. Nitrogen (N), potassium (K), phosphorus (P), magnesium and sulphur are the major nutritional requirements for the bacteria strains. Magnesium and sulphur requirements are mostly met from the ore and water used for pulping whereas N, P, and K are found in trace quantities in the ore.

Nitrogen is an essential element that is required for protein cell development via the amino acid building blocks (Clarke, 1989; Madigan and Martinko, 2006). It is supplied in the form of ammonium nitrate which dissolves in acid solution. Atmospheric ammonia readily dissolves in leach solutions and may provide most, if not all, of the nitrogen required by the bacteria for growth (Clarke, 1989; Madigan and Martinko, 2006) . Notwithstanding, the study of the nitrogen requirement for bioleaching organisms is a complex phenomenon.

Phosphorus is supplied in the form of phosphate. Phosphate in the BIOX® stream is necessary for both energy metabolism and the initial steps of Fe^{2+} oxidation outside the cell wall of *A. ferrooxidans* (Clarke, 1989). The total removal of PO_4^{3-} from the media will result in a slower rate of iron oxidation but the bacteria will still remain active.

Potassium is the principal inorganic cation in the cell and is the co-factor of some enzymes. The supply of potassium to the bacteria is somewhat controversial (Clarke, 1990; Madigan and Martinko, 2006). On the other hand, high concentrations of K^+, coupled with a high Fe^{3+} concentration, enhances jarosite ($KFe_3(OH)6(SO_4)^2$) formation, which will deplete the available K^+ from the solution and cause coating of the sulphide mineral. Since the pH in a BIOX® reactor should be between 1.2 - 1.8 and the temperature is below 45 °C the degree of jarosite formation is limited (Habashi, 1999; Marsden and House, 2006). However, occasional process control failures may result in jarosite formation and hence loss of potassium.

The Bogoso/Prestea deposit is predominantly refractory with untapped reserves at Dumasi, Chujah, Buesichem, Mampong and other satellites pits. Though the plant is under care and maintenance due to high cost of operation, research into reducing the cost of operation will help revamp the sulphide operations again. One of the cost drivers in a typical BIOX® plant like that of GSBPL are those of reagents; sulphuric acid and nutrients. The former can be regulated using the proportion of arsenic and pyrite in the concentrate. The latter however cannot be done away with and an optimisation study is usually conducted to determine their optimum concentrations for the best bacteria activity. The role of nutrients (NPK) is of both economic and biological importance when it comes to biological oxidation of refractory (sulphidic) ores as a pre-treatment process and their requirement varies from one ore body to the other. Therefore, this work seeks to optimise NPK usage that will help manage the high operating cost of BIOX® without trading-off oxidation and subsequent metal recovery.

It is therefore hypothesised that a step-wise reduction of the NPK in BIOX® operation will yield similar results to or even better results than the standard addition rate with high metal recovery in the subsequent leaching of BIOX® products.

1.3 Objectives of the Research

The objectives of this thesis were to:

i. To ascertain the refractory nature of the Chujah concentrate
ii. Determine the levels of bio-available nutrients in the Chujah concentrate
iii. Determine the best NPK combination for optimum sulphide sulphur oxidation; and
iv. Establish the best sulphide sulphur oxidation conditions that result in optimum cyanidation recovery.

1.4 Methods Used

The methods used for this research included:

i. Review of relevant literature on refractory ore pre-treatment and BIOX® nutrient optimisation;
ii. Chemical and mineralogical analysis of Chujah concentrate
iii. Acid leach tests on the Chujah concentrate;
iv. Batch BIOX® amenability tests using different combinations for NPK; and
v. Cyanidation of the biooxidised concentrates.

1.5 Thesis Organisation

This thesis is organised into five chapters. Chapter 1 contains the introduction of the thesis, statement of the problem, objectives of the research, and methods used. Chapter 2 covers the relevant literature on gold ores, pre-treatment processes for refractory ores and biohydrometallurgy technology. Chapter 3 deals with the experimental work. Chapter 4 deals with the results and discussions obtained from the interpretation of data from the experimental work. Chapter 5 takes care of the conclusions and recommendations drawn from the research.

CHAPTER 2

LITERATURE REVIEW

2.1 Geochemistry and Mineralogy of Gold

Gold which is usually concentrated in residual hydrothermal fluids is precipitated when the solution encounters a reducing environment, that is, regions of high carbonate, carbon or reducing sulphide content (Marsden and House, 2006). Its occurrence in the host rock depends mainly on temperature, pressure, pH, chloride concentration, and fugacity of H_2S of the hydrothermal system. Gold mostly occurs as $AuCl_2$- in a system with temperature higher than 400 ^oC (Gammons and Williams-Jones, 1997), with the primary mechanism for gold deposition being the decrease in temperature as shown in Equations 2.1- 2.3.

$$2Au(HS)_2^- + H_2O \rightarrow 2Au + 4HS^- + 2H + 0.5O_2 \quad (2.1)$$

$$2Au(HS)_2^- + 8H_2O \rightarrow 2Au + 4SO_4^{2-} + 4H^+ + 8H_2 \quad (2.2)$$

$$FeCO_3 + Au(HS)_2^- \rightarrow FeS_2(pyrite) + CO_2 + H_2O + Au \quad (2.3)$$

Indeed, Prasad *et al.* (1991) and Ampofo, (2015) said gold occurs naturally in a metallic form and is the most noble of all the metals. In air it is not reactive and is not corroded by strong acids or bases. The chemistry of gold in aqueous solution is related to its very low electro positivity, which is related to its standard reduction potential. The two most prevalent oxidation states of gold are +1 (aurous) and +3 (auric) for which the reduction potentials are 1690 and 1500 mV versus Standard Hydrogen Electrode (SHE) respectively (Pourbaix, 1966; Ampofo, 2015). These values are more positive than the standard reduction potential for water, 1230 mV, indicating that Au+ and Au^{3+} are thermodynamically unstable in aqueous solution (Nicol *et al.* 1987; Ampofo, 2015). Adams (1998) indicated that the aurous and auric cations can be stabilised by a number of ligands, including chloride, cyanide, thiourea and thiosulphate.

2.2 Classification of Gold Ores

Classification of gold ores could be done in several ways; according to either the extraction technique or the associated geological environment (McQuiston and Shoemaker, 1975; Boyle, 1979; 1987, Guay, 1981; Yannopoulos, 1991; Afidenyo, 2008). One generally accepted classification method based on the extraction techniques places gold ores under two main groups; refractory or non-refractory. Generally, classification of gold ores can also be based on the extraction method classifying gold ores as either refractory or non-refractory.

2.2.1 Non-refractory Ores

Non-refractory ores undergo alteration following initial deposition and include placer, free-milling and oxidised ores from which about 95% of the gold is recoverable by simple gravity concentration and/or direct cyanidation.

Placer Ores

In geology, a placer deposit or placer is an accumulation of valuable minerals formed by gravity separation during sedimentary processes. Primary deposits of gold undergo weathering overtime causing liberation and hydraulic transport of gold particles from the parent rock. Due to the dense and chemically inert nature of gold, the gold particles accumulate depending on the distance from the primary deposit to form the different classes of placers (eluvial, colluvial, fluvial and marine). Placer is Spanish word which means "alluvial sand" and although the quantity of gold found in placers are usually low as compared to the primary deposits; they have economically been mined over the years due to its simple processing route and low operating cost (Silva, 1986).

Placer mining is an ancient method of extracting gold and typically employs the use of water to excavate, transport, concentrate, and recover heavy minerals from placer deposits. Placer gold mining is potentially a relatively "clean" industry, not producing the highly toxic ore

separation chemical; effluent, acid drainage, and smelter emissions common to the mining and processing of other metal deposits. With placer mining, recovery of the gold from the ore is usually the most expensive phase of the mining operation and can be the most difficult to implement properly. Placer mining techniques such as simple gold panning and sluicing have generally only been the practice of artisan miners resulting in direct recovery of small gold nuggets and flakes. Placer mining takes advantage of gold's high density, which causes it to sink more rapidly from moving water than the lighter siliceous materials with which it is found. Though the basic principles of placer mining have not altered since early times, methods have improved considerably (Silva, 1986; Farrington, 2000).

Free Milling Gold Ores

In the case of free milling, the most common method of extraction is dissolving the gold by direct alkaline cyanide (Mousoulous *et al.,* 1984; Prasad, *et al.* 1991). With this method, about 95% of the gold can be extracted when the ore is ground to about 80% passing 75 um. Gravity concentration may be used to produce a concentrate that can be treated separately (Marsden and House, 2006; Ampofo, 2015) and the gangue mineral present do not significantly affect the processing requirement of such deposits.

Oxidised Gold Ores

Baker, (1997) indicated that oxide gold deposits, refers to gold-bearing veins, faults, and shear zones that typically contain appreciable amounts of oxidised ore or gossanous material, resulting from oxidation of sulphides. The concept of oxide gold deposits underwent a dramatic change after the discovery of the Carlin gold mine in North Central Nevada in early 1960's, and now implies large tonnage, low-grade bulk minable deposits that are processed by milling and/or heap leach methods. Oxide gold deposits may be classified into two namely primary oxide ores which are derived directly from the effects of hydrothermal alteration on oxidised host rocks, and secondary oxide ores which derived from the epigenetic effects of circulating post mineral fluids oxidising gold-bearing sulphides. Each type of oxide gold deposit has unique diagnostic mineralogical and metallurgical characteristics that must be addressed in order to achieve a successful exploration strategy.

2.2.2 Refractory Gold Ores

Refractory ores are of hydrothermal origin and are deposited by hot aqueous solutions obtained from an external source such as a volcanic intrusion. These hydrothermal solutions usually act as carrier for several dissolved mineral components such as sulphides, tellurides, and arsenides which are deposited along with gold. Depending on the extent to which the mineral of interest is associated with these minerals, extraction by conventional methods is rendered inefficient as the dissolution of gold in such systems is impeded. The term refractory is thus applied when a significant proportion of the gold cannot be recovered by conventional gravity concentration or direct cyanidation even after fine milling. The nature of the gold bearing minerals and the gangue minerals present are the two main mineralogical factors that determine the refractoriness of gold ores. Compounds in which gold is in chemical combination with other elements such as tellurides ($AuTe_2$– calaverite), antimonides ($AuSb_2$ – aurostibnite) and selenides (Ag_3AuSe_2 – fischesserite) cause chemical refractoriness. These gold compounds are naturally occurring and their rate of dissolution in cyanide solution is relatively low (Fersman, 1939; McQuiston and Shoemaker, 1975; Boyle, 1979; Guay, 1981).

Sulphides, silicates, carbonaceous matter and compounds of bismuth are also gold bearing minerals that cause refractoriness. The occlusion and inclusion of fine or colloidal gold particles in the matrix of these minerals cause physical refractoriness since this association makes it impossible to fully liberate gold by mechanical means (Boyle, 1979; Guay, 1981; Afenya, 1991; Linge, 1991; Henley 1993; Marsden and House, 2006).

Refractoriness of gangue minerals on the other hand is seen when associated minerals react with and deplete the free cyanide and oxygen required for gold dissolution or by adsorbing dissolved gold from the solution (preg-robbing). These minerals may also passivate the gold surface thereby preventing contact with the cyanide solution. These conditions retard the dissolution process; it is usually not economical for such ore types to be treated. Based on the degree of refractoriness, Linge, (1991) suggested the classification in Table 2.1.

Table 2.1: Classification of Refractory Gold Ores

Classification	Gold Recovery
Free milling	More than 95 %
Mildly refractory	80-95%
Moderately refractory	50-80%
Highly refractory	Less than 50%

(Source: Linge, 1991)

In cases where both sulphides and Carbonaceous Matter (CM) are associated with an ore, that ore type is termed double refractory (Nyavor and Egiebor, 1992). Double refractory gold ores are found in several mining areas around the world including Prestea and Obuasi in Ghana, Witwatersrand in South Africa, Nevada and the mother lode districts of California in the United States of America, McIntyre Porcupine and Kerr Anderson in Canada and Bakyrchik and Natalinsk in the Soviet bloc. Others are Morro Velho and Queiroz Mine in Brazil, Cosmo Howley and Fortnum in Australia, Waihi/Paeroa in New Zealand and Laizhou and Neilanggou in China (Radtke and Scheiner, 1970; Zhuchkov *et al.*, 1968; Nice, 1971; Adamson, 1972; Afenya, 1976; Kesse, 1985).

2.3 Pre-treatment Processes

Most refractory gold deposits are large with richer grades and with depletion of non-refractory ores over the years, extensive research interest has been sparked in this area focused on eliminating the source of refractoriness in a pre-treatment step. The purpose of pre-treatment is to target and decompose selected associated minerals; a process step which opens up the molecular structure so that leaching reagents can make contact with and extract gold (Ofori-Sarpong *et al*., 2011).

It also eliminates, passivates or drastically reduces the presence of the gangue minerals, which cause refractoriness. These pretreatment methods are broadly classified into pyrometallurgical and hydrometallurgical oxidation processes (Guay, 1981; Hayden *et al.*,

1987; Taylor *et al.*, 1989; Yannopoulos, 1991; Afenya, 1991; Linge, 1991; Marsden and House, 2006).

2.3.1 Pyrometallurgical Oxidation

Roasting is the main pyrometallurgical oxidation method for treating refractory ores. It has been adjudged by researchers and operating mines as a reliable method of treating refractory gold ores to achieve significant recovery (Yannopoulous, 1991; Marsden and House, 2006). Roasting involves the heating of refractory ores and concentrates in a controlled oxidising atmosphere resulting in the simultaneous decomposition of volatile components and oxidation of sulphide minerals. Sulphur and arsenic are converted to volatile components with this process whereas iron is converted to hematite. The liberated gold particles are concentrated in the calcine, which may consist mainly of hematite and siliceous gangue minerals. For most refractory gold ores, roasting is achieved at temperatures between 450 ^{o}C to 750 ^{o}C with ample supply of oxygen (Arriagada and Osseo-Asare, 1984; Stanley, 1987; Robinson, 1988). The major reactions of pyrite, arsenopyrite, pyrrhotite and carbonaceous materials can be expressed in Equations 2.4-2.7:

$$4FeS_2 + 11O_2 \rightarrow 2Fe_2O_3 + 8SO_2 \qquad 2.4$$

$$2FeAsS + 5O_2 \rightarrow Fe_2O_3 + As_2O_3 + 2SO_2 \qquad 2.5$$

$$4FeS + 7O_2 \rightarrow 2Fe_2O_3 + 4SO_2 \qquad 2.6$$

$$C + O_2 \rightarrow CO \qquad 2.7$$

The roasting process is quite complex and due to the abundance of other minerals in refractory ores; the desired end product may not necessarily be achieved. These minerals present decompose at different temperatures making temperature control very difficult. High temperatures are experienced if the ore contains carbonaceous material which oxidises at a much slower rate and at high temperatures. When carbonaceous material is not completely oxidised, they become activated making it a more efficient preg-robber. Conversely, high temperatures lead to the recrystallisation of hematite and subsequent physical encapsulation of gold; a calcine with a low porosity is produced and gold leaching is highly retarded (Arriagada and Osseo-Asare, 1984; Swash and Ellis, 1986; Grimsey and

Aylmore, 1990). Notwithstanding, other options such as very fine grinding, the two-stage roasting process and quenching of calcine have been proposed to help produce calcine that is more amenable to leaching.

Roasters are mostly operated at temperatures between 600 ^{o}C and 700 ^{o}C. However, the roasting temperature depends very much on the type of sulphide minerals present. The decomposition reaction during the roasting step is an exothermic one and so lime is sometimes added to the roaster feed to act as a heat sink and allow better temperature control. In addition, it reacts with sulphur dioxide and helps to reduce emissions by about 75%.

Although considered a very effective pretreatment option over the years, the use of roasting has suffered a setback in recent times due to heightened environmental concerns over its by-products and high-power cost. However, it may still be considered as a suitable pre-treatment option due to recent improvements in furnace and process design such as scrubbing of roaster gases and development fluo-solids roasters.

2.3.2 Hydrometallurgical Oxidation

Currently much of research and development activities in the mineral processing industry is focused on hydrometallurgical methods of treatments and this has the potential to eventually replace the traditional roasting (Marsden and House, 2006). The three main hydrometallurgical oxidation techniques are: hydrochemical oxidation, pressure oxidation (autoclave leaching) and bioleaching (biochemical leaching).

2.3.3 Chlorine Oxidation

Aqueous solutions of chlorine have the potential of oxidising carbonaceous material and associated sulphides especially pyrites and dissolving the free gold in the process. Chlorine gas easily dissolves in water to form hydrochloric acid and hypochlorous acid as shown in Equation 2.8 (Snoeyink and Jenkins, 1979).

$$Cl_2 + H_2O \rightarrow HCl + HOCl \qquad 2.8$$

Both products from the dissolution reaction; chlorine gas and hypochlorous acid have high oxidising abilities but hypochlorous acid is the preferred agent for the oxidation and deactivation of carbon and sulphides (Marsden and House, 2006). Hypochlorous acid is added in the form of sodium hypochlorite, which is inexpensive, readily available, easy to handle and can be added in specific quantities. The activity of hypochlorous is highest at temperatures between 50 ^{o}C and 60 ^{o}C and a pH range of 3 and 5 making it the best pair of conditions for effective chlorine oxidation.

Sodium hypochlorite could be regenerated directly in the pulp by electrolysis to maintain a high level of the reactant at all times (Marsden and House, 2006). The exact mechanism involved in the deactivation of carbonaceous matter is not fully comprehended but it is believed that the surfaces of the carbonaceous matter are modified by chlorine either by the formation of a chlorohydrocarbon layer or by the formation of carbonyl structures composed mainly of carboxyl groups. These surface groups passivate the carbon by blocking active adsorption sites (Brunk *et al.,* 1987; Sibrell *et al.,* 1990). For sulphidic and carbonaceous ores, chlorine consumption during the process could be over 50 kg/t. The consumption of the oxidant can be attributed not only to the oxidation of sulphides and deactivation of carbonaceous material but also with gold's reaction with the oxidant (Nagy *et al.,* 1968; Brunk and Atwood, 1987; Birak and Deter, 1987). Pyrite reacts according to the Equation 2.9:

$$2FeS_2 + 15HOCl + 7H_2O \rightarrow 2Fe(OH)_3 + 23H^+ + 4SO_4^{2-} + 15Cl^- \qquad 2.9$$

The very high operating cost of the process is due to excessive consumption of chlorine making the process less attractive and thus the double oxidation process was introduced to help reduce the chlorine consumption.

2.3.4 Double Oxidation

This is a two-stage oxidation process; the initial oxidation process involves vigorous agitation of the pulp with air at temperature of 70 ºC to 80 ºC and air pressure of 6-8 psi until a considerable amount of pyrite and other sulphides are oxidised. The oxidation of pyrite produces sulphuric acid as an end product and hence lime or sodium hydroxide may be introduced during the initial aeration step for any sulphuric acid produced by pyrite oxidation to be neutralised. After the pre-aeration step, chlorine consumption is significantly reduced in the subsequent step.

The process of aeration followed by chlorination is thus referred to as double oxidation (Marsden and House, 2006). The high gains of gold recoveries of (90-93%) and current technologically modification has not erased the negative impact of this process technique due to excessive consumption of chlorine, the toxic and corrosive nature of chlorine coupled with the expensive tailings disposal as ores becomes more refractory.

2.3.5 Arseno Process

The arseno process is a pressure oxidation process that utilises a nitrate catalysed oxygen leach system. It is best suited for ores or concentrates in which the gold is locked in minerals such as marcasite or arsenopyrite. The catalytic nature of the process results in very high reaction rates and sulphide decomposition is completed in less than 15 minutes (Fair *et al.*, 1986; Beattie and Raudsepp, 1988; Van Weert *et al.*, 1988; Beattie and Ismay, 1990). Temperatures of up to 120 ºC and an oxygen pressure of 700 kPa are required to maintain the process reactions. During the process, nitric acid dissociates completely in water to produce the nitrate ion, which is a strong oxidising agent (Beattie and Raudsepp, 1988; Foo and Bath, 1989) as shown in Equation 2.10:

$$HNO_3 \rightarrow H^+ + NO_3 \qquad 2.10$$

The nitrate ion also undergoes reactions to form another oxidising agent according to the Equation 2.11:

$$NO_3^- + 3H^+ + 3e^- \rightarrow HNO_2 + H_2O \qquad 2.11$$

Both the nitrate and nitrite species are strong oxidising agents capable of oxidising sulphides yielding nitrous oxide (NO) gas as the reduced species. Nitrous oxide reacts rapidly with oxygen to form nitrogen dioxide, which dissolves in water to regenerate the reactants as shown in Equation 2.12 – 2.14:

$$2NO + O_2 \rightarrow 2NO_2 \qquad 2.12$$

$$4NO_2 + +2H_2O \rightarrow 2HNO_2 + 2HNO_3 \qquad 2.13$$

$$3NO_2 + H_2O \rightarrow 2HNO_3 + NO \qquad 2.14$$

The reactions with pyrite and arsenopyrite proceed at a very fast rate at temperatures above 60 ^oC, pH below 1.7 and an acid strength between 70 and 180 g/l. The reactions are given by Equations 2.15 and 2.16 (Fair *et al.*, 1986):

$$3FeAsS + 14HNO_3 \rightarrow 3FeAsO_4 + 4H_2O + 3H_2SO_4 + 14NO \qquad 2.15$$

$$2FeS_2 + 10HNO_3 \rightarrow Fe_2(SO_4)_3 + H_2SO_4 + 10NO + 4H_2O \qquad 2.16$$

The drawbacks for this process are the generation of arsenide and elemental sulphur which are detrimental to the cyanidation process. However, high pressure and temperature (180 ^oC) are employed to arrest these problems.

2.3.6 Nitrox Process

The nitrox process utilises nitric acid as an oxidant for the decomposition of pyrite and arsenopyrite concentrates under atmospheric conditions. Neither air nor oxygen is involved in these reactions and temperatures are relatively lower. The reactions of pyrite and arsenopyrite in the process are given by Equations 2.17 and 2.18:

$$3FeS_2 + 18HNO_3 \rightarrow Fe_2(SO_4)_3 + Fe(NO_3)_3 + 3H_2SO_4 + 15NO + 6H_2O \qquad 2.17$$

$$3FeAsS + 14HNO_3 + 2H_2O \rightarrow 3FeAsO_4 + 2H_2O + 3H_2SO_4 + 14NO \qquad 2.18$$

Unlike the arseno process, iron nitrate is one of the leach products. The nitric acid used is regenerated and recycled according to the reactions provided in Equations 2.19 and 2.20:

$$2Fe(NO_3)_3 + 3CaCO_3 + 3H_2O \rightarrow 2Fe(OH)_3 + 3Ca(NO_3)_2 + 3CO_2 \qquad 2.19$$

$$3Ca(NO_3)_2 + 3H_2SO_4 \rightarrow 6HNO_3 + 3CaSO_4 \qquad 2.20$$

Due to the very corrosive nature of nitric acid, processes that use this oxidant are not very popular in the gold industry and no plant uses either the nitrox or the arseno process.

2.3.7 Pressure Oxidation

Pressure oxidation techniques employ high pressure, oxygen and temperatures to treat refractory ores or concentrates. This treatment method oxidises the gold bearing sulphide minerals followed by subsequent precipitation of leach products. The process can take place in highly acidic or alkaline medium depending on the mineralogy of the ore. However, high-pressure acidic oxidation is the most widely used pressure oxidation process (Berezowsky and Weir, 1984, 1987; Peters, 1986; Berezowsky and Weir, 1989a, 1989b). Non-acidic or alkaline pressure oxidation is applied to acid-consuming carbonate rich refractory ores. The process is best suited for ores with low sulphide content. The principal reactions involve the total oxidation of sulphidic sulphur and arsenic to sulphates and arsenate respectively (Berezowsky and Weir (1989a, 1989b.) These reactions take place at higher temperatures (between 180 and 225 ^{o}C) and in strongly acidic environments (pH less than 2). The major reactions are represented by Equation 2.21-2.25 below:

$$4FeAsS + 11O_2 + 2H_2O \rightarrow 4HAsO_2 + 4FeSO_4 \qquad 2.21$$

$$2FeS_2 + 7O_2 + 2H_2O \rightarrow 2FeSO_4 + 2H_2SO_4 \qquad 2.22$$

$$2Fe_7S_8 + 31O_2 + 2H_2O \rightarrow 14FeSO_4 + 2H_2SO_4 \qquad 2.23$$

$$4FeSO_4 + 2H_2SO_4 + O_2 \rightarrow 2Fe_2(SO_4)_3 + 2H_2O \qquad 2.24$$

$$2HAsO_2 + O_2 + 2H_2O \rightarrow 2H_3AsO_4 \qquad 2.25$$

Oxidation is carried out at temperatures above 160 ^{o}C to prevent the formation of hydronium jarosite $(H_3OFe_3(SO_4)_2(OH)_6)$ and elemental sulphur and promote complete oxidation of the sulphides to sulphates. Refractory ore treatment by this method normally gives gold extraction in excess of 95%.

2.3.8 Bacteria Oxidation

The Bacteria Oxidation (BIOX®) process is the latest processing technique developed for pre-treating double refractory gold ores. The technology, which was commercialised in 1986 under the name, BIOX®, employs a mixed population of chemolithotrophic bacteria to break down the sulphide mineral matrix, thereby liberating the occluded gold for subsequent cyanidation (Marsden and House, 2006). Bacteria oxidation of gold ores is differentiated from bacteria leaching in that the bacteria used do not dissolve the gold. This technology has gain widespread acceptance due to its environmental friendliness and relatively low operating cost. Several types of bacteria are known to oxidise sulphides but the most common ones in biomining are *Acidithiobacillus thiooxidans*, *Acidithiobacillus ferrooxidans* and *Leptospirillum ferrooxidans* (Livesey-Goldblatt *et al.*, 1983; Hutchins *et al.*, 1987; Rawlings, 1997).

Characterisation of bacteria community in operating BIOX® plants by Van Hille *et al.* (2013) shows marked presence of *Acidithiobacillus caldus*, *Leptospirillum ferriphilum* and *Ferroplasma acidiphilum*. Hence, the mixed culture can now boast of six (6) predominant microbes. These bacteria are acidophilic mesophiles thus perform optimally within the pH range of 1.2-1.8 and temperature range of 35 and 45 ^{o}C. Thermophiles having growth activity from 45 to 55 ^{o}C and from 50 to 80 ^{o}C have also been studied. The extremely thermophilic Sulphobulus species which grow at temperatures of 50 to 80 ^{o}C have been used to pre-treat both sulphidic and carbonaceous gold ores and recoveries in excess of 80% were achieved. Using Sulphobulus for treating various carbonaceous ores and blanking the carbon before cyanidation, recoveries of 84 to 94% were achieved (Hutchison *et al.*, 1988).

Biooxidation of gold ores may take place in continuously stirred tank reactors (CSTR) or in heaps and dumps but the use of stirred tank bioreactors is more prominent. Feed to

bioreactors is mainly flotation concentrate and the processing time is between 3 and 5 days if high sulphide oxidation levels are expected. Sulphide minerals oxidation reactions are exothermic and hence by practice, the process is cooled using water to maintain the operating temperature within range considered optimum. CSTR employ the use of cooling coils for this purpose. The oxygen demand for sulphide oxidation is high and thus large volumes of air has to be injected and dispersed into the slurry and bioreactors are designed to provide efficient and rapid oxygen uptake. Dissolved oxygen levels are kept at 2 ppm and above. During biooxidation, the bacteria require nutrients to sustain growth. Essential elements such as nitrogen, phosphorus and potassium are added into the primary reactors as solution of fertiliser grade ammonium sulphate and potassium phosphate or phosphoric acid (Hackl, 1997; Rawlings, 1997).

Biooxidation process is efficient and cost effective making it an attractive alternative to the conventional roasting and pressure oxidation. The bacteria culture is robust; the plants are simple to operate and have proven high scale up potential. More so the process is environmentally friendly as neutralisation of plant effluents produces precipitates that meet the most stringent environmental regulations.

2.3.9 Post Bacteria Oxidation Processes

High concentrations of sulphuric and arsenic acid, ferrous and ferric compounds and other dissolved ions are produced as end products of the bacteria oxidation. This is usually washed in a three or four stage Counter Current Decantation (CCD) circuit to separate the oxidised gold bearing solids from the acidic liquor before conditioning to pH 10-11 for cyanide leaching (Marsden and House, 2006). The washed biooxidised product would normally contain less than 1 g/l total iron in solution with an acid pH of 1-3. Removal of iron which is known to retard gold dissolution is necessary before cyanide leaching to promote gold recovery and reduce cyanide consumption. The arsenic-rich CCD effluent is acidic and very toxic, and it is a known carcinogen. It is neutralised with limestone and lime in a two-stage process to a pH between 6 and 7 to precipitate arsenic, iron and sulphates as a stable precipitate of ferric arsenates. The environmentally acceptable waste sludge is then impounded safely at the tailings dam. Extensive research and development have gone into the optimisation of the design of the neutralisation section of the BIOX® process in recent

years. This was to ensure that the effluents comply with the most stringent international environmental regulations with respect to product stability and residual arsenic concentrations (Marsden and House, 2006; Afidenyo, 2008). The chemistry of the arsenic precipitation process is presented in Equations 2.26-2.30 as follows:

$$H_2SO_4 + CaCO_3 + H_2O \leftrightarrow CaSO_4.2H_2O + CO_2 \quad 2.26$$

$$2Fe_2(SO)_4 + 2H_3AsO_4 + H_2SO_4 + 7CaCO_3 \leftrightarrow 2FeAsO_4 + 2Fe(OH)_3 + 7CaSO_4 + 7O_2 + H_2O \quad 2.27$$

$$Fe_2(SO)_4 + 2H_3AsO_4 + H_2SO_4 + 7CaCO_3 \leftrightarrow FeAsO_4 + 2Fe(OH)_3 + 7CaSO_4 + 8H_2O \quad 2.8$$

$$MSO_4 + Ca(OH)_2 + 2H_2O \leftrightarrow M(OH)_2 + CaSO_4.2H_2O \quad 2.29$$

Where,

M= Mg, Cu, Ni, Zn, etc.

$$H_2SO_4 + CaO \leftrightarrow CaSO_4 + H_2O \quad 2.30$$

Equation 2.26 represents free acid neutralisation with limestone or seashells (pH 1.4 to 2.0) to form gypsum and Equation 2.27 is the precipitation of ferric arsenate with limestone (pH 2.0 to 4.0) during the first stage (Marsden and House, 2006). Ferric iron and other dissolved tramp metals are precipitated in the second stage as the pH is further raised with lime to 7 as represented by Equations 2.28 and 2.29. Equation 2.30 represents neutralisation of the excess acid present by further lime addition to pH of 8. The thickened biooxidised product is conditioned to pH 10-11, and gold recovered by traditional cyanidation at the leaching circuit. The cyanide tailing which is the second waste stream generated by the process joins the neutralised sludge at the tailings pond. Some of the plants now use part of the leach circuit tailings or flotation tails by virtue of its alkalinity as a neutralising media to cut down on cost of lime consumption. The gold is adsorbed onto activated carbon by either the Carbon in Leach (CIL) or Carbon in Leach Process (CIP). The carbon is eluted with strongly basic cyanide solution, and the gold is electrowon from solution onto stainless steel wool cathodes, then calcined, smelted and cast into bullion.

2.4 Nutrition and Energy

Bacteria used in bioleaching are remarkable in that they have very modest nutritional requirements. Aeration of a sample of iron pyrite in acidified water is sufficient to support the growth of *A. ferrooxidans* and *L. ferrooxidans*. Air provides the source of carbon; nitrogen and oxygen, pyrite the energy source whereas trace elements and acidified water provide an environment that stimulates growth (Madigan and Martinko, 2006; Marsden and House, 2006). Nitrogen (N), potassium (P), phosphorus (K), magnesium and sulphur are the major nutritional requirements for the bacteria strain. Magnesium and sulphur requirements are mostly met from quantities present in the ore and water whiles NPK are found in trace quantities in the ore. Nitrogen is an essential element that is credited for protein cell development via the amino acid building blocks (Clarke, 1989).

2.4.1 Carbon Sources

A. ferrooxidans strains that have been confirmed as being pure are obligate autotrophs (Madigan and Martinko, 2006). Some early studies appeared to show that after a period of adaptation, A. *ferrooxidans* was able to grow on organic substrates and that this was followed by a permanent loss of the ability to oxidise iron. However, the G+C mole % ratio of the cultures changed under these conditions and heterotrophic growth was almost certainly due to the inability of researchers to free their cultures from the presence of the closely associated heterotrophic bacteria belonging to the genus *Acidiphilium* (Harrison, 1984).

The iron-dependent, mixotrophic growth of one strain has been reported but unfortunately that isolate has been lost (Barros et al., 1984). Carbon dioxide fixation in *A. ferrooxidans* takes place via the Calvin reductive pentose phosphate cycle. One of the most important enzymes in this process, ribulose l,5-biphosphate carboxylase (RuBPCase) has been characterised (Holuigue *et al.,* 1987). The growth on ferrous iron was reduced unless the concentration of CO_2 in the air was increased. This observation is in contrast to the other statements (Kelly and Jones, 1978; Norris, 1989) in which it was found that the concentration of CO_2 in the air was sufficient to avoid limitation on growth on ferrous iron and mineral sulphide oxidation by *A. ferrooxidans* (Rawlings, 1997).

The bacterium responds to CO_2 limitation by increasing the cellular concentration of RuBPCase. Indeed, *A. ferrooxidans* strain Fe_1 has two sets of the structural genes for RuBPCase (Kusano *et al.*, 1991). The two sets are separated by more than 5 kb and the nucleotide sequence of the coding region of each set is identical although the flanking regions varied substantially. The RuBPCase gene regulator, RbcR, has been isolated and sequenced (Kusano and Sugawara, 1993). Very little work has been carried out on the enzymology or genetics of CO_2 fixation by either *L. ferrooxidans* or *A. thiooxidans* (Rawlings, 1997).

2.4.2 Nitrogen Sources

The study of the nitrogen requirements of bioleaching organisms is complicated by the phenomenon that ammonia is highly soluble in acid solutions. Atmospheric ammonia readily dissolves in leach solutions and may provide most, if not all, of the nitrogen required for growth. As little as 0.2 M ammonium has been reported to be sufficient to satisfy the nitrogen requirement of *A. ferrooxidans* (Tuovinen *et al.*, 1971). However, this concentration is dependent on the amount of ferrous iron or mineral present in the medium or leach liquor. High concentrations of inorganic or organic nitrogen are inhibitory to iron oxidation. *A. ferrooxidans* is diazotrophic and is able to reduce atmospheric nitrogen to ammonia. This property was first reported by Mackintosh (1978) who demonstrated that *A. ferrooxidans* was able to incorporate $^{15}N_2$ 1abel into cellular material. It has since been shown that all fifteen isolates of *A. ferrooxidans* tested contain the structural genes *(nifHDK)* for the nitrogen fixing enzyme, nitrogenase (Rawlings, 1997).

The ability to fix nitrogen is therefore almost certainly a general property of *A. ferrooxidans*. The *nifHDK* genes from *A. ferrooxidans* ATCC 33020 have been cloned and sequenced (Pretorius *et al.*, 1985; Pretorius *et al.*, 1997). There is evidence that *A. ferrooxidans* is also capable of fixing atmospheric nitrogen. Genomic DNA from the *A. ferrooxidans* type strain was reported to give a positive hybridisation signal with a nifHDK gene probe from Klebsiella pneumoniae (Norris *et al.*, 1995).

A. ferrooxidans was also shown to reduce acetylene to ethylene and oxidise ferrous iron to ferric iron at low oxygen concentrations. This ability was repressed by added ammonium

ions, behavior which is indicative of the ability to fix nitrogen. The ability of *A. thiooxidans* to fix nitrogen is uncertain. No hybridisation signal was obtained when a *nifHDK* gene probe from *Klebsiella pnuemoniae* was used against chromosomal DNA from *A. thiooxidans* ATCC 8085, but a positive signal was obtained when a *A. ferrooxidans nifHDK* probe was hybridised to an unidentified *A. thiooxidans* isolate (Dew *et al.*, 1997). The role of nitrogen fixation in bioleaching operations is difficult to predict. The dissolution of atmospheric ammonia in acid solutions could provide sufficient ammonium to suppress nitrogen fixation. Furthermore, nitrogen fixation is inhibited under fully aerobic conditions therefore might not occur in a well-aerated leaching operation. In the highly aerated, high oxidation rate, BIOX® tanks used to pre-treat gold-bearing arsenopyrite ores, addition of a small amount of ammonia in the form of low-grade fertiliser is required to enhance mineral oxidation (Rawlings, 1997).

2.4.3 Phosphorus Sources

Phosphorus is supplied in the form of phosphate. Phosphate seems to be necessary for both energy metabolism and the initial steps of Fe^{2+} oxidation outside the cell wall of *Acidithiobacillus ferrooxidans* (Clarke, 1989). The total removal of PO_4^{3-} from the media will result in a slower rate of iron oxidation but the bacteria will still remain active.

2.4.4 Potassium Sources

Potassium is the principal inorganic cation in the cell and is the co-factor of some enzymes. The supply of potassium to the bacteria is somewhat controversial (Clarke, 1990). High concentrations of K^+, coupled with a high Fe^{3+} concentration, enhances jarosite ($KFe_3(OH)6(SO_4)^2$) formation, which will deplete the available K^+ from the solution and cause coating of the sulphide mineral. Since the pH in a BIOX® reactor is between 1.2 - 1.8 and the temperature is below 45°C the degree of jarosite formation is limited (Habashi,

1999). However, occasional process control failures may result in jarosite formation and hence loss of potassium.

2.5 Optimisation of Nutrients

During biooxidation, the bacteria require nutrients to sustain growth. Essential elements such as nitrogen; phosphorus and potassium are added into the primary reactors as solution of fertiliser grade ammonium sulphate and potassium phosphate or phosphoric acid (Hackl, 1997: Rawlings, 1997). The role of nutrients (NPK) is of economic and biological importance when it comes to biological oxidation of refractory (sulphidic) ores as a pre-treatment process. Like the Golden Star (Prestea/Bogoso) plant, BIOX® plants spend huge sums of money in procuring NPK for the process. Therefore, this work seeks to elucidate a good optimisation process that will keep the already high operating cost of BIOX® relatively low.

2.6 Relevant information about Golden Star (Bogoso/Prestea) Limited

Golden Star (Bogoso/Prestea) Limited (GSBPL) is a subsidiary of Golden Star, a mid-tier gold mining company with two operating mines Wassa and Bogoso/Prestea in Ghana (Anon., 2013; Adam, 2015).The Bogoso/Prestea concession is located on the Ashanti belt (Kesse, 1985) in the Tarkwa Municipality of the Western region of Ghana, which runs from Axim in the south to Konogo in the north, over a distance of some 240 km. It is located some 35 km northwest of Tarkwa (85 km northwest of the port city of Takoradi) with good access roads and an established infrastructure. The property is some 360 km by road west of the capital, Accra and 130 km by road from the port city of Takoradi on the Atlantic coast.

The Prestea property is also located near Prestea, 20 km southwest of Bogoso. A paved road from Takoradi to Kumasi passes through the concession with the mining areas connected by gravel haul roads. The village of Bogoso is located at the junction of this road with a major east-west artery from Prestea to Enchi on the Côte D'Ivoire border. The mine

is in the neighborhood of Bogoso, a small town in the Tarkwa Municipality. The mine occupies an area of 100 km^2 and lies entirely within the Ankobra basin. It stretches between latitude 050 29'N and 050 37'N and longitude 0010 59'W and 0020 11'W. Bogoso/Prestea mine has its own back-up power system (6.4 MW) to the national hydroelectric power grid to which it is connected (Keith and Alan, 2001).

2.6.1 Local Geology

In the Bogoso area the Ashanti Belt structural discontinuity is referred to as the central structural corridor or central fault zone. This corridor passes through the centre of the concession and gold mineralisation occurs along the 18.5 km strike length. The central structural corridor separates the Birimian structural domain in the west from the Tarkwaian structural domain in the east with both suites hosting a network of faults and imbricate fault slices. The Tarkwaian rocks are not considered to be prospective for gold mineralisation in the Bogoso area (Anon., 1998).

The most intense faulting occurs along the western edge of the central structural corridor and is referred to as the Main Crushed Zone (MCZ). The MCZ varies in width from 1 000 m to 2 500 m with major gold deposits occurring at bends or junctions within the fault system. The fault system is dominated by early thrusting, which resulted from northwest-southeast compression. The fault was later re-activated and appears to have undergone sinistral wrench type movement (Allibone *et al.,* 2002). Numerous interlocking splays occur within the MCZ. About 90% of gold production to date has come from the MCZ. Mineralised splay structures outside of the MCZ have been identified but there has been little exploration of these targets to date. Gold occurs in two primary ore types; principally, as refractory sulphide ores in the host rocks where arsenopyrite is the main ore mineral (sub-types of this include; graphitic shear zones, siliceous ores, carbonate alteration zones and wall rock hosts), and, as quartz±sulphide±carbonate veins (limited to the Bogoso North area) with free-milling gold. Both quartz veins and sulphide ores are restricted or confined to structures that cross-cut pre-existing foliations in the rock fabric. The quartz veins are probably younger than the gold-bearing arsenopyrite (Allibone *et al.,* 2002). A number of gold deposits have been delineated on the Bogoso Concession.

The principal deposits from south to north are: Chujah South, Chujah Main, Chujah Stage 3, Chujah North, Dumasi, Dumasi North (Nankafa), Marlu and Bogoso North. Oxide resources have largely been exhausted on the property and the resource base, while substantial, is limited to generally, refractory sulphides. All of the deposits are open at depth.

Mineralisation in the Chujah South deposit is associated with two moderate to steeply dipping structures, the Chujah footwall fault which runs sub-parallel to the MCZ and an easterly trending splay which intersects the bounding fault with the Tarkwaian (Chujah fault). Mineralisation is associated with the intersection of these structures and minor northeast trending faults.

A series of westerly dipping faults host oxide mineralisation. Faulting in Chujah South is extremely complex and controlled by fault bounded volcanic bodies (Allibone *et al.*, 2002). The Chujah Main and Chujah Stage 3 deposits are separated by a thinning or pinching out of the MCZ mineralisation. Gold is associated with finely disseminated pyrite and arsenopyrite. Increased gold grade is associated with an increased concentration of arsenopyrite. Sulphide in the Chujah Main deposit has been mined prior to the closure of the roaster. Gold in both deposits is associated with graphitic mylonites and carbonate alteration (Anon., 1998). A substantial portion of the known sulphide resources occurs in the Chujah deposits.

At Dumasi, gold is also hosted in graphitic mylonites and carbonate altered rock. The deposit is associated with a sinistral flexure in the MCZ and hanging wall and footwall splays. The deposit is approximately 380 m along strike, 30 m - 35 m wide and tested to 100 m in depth. The dips are gentler (50 – 600) than those at Chujah with the gold grade being slightly lower. The Dumasi North deposit (Nankafa) is 400 m along strike 30 m - 70 m perpendicular to strike and drilled to a vertical depth of 200 m. The Dumasi deposits also account for a substantial proportion of the gold resources on the Bogoso concessions (Allibone *et al.,* 2002).

The Marlu deposit over the last decade has been a hub for mining activities employing the use of both underground and open-pit methods. Higher grades are associated with sporadic quartz veins and graphitic mylonite. The deposit is approximately 450 m along strike and

15 m - 25 m in width and untested at depth. The deposit dies out northward in a low-grade graphitic fault zone and opens up in the Bogoso North deposit. At Bogoso North two splays of the MCZ, a graphitic footwall structure and a silicified, quartz vein dominated hanging wall structure extend for 500 m along strike. Quartz vein mineralisation is associated with higher gold grades of 5 g/t-15 g/t. Gold is associated with abundant arsenopyrite (Allibone *et al.*, 2002).

2.7 The Sulphide Processing Plant

GSBPL treats sulphide and transition ores as well as tailings. The sulphide ores treated are refractory in nature. The three major sulphide minerals in the refractory ore are pyrites (FeS_2), arsenopyrite (FeAsS) and Arsenian Pyrite (Pyrite with arsenic in solid solution) (Anon, 1999; Anon, 2005a; MacIntyre, 2005; Adam, 2015).

Sulphide ore treatment started in February 1991 at Bogoso/Prestea through the flotation and concentrate roasting process route using the Dorr Oliver fluo-solids roasters. In January, 1994 the roaster failed and coupled with poor gold recoveries and environmental challenges associated with arsenic disposal, a decision was made to discontinue the treatment of sulphide ores. Oxide and transition ores were treated until March 2007 when the BIOX® plant was commissioned (Anon., 1999; Adam, 2009; 2015).

2.7.1 Process Flow Diagram

The Bogoso/Prestea sulphide plant was designed to treat 3.5 Mta ore refractory ore. The initial stages involved in the processing are Crushing, Milling and Flotation (CMF), BIOX®, CIL, Elution and Electrowinning. The process flow diagram is shown in Figure 2.1.

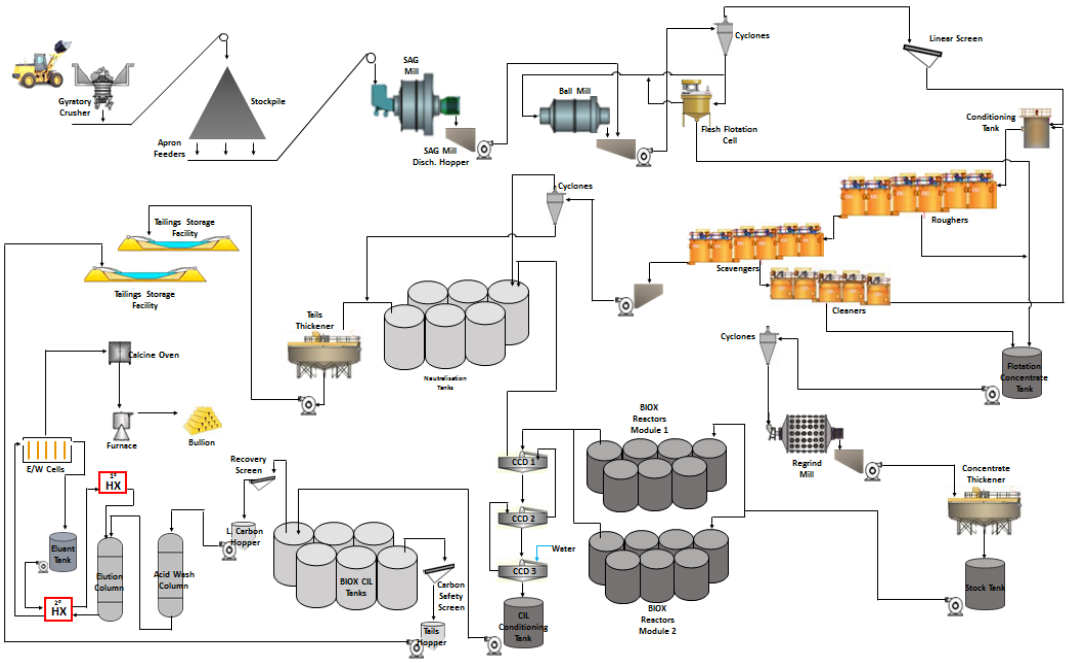

Figure 2.1 Process Flow Diagram of Bogoso/Prestea Sulphide Plant

2.7.2 Crushing Circuit

A 2000 tph 60″x89″ Fuller Traylor Gyratory Crusher is employed in crushing. The 500 mm ROM ore is reduced to sizes of about 80% passing 120 mm. Size reduction occurs as the ore passes downwards being crushed against the faces of the concave shells and the mantle. The ore discharges into a rock box down to a variable speed conveyor (CV 11) which help regulate the amount of feed being discharged through the transfer chute to CV 13.

The ore on CV 13 then discharges onto the stockpile (which is able to take 25,000 t of crushed ore) which has four vibrating feeders underneath, which discharges feed ore onto the mill feed conveyor (CV 14). The stockpile is shown in Figure 2.2.

Figure 2.2 Sulphides and Transition Ore Stockpile

2.7.3 Milling Circuit

The SAG (5.5 MW) mill operates in open circuit whiles the ball mill operates in closed circuit with a bank of cyclones. The SAG mill shown in figure 2.3 grinds to a P80 of 350 µm while the Ball Mill grinds to a P80 of 140 µm. The Ball Mill cyclones underflow is fed to the flash flotation cell and the flash tails returns to the Ball mill (MacIntyre, 2005; Adam, 2009; 2015).

Figure 2.3 SAG Mill

2.7.4 Flotation Circuit

Flotation is used to concentrate the sulphides in the slurry for further treatment in the BIOX® section. The ball mill cyclone overflow discharges into two conditioning tanks each with a capacity of 160 m^3 via a 12 m^2 trash removal screen. The conditioning tanks are equipped with agitators for effective mixing of slurry before they get into the flotation cells. The residence time for the feed in the conditioning tanks are 4 minutes.

The flotation circuit consists of a flash cell, two parallel tank cell streams (flotation 1 and flotation 2). There are 3 roughers and 3 scavengers each with a capacity of 70 m^3 for each stream. Scavenger concentrates are polished in five cleaner cells. Concentrate from the flash, roughers, and cleaners are classified by a bank of cyclones in closed circuit with a re-grind mill. This produces a product of P80 of 45 μm from which is thickened to 50 % w/w solids and stocked in a 1130 m^3 capacity Surge tank which provides about 24 hours surge capacity for BIOX® (MacIntyre, 2005; Adam, 2009; Adam, 2015). The Figure 2.4 shows a rougher cell.

Figure 2.4 Rougher Flotation Cell

2.7.5 Biological Oxidation Circuit

The BIOX® circuit consists of two modules of seven reactors each. The reactor configuration on each module is four primaries, three secondaries, with the flexibility of using the fourth primary reactor as a secondary. The Bogoso/Prestea BIOX® reactors are the largest reactor sizes in the BIOX® Community with a capacity of 1500 m^3 (Anon., 2005b; Adam, 2009; Jan van, 2009; Adam 2015). Concentrate is fed to the two BIOX® modules by means of variable speed pumps equipped with automatic dilution systems. High density pulp from the surge tank is diluted to 20% solids with process water and fed to the feed box on top of the BIOX® tanks. Nutrients mix (trade named Alimentor) is fed together with the concentrate feed to the primary reactors following BIOMIN's standard nutrient addition rate of N, P and K of 1.7 kg/t, 0.3 kg/t and 0.9 kg/t respectively. The primary reactors overflow to the secondaries operating in series (Anon 2005b; MacIntyre, 2005; Adam, 2009, 2015).

The temperature of the BIOX® culture is controlled between 40-45 ºC by temperature control valves on the inlet manifolds on the cooling coils and the heat removed via heat exchangers on Suzdar evaporative cooling towers. Low pressure air is supplied by 6 blowers

into the sparge rings below the axial flow agitator turbines to maintain dissolved oxygen levels greater than 3 ppm. Slurry pH is controlled manually between pH 1.3-1.8 by the addition of milk of lime or sulphuric acid (Anon, 2005b, MacIntyre, 2005, Adam, 2009, 2015). A flow diagram of the Bogoso/Prestea BIOX® plant is shown in Figure 2.5.

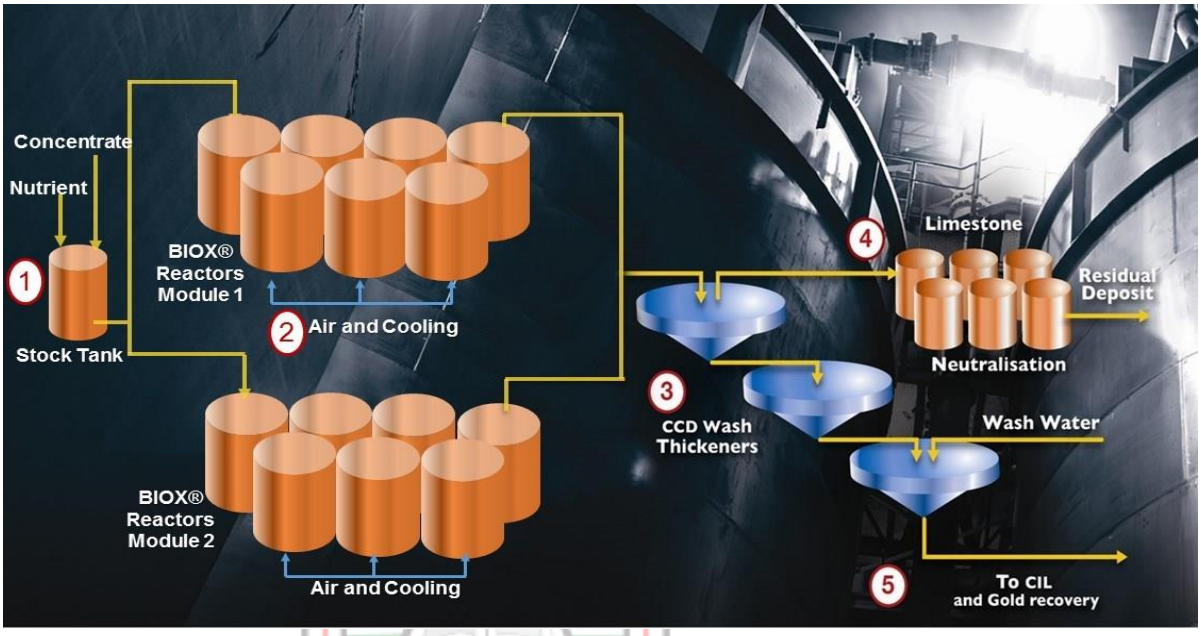

Figure 2.5 Biological Oxidation Circuit

2.7.6 Counter-Current Decantation (CCD) Thickeners and neutralisation

The BIOX® product from the last secondary reactor on both modules is washed in a series of three Counter Current Decantation (CCD) 14 m High-Rate thickeners. The acidic CCD overflow solution is neutralised in the neutralisation circuit with flotation tails and pulp of grounded seashells in the first stage (pH of 5) and milk of lime in the second stage to pH 7.0. The neutralised effluent is thickened, and water recovered for re-use in the plant (Anon, 2005b; Adam, 2009, 2015). The neutralisation reactions are shown in Equations 2.31 to 2.33.

Stage 1: pH 5

$$2H_3AsO_4 + Fe_2(SO_4)_3 + 3CaCO_3 \rightarrow 2FeAsO_4 + 3CaSO_4 + 3CO_2 + 3H_2O \quad 2.31$$

$$Fe_2(SO_4)_3 + 3CaCO_3 + 3H_2O \rightarrow 2Fe(OH)_3 + 3CaSO_4 + 3CO_2 \quad 2.32$$

Stage 2: pH 7

$$H_2SO_4 + CaO \rightarrow CaSO_4 + H_2O \quad 2.33$$

2.7.7 CIL, Elution and Electrowinning

The thickened CCD underflow is conditioned to pH 10.5 with lime and then fed to the CIL circuit consisting of six 760 m^3 tanks. About 50 g/l of regenerated carbon is added to the head tank to combat preg-robbers in the concentrate. Sixteen (16) tonnes of recovered carbon is treated with 3% HCl acid at 1 bed volume (1BV) and gold recovered in a Zadra elution and electrowinning circuit with 3% NaOH.

2.7.8 Tailings Management

Tailings from leach circuit and neutralised effluent are pumped to the tailings dam. Golden Star Resources, Bogoso divides the dam into cells namely Cell 1, Cell 2 and Cell 3. All of these cells are non-cyanide with the exception of Cell 3. Pollution to Water (P2W) has been contracted to treat the cyanide water from Cell 2. Deposition is by the method of spigot rather than cyclone. Spigots disposals (sub aerial) are fixed along the perimeter of the facility of 50 by 80-meter dams. The spigots deposit materials from the Sulphide Treatment Plant (STP) and materials settle from coarse to ultra-fines creating a beach between the embankment and the supernatant pond surrounded by beached tailings.

Penstocks are installed at the pools to limit the pool water by pumping the supernatant water back to the plant. Emergency penstocks are installed to kick in case of the rare event (1-in-100 years) storm when the pool is within 100 m from the wall. Water management is also by Decant Barge which decants the day to day demands. A minimum free-board of about 4 m is allowed. Seepage drains are installed to cater for the inevitable discharges from the toe

of the tailings storage facility into a sump for monitoring changes and anomalies of seepage effluents. Routine monitoring is done on the free board, tonnage of material coming into the dam, the density or percentage water coming into the dam, and rainfall. Water from the tailing dam is held in a holding pond from where it is pumped to the plant for use in the various plant processes. The aerial view of tailings storage facility is shown in Figure 2.6.

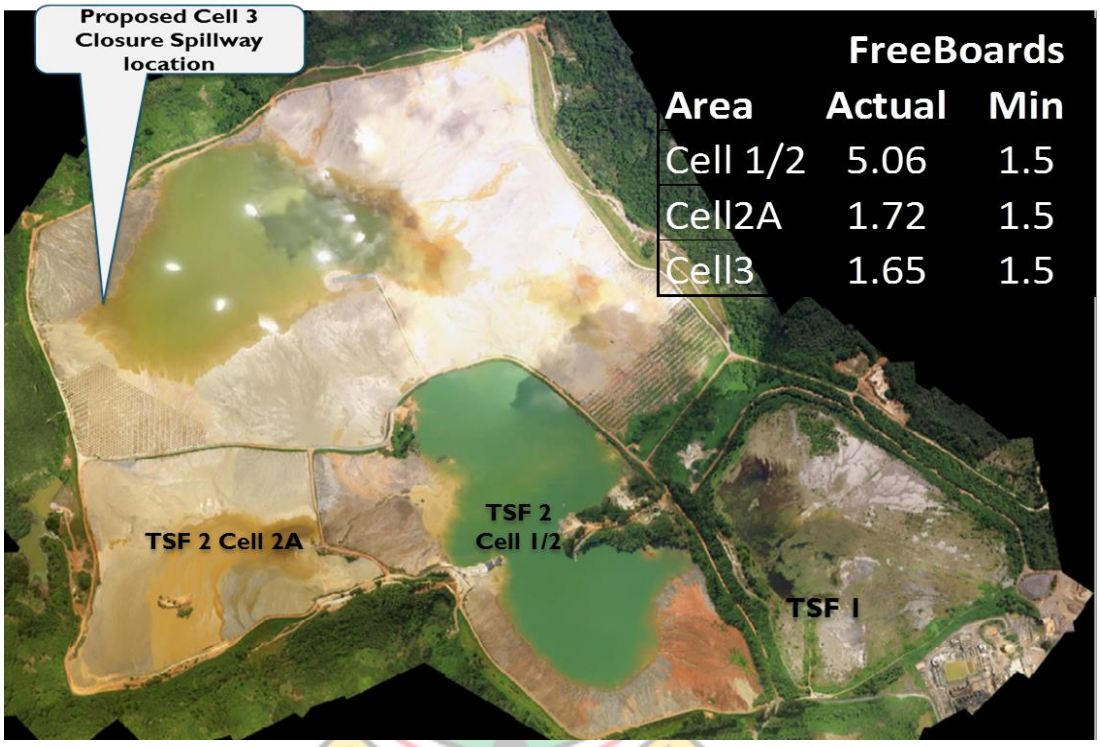

Figure 2.6 Bogoso/Prestea Tailings Storage Facility

CHAPTER 3

MATERIALS AND METHODS

3.1 Materials

The flotation concentrates for this study were composited over a period of four weeks from the Golden Star (Bogoso/ Prestea) Limited (GSBPL) Sulphide plant. About 30 kg of double refractory flotation concentrate filter cake was dried in an oven at a temperature of 80 ^{o}C for 12 hours. The dried cake was crushed, homogenised and split into 12 samples using the rotary sample splitter. Each of the 12 samples were sub-sampled and 50 g each taken as head samples for gold, sulphide sulphur and organic carbon analysis. The reject samples were homogenised and used for this investigation.

The bacteria inoculums for the BIOX® test were collected from a secondary reactor at the GSBPL BIOX® plant. About 10 L of inoculum was collected and preserved in a sample refrigerator at the University of Mines and Technology, Minerals Engineering Laboratory. Analytical grade of ammonium sulphate, potassium sulphate and phosphoric acid were obtained from the Analytical Laboratory of GSBPL to supply NPK. Potassium dichromate solution, spekker acid mixture, ferric acid, stannous chloride, saturated mercuric chlorides and barium diphenylamine sulphonate indicator solution (BIOX® reagents for various titrations) were prepared at the Analytical Laboratory of GSBPL.

3.2 Laboratory Equipment

Standard bottle roll set-up (Figure 3.1) was used for the BIOX® product cyanide amenability leach tests. A Metrohm 826 pH meters and Syland TEMP-O_2-MAT-4000 Dissolved Oxygen (DO) meters were used for pH and DO measurements respectively. Labtec Essa drying oven (Figure 3.2), Rocklabs laboratory crusher (Figure 3.3), Rotary sample splitter (Figure 3.4) and Micro-analytical weighing balance (Figure 3.5) were used for the sample preparations.

Figure 3.1 Bottle Roll Set-Up

Figure 3.2 Labtec Essa Drying Oven

Figure 3.3 Rocklabs Laboratory Crusher

Figure 3.4 Rotary Sample Splitter

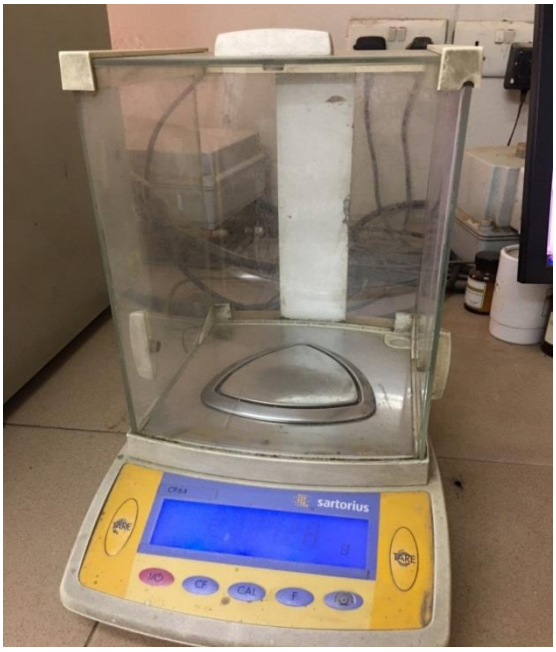

Figure 3.5 Micro Analytical Weighing Balance

3.3 Chemical Analysis

Sulphur and carbon contents were determined by the volumetric combustion method using a LECO titrator SC-444DR (Figure 3.6). The LECO was first calibrated with at least three replicates of the standards used and about 0.1 -0.2 grams of the sample was weighed into a 529-204 crucible and mixed well with about 1 gram of 502-321 COM-CAT. The mouse was clicked to analyse and the sample was slided into the combustion tube when the "load furnace" message appeared on the screen.

Gold content was determined by conventional fire assaying with AAS (Figure 3.7) finish. After the sample was dried, crushed, splitted and pulverised, about 30 g was weighed into the crucible and fluxed. The sample was then taken to fusion and de-slagging followed by cupellation and button preparation. The button was digested with acqua-regia and the resulting solution was sent for atomic absorption spectroscopy (AAS) analysis.

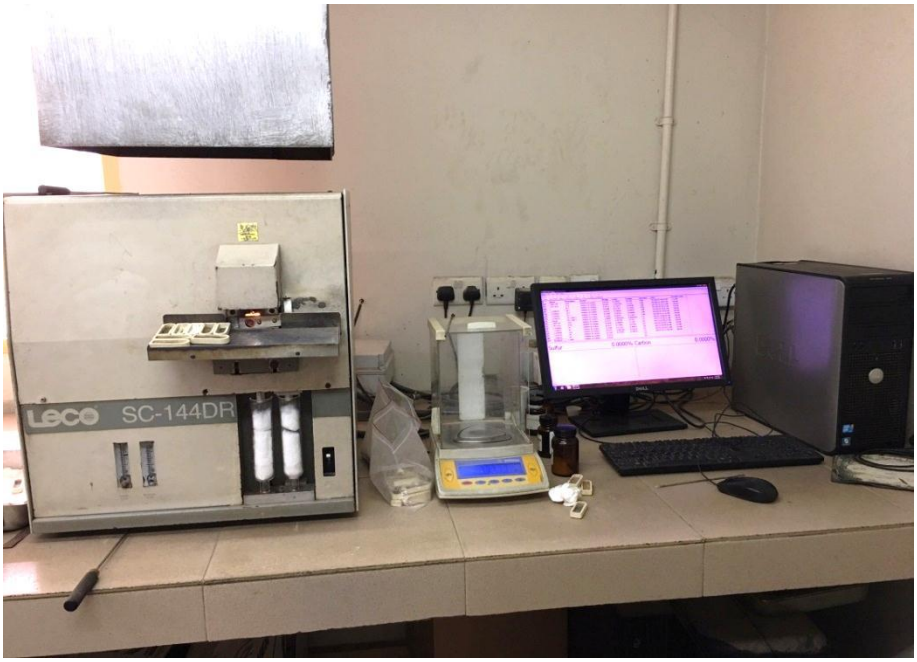

Figure 3.6 LECO (Carbon and Sulphur Analyser)

Figure 3.7 AAS (Atomic Absorption Spectroscopy) Machine

3.4 Mineralogical Analysis

The samples were dried in an oven at 60 ºC for overnight and grounded into fine powder. The samples (about 1-2 grams each) were packed in a powder mount, against glass, to provide a stable surface for the analysis. The samples were analysed using an Empyrean

Powder X-ray Diffractometer (XRD) to determine which mineral phases were present and the peak heights associated with a particular mineral were measured.

3.5 Acid Leaching and Nutrient Concentration Evaluation

Flotation concentrate sample weighing 1000 g was pulped to 20% solid with process water and pH lowered to values between 1.3-1.6 with sulphuric acid. The tests were performed in 4-litre Pyrex volumetric cylinder (Figure 3.8) on a Single Stirred Bench (SSB) at GSBPL Metallurgical Laboratory. The sample was placed on a hotplate and the temperature was maintained at 40 - 41^{o}C throughout the acid leaching periods as shown in Figure 3.8. Leaching was done at 40 °C +/- 1 °C and pH 1.3 - 1.6 with an agitation speed of 400 rev/min to simulate the conditions used in the BIOX® reactor. Dissolved Oxygen (DO) concentrations were maintained at 3 ppm.

A total of six tests were performed and the NPK concentrations in the leach solution was analysed after 1, 3 and 5 days.

Figure 3.8 Acid Leaching Set-Up

3.6 Biooxidation of Chujah ore concentrate (BIOX® Test)

A series of 18 batch bioleach tests were performed on a Chujah ore flotation concentrate sample using the LOM-150 DIG/500 Low Temperature shaker Incubator (Figure 3.11). The aim of the test work was to determine the relative importance and minimum requirements of the nutrient elements N, P and K to ensure good bacteria activity and sulphide oxidation. The batch tests were divided into 4 groups (Table 3.1): Nitrogen, Potassium, and Phosphorus and stepwise changes. For Groups 1 to 3, the concentration of the focus element was lowered while the concentration of the remaining nutrient elements remained constant. The tests in Group 4 were performed at progressively lower additions of NPK. The final test was done with no nutrient addition to confirm the availability of the bio-leachable nutrient with the residence time frame of the BIOX®. Table 3.1 shows the nutrient addition chart for the various BIOX® tests.

Table 3.1 Series of 18 Batch Bioleach Test Performed with the Nutrient Addition

Sample ID	Description/Group	Nutrient Additions, kg/t		
		Nitrogen	Phosphorus	Potassium
Test 1	Nutrient Blank	0.00	0.00	0.00
Test 2	Genmin Standard	1.700	0.300	0.900
Test 3	Group 1 *(stepwise drop in nitrogen)*	1.190	0.300	0.900
Test 4		0.833	0.300	0.900
Test 5		0.580	0.300	0.900
Test 6		0.406	0.300	0.900
Test 7	*Group 2 (stepwise drop in phosphorus)*	1.700	0.210	0.900
Test 8		1.700	0.147	0.900
Test 9		1.700	0.103	0.900
Test 10		1.700	0.072	0.900
Test 11	*Group 3 (stepwise drop in potassium)*	1.700	0.300	0.630
Test 12		1.700	0.300	0.441
Test 13		1.700	0.300	0.309
Test 14		1.700	0.300	0.216
Test 15	*Group 4*	1.190	0.210	0.630

Test 16	*(progressive drop in NPK)*	0.833	0.147	0.441
Test 17		0.580	0.103	0.309
Test 18		0.406	0.072	0.216

The conditions utilised for the bioleach test are given below :

i. pH: Between 1.2 - 1.6 by addition of lime and/or sulphuric acid;
ii. Temperature : 40 ^{o}C +/- 1^{o}C;
iii. Inoculum : 11% (v/v) of active Bogoso inoculum;
iv. Nutrients: $(NH_4)_2SO_4$, K_2SO_4 and H_3PO_4;
v. Slurry density: 20% solids; and
vi. Monitoring: Fe^{2+} and Fe^{3+} titrations, pH, redox concentration and sulphide sulphur oxidation.

Each sample was subjected to a 10-day-biooxidation period. The first 4 days were incubation periods and monitoring the next 6 days. The biological oxidation set-up and incubator used are shown in Figures 3.10 and 3.11 respectively.

Figure 3.9 Bio-Leaching Set-Up

Figure 3.10 LOM-150 DIG/500 Low Temperatures Shaker Incubator at UMaT Minerals Engineering Laboratory

3.7 Bacteria Activity Monitoring

3.7.1 Total Iron in Solution Determination

Total iron in solution was determined as follows:

i. Five (5.0) ml aliquot of BIOX® liquor was pipetted into a 100 ml conical flask.
ii. Approximately, 30 ml of ferric acid was added to the aliquot.
iii. The solution was boiled for 5 minutes and stannous chloride was added drop-wise until the solution turned colourless. One extra drop was added.
iv. The boiled solution was cooled to room temperature before approximately 10 ml of mercuric chloride solution was added.

v. About 4 to 8 drops of barium diphenylamine sulphonate (indicator) solution was added and titrated with the standard potassium dichromate solution. The potassium dichromate was added drop-wise until the end point while the conical flask was swirled. The end point was observed to be an intense permanent purple colour.
vi. The total iron in the 5 ml aliquot BIOX® solution was calculated as: Titre (ml) = Total iron in solution (g/l).

3.7.2 Ferrous Iron in Solution Determination

Ferrous Iron in solution determined as follows:

i. Five (5.0) ml aliquot of BIOX® liquor was pipetted into a 100 ml conical flask.
ii. About 10 ml of spekker acid was added to the aliquot (A clear yellow colouration was observed)
iii. About 3 to 4 drops of barium diphenylamine sulphonate (indicator) solution was added and titrated with the standard potassium dichromate solution. The potassium dichromate was added drop-wise until the end point while the conical flask was swirled. The end point was observed to be an intense permanent purple colour.
iv. The ferrous iron in the 5 ml aliquot BIOX® solution was calculated as: Titre (ml) = Ferrous iron in solution (g/l).

The ferric iron concentration in the solution was determined by the difference as shown in Equation 3.1 (Musonda, 2006):

$$[Fe^{3+}]\ (g/l) = [Fe\ T]\ (g/l) - [Fe^{2+}]\ (g/l) \qquad 3.1$$

The redox potential, slurry pH and DO levels were measured using their respective meters.

3.8 Cyanidation of Bacteria-Treated Material

After 200 g each of flotation concentrate was pre-treated with bacteria in 18 different test conditions, it was very necessary to test the amenability of each BIOX product to cyanidation since the main purpose of pre-treatment is to enhance gold recovery.

30 g of each BIOX products in triplicates were weighed into leach bottles and pulped with process water to 30% solids. The first leach bottle was to test for preg-robbing index (PRI) and the last two bottles were for Carbon-in-Leach (CIL). The pH was maintained between 10.5 to 11.5 with quick lime and the dissolved oxygen levels, above 15 ppm with pure oxygen for the pre-leach stage. Sodium cyanide was used to keep the cyanide ions in solution at 4000 ppm and activated carbon was added to the CIL samples at 50 g/l. These conditions were already established in the MacIntyre, (2005) Bogoso Feasibility Report. With exception of the PRI bottles which were rolled for 40 hours, all the CIL samples were leached at 12, 24 and 40 hours to monitor the leach kinetics.

CHAPTER 4

RESULTS AND DISCUSSION

This thesis set out to investigate how step-wise reduction in Genmin standard Nitrogen-Phosphorus-Potassium (NPK) in Bacteria Oxidation (BIOX®) operation will yield similar results to or even better results than the standard addition rate with high gold recovery in the subsequent leaching of BIOX® products. The results of the experimental work are presented and discussed in four (4) groupings as follows: chemical and mineralogical analyses of the flotation concentrate, acid leaching of the flotation concentrate, biooxidation of the flotation concentrate (BIOX®), and cyanidation of the BIOX® product.

4.1 Chemical and Mineralogical Analysis of Samples

4.1.1 Partial Chemical Analysis of Flotation Concentrate

The results (Table 4.1) confirm that the concentrate is sulphidic (13.83%) and the about 3.0% difference between total sulphur and sulphide sulphur suggests the presence of elemental sulphur and sulphates due to natural oxidation. Adam, (2015) attributed the presence of sulphates to the buffer stockpile kept by the Bogoso operations to arrest mill downtimes during low mining periods. Again, the sample for the test work was stored at the laboratory for about a year before the test work campaign commenced; hence, further oxidation is probable.

The arsenic (3.33%) compared to the iron (34.12%) confirms the ore has more pyrite than arsenopyrite. Consistent with expectation, Musonda, (2006) predicted the main minerals in the flotation concentrate are pyrite (38%) and arsenopyrite (6%), hence, this work confirms his predictions.

Two factors critically considered before an ore is said to be suitable for BIOX® treatment are the ratios of Au: S^{2-} and S^{2-}: CO_3^{2-}. The industry rule of thumb is Au: S^{2-} of higher than 0.7, as this renders the project viable with low downstream operating costs (Aylmore and Jaffer, 2012; Adam, 2015). The concentrate has Au:S^{2-} ratio being more than 2:1 and hence,

satisfying one condition. Again, S2-:CO_3^{2-} ratio of the concentrate is about 1.77. Adam, (2015) ascribed that lower ratios render the BIOX® reactions net acid consuming, and the reverse is also true. Sulphuric acid was added to keep the pH within the required range (1.2-1.6) resulting in a mass loss of concentrate across the BIOX® circuits. This was evident with the high-grade difference between flotation concentrate of 39.47g/t and maximum BIOX® product grade of 54.04g/t. The addition of quick lime to correct low pH results in the precipitation of gypsum and a mass gain after BIOX®. Gypsum passivates gold surfaces and reduces the amenability to cyanidation recovery.

The concentrate contains organic carbon ranging between 2.76 and 3.05, which is indicative of a high preg-robbing material. These values are in agreement with those obtained by previous researchers who characterised some ores/concentrates from Golden Star Bogoso Resources (Ofori-Sarpong *et al*., 2013; Adam *et al*., 2017).

Table 4.1 Partial Chemical Analysis of the Flotation Concentrate

Sample ID	Au g/t	S^{2-} %	TS %	Org C %	TC %	CO_3^{2-} %	As %	Fe %
Met 1	41.18	12.66	19.15	3.05	4.71	8.30	3.50	31.27
Met 2	42.59	14.68	20.35	3.04	4.53	7.45	3.42	38.78
Met 3	38.85	14.37	20.57	2.84	4.38	7.70	3.51	26.27
Met 4	42.14	12.73	18.98	2.89	4.59	8.50	3.67	37.86
Met 5	42.76	13.83	20.78	2.95	4.70	8.75	3.50	30.10
Met 6	36.72	13.74	15.05	2.97	4.62	7.25	3.53	30.29
Met 7	37.84	13.64	14.48	2.95	4.35	7.00	2.90	39.87
Met 8	35.45	13.31	14.27	2.81	4.31	7.50	3.54	38.14
Met 9	37.82	14.58	15.65	2.76	4.61	9.25	2.37	28.89
Met 10	39.46	13.87	14.14	2.90	4.26	6.80	3.12	43.81
Met 11	39.05	14.63	17.34	2.83	4.34	7.55	3.31	38.77
Met 12	39.79	13.96	13.48	2.90	4.44	7.70	3.53	25.38
Average	***39.47***	***13.83***	***17.02***	***2.91***	***4.49***	***7.81***	***3.33***	***34.12***

4.1.2 X-Ray Diffraction (XRD) Analysis of Samples

X-Ray Diffraction results of the BIOX feed, BIOX product of the Genmin Standard (Test 2) and the BIOX product of test 5 (Higher SOX and CIL) shows that the major phases in the samples were pyrite (FeS_2), quartz (SiO_2) and magnetite (Fe_3O_4). The phases in the BIOX feed were pyrite (FeS_2), arsenopyrite (FeAsS), dolomite [$CaMg(CO_3)_2$] and karrooite ($MgTi_2O_5$).

The presence of these gangue minerals poses challenges to gold recovery in downstream processes. Figure 4.1 is the graphical presentation of the X-Ray diffraction analysis of the BIOX feed. Again, the BIOX product of the Genmin standard (Test 2) also have all the three main minerals found in all the samples. The XRD of the test 2 is also presented in Figure 4.2. Lastly, test 5 (Higher SOX and CIL) had the presence of the carbonate dolomite and the XRD analysis is presented in Figure 4.3. The results show that biooxidation was not complete as the product contained some amount of sulphide minerals.

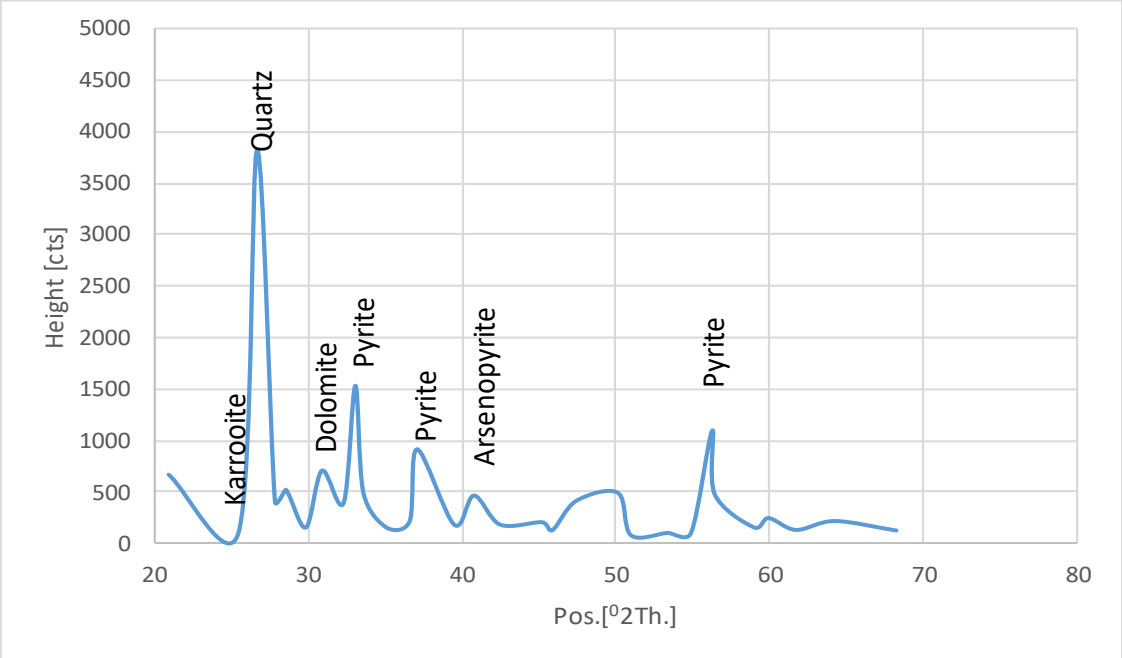

Figure 4.1 XRD of BIOX® feed

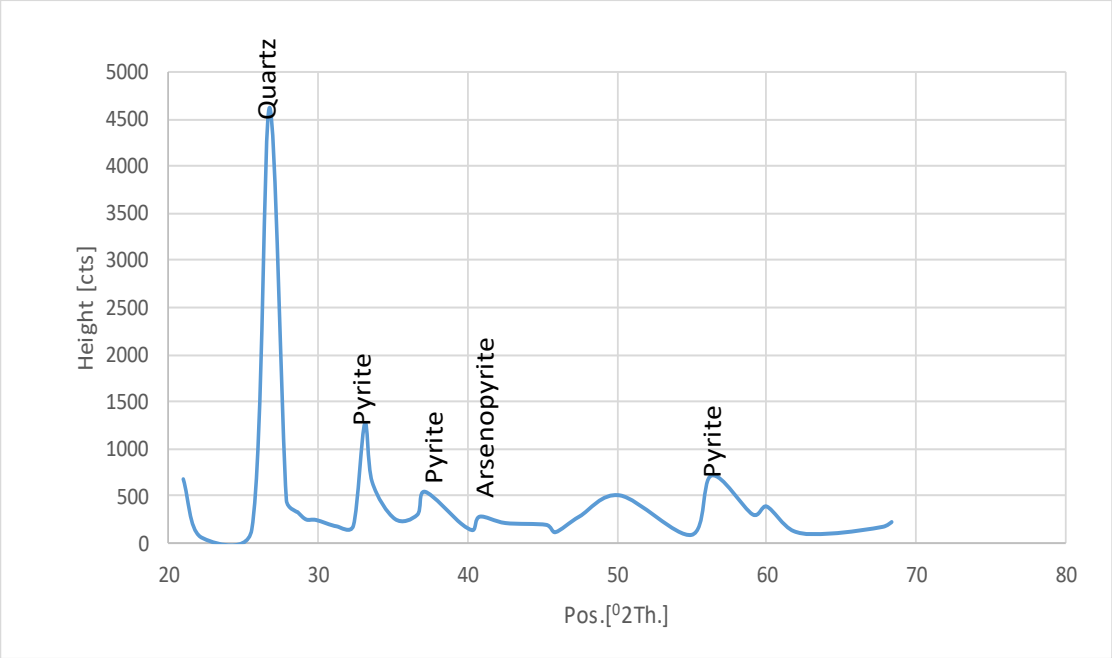

Figure 4.2 XRD of BIOX® product of the Genmin standard (test 2)

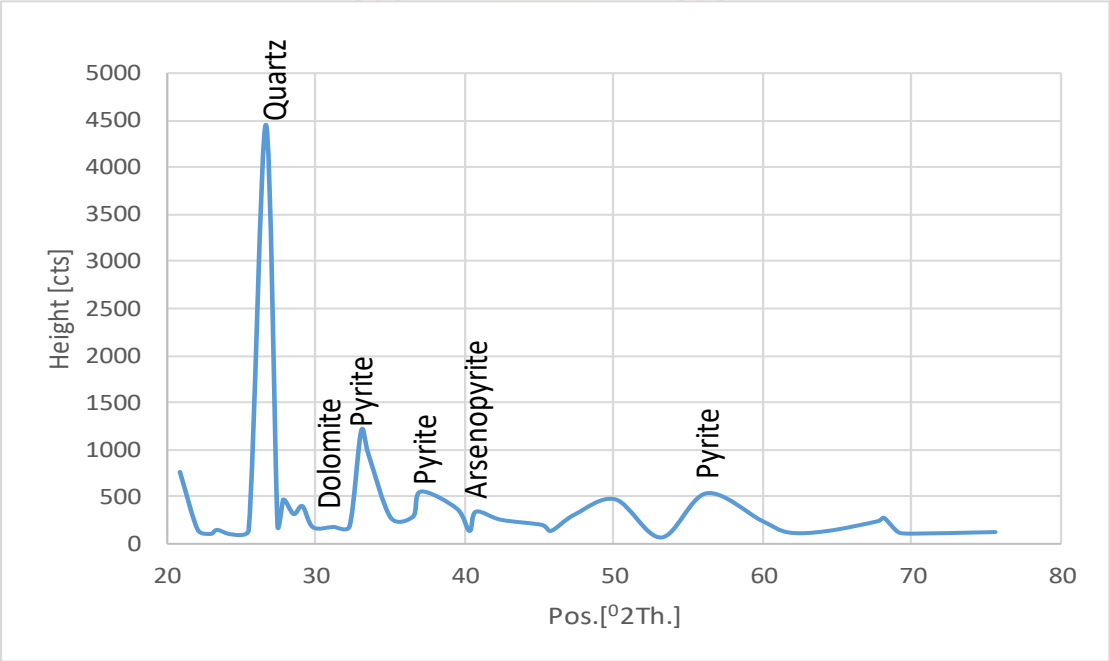

Figure 4.3 XRD of BIOX® product of test 5 (Higher SOX and CIL)

4.2 Acid Leaching of Nutrients from Flotation Concentrate

An investigation into possible nutrient sources within the flotation concentrate of Bogoso Chujah ore was conducted. The availability of N, P and K were determined by acid leaching a representative sample from the Chujah flotation concentrate and analysing the solution products for traces of the nutrient elements. Preliminary acid leaching of the BIOX® feed revealed substantial amount of bio-leachable nutrients (NPK) in the ore. At the end of 5 days residence time, 0.362 kg/t nitrogen, 0.702 kg/t phosphorus and 0.417 kg/t potassium were made bio-available (refer to Table 4.2) in the BIOX® stream as analysed by SGS Analytical Laboratory, Tema. This represents about 21%, 234% and 46% respectively of nitrogen, phosphorus and potassium with reference to the Standard BIOMIN nutrient addition of 1.7 kg/t, 0.3 kg/t and 0.9 kg/t of NPK respectively as presented in Table 4.2.

Table 4.2 Nutrient Concentration Evaluation

SUMMARY TABLE FOR BIOAVAILABLE NUTRIENTS (NPK) IN CHUJAH ORE						
Sample	***NUTRIENT, mg/L***			***NUTRIENT,Kg/t of conc.***		
ID	***N***	***P***	***K***	***N***	***P***	***K***
24 HRS	158	296	123	0.201	0.375	0.153
72 HRS	275	309	197	0.349	0.387	0.239
120 HRS	292	567	332	0.362	0.702	0.417

The result (Table 4.2) clearly indicates that about 27% of the K needed by the bacteria for oxidation as defined by BIOMIN was bio-available in the primary reactors (3 days residence) and cumulatively 46% in the secondary reactors of the Bogoso/Prestea BIOX® streams. Adam, (2015) predicted the formation of jarosite ($KFe_3(OH)_6(SO_4)_2$ of the Bogoso/Prestea BIOX® streams after extensive studies on the BIOX® products. The formation of jarosite was attributed to excessive supply of potassium in the presence of high Fe^{3+} concentrations at pH above 2.

With the addition of 0.9 kg/t (BIOMIN standard) of K to the Bogoso/Prestea BIOX® streams and the levels of K recorded by these studies, prediction of jarosite formation in the Bogoso/Prestea BIOX® streams has been confirmed by this work. Habashi, (1999) also

ascribed the depletion of K from solution and subsequent coating of the sulphide mineral to jarosite formation. pH of slurry above the value of 2 and longer residence times typified in BIOX® circuits due to operational inefficiencies enhances the formation of jarosite. (Dew, 1995; Chetty *et al.*, 2000; Deveci *et al.*, 2004; Márquez *et al.*, 2006; Gahan, 2008; Fantauzzi *et al.*, 2011; Gagliardi and Cashion, 2013). Again, Fe^{3+} undergoes precipitation at high pH (above 2.0) and temperature (above 45 ^{o}C) to form jarosite.

From the results of this work, about 21% of the N needed by the bacteria for proper oxidation was bio-available in the primary reactors and did not significantly change in the secondary reactors of the Bogoso/Prestea BIOX® streams. Carbon and nitrogen are to a large extent responsible for the cell growth and activity of the bacteria (Afidenyo, 2008). Again, the amount of bio-available concentrations of P in the primary reactors was found to be 29% (0.375 kg/t) and 34% (0.387 kg/t) more than the required quantity (0.300 kg/t) in the secondary reactors required for bacteria activity. Thus, the Bogoso/Prestea BIOX® streams can afford a reduction or total elimination of the P component in the nutrient mix. This can be explained by the high concentration of available phosphorus present in Chujah concentrate. Decreasing the phosphorus addition had no negative effect on the bacteria activity and final sulphide sulphur oxidation. In some cases, sulphide sulphur oxidation (SOX) rather improved with decreasing P concentration, suggesting that excess P concentration could be detrimental to bacteria activity and overall reduction in SOX.

4.3 Bacteria Oxidation of Sulphide Sulphur in Flotation Concentrate

In order to examine the leaching behaviour of the concentrate, 18 series of experiments at different NPK additions were conducted, and oxidation rate of the sulphide minerals was studied. Bacteria Oxidation (BIOX®) process, make use of chemolithotropic or 'autotrophic bacteria". Autotrophic bacteria, the most common mineral-oxidising bacteria, derive their energy from oxidation of inorganic compounds. These bacteria oxidise ions such as ferrous and sulphur. Hence, gold locked up in iron and sulphur containing minerals can be liberated when subjected to bacteria oxidation, as a result of the feeding habit of these microbes (Amankwah *et al.*, 2005). Sulphide sulphur monitoring and sulphide sulphur oxidation for the test works are presented in Figures 4.4, 4.5 and 4.6, 4.7 respectively.

The bacteria activity monitoring for the first batch of the test works is also presented in Table 4.3. The pH was maintained between 1.2 -1.6 with sulphuric acid and the DO increased from about 1.8 ppm during the incubation period to above 3.0 ppm at the monitoring stage. The redox at the incubation stage was about 450 but gradually increased to above 500 which shows the activity of the bacteria. Again, as ferrous iron was reducing from the incubation stage to the monitoring stage (0.50 to 0.15 g/l), total iron in solution was increasing from about 6 to 9 g/l. This is a clear indication of the convention of ferrous iron to ferric iron. The convention of ferrous to ferric is also a measure of the bacteria activity.

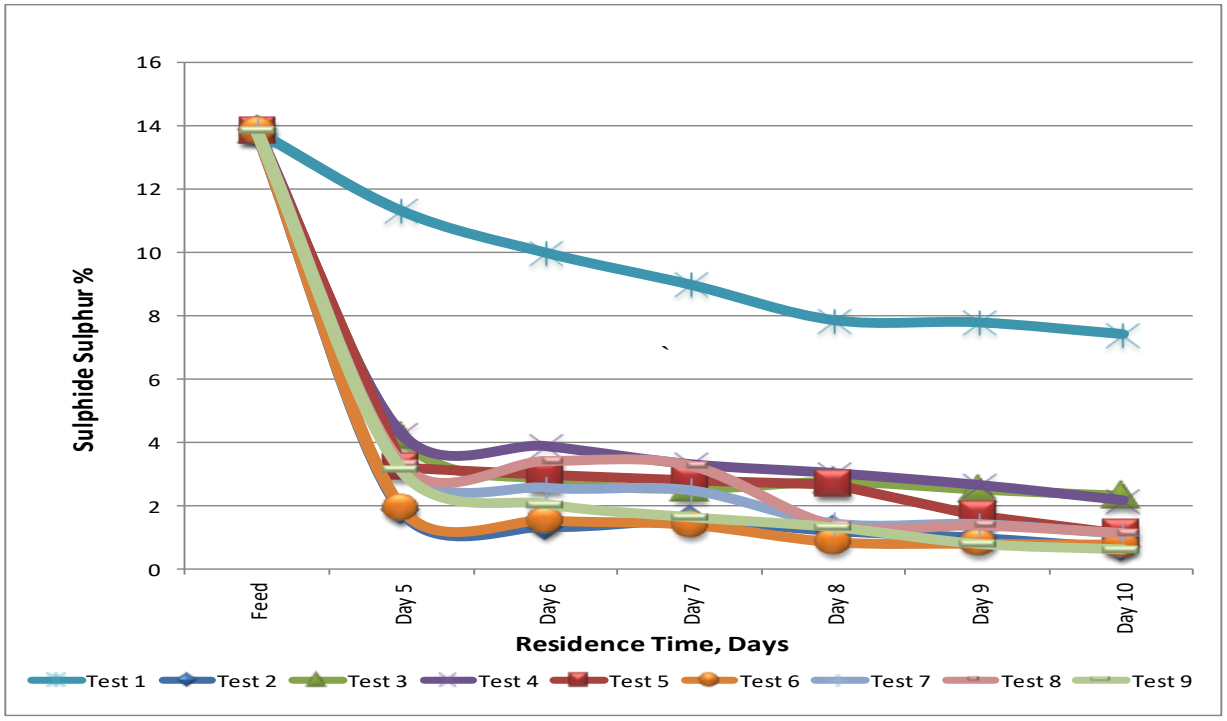

Figure 4.4 BIOX® Tests 1-9 Sulphide Sulphur Monitoring

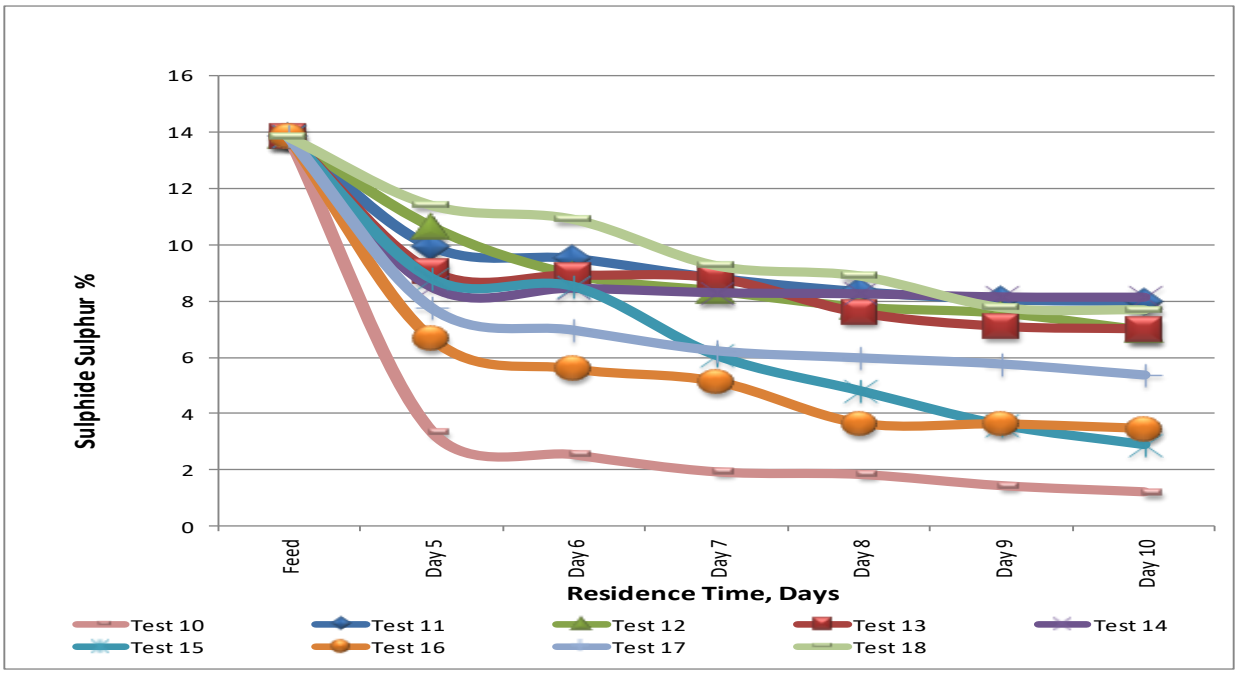

Figure 4.5 BIOX® Tests 10-18 Sulphide Sulphur Monitoring

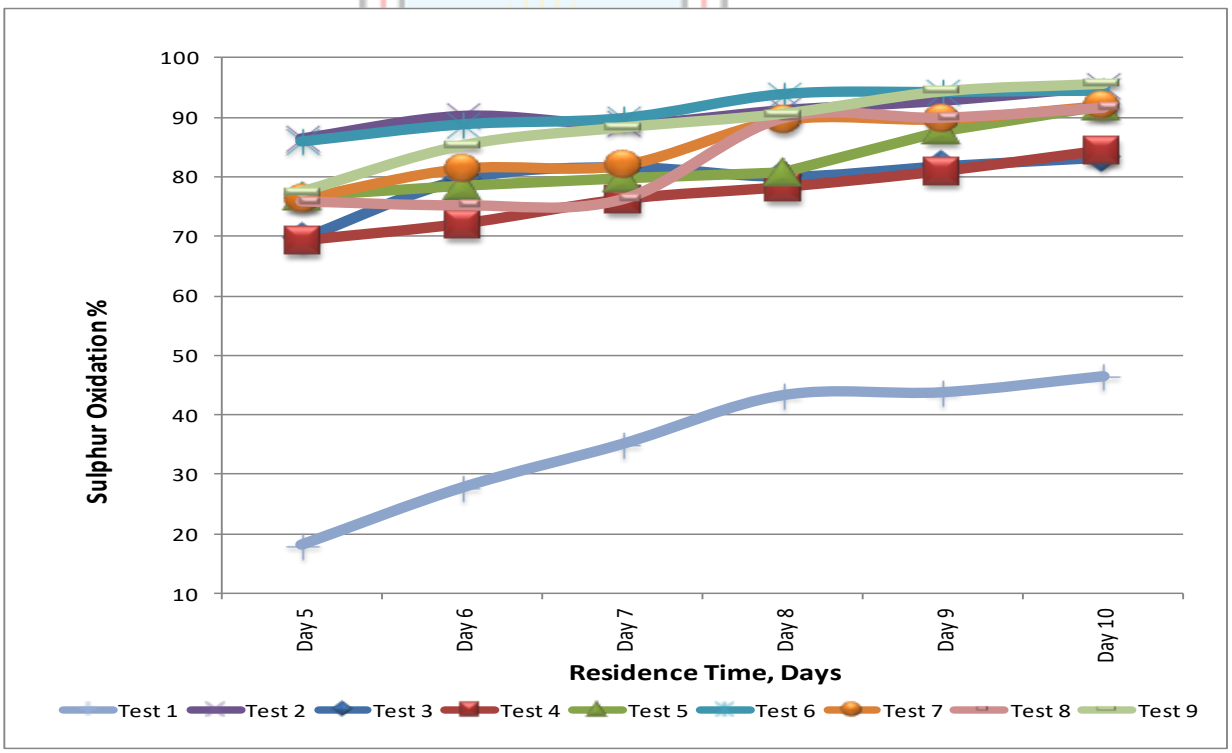

Figure 4.6 BIOX® Tests 1-9 Oxidation Monitoring

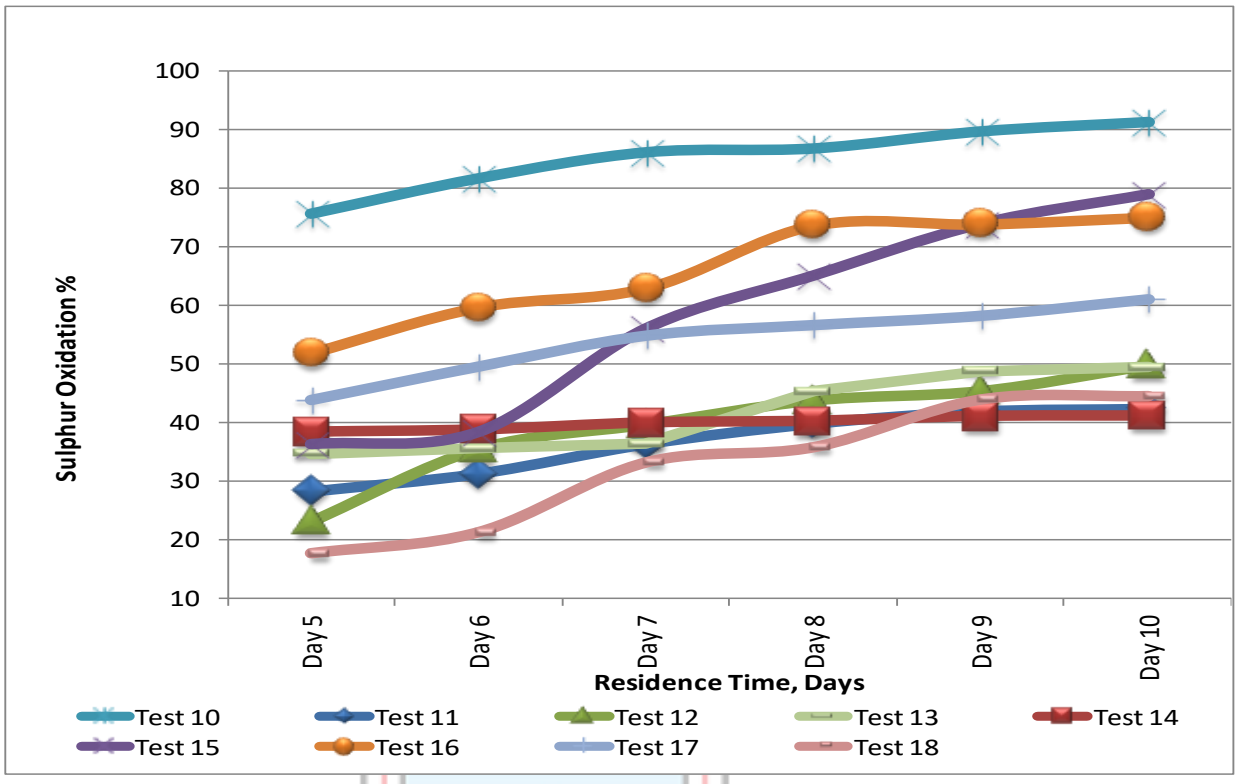

Figure 4.7 BIOX® Tests 10-18 Oxidation Monitoring

Table 4.3 Bacteria Activity Monitoring

INCUBATION PERIOD (DAY 1 TO 4)					
SAMPLE ID	PH	REDOX	DO	FERROUS	TOTAL IRON
A	1.18	448	2.9		
B	1.13	448	1.9		
C	1.16	457	1.4		
D	1.16	455	1.5		
E	1.23	449	1.4		
MONITORING PERIODS (DAY 5 TO 10)					
DAY 5					
A	1.44	504	1.2	0.10	6.80
B	1.74	498	2.2	0.55	6.20
C	1.30	449	1.8	0.75	6.75
D	1.51	485	1.6	0.50	5.45
E	1.50	501	1.7	0.40	4.50
DAY 6					
A	1.45	536	2.3	0.09	6.45
B	1.33	512	2.9	0.35	6.65
C	1.31	506	2.4	0.50	7.10
D	1.52	506	2.6	0.35	5.80
E	1.54	491	2.7	0.30	4.90
DAY 7					
A	1.41	541	2.8	0.43	7.50
B	1.26	514	3.2	0.30	7.30
C	1.30	516	3.0	0.45	7.70
D	1.50	506	2.9	0.45	6.30
E	1.43	518	3.1	0.20	6.50
DAY 8					
A	1.35	502	2.9	0.28	9.20
B	1.20	514	3.4	0.25	8.35
C	1.22	513	3.2	0.30	9.20
D	1.38	509	3.0	0.45	7.90
E	1.32	539	3.4	0.25	7.25
DAY 9					
A	1.35	545	3.2	0.30	9.00
B	1.15	529	3.6	0.10	7.65
C	1.18	534	3.5	0.20	9.45
D	1.32	508	2.9	0.50	6.80
E	1.29	531	3.7	0.10	8.00
DAY 10					
A	1.42	568	3.8	0.08	9.10
B	1.19	525	3.8	0.10	8.20
C	1.21	527	3.6	0.20	9.80
D	1.33	511	3.2	0.25	7.40
E	1.21	533	3.7	0.10	9.80

In Test 1, SOX of 46% was achieved with no nutrient addition hence confirming the presence of in-situ bio-leachable NPK which sustained the bacteria to achieve that SOX. The NPK addition recommended by Genmin (Test 2) yielded SOX of 95% in accordance with the findings of Adam, (2015).

Sulphur oxidation was generally not affected even as the nitrogen addition reduced (Tests 3-6). This is in accordance with works done by Oliveir *et al*, (2005). Notwithstanding the above, nitrogen addition is very essential for the activity of the bacteria culture used and hence the authors suggested that the ore had enough in-situ nitrogen which could meet the demand for the bacteria activity. Again, it was observed that Tests 5 with a 66% reduction of the Genmin standard rate of N yielded SOX of 92% and Test 6 with a 76% reduction of the Genmin standard rate of N gave SOX of 94%. Clarke, (1989) confirms that the study of the nitrogen requirement for bio-leaching organisms is a complex phenomenon. Autotrophic bacteria, the most common mineral-oxidising bacteria, derive their energy from oxidation of inorganic compounds like sulphur and iron. Nitrogen is an essential element that is required for protein cell development via the amino acid building blocks (Clarke, 1989). From Figure 4.8, a further reduction in N leads to starvation which leads to higher SOX and subsequent higher CIL recovery. Further reduction in N below 0.580 kg/t is led to a reduction in CIL recovery implying that the bacteria cannot be starved of N below 0.580 kg/t. The highest CIL recovery is achieved with N reduction from 1.700 kg/t to 0.580 kg/t.

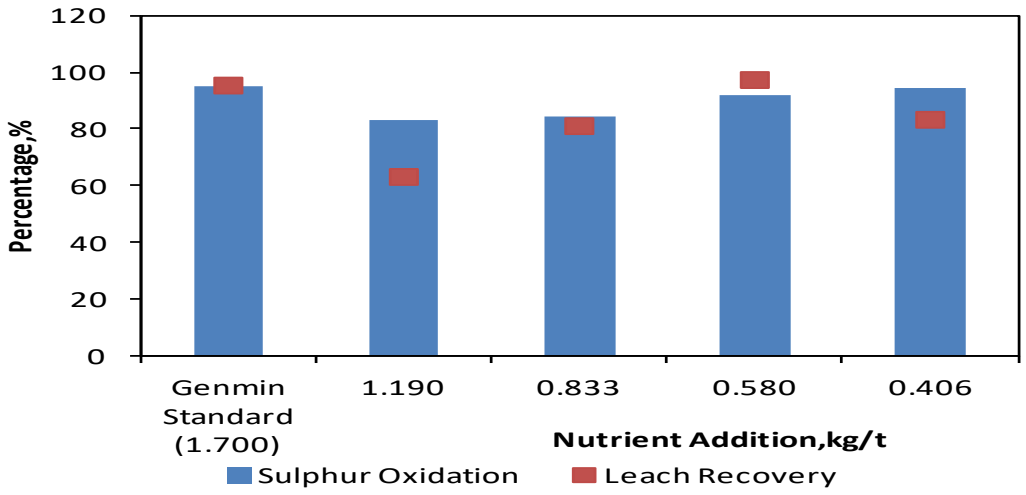

Figure 4.8 Stepwise Reduction of Nitrogen (Tests 3-6).

Tests 7-10 show that SOX was not affected when phosphorus addition was reduced by as much as 75% of the quantity proposed by Genmin for efficient bio-oxidation. The concentrate sent for bio-oxidation had enough phosphorus (Bio-available P in 24 and 120 hours were 0.375 and 0.702 kg/t respectively) to sustain bacteria activity and subsequent oxidation of sulphide sulphur. In addition, the oxidation rate proceeded steadily in all progressive reduction in the amount of phosphorus added. Addition of the proposed standard quantity would be an oversupply. Phosphate seems to be necessary for both energy metabolism and the initial steps of Fe^{2+} oxidation outside the cell wall of *A. ferrooxidans* (Clarke, 1989). The total removal of PO_4^{3-} from the media will result in a slower rate of iron oxidation but the bacteria will still remain active. This implies liberation of gold occluded in the iron matrix is slowed, hence resulting in possible gold loss to iron at the end of the five days BIOX® residence time. Again, SOX was not affected with reduction of phosphorus as the sulphur oxidising bacteria were not impacted negatively by phosphorus reduction. From Figure 4.9, for as much as a 75% drop in the quantity proposed by Genmin for efficient bio-oxidation, 91% SOX was achieved as compared to 92% realised for no reduction in concentration of phosphorus added.

Phosphorus reduction leads to lower CIL recovery. These could be attributed to slimes/precipitates such as iron hydroxide formed which coat the surfaces of the gold particle hence reducing the amenability to cyanidation. For BIOX® solutions with pH less than 5, iron concentrations are very high and react very quickly with phosphorus to create iron phosphate which is easily washed out at CCD because it is soluble (Anon., 2017). At low phosphorus levels however, most of the iron which find its way to the CIL circuit forms iron hydroxide which precipitates and coats the surfaces of the gold reducing its amenability to cyanidation in the process.

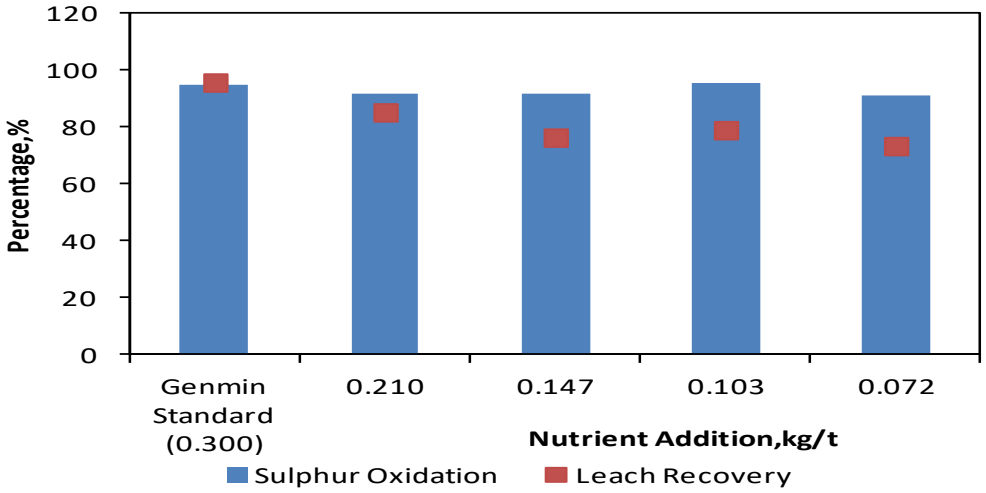

Figure 4.9 Stepwise Reduction of Phosphorus (Tests 7-10).

Conversely, a drop in the standard addition masses in potassium (Tests 11-14) proposed by Genmin led to a drastic drop in oxidation typified by low sulphide sulphur oxidation conditions. As low as a 30% drop in potassium concentration drastically affected the SOX; oxidation dropped from about 95.2% (Standard Addition) to as low as 42.3% as shown in Figure 4.10.

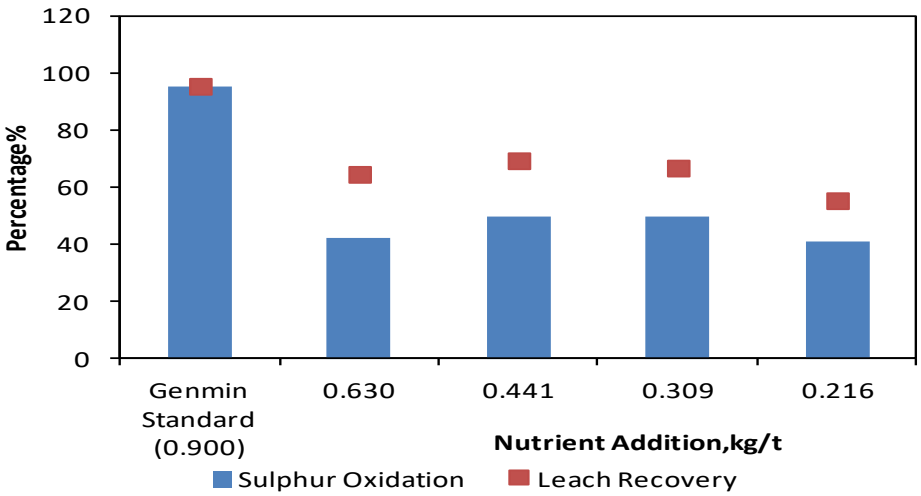

Figure 4.10 Stepwise Reduction of Potassium (Tests 11-14).

SOX dropped drastically with combined drop in concentration of NPK, which moves to confirm the fact that K is very essential in the activity of the bacteria. Even the high performance of the bacteria in low nitrogen and phosphorus conditions could not lead to high oxidations when the mass of potassium added was reduced. The results of the impact of K on N and P is presented graphically in Figure 4.11.

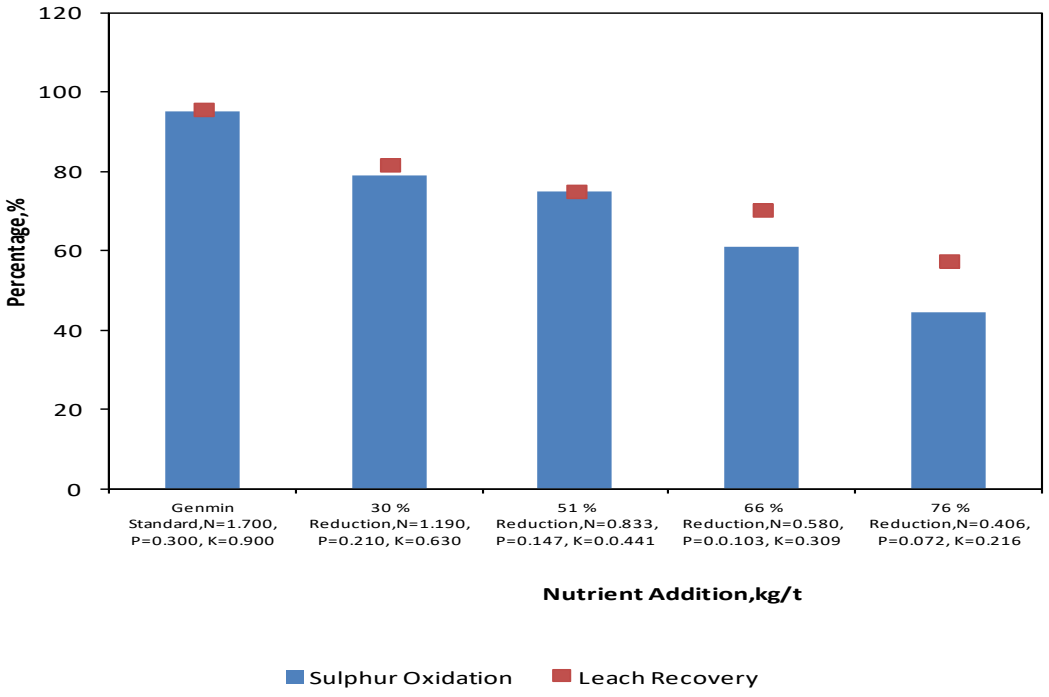

Figure 4.11 Stepwise Reduction of NPK (Tests 15-18).

4.4 Cyanidation of Biological Oxidation (BIOX®) Product

Laboratory investigation was carried out in an attempt to see how the optimum CIL recovery can be attained when processing the various BIOX products. To establish the impact of leach residence time on recovery, samples of the various BIOX products were leached at 12, 24 and 40 hours. MacIntyre, (2005) established an optimum residence time of 40 hours during the Bogoso feasibity study because the project revenue diminishes with further residence time or addition of leach tanks. Cyanidation was conducted at an average grind size of 91.3% passing 75 µm, and respective lime and cyanide consumptions of 6.22 kg/t and 17.35 kg/t were the condition for the cyanidation.

The average preg-robbing index (PRI) is about 59% (refer to Table 4.4). The preg-robbing index of all the BIOX products were evaluated by comparing the direct CIL leach (CIL- 40 hours) with cyanidation without carbon (CN-40 hours). The difference in gold recovery performance indicates the preg-robbing index (PRI).

The results of the cyanidation tests are presented graphically in Figures 4.12 and 4.13

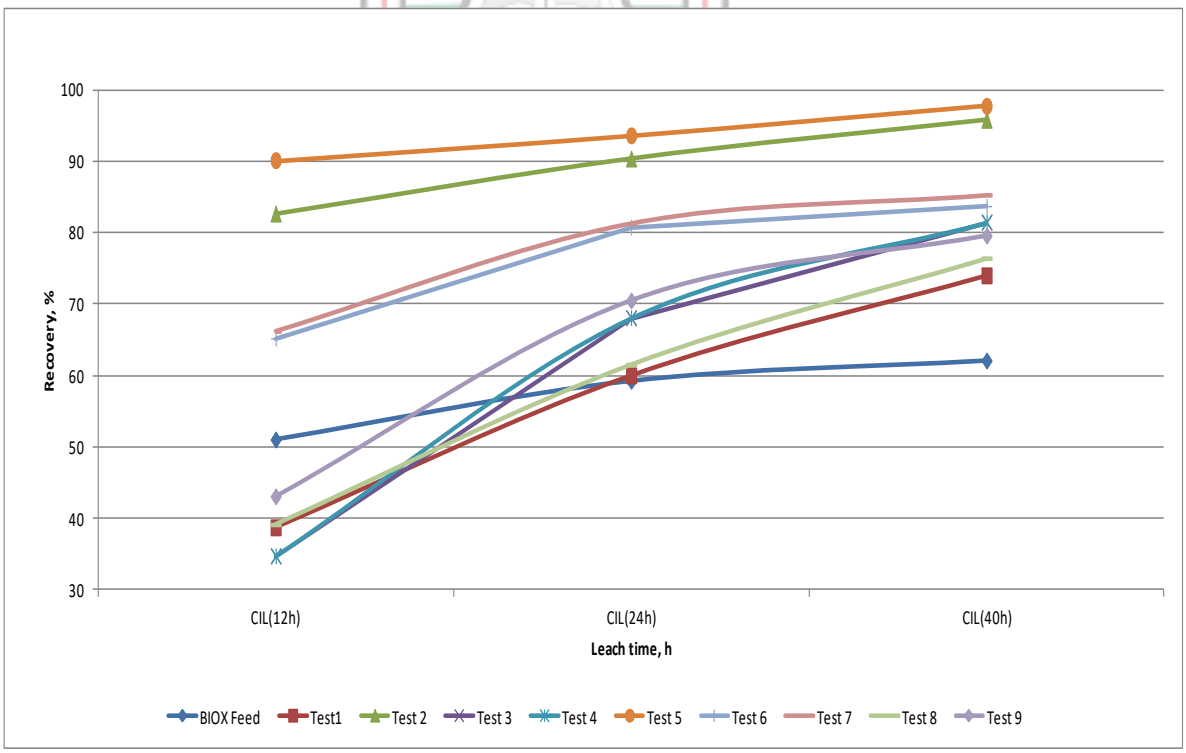

Figure 4.12 BIOX® Leach Test Results (1-9)

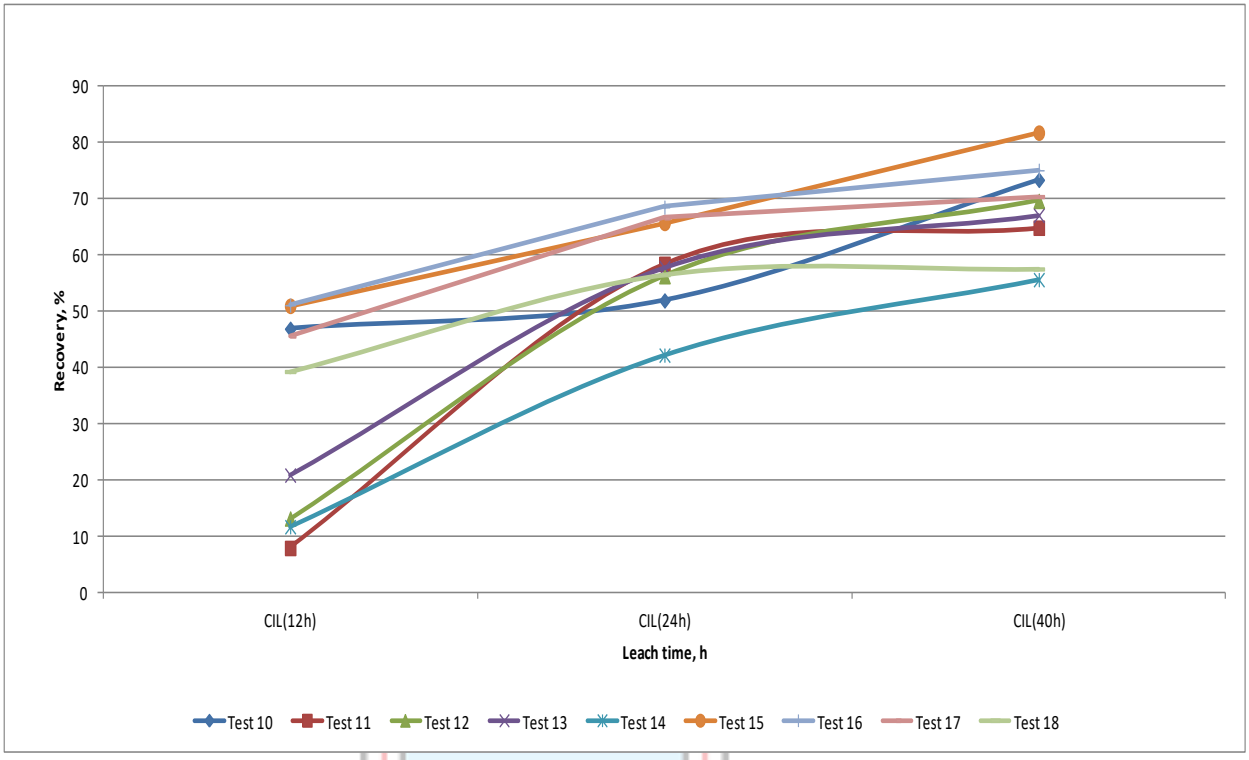

Figure 4.13 BIOX® Leach Test Results (10-18)

Comparatively, the results of the SOX and leach recovery is presented in Table 4.5. The "as is" BIOX feed leach recovery is about 62%. This is higher than the usual less than 50% achieved for most double refractory ores and can be attributed to partial oxidation due to the buffer stockpile kept at GSBPL and the storage of the BIOX feed sample under-laboratory condition for about a year before the set-up of this testwork. A recovery of 95.72% was achieved after biooxidation of the ore at the Genmin standard of 1.7:0.3:0.9 N:P:K. This recovery is about 3% higher than the Bogoso target leach recovery established in the MacIntyre, (2005) Feasibility Report. After stepwise reductions in NPK for biooxidation, a combination of 0.58:0.30:0.90 N:P:K led to an optimum sulphide oxidation of 91.83%, which resulted in the best cyanidation recovery of 97.68% higher than the recovery obtained after biooxidation at the Genmin standard. This record leach recovery can be attributed to starvation of the bacteria leading to higher SOX and subsequent higher leach recovery explained under the stepwise reduction of N above.

Table 4.4 Preg-robbing index (PRI)

Sample	Head Assays			Residue, g/t		Recovery, %		PRI
ID	Au, g/t	S2-, %	Org C, %	CN (40h)	CIL (40h)	CN (40h)	CIL (40h)	%
BIOX Feed	39.47	13.83	2.91	24.41	14.99	38.15	62.01	*38.43*
Test1	41.37	1.78	3.96	28.32	10.78	31.54	73.94	*57.33*
Test 2	55.94	0.67	3.74	28.99	2.40	48.19	95.72	*49.66*
Test 3	46.03	2.32	3.53	29.32	16.82	36.30	63.46	*41.71*
Test 4	47.66	2.16	3.60	30.45	8.87	36.12	81.40	*55.61*
Test 5	49.28	1.13	3.23	15.58	1.15	68.38	97.68	*29.99*
Test 6	54.14	0.77	4.05	33.63	8.86	37.89	83.64	*54.69*
Test 7	51.38	1.11	4.44	25.48	7.60	50.42	85.21	*40.83*
Test 8	48.21	1.15	3.55	33.22	11.41	31.10	76.34	*59.22*
Test 9	55.04	0.60	3.54	34.26	11.58	37.75	78.96	*52.18*
Test 10	50.48	1.21	3.59	39.70	13.45	21.35	73.36	*70.88*
Test 11	40.38	7.98	3.40	39.01	14.19	3.39	64.86	*94.76*
Test 12	43.67	6.94	3.40	39.27	13.27	10.08	69.62	*85.53*
Test 13	44.93	6.99	3.52	37.77	14.80	15.95	67.06	*76.22*
Test 14	41.05	8.15	4.05	38.65	18.21	5.85	55.63	*89.47*
Test 15	48.99	2.92	3.94	26.14	8.94	46.64	81.76	*42.91*
Test 16	47.66	3.45	3.50	27.76	11.88	41.75	75.06	*44.28*
Test 17	41.58	5.38	3.64	32.05	12.31	22.93	70.39	*67.42*
Test 18	40.89	7.68	3.45	32.79	17.37	19.81	57.53	*65.55*
AVERAGE PRI								58.77

Table 4.5 BIOX® and Leach Tests Summary

Sample ID	NUTRIENT ADDITIONS (Kg/t)			Sulphur Oxidation	Leach Recovery
	N	P	K	%	%
BIOX Feed	0.00	0.00	0.00	0.00	62.01
Test 1	0.00	0.00	0.00	46.421	73.942
Test 2	1.700	0.300	0.900	95.155	95.719
Test 3	1.190	0.300	0.900	83.225	63.459
Test 4	0.833	0.300	0.900	84.382	81.399
Test 5	0.580	0.300	0.900	91.829	97.677
Test 6	0.406	0.300	0.900	94.432	83.635
Test 7	1.700	0.210	0.900	91.974	85.213
Test 8	1.700	0.147	0.900	91.685	76.343
Test 9	1.700	0.103	0.900	95.662	78.956
Test 10	1.700	0.072	0.900	91.251	73.361
Test 11	1.700	0.300	0.630	42.299	64.859
Test 12	1.700	0.300	0.441	49.819	69.624
Test 13	1.700	0.300	0.309	49.458	67.060
Test 14	1.700	0.300	0.216	41.070	55.633
Test 15	1.190	0.210	0.630	78.886	81.756
Test 16	0.833	0.147	0.441	75.054	75.065
Test 17	0.580	0.103	0.309	61.099	70.388
Test 18	0.406	0.072	0.216	44.469	57.526

CHAPTER 5

CONCLUSIONS AND RECOMMENDATIONS

5.1 Conclusions

This study was set up to determine the levels of bio-available nutrient (NPK) in the ore and to deduce the impact of reducing the Genmin NPK nutrient combination on BIOX® and CIL processes. The following conclusions can be drawn from the studies as:

i. The ore sample treated is double refractory (contains an average of 13.83% of sulphide sulphur and 2.91% of organic carbonaceous matter) with high preg-robbing index (PRI of 59%);

ii. About 0.362 kg/t of N, 0.702 kg/t P and 0.417 kg/t K were bio-available in the ore by the end of the 5 day residence time of the BIOX® stream. This represents 21%, 234% and 46% respectively of nitrogen, phosphorus and potassium as proposed by BIOMIN as the standard nutrient requirement for the BIOX® circuit. The excess level of phosphorus could react with the high concentration of iron to create iron phosphate which is easily washed at CCD because it is soluble;

iii. Sulphur oxidation was generally not affected even as the nitrogen addition reduced, further reduction in N resulted in starvation which led to higher SOX and subsequent higher CIL recovery;

iv. Sulphide sulphur oxidation was not affected when phosphorus addition was reduced by as much as 75% of the quantity proposed by Genmin for efficient bio-oxidation;

v. Phosphorus reduction led to lower CIL recovery. These could be attributed to slimes/precipitates such as iron hydroxide formed to coating the surfaces of the gold particle hence reducing the amenability to cyanidation;

vi. Potassium reduction led to lower SOX and CIL recovery;

vii. Sulphide sulphur oxidation dropped drastically with combined drop in concentration of NPK, which moves to confirm the fact that K is very essential in the activity of the bacteria;

viii. Even the high performance of the bacteria in low nitrogen and phosphorus conditions could not lead to high oxidations when potassium mass added was reduced;

ix. CIL recovery dropped steadily with stepwise reduction in NPK; and

x. The best combination of NPK that resulted in the highest cyanidation gold recovery of 97.68% was 0.58:0.30:0.90, and this was at a SOX of 91.83%.

5.2 Recommendations

Based on the research findings and conclusions, the following recommendations are made:

i. The mine should purchase the individual nutrients (Ammonium sulphate [$(NH_4)_2SO_4$], potassium sulphate [K_2SO_4] and phosphoric acid [H_3PO_4]) instead of the pre-mix nutrient. This will allow the 0.58:0.30:0.90 NPK optimisation to be successfully implemented on the BIOX Plant thereby reducing the cost of nutrient;

ii. Study to ascertain the levels of bio-available nutrient (NPK) of the other sulphide deposits of the Bogoso operations. This will help with future ore blends to the sulphide plant;

iii. Nutrient optimisation must be an on-going development on BIOX® operations since bio-available nutrient levels vary with ore sources; and

iv. Strict QA/QC must be done on nutrient product supplied to the mine to ensure value for money.

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APPENDICES

APPENDIX A

Table 1.1 Acid Leach of Concentrate Tests 1 -6

Test No.	Sample	Nutrient, mg/L			Nutrient, g/t of Concentrate		
	ID	N	P	K	N	P	K
Test 1	24 HRS	48	114	110	49	116	112
	72 HRS	75	214	317	77	219	324
	120 HRS	292	567	402	299	579	411
Test 2	24 HRS	180	332	161	184	340	165
	72 HRS	314	327	180	321	335	184
	120 HRS	292	567	304	299	579	311
Test 3	24 HRS	180	332	108	184	340	110
	72 HRS	314	327	173	321	335	177
	120 HRS	292	567	402	299	579	411
Test 4	24 HRS	180	332	139	184	340	142
	72 HRS	314	327	168	321	335	172
	120 HRS	292	567	286	299	579	292
Test 5	24 HRS	180	332	108	184	340	110
	72 HRS	314	327	170	321	335	174
	120 HRS	292	567	195	299	579	199
Test 6	24 HRS	180	332	110	184	340	112
	72 HRS	314	327	173	321	335	177
	120 HRS	292	567	402	299	579	411

BIOX® Monitor Sheet

		Test 1			Test 2			Test 3		
Module	S^{2-} Target	**No Nutrient Addition** S2-, %	As, %	Oxdn %	**N=1.700 k g/t, P=0.300 kg/t, K= 0.900 kg/t** S2-, %	As, %	Oxdn %	**N=1.190 k g/t, P=0.300 kg/t, K= 0.900 kg/t** S2-, %	As, %	Oxdn %
Feed		13.83			13.83			13.83		
Day 5		11.32		18.2	1.86		86.6	4.20		69.6
Day 6		9.99		27.8	1.34		90.3	2.83		79.5
Day 7		8.98		35.1	1.53		88.9	2.54		81.6
Day 8		7.85		43.2	1.21		91.3	2.76		80.0
Day 9		7.78		43.7	0.99		92.8	2.52		81.8
Day 10		7.41		46.4	0.67		95.2	2.32		83.2

BIOX® Monitor Sheet

		Test 4			Test 5			Test 6		
Module	S^{2-} Target	**N=0.833 kg/t, P=0.300 kg/t, K= 0.900 kg/t** S2-, %	As, %	Oxdn %	**N=0.580 kg/t, P=0.300 kg/t, K= 0.900 kg/t** S2-, %	As, %	Oxdn %	**N=0.406 kg/t, P=0.300 kg/t, K= 0.900 kg/t** S2-, %	As, %	Oxdn %
Feed		13.83			13.83			13.83		
Day 5		4.26		69.2	3.22		76.7	1.93		86.0
Day 6		3.87		72.0	2.99		78.4	1.54		88.9
Day 7		3.30		76.1	2.82		79.6	1.40		89.9
Day 8		3.02		78.2	2.68		80.7	0.85		93.9
Day 9		2.64		80.9	1.69		87.8	0.80		94.2
Day 10		2.16		84.4	1.13		91.8	0.77		94.4

BIOX® Monitor Sheet

		Test 7			Test 8			Test 9		
Module	S^{2-} Target	**N=1.700 kg/t, P=0.210 kg/t, K= 0.900 kg/t** S2-, %	As, %	Oxdn %	**N=1.700 kg/t, P=0.147 kg/t, K= 0.900 kg/t** S2-, %	As, %	Oxdn %	**N=1.700 kg/t, P=0.103 kg/t, K= 0.900 kg/t** S2-, %	As, %	Oxdn %
Feed		13.83			13.83			13.83		
Day 5		3.27		76.4	3.33		75.9	3.12		77.4
Day 6		2.57		81.4	3.42		75.3	2.04		85.2
Day 7		2.49		82.0	3.28		76.3	1.62		88.3
Day 8		1.44		89.6	1.39		89.9	1.30		90.6
Day 9		1.42		89.7	1.38		90.0	0.78		94.4
Day 10		1.11		92.0	1.15		91.7	0.60		95.7

BIOX® Monitor Sheet		Test 10			Test 11			Test 12		
Module	S^{2-} Target	**N=1.700 kg/t, P=0.072 kg/t, K= 0.900 kg/t** S2-, %	As, %	Oxdn %	**N=1.700 kg/t, P=0.300 kg/t, K= 0.630 kg/t** S2-, %	As, %	Oxdn %	**N=1.700 kg/t, P=0.300 kg/t, K= 0.441 kg/t** S2-, %	As, %	Oxdn %
Feed		13.83			13.83			13.83		
Day 5		3.36		75.7	9.93		28.2	10.64		23.1
Day 6		2.53		81.7	9.51		31.2	8.92		35.5
Day 7		1.92		86.1	8.81		36.3	8.34		39.7
Day 8		1.83		86.8	8.33		39.8	7.77		43.8
Day 9		1.43		89.7	8.04		41.9	7.55		45.4
Day 10		1.21		91.3	7.98		42.3	6.94		49.8

BIOX® Monitor Sheet		Test 13			Test 14			Test 15		
Module	S^{2-} Target	**N=1.700 kg/t, P=0.300 kg/t, K= 0.309 kg/t** S2-, %	As, %	Oxdn %	**N=1.700 kg/t, P=0.300 kg/t, K= 0.216 kg/t** S2-, %	As, %	Oxdn %	**N=1.190 kg/t, P=0.210 kg/t, K= 0.630 kg/t** S2-, %	As, %	Oxdn %
Feed		13.83			13.83			13.83		
Day 5		9.04		34.6	8.52		38.4	8.82		36.2
Day 6		8.91		35.6	8.47		38.8	8.53		38.3
Day 7		8.80		36.4	8.31		39.9	6.09		56.0
Day 8		7.57		45.3	8.27		40.2	4.84		65.0
Day 9		7.08		48.8	8.16		41.0	3.62		73.8
Day 10		6.99		49.5	8.15		41.1	2.92		78.9

BIOX® Monitor Sheet		Test 16			Test 17			Test 18		
Module	S^{2-} Target	**N=0.833 kg/t, P=0.147 kg/t, K= 0.441 kg/t** S2-, %	As, %	Oxdn %	**N=0.580 kg/t, P=0.103 kg/t, K= 0.309 kg/t** S2-, %	As, %	Oxdn %	**N=0.406 kg/t, P=0.072 kg/t, K= 0.216 kg/t** S2-, %	As, %	Oxdn %
Feed		13.83			13.83			13.83		
Day 5		6.65		51.9	7.76		43.9	11.40		17.6
Day 6		5.57		59.7	6.97		49.6	10.90		21.2
Day 7		5.12		63.0	6.24		54.9	9.25		33.1
Day 8		3.64		73.7	5.99		56.7	8.89		35.7
Day 9		3.61		73.9	5.77		58.3	7.76		43.9
Day 10		3.45		75.1	5.38		61.1	7.68		44.5

Cyanidation monitor sheet

	Sample	Head Assays			Grind	Residue, g/t				Recovery, %				NaCN, kg/t		Lime, kg/t		PRI
	ID	Au, g/t	S2-,%	Org C, %	80% pa	CN(40h	CIL(12h	CIL(24h	CIL(40h	CN (40h	CIL(12h	CIL(24h	CIL(40h	CN	CIL	CN	CIL	%
BIOX FEED	Test	39.47	13.83	2.91	91.30	24.41	20.60	15.26	14.34	38.15	47.81	61.34	63.67	16.41	17.33	4.00	4.00	40.08
	repeat	39.47	13.83	2.91	91.30	24.41	18.12	16.98	15.65	38.15	54.09	56.99	60.36	16.41	17.28	4.00	4.00	36.79
	BIOX Feed	**39.47**	**13.83**	**2.91**	**91.30**	**24.41**	**19.36**	**16.12**	**14.99**	**38.15**	**50.95**	**59.16**	**62.01**	**16.41**	**17.31**	**4.00**	**4.00**	**38.43**
NO NUTRIENT	Test	41.37	1.78	3.96	91.30	28.32	22.20	17.59	11.15	31.54	46.35	57.48	73.05	15.44	17.24	5.00	6.00	56.82
	repeat	41.37	1.78	3.96	91.30	28.32	28.56	15.56	10.41	31.54	30.96	62.39	74.84	15.44	17.33	5.00	6.00	57.85
	Test1	**41.37**	**1.78**	**3.96**	**91.30**	**28.32**	**25.38**	**16.58**	**10.78**	**31.54**	**38.66**	**59.93**	**73.94**	**15.44**	**17.29**	**5.00**	**6.00**	**57.33**
N=1.70 kg/t, **P**=0.30 kg/t, **K**=0.9 kg/t	Test	55.94	0.67	3.74	91.30	28.99	10.66	5.51	2.57	48.19	80.94	90.15	95.41	17.02	17.20	5.00	6.00	49.49
	repeat	55.94	0.67	3.74	91.30	28.99	8.82	5.34	2.22	48.19	84.23	90.45	96.03	17.02	17.20	5.00	6.00	49.82
	Test 2	**55.94**	**0.67**	**3.74**	**91.30**	**28.99**	**9.74**	**5.43**	**2.40**	**48.19**	**82.59**	**90.30**	**95.72**	**17.02**	**17.20**	**5.00**	**6.00**	**49.66**
N=1.19 kg/t, **P**=0.30 kg/t, **K**=0.9 kg/t	Test	46.03	2.32	3.53	91.30	29.32	30.54	29.32	20.80	36.30	33.65	36.30	54.81	16.67	17.33	3.00	4.00	33.77
	repeat	46.03	2.32	3.53	91.30	29.32	35.96	22.48	12.84	36.30	21.88	51.16	72.11	16.67	17.33	3.00	4.00	49.65
	Test 3	**46.03**	**2.32**	**3.53**	**91.30**	**29.32**	**33.25**	**25.90**	**16.82**	**36.30**	**27.76**	**43.73**	**63.46**	**16.67**	**17.33**	**3.00**	**4.00**	**41.71**
N=0.83 kg/t, **P**=0.30 kg/t, **K**=0.9 kg/t	Test	47.66	2.16	3.60	91.30	30.45	31.91	13.42	9.47	36.12	33.06	71.84	80.13	16.67	17.33	4.00	5.00	54.92
	repeat	47.66	2.16	3.60	91.30	30.45	30.38	17.08	8.26	36.12	36.27	64.16	82.67	16.67	17.26	4.00	5.00	56.31
	Test 4	**47.66**	**2.16**	**3.60**	**91.30**	**30.45**	**31.14**	**15.25**	**8.87**	**36.12**	**34.66**	**68.00**	**81.40**	**16.67**	**17.30**	**4.00**	**5.00**	**55.61**
N=0.58 kg/t, **P**=0.30 kg/t, **K**=0.9 kg/t	Test	49.28	1.13	3.23	91.30	15.58	4.96	2.50	1.10	68.38	89.94	94.93	97.77	16.76	17.28	4.00	4.00	30.05
	repeat	49.28	1.13	3.23	91.30	15.58	4.83	3.86	1.19	68.38	90.21	92.17	97.59	16.76	17.31	4.00	4.00	29.92
	Test 5	**49.28**	**1.13**	**3.23**	**91.30**	**15.58**	**4.89**	**3.18**	**1.15**	**68.38**	**90.07**	**93.55**	**97.68**	**16.76**	**17.30**	**4.00**	**4.00**	**29.99**

Cyanidation monitor sheet

														Consumptions				
	Sample	Head Assays			Grind	Residue, g/t				Recovery, %				NaCN, kg/t		Lime, kg/t		PRI
N=0.406 kg/t, P=0.30 kg/t, K=0.9 kg/t	Test	54.14	0.77	4.05	91.30	33.63	18.29	10.48	9.01	37.89	66.23	80.64	83.36	16.82	17.46	5.00	6.00	54.54
	repeat	54.14	0.77	4.05	91.30	33.63	19.48	10.42	8.71	37.89	64.02	80.76	83.91	16.82	17.42	5.00	6.00	54.84
	Test 6	**54.14**	**0.77**	**4.05**	**91.30**	**33.63**	**18.88**	**10.45**	**8.86**	**37.89**	**65.12**	**80.70**	**83.64**	**16.82**	**17.44**	**5.00**	**6.00**	**54.69**
N=1.70 kg/t, P=0.210 kg/t, K=0.9 kg/t	Test	51.38	1.11	4.44	91.30	25.48	17.15	9.59	7.84	50.42	66.63	81.34	84.75	17.22	17.44	5.00	6.00	40.51
	repeat	51.38	1.11	4.44	91.30	25.48	17.65	9.68	7.36	50.42	65.65	81.16	85.68	17.22	17.48	5.00	6.00	41.15
	Test 7	**51.38**	**1.11**	**4.44**	**91.30**	**25.48**	**17.40**	**9.63**	**7.60**	**50.42**	**66.14**	**81.25**	**85.21**	**17.22**	**17.46**	**5.00**	**6.00**	**40.83**
N=1.70 kg/t, P=0.147 kg/t, K=0.9 kg/t	Test	48.21	1.15	3.55	91.30	33.22	26.79	19.03	10.35	31.10	44.43	60.54	78.53	17.15	17.42	5.00	6.00	60.39
	repeat	48.21	1.15	3.55	91.30	33.22	31.97	18.14	12.46	31.10	33.69	62.37	74.15	17.15	17.39	5.00	6.00	58.06
	Test 8	**48.21**	**1.15**	**3.55**	**91.30**	**33.22**	**29.38**	**18.58**	**11.41**	**31.10**	**39.06**	**61.46**	**76.34**	**17.15**	**17.41**	**5.00**	**6.00**	**59.22**
N=1.70 kg/t, P=0.103 kg/t, K=0.9 kg/t	Test	55.04	0.60	3.54	91.30	34.26	31.58	17.88	11.88	37.75	42.63	67.51	78.42	17.18	17.44	5.00	5.80	51.85
	repeat	55.04	0.60	3.54	91.30	34.26	31.38	16.29	11.29	37.75	42.99	70.41	79.50	17.18	17.42	5.00	5.80	52.51
	Test 9	**55.04**	**0.60**	**3.54**	**91.30**	**34.26**	**31.48**	**17.08**	**11.58**	**37.75**	**42.81**	**68.96**	**78.96**	**17.18**	**17.43**	**5.00**	**5.80**	**52.18**
N=1.70 kg/t, P=0.072 kg/t, K=0.9 kg/t	Test	50.48	1.21	3.59	91.30	39.70	27.05	23.54	13.97	21.35	46.41	53.37	72.33	17.13	17.33	5.00	5.60	70.47
	repeat	50.48	1.21	3.59	91.30	39.70	26.57	25.00	12.93	21.35	47.37	50.48	74.40	17.13	17.37	5.00	5.60	71.30
	Test 10	**50.48**	**1.21**	**3.59**	**91.30**	**39.70**	**26.81**	**24.27**	**13.45**	**21.35**	**46.89**	**51.92**	**73.36**	**17.13**	**17.35**	**5.00**	**5.60**	**70.88**

														Consumptions				
	Sample	Head Assays			Grind	Residue, g/t				Recovery, %				NaCN, kg/t		Lime, kg/t		PRI
N=1.70 kg/t, P=0.30 kg/t, K=0.63 kg/t	Test	40.38	7.98	3.40	91.30	39.01	36.18	16.37	14.95	3.39	10.40	59.46	62.98	17.29	17.42	5.00	6.40	94.61
	repeat	40.38	7.98	3.40	91.30	39.01	38.18	17.20	13.43	3.39	5.45	57.40	66.74	17.29	17.44	5.00	6.40	94.92
	Test 11	**40.38**	**7.98**	**3.40**	**91.30**	**39.01**	**37.18**	**16.79**	**14.19**	**3.39**	**7.92**	**58.43**	**64.86**	**17.29**	**17.43**	**5.00**	**6.40**	**94.76**
N=1.70 kg/t, P=0.30 kg/t, K=0.441 kg/t	Test	43.67	6.94	3.40	91.30	39.27	38.19	18.55	13.61	10.08	12.55	57.52	68.83	17.22	17.42	5.00	7.00	85.36
	repeat	43.67	6.94	3.40	91.30	39.27	37.65	19.67	12.92	10.08	13.79	54.96	70.41	17.22	17.39	5.00	7.00	85.69
	Test 12	**43.67**	**6.94**	**3.40**	**91.30**	**39.27**	**37.92**	**19.11**	**13.27**	**10.08**	**13.17**	**56.24**	**69.62**	**17.22**	**17.41**	**5.00**	**7.00**	**85.53**
N=1.70 kg/t, P=0.30 kg/t, K=0.309 kg/t	Test	44.93	6.99	3.52	91.30	37.77	35.51	19.58	14.66	15.95	20.97	56.42	67.37	17.24	17.44	5.00	5.80	76.33
	repeat	44.93	6.99	3.52	91.30	37.77	35.59	18.37	14.94	15.95	20.79	59.11	66.75	17.24	17.39	5.00	5.80	76.11
	Test 13	**44.93**	**6.99**	**3.52**	**91.30**	**37.77**	**35.55**	**18.98**	**14.80**	**15.95**	**20.88**	**57.77**	**67.06**	**17.24**	**17.42**	**5.00**	**5.80**	**76.22**
N=1.70 kg/t, P=0.30 kg/t, K=0.216 kg/t	Test	41.05	8.15	4.05	91.30	38.65	36.12	23.35	17.27	5.85	12.01	43.13	57.94	17.33	17.42	5.00	7.20	89.91
	repeat	41.05	8.15	4.05	91.30	38.65	36.37	24.11	19.16	5.85	11.40	41.28	53.33	17.33	17.44	5.00	7.20	89.04
	Test 14	**41.05**	**8.15**	**4.05**	**91.30**	**38.65**	**36.25**	**23.73**	**18.21**	**5.85**	**11.71**	**42.20**	**55.63**	**17.33**	**17.43**	**5.00**	**7.20**	**89.47**
N=1.190 kg/t, P=0.210 kg/t, K=0.630 kg/t	Test	48.99	2.92	3.94	91.30	26.14	24.26	17.26	9.98	46.64	50.49	64.78	79.63	17.31	17.44	5.00	6.20	41.43
	repeat	48.99	2.92	3.94	91.30	26.14	23.79	16.34	7.90	46.64	51.45	66.65	83.88	17.31	17.48	5.00	6.20	44.40
	Test 15	**48.99**	**2.92**	**3.94**	**91.30**	**26.14**	**24.02**	**16.80**	**8.94**	**46.64**	**50.97**	**65.71**	**81.76**	**17.31**	**17.46**	**5.00**	**6.20**	**42.91**

Cyanidation monitor sheet

														Consumptions				
	Sample	Head Assays			Grind	Residue, g/t				Recovery, %				NaCN, kg/t		Lime, kg/t		PRI
N=0.833 kg/t, P=0.147 kg/t, K=0.441 kg/t	Test	47.66	3.45	3.50	91.30	27.76	23.17	15.67	13.34	41.75	51.40	67.12	72.00	16.82	17.24	8.00	9.20	42.01
	repeat	47.66	3.45	3.50	91.30	27.76	23.46	14.36	10.43	41.75	50.78	69.88	78.13	16.82	17.33	8.00	9.20	46.56
	Test 16	**47.66**	**3.45**	**3.50**	**91.30**	**27.76**	**23.31**	**15.01**	**11.88**	**41.75**	**51.09**	**68.50**	**75.06**	**16.82**	**17.29**	**8.00**	**9.20**	**44.28**
N=0.580 kg/t, P=0.103 kg/t, K=0.309 kg/t	Test	41.58	5.38	3.64	91.30	32.05	22.54	14.88	12.58	22.93	45.80	64.23	69.76	16.78	17.22	6.00	9.00	67.13
	repeat	41.58	5.38	3.64	91.30	32.05	22.70	12.90	12.05	22.93	45.41	68.98	71.02	16.78	17.28	6.00	9.00	67.71
	Test 17	**41.58**	**5.38**	**3.64**	**91.30**	**32.05**	**22.62**	**13.89**	**12.31**	**22.93**	**45.60**	**66.60**	**70.39**	**16.78**	**17.25**	**6.00**	**9.00**	**67.42**
N=0.406 kg/t, P=0.072 kg/t, K=0.216 kg/t	Test	40.89	7.68	3.45	91.30	32.79	24.02	17.04	16.87	19.81	41.26	58.33	58.76	16.76	17.20	6.00	9.00	66.29
	repeat	40.89	7.68	3.45	91.30	32.79	25.62	18.52	17.87	19.81	37.34	54.71	56.30	16.76	17.17	6.00	9.00	64.81
	Test 18	**40.89**	**7.68**	**3.45**	**91.30**	**32.79**	**24.82**	**17.78**	**17.37**	**19.81**	**39.30**	**56.52**	**57.53**	**16.76**	**17.19**	**6.00**	**9.00**	**65.55**

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