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ASSESSMENT OF LEACHING RESPONSE AND RECOVERY ENHANCEMENT FACTORS OF A GOLD REFRACTORY ORE

MSc (50/50) DISSERTATION

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Submitted to

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DECLARATION

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Summary

This research aimed to assess the leaching response and recovery enhancements on pyrite refractory gold ores from North Mara Gold Mine in Tanzania.

Gold is found in a variety of geological settings and can occur in both its native, pure state and as part of other minerals. It is most found in quartz veins, typically formed by hydrothermal fluids, and in placer deposits, which are concentrations of heavy minerals eroded from source rocks and transported by water.

Enhancing gold leaching involves optimizing various factors to increase gold dissolution rates and overall recovery. Key strategies include improving reagent efficiency, optimizing leaching conditions (like grind size, pH, cyanide level, temperature, and oxygen levels), and implementing pre-treatment methods to expose gold more effectively to the leach solution.

Based on the grinding establishment results, a four-point milling curve showed that the optimal milling time for P80 <75 µm and P80 <150 µm was 18.4 and 9 min, respectively. The gravity test work conducted via Knelson concentrator for the P80 <150 µm grind showed that the decrease in pulp density from 38% to 25% resulted in an increase in the gravity concentrate mass pull from 121.65g to 126.14g indicating that at lower pulp density the gold bearing particles migrate much easier to the concentrate slots.

Lower pulp density resulted in higher gold concentrate mass pull and higher concentrate grades in a Knelson concentrator. A grind of P80 <150 µm showed the best leaching results indicating that it is the optimum grind for effective liberation for the ore under investigation. An increase in the cyanide concentration from 0.023 - 0.025% to 0.030 - 0.035% resulted in a decrease in the gold recovery, indicating that higher leaching agent concentration does not necessarily increase the recovery considering the cyanide concentration range tested.

Increase in dissolved oxygen (DO) from the range of 10-15 ppm to 15-20 ppm resulted in an increase in gold recovery, however, further increase in DO beyond 15-20 ppm led to approximately 30% decrease in gold recovery efficiency.

Dedication

This research is lovingly dedicated to my parents, Pascal Mutombo and Justine MUSHIYA; my wife, Noella KAPINGA and my sons, Ray BADIBANGA; Atem BADIBANGA and Hisham BADIBANGA. They have given me the drive and discipline to tackle any task with enthusiasm and determination. Without their love and support this research would not have been made possible.

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TABLE OF CONTENTS

CHAPTER 1: INTRODUCTION 9
1.1 Background 9
1.2 Problem identification 11
1.3 Research objective 11
1.4 Research question 12
1.5 Research approach 12
CHAPTER 2: LITERATURE REVIEW 13
2.1 Introduction 13
2.1.1 Leaching 13
2.2.2 Comminution 13
2.2 Overview of Gold Refractory Ores 14
2.2.1Definition and Characteristics of Refractory Gold Ores 14
2.2.2Processing of Refractory Gold Ores 15
2.2.3 Challenges in Processing Refractory Gold Ores 16
2.3 Leaching Techniques 17
2.3.1 Overview of Leaching Methods 17
2.3.2 Thiosulfate Leaching of Refractory Ores 19
2.3.3 Pressure Oxidation as a Pretreatment Method 20
2.3.4 Roasting as a pretreatment method 20
2.3.5 Bio-oxidation as a pretreatment method 23
2.3.6 Pressure oxidation as a pretreatment method 25
2.3.7 Ultra fine grinding as pretreatment method 26
2.3.8 Other leaching methods 26
2.3.9 Effects of Mechanical Activation on Recovery Rates 27
2.4 Role of metallurgical test works 28
2.4.1 Bottle roll test works 28
2.5 Factors Affecting Leaching Efficiency 28
2.5.1 Particle Size 29
2.5.2 Dissolved Oxygen 29

2.5.3 pH 30
2.5.4 Residence Time 30
2.5.5 Cyanide Concentration 30
2.5.6 Slurry Density 31
2.5.7 Mineralogy 31
2.6 Alternative Leaching Reagents and Their Efficacy 33
2.6.1 Non-Cyanide Leaching Reagents 33
2.6.2 JinchanTM Gold Leaching Reagent 33
2.6.3 Glycine-Based Leaching System 33
2.7 Case Studies 34
CHAPTER 3: METHODOLOGY 35
3.1 Sampling and submission of sample to the laboratory 35
3.2 Sample preparation 35
3.3 Characterization 35
3.3.1 Head grade analysis 35
3.3.2 Multi element analysis 36
3.3.3 Mineralogy 36
3.3.4 Gold deportment 36
3.4 Grind establishment 37
3.5 Gravity test-work 38
3.6 CIL leaching test work 38
CHAPTER 4: RESULTS, DISCUSSION AND COST ANALYSIS 43
4.1 Samples 43
4.2 Sample preparation 44
4.3 Characterization of ore 45
4.3.1 Head grade analysis 45
4.3.2 Multi element analysis 46
4.3.3 Mineralogy 48
4.3.4 Gold deportment 50
4.4 Grind establishment 53
4.5 Gravity determination 55
4.6 CIL leaching amenability for the gravity tailings 56

4.6.1 Effect of Leaching time on gold head assay in an aerated bottle roll leach result..... 56
4.6.2 Effect of leaching time on gold recovery in CIL leaching..... 57
4.6.3 Effect of particle sizes on gold recovery in Direct Leaching..... 59
4.6.4 Effect of dissolved oxygen on gold recovery in CIL leaching 60
4.6.5 Effect of cyanide concentration on gold recovery in CIL leaching..... 61
CHAPTER 5: CONCLUSIONS AND RECOMMENDATIONS..... 64
REFERENCES 67

CHAPTER 1: INTRODUCTION

1.1 Background

Tanzania is among the richest African countries endowed with many valuable mineral deposits. The mining industry contributes significantly to the national economy. This is mainly through the extraction of precious metals (gold, silver and PGM), metallic minerals (copper, nickel, iron, etc.), gemstones (diamond, tanzanite, ruby, garnet etc.), fuel minerals (coal, uranium, petroleum) and industrial minerals (gypsum, phosphate, limestone etc.) (Mwaipopo, et al 2004).

The precious metal of interest in this current work is gold. The chemical element gold (Au) is classified as a noble metal due to its inertness to chemical reactions in non-complex media. Gold is found in a variety of geological settings and can occur in both its native, pure state and as part of other minerals. It is most commonly found in quartz veins, typically formed by hydrothermal fluids, and in placer deposits, which are concentrations of heavy minerals eroded from source rocks and transported by water. It does, however, react with numerous reagents, i.e., chlorine, bromine, fluorine and aqua regia which is a mixture of nitric and hydrochloric acid. It belongs to the same group as copper and silver in the periodic table and it is commonly found to be associated with these elements in rocks. Gold is also found in host minerals such as calaverite ($AuTe_2$), montbrayite (Au_2Te_3) and sylvanite ($AuAgTe_4$), in varying concentrations and occurs in association with minerals like sulfide. The average concentration of gold in the earth's crust is 0.005 g/t, which is much lower than most other metals, for example, 0.07 g of silver/t and 50 g of copper/t. The gold content in the ores is dependent on gold bearing minerals as well as gold properties. For instance, electrum, with a specific gravity of 16-19.3, is a mixture of silver and 45-75% gold. Gold extraction from its various sources is well established. The many possible methods to recover gold from ores include gravity concentration, leaching and flotation. Leaching by cyanide solutions or gold cyanidation has been the main metallurgical process for gold extraction for more than a century (Monhemius 1986).

Since the invention of the gold cyanidation process in 1887, its chemistry and leaching kinetics have been the subject of considerable investigation that has led to several theories being proposed to explain the reaction mechanism. Various variables affecting gold cyanidation, such as dissolved oxygen concentration, free cyanide concentration, temperature, pH and particle size, have been

studied. Their effects on the optimal gold conversion have been investigated since they could lead to improvements on the industrial scale as well as boosting the operational profits.

However, gold cyanidation is a complex process because gold particles occur as alloys or compounds which are embedded in mineral matrices and complex galvanic interactions take place between the phases. A number of these studies remain contradictory in their conclusions concerning the mechanism of dissolution. Some studies propose that the diffusion of reactants to the gold surface controls the rate of reaction, while others claim that the chemical reaction is slow and in control (Monhemius 1986).

Gold ores can be broadly categorized as "free milling" and "refractory" depending on their response to cyanide leaching. A gold recovery of >90% can be readily achieved with a conventional cyanide leaching of free milling ores. However, refractory gold ores are often characterized by the low gold extractions (<50-80%) in cyanide leaching. The refractoriness of gold ores and concentrates stems primarily from the inherent mineralogical features with reference to the mode of presence and association of gold and carbonaceous matter present. In practice, refractoriness has been reported to be caused by the following reasons:

1. Very fine dissemination (<10 μm) of gold particles or presence of gold particles as solid solution within mostly pyrite and arsenopyrite.

2. Association and presence of gold with tellurides and base metal sulfides of lead, copper and zinc.

3. Presence of carbonaceous matter and silicate minerals.

4. Ultrafine gold locked up in the gangue matrix (mostly quartz) or complexes with manganese oxides.

This research aims to study the leaching response and recovery enhancement of refractory gold ore from the North Mara gold mine in Tanzania. This aim will be achieved by gathering information from existing cyanide leaching processes and experimentally investigating the important factors which impact the reaction kinetics of the gold leaching process and potentially optimize the process for the ore under consideration.

1.2 Problem identification

The average concentration of gold in the earth's crust is 0.005 g/t, much lower than most other metals. It is usually found in host minerals such as calaverite ($AuTe_2$), montbrayite (Au_2Te_3) and sylvanite ($AuAgTe_4$) in varying concentrations and sometimes in association with sulfides and copper. Since the global demand for gold is huge, it becomes imperative that most of the gold in the ores has to be recovered. To achieve this, the extraction method of gold from ores must be chosen carefully. Leaching by cyanide solutions or gold cyanidation has been the main metallurgical process of choice for more than one century. However, refractory gold ores are often characterized by low gold extractions (<50-80%) in cyanide leaching. This low yield makes it critical that the leaching response characteristics and recovery enhancement of refractory gold ore by optimizing liberation and recovery parameters needs to be further studied to improve its yield.

1.3 Research objective

This research aims to assess the leaching response and recovery enhancements on pyrite refractory gold ores from North Mara Gold Mine which is a combined underground and open pit gold mine located in Tarime district of the Mara region in the northern part of Tanzania.

The specific objectives are:

1. To optimize the size reduction steps and run the comminution test works by checking the response to liberation.
2. To run the GRG (Gravity recoverable gold) tests at different liberation levels.
3. To run the leaching tests on the GRG residue and optimize the conditions.
4. To run the leaching tests on the ROM without GRG and optimize the conditions.
5. To perform an economic valuation of the process.

1.4 Research question

The questions that need to be answered in this research are:

1. What is the required grinding size for optimal leaching of refractory gold ores?
2. What are the optimal leaching parameters for extraction of refractory gold ores using cyanide?
3. Is the optimized process under investigation economically viable?

1.5 Research approach

To arrive at a meaningful answer to the research questions, the following methodology is necessary:

- Characterization of the ROM feed,
- Conducting grinding tests of the ROM Composite feed and produce grinds with P80:150, 125, 106, 75 and 53 microns.
- Conducting GRG of the ROM from the different liberation sizes with P80:150, 125, 106, 75 and 53 microns.
- CIL Leaching test works on GRG residue and from ROM composite, determines the effect of cyanide dosage, the effect of oxygenation reagent, the effect of particle size, and the effect of pH.
- Evaluation of the economics of the process.

CHAPTER 2: LITERATURE REVIEW

2.1 Introduction

This literature review aims to discuss the leaching response and the enhancement of recovery on the refractory gold ores. It will also review previously published research conducted on refractory gold ores to provide insight into this research. This comprehensive review is designed to delve into the intricacies of leaching kinetics and the augmentation of extraction efficiency of refractory gold ores. In addition, this segment will meticulously scrutinize and interpret previously published investigations on refractory gold ores, thereby enriching the knowledge base for this ongoing study. The focus will be on mineral processing aspects such as comminution, liberation, concentration, and extraction techniques, with an emphasis on understanding the complex nature of refractory gold ores and the challenges they present in mineral processing. This will provide a solid foundation for further research and innovation in this field.

2.1.1 Leaching

The dissolution process of the soluble ingredient(s) contained in a solid material by a solvent is the most popular and accepted definition of leaching. In its most common form, leaching carries out the task of dissolving the valued mineral(s) of an ore or concentrate, generally by an aqueous solution of the lixiviating agent. The term, leaching, may be extended to include the dissolution of secondary substances (scrap, residues and wastes).In an ideal situation, leaching gives rise to two fractions: one is the worthless material that is largely free from value minerals and is mostly destined for dumping, whilst the other is the metal-laden solution that is invariably advanced for further processing (Marsden & House 1992).

2.2.2 Comminution

Comminution is a process whereby particulate materials are reduced by blasting, crushing, and grinding which refers to the breakage of particles by abrasion and impact with unconnected media such as rods, balls and other particles in a tumbling mill, to the product sizes required for downstream processing or end use.One of the major objectives of comminution is the liberation or release of the valuable minerals from the associated gangue minerals at the coarsest possible

particle size. Grinding is usually performed "wet" to provide a slurry feed to the concentration process, although dry grinding exists but has limited applications. For a good leaching process, the particle size range should be from 75microns to 106 microns depending on the technique used. This increases the surface area and exposes the mineral grains to attack by the solvent (Wills 2005).

In mineral processing, comminution operations are used to ensure that valuable constituents are physically liberated from waste constituents before physical or chemical separations are attempted. The mineral grains may not, however, be liberated completely from the associated gangue material by this process. If direct contact between some portions of the mineral grain and the leachant is not inhibited or lost, the mineral leaching process proceeds in an uninterrupted manner (Gupta 2003).

2.2 Overview of Gold Refractory Ores

2.2.1Definition and Characteristics of Refractory Gold Ores

Refractory gold ores are those ores that do not give up their gold to conventional cyanide leaching. The gold in these ores is often physically locked in sulphide minerals such as pyrite, arsenopyrite, and stibnite. Sometimes, it is chemically bound within the crystal structure of these minerals or other compounds such as tellurides (Marsden & House, 1992). Refractory ores, as defined by Marsden and House (1992), are those ores that resist simple chemical treatment and require more complex methods for effective treatment. These ores can be categorized into sulphidic, carbonaceous, telluride, or a combination of these. Recently, roasting has been employed to fully oxidize the refractory portion of the ore, thereby rendering the contained ore leachable.

The characteristics of refractory gold ores are largely determined by the mineralogical composition of the ore. The presence of carbonaceous matter, which can adsorb gold from the leach solution, can also render an ore refractory. Other factors that contribute to an ore's refractoriness include the fineness of the gold particles. Gold particles that are less than 1 μm in size are particularly problematic. The presence of naturally occurring chemicals also interferes with the leaching process (Dunne, 2016). Consequently, refractory gold ore is naturally resistant to recovery by standard cyanidation and carbon adsorption processes. These ores require pre-treatment for

cyanidation to be effective. This resistance to conventional recovery methods underscores the complexity of refractory gold ores.

Sulfide minerals, which are common in these ores, often trap or occlude gold particles, making it challenging for the leach solution to complex with the gold. This occlusion hinders the extraction process, necessitating more complex and intensive treatment methods.

Organic carbon present in gold ore may also adsorb dissolved gold-cyanide complexes, like the way activated carbon does in solution purification. This so-called “preg-robbing” carbon is significantly finer than the carbon recovery screens typically used to recover activated carbon, resulting in reduced gold extraction.

Pyrite, also known as iron sulfide, plays a significant role in the process of gold leaching. Gold leaching refers to the extraction of gold from gold-bearing ores, an essential step in gold mining. Pyrite is commonly found in these ores, and its influence on gold leaching is of considerable interest. Pyrite has been found to catalyze the decomposition of thiosulfate, a common gold leaching agent. This catalytic action accelerates the decomposition rate of thiosulfate and increases its consumption. The presence of pyrite can slow down the gold-leaching kinetics and reduce the overall gold dissolution.

2.2.2Processing of Refractory Gold Ores

Many gold deposits contain finely disseminated gold particles in iron sulfide minerals such as pyrite. These minerals are called refractory gold ores because of the presence of gold, which occurs either as disseminated fine gold particles, typically <1 μm or as fine particles of gold encapsulated in the ore host. In certain ores, extraction rates corresponding to lower than 60% gold recovery occur when using direct cyanide leaching, even after grinding, hence indicating that the concentrate is refractory (MacMaster and kenny 1990).

These types of ores are much more difficult to treat by cyanidation because gold is locked inside the sulphide mineral matrix hence cyanide cannot effectively dissolve the gold particles. This results in a low gold recovery of approximately 50%-90%. The main sources of the refractory ores are sub-microscopic gold particles, tellurides (gold ore with a high level of Au-Ag) and very fine inclusion of native gold in sulphides, where the size of a particle is less than 10μm. In such cases,

gold is difficult to liberate, hence the cost of energy required to grind and liberate is relatively high. To recover gold from these ores, the ores should be pretreated prior to dissolution. For example, if gold ore contains pyrite, the pyrite should be oxidized before leaching. It has been reported that pre-oxidation of pyrite containing gold ore increases gold recovery from 20% to 70%, while cyanide consumption decreases from 2.5 kg/t ore to 1.5 kg/t ore (Monhemius 1986).

This highlights the importance of understanding the mineralogical characteristics of the ore and selecting the appropriate processing methods to efficiently extract gold from refractory ores. It also underscores the need for further research and technological advancements to improve the efficiency and sustainability of gold extraction processes.

Treatment of refractory gold ores requires additional processing to liberate the gold from the host mineral. This can involve physical processes such as ultrafine grinding, pressure oxidation, and roasting, or biological processes such as bacterial oxidation. Most of these processes can be energy-intensive and costly, adding to the challenges of processing refractory gold ores (Fleming, 2006).

2.2.3 Challenges in Processing Refractory Gold Ores

The processing of refractory gold ores presents a significant challenge due to the nature of the ores. The gold in refractory ores is often encapsulated in sulphide minerals such as pyrite and arsenopyrite or is present as fine particles or submicroscopic gold within the ore matrix, making it inaccessible to conventional leaching techniques (Marsden & House, 1992).

One of the primary challenges in processing refractory gold ores is the need for pre-treatment before leaching. This often involves roasting, pressure oxidation, or bacterial oxidation to oxidize the sulphide minerals and liberate the gold for subsequent leaching. These pre-treatment processes can be complex, energy-intensive, and costly, adding to the overall processing costs (Dunne, 2016). Another challenge is the potential environmental impact of certain chemicals in the leaching process, such as cyanide, which can pose environmental risks if not managed properly. Additionally, the oxidation of sulphide minerals during pre-treatment can generate acid, which can lead to acid mine drainage if not properly controlled (Fleming, 2006).

The variability of refractory gold ores also poses a challenge. The mineralogical characteristics of the ore, including the type and distribution of sulphide minerals and the size and distribution of gold particles, can vary widely. This variability can affect the efficiency of both pre-treatment and leaching processes, making it difficult to design and optimize the processing plant (Adams, 2005). Despite these challenges, the processing of refractory gold ores is a crucial aspect of the gold mining industry. With the decline in the availability of free-milling gold ores, more and more companies are having to deal with refractory gold ores. As such, research into more efficient and cost-effective methods of processing these ores is an active area of research (Yarar, 1991; Deschênes, 2016).

2.3 Leaching Techniques

2.3.1 Overview of Leaching Methods

Leaching, a fundamental process in hydrometallurgy, involves the extraction of valuable metals from ores through the application of a leaching solution. The choice of a leaching method is contingent on the nature of the ore and the specific metal to be extracted. Heap leaching, for instance, is a low-cost method suitable for low-grade ores. The ore is stacked on a liner, and the leaching solution percolates through the heap to leach the metal. In contrast, tank leaching, also known as agitation leaching, involves the ore being finely ground and leached in a tank. This method is used for higher grade ores. Vat leaching, where the ore is mixed with the leaching solution in large vats, is used for ores that are amenable to this treatment. In-situ leaching, where the leaching solution is injected into an ore body and the resultant solution is pumped to the surface, is used for mineral deposits that are not economically recoverable by other methods. Each method has its advantages and challenges, and the choice of method depends on a variety of factors including the nature of the ore, the type of mineral, the local environmental regulations, and economic considerations (Marsden & House, 2006).

Various leaching reagents for the leaching of gold have been investigated which include sodium cyanide, ammonia, glycine, thiourea and thiosulphate. For refractory ores, a pretreatment process which may be preceded by concentration (e.g. sulfide flotation) should be carried out in order to ensure that gold is liberated from the ores.

Pretreatment of ore

Pretreatment of refractory ores is necessary to liberate gold from the ores and/or to transform the host matrix into a leachable metal oxide phase. Usually, pretreatment processes may be preceded by concentration methods such as sulfide flotation. This step is crucial in preparing the ore for subsequent treatment and ensuring the effectiveness of the extraction process. Common pretreatment options for refractory ore include:

- Roasting.
- Bio-oxidation.
- Pressure oxidation.
- Ultra fine grinding.

Each of these methods has its advantages and challenges, and the choice of method depends on the specific characteristics of the ore and the requirements of the extraction process. Before gold ore is leached, the ore must first be crushed and ground into small particles, usually particle size between 30 and 180 μm. Free milling ores would be ready for leaching after crushing and grinding whereas refractory ores need pretreatment. It is important to pretreat a refractory ore before leaching to achieve better gold extraction and improve the performance of cyanidation. Pretreatment methods are for example chemical blanking, roasting, chlorination, bio-oxidation, pressure oxidation, chemical oxidation and ultrafine grinding. Many minerals in refractory gold ores have a detrimental effect on gold leaching. These minerals may dissolve in alkaline cyanide and consume the reagent and oxygen. Certain sulfide minerals are characteristically associated with gold i.e. pyrite, chalcopyrite, pyrrhotite, galena and arsenopyrite and applying a pretreatment prior to cyanidation can significantly reduce the dissolution of sulfide minerals. Pre-aeration or pre-leaching is used for ores containing reactive sulfide minerals such as pyrrhotite, marcasite and high concentrations of pyrite (Tesh 2020).

If gold ore contains pyrite, the pyrite should be oxidized before leaching. It is reported that pre-oxidation of pyrite containing gold ore increased gold recovery from 20% to 70% (Tesh 2020) It has been reported in other studies that the best way to reduce the dissolution of sulphide minerals

during cyanidation depends on the type of sulphide minerals present in the respective ore. In the case of iron sulphide minerals, sufficient amount of lime and oxygen concentration are used and in the case of arsenic and antimony sulphides, the use of lead nitrate under aerated condition is proposed (Figure 1). (Marsden & House 1992).

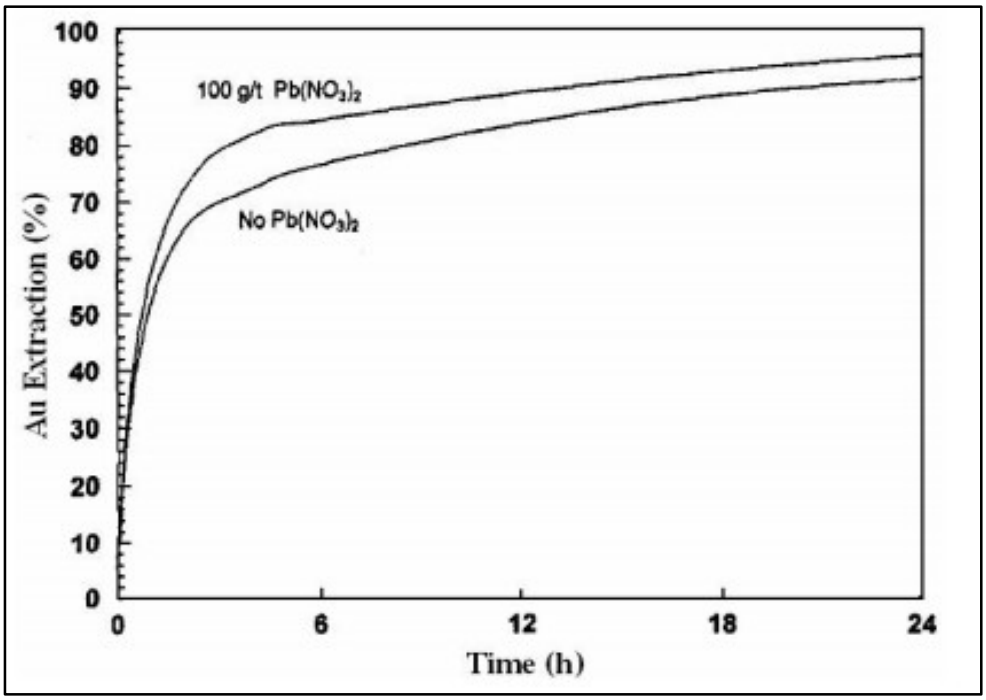


Figure 1: Effect of lead nitrate on gold recovery

2.3.2 Thiosulfate Leaching of Refractory Ores

Thiosulfate leaching is a promising alternative to cyanide leaching due to its lower toxicity and greater rate of gold and silver dissolution (Mhandu et al., 2023). It is particularly useful for treating refractory gold ores, which are ores that do not readily give up their gold to traditional cyanide leaching. The gold in these ores is often encapsulated in sulphide minerals such as pyrite and arsenopyrite or is present as fine particles or submicroscopic gold within the ore matrix, making it inaccessible to conventional leaching techniques. The use of thiosulfate for the extraction of gold from refractory ores is promising because of its non-toxicity and high selectivity. However, sulfide minerals, which are major gold carriers in refractory gold ores, can hinder gold extraction due to the high consumption of the lixiviant (Mhandu et al., 2023). This presents a significant challenge in the processing of refractory gold ores. To overcome this challenge, a pretreatment step is often necessary before leaching. This can involve roasting or pressure oxidation to oxidize the sulphide

minerals and liberate the gold for subsequent leaching. In a study by Mhandu et al. (2023), it was found that pretreating the ore with ammonium solutions containing cupric ions could oxidize the sulfide minerals and minimize thiosulfate consumption in the subsequent gold leaching step, resulting in an improvement in gold extraction from 10% to 79%.

However, thiosulfate leaching is not without its challenges. One of the main issues is the instability of the thiosulfate. It can decompose under certain conditions, leading to lower gold recovery. Additionally, the process can be affected by the presence of certain impurities in the ore, such as copper and nickel, which can consume thiosulfate and lower gold recovery (Mhandu et al., 2023).

Despite these challenges, thiosulfate leaching remains a promising method for the extraction of gold from refractory ores. With ongoing research and technological advancements, it is hoped that the efficiency and effectiveness of thiosulfate leaching can be improved further, making it a viable alternative to cyanide leaching for refractory gold ores (Mhandu et al., 2023).

2.3.3 Pressure Oxidation as a Pretreatment Method

According to Marsden and House (1992), refractory ores are those ores that could not be effectively treated by simple chemical means and can be classified as sulphidic, carbonaceous, telluride or a combination of these. Recently, they have been roasted to completely oxidize the refractory portion of the ore and render the contained ore leachable (Wikipedia, 2009). This suggests that sulfide minerals often trap or occlude gold particles, making it difficult for the leach solution to complex with gold. Organic carbon present in gold ore may also adsorb dissolved gold-cyanide complexes in much the same way as activated carbon does in solution purification. This so-called "preg-robbing" carbon is washed away because it is significantly finer than the carbon recovery screens typically used to recover activated carbon and thus can result in reduced gold extraction. Consequently, refractory gold ore is naturally resistant to recovery by standard cyanidation and carbon adsorption processes and these ores require pre-treatment for cyanidation to be effective in recovering the gold.

2.3.4 Roasting as a pretreatment method

Roasting is used to oxidize both the sulfur and organic carbon at high temperatures using air and/or oxygen. According to Kadenhe and Makande (1985), the objective of roasting is to expel sulphur,

arsenic, antimony and other volatile substances from the concentrates and to oxidize the remaining metals to transforms it to a metal oxide which is easily leachable.Roasting is a high temperature process in which the feed is exposed to temperatures and gas compositions that will bring about physical and chemical changes which will make the feed more suitable for further processing. It is a gas-solid reaction and is usually the first operation of pyrometallurgy used to make the subsequent operations more successful. A chemical roast can be classified as dead or sweet. In a dead roast, the reactions go to completion. For example, dead roasting of chalcopyrite concentrate ($CuFeS_2$) would contain no sulphur, just FeO and Cu_2O. For a sweet roast, also called a partial roast, the reactions are not completed. For example, a sweet roast of $CuFeS_2$ would give a mixture of CuS, FeS, Cu_2O and FeO. Roasting produces a chemical change in a material and often a physical change as well, so that the material will be more amenable to further processing and these processes begin at the particle surface or the gas-solid interface. The sulphide changes to an oxide on the surface first and the oxide layer slowly grows inwards onto the particle until it is completely oxidized. For the reaction to occur in the later stages, it must occur at the sulphide-oxide interface. For this to happen, oxygen must diffuse outwards through the oxide layer. As the oxide layer gets thicker, the diffusion path is longer and so the reaction slows down. If the oxide layer is particularly dense, it can prevent the passage of the gases or slow them down drastically. This so-called ‘‘Kernel Effect’’ gives high sulphur calcine where the centres of the particles remain unchanged. Methods of roasting include:

- Oxidizing roasting

Used to convert sulphides to oxides, for example:

Sphalerite (ZnS): $2ZnS + 3O_2 \rightarrow 2ZnO + 2SO_{2\uparrow}$

Chalcopyrite ($CuFeS_2$): $4CuFeS_2 + 11O_2 \rightarrow 2CuO + 4FeO + 8\ SO_{2\uparrow}$

- Sulphatising roasting:

This method is used to convert sulphides to sulphates, usually to process a calcine for leaching since many sulphates are water soluble. The sulphatising roast requires more oxygen. Each

sulphate has a temperature at which it vigorously decomposes. If the roaster is operated above this temperature, then sulphates cannot be formed. Below this temperature, sulphates will tend to be formed rather than oxides. The presence of high concentrations of SO_3 in the roaster will also promote the formation of sulphates. This is because the overall sulphating reaction is made up of two-part reactions:

- Oxidation of metal: $MS + 3/2O_2 \rightarrow MO + SO_2\uparrow$

  Oxidation of SO_2: $SO_2 + 1/2O_2 \rightarrow SO_3\uparrow$

- Sulphation: $MO + SO_3 \rightarrow MSO_4$

  Adding: $MS + 2O_2 \rightarrow MSO_4$

» High O_2 and SO_2 concentrations will help the reactions move forward.

- Chloridising roasting

To produce metal chlorides for leaching or other processes, the metal sulphide can be roasted with a chloride salt like NaCl, for example, copper sulphide is volatile and can be removed from iron concentrates by chloridising roast.

$$CuS + 2NaCl + 2O_2 \rightarrow CuCl_2\uparrow + Na_2SO_4$$

- Chlorinating roasting

This method employs chlorine to achieve the same results as the chloridising roast and produces a leachable product $CuCl_2$.

$$2CuO + 2Cl_2 \rightarrow 2CuCl_2\uparrow + O_2\uparrow$$

- Reducing roasting

This is when reducing gases such as CO and H_2 are used for roasting. For example, to produce magnetite from haematite in:

Inco`s iron ore recovery plant: $3FeO_3 + CO \rightarrow 2Fe_3O_4 + CO_2\uparrow$

The hydrogen reduction of iron: $Fe_2O_3 + 3H_2 \rightarrow 2Fe + 3H_2O\uparrow$

- Arsenic elimination roasting

The oxides of arsenic and antimony are volatile and can be eliminated from concentrates by an oxidizing roast. The impurities may be in the form of metal arsenides or antimonides.

$$4MAS + 5O_2 \rightarrow 4MO + 2AsO_3\uparrow$$

$$4MAS + 5O_2 \rightarrow 4MO + 2Sb_2O_3\uparrow$$

» This is often a useful side reaction in a regular oxidizing roasting.

Although Roasting is effective in increasing porosity, it may still leave calcined material refractory due to operational fluctuations in roast temperatures, oxygen flow rates and levels of elements such as antimony and lead in the feed material. Also, gold recoveries after roasting of ore are around 70- 80% after cyanidation which are rather low considering the cost incurred in reagents. Lastly the major constraint to the process is the safe disposal and handling of environmentally unfriendly gaseous emissions such as SO_2 ,AsO_3 and Sb_2O_3.

2.3.5 Bio-oxidation as a pretreatment method

Bio-oxidation generally involves the use of thiobacillus (thio and ferro-oxidans) bacteria which thrive by oxidizing ferrous iron found in pyrite, arsenopyrite, etc. to ferric ion, which, in turn oxidizes the sulphides components of the mineral, thereby releasing the encapsulated gold. Bacteria help in the oxidation and removal of elemental sulphur which builds up on the sulphide surface if not removed, hindering further oxidation of the mineral by occluding gold values and increasing cyanide consumptions in leaching.

Bio-oxidation is a process that leverages the metabolic activities of specific bacteria, notably Thiobacillus ferrooxidans and Thiobacillus thiooxidans, to facilitate the extraction of gold from refractory ores. These bacteria thrive in environments where they can oxidize ferrous iron, a

component commonly found in minerals such as pyrite and arsenopyrite. The oxidation process transforms the ferrous iron into ferric ions. These ferric ions, in turn, oxidize the sulphide components of the mineral matrix. This oxidation process is crucial as it leads to the release of the encapsulated gold particles, making them accessible for further processing.

The role of these bacteria extends beyond the initial oxidation of ferrous iron. They also aid in the oxidation and subsequent removal of elemental sulphur. Elemental sulphur can accumulate on the surface of the sulphide minerals if not removed. This build-up forms a layer that occludes the gold values, thereby hindering further oxidation of the mineral. Moreover, the presence of this sulphur layer can lead to an increase in cyanide consumption during the leaching process, which is not economically desirable. Therefore, the bacteria-assisted removal of elemental sulphur is a critical step in enhancing the efficiency of gold extraction from refractory ores.

For chalcopyrite and arsenopyrite oxidation, the general equations relying on bacterial catalysis are:

1. $CuFeS_2 + 17O_2 + 2H_2SO_4 = 4CuSO4 + 2Fe_2 (SO_4)_3 + 2H_2O$

2. $2FeAsS + 7O_2 + H_2SO_4 + 2H_20 = Fe_2(SO_4)_3 + 2H_3AsO_4$

The operation is carried out in acidic conditions in less aerated tanks at 35-45°C. The reaction is exothermic, and the system must be cooled to maintain an acceptable temperature for the bacteria to exist (Marsden and House, 1992).

Wall and Pattinsone (1997) have noted that the efficacy of bacterial activity can diminish rapidly and may even cease entirely under adverse conditions. In such scenarios, the use of thermophilic bacteria, which can operate at elevated temperatures exceeding 45°C, has been considered. This is particularly relevant in instances where the quality of water is not optimal. Under these conditions, gold recoveries of up to 90% have been reported in leaching processes. However, to accommodate specific conditions, a mixed culture approach has been suggested. Johnston (1994) recommended the use of Thiobacillus ferrooxidans as the primary organism, supplemented with heptospirillum ferrooxidans, Thiobacillus thiooxidans, and sulfolobus. This combination aims to enhance the bio-oxidation process under a variety of conditions.

Bio-oxidation presents a highly economically viable method for treating low-grade ores. Post-cyanidation recoveries can reach up to 90%, depending on the presence of other toxic elements such as arsenic in the ore. The technology is straightforward, the capital costs are low, and the process is environmentally friendly, making it an attractive option. However, it is important to note that bio-oxidation is a slow process. The operation is sensitive to conditions such as pH and temperature. Additionally, the costs associated with bacterial culturing and the supply of nutrients can make the process less appealing. Despite these challenges, bio-oxidation remains a critical component in the treatment of refractory gold ores.

2.3.6 Pressure oxidation as a pretreatment method

Pressure oxidation is an aqueous process for oxidizing and/or passivating some of the more reactive and reagent-consuming sulphides such as pyrrhotite and marcasite. It is carried out in a continuous autoclave operating at high pressures and somewhat elevated temperatures in acidic media. The principal oxidants are oxygen and iron III (Fe^{3+}) species which are playing an important role in oxidation reactions. With sulphide ores containing slightly greater than 4% sulphide sulphur, the reactions are autogenous as the sulphur is a principal heat source. Reactions involved are:

Chalcopyrite: $4CuFeS2+ 3O_2 + 12H^+ = 4Cu^+ + 4Fe^{2+} +8S + 6H_20$

Iron oxidation from iron II to iron III also occurs under these conditions which are:

$$4Fe^{2+} + O_2 + 4H^+ = 4Fe^{3+} +2H_2O$$

And the iron III so formed can also take part in oxidation of the ore:

$$4CuFeS_2 + 10Fe^{3+} + 4H_2O = 11Fe^{2+} + Cu^{2+} + 8H^+ + S + SO_4^{2-}$$

Usually, refractory minerals are decomposed in just a few hours, exposing the gold in a residue that is washed, conditioned with lime and leached in cyanide. Most base metal sulphides are also soluble in severely oxidizing condition media and remain in solution following oxidation. This permits them to be partially removed from the solid phase during the solid/liquid separation before cyanidation. Arsenic (As) and antimony (Sb) are precipitated from the solution if there is sufficient iron present.

According to Chifamba (1994), pressure oxidation takes just a few hours to complete giving downstream recoveries as high as 95-98% and the resultant solutions are much easier to handle and dispose of than the volatile gases from roasting. However, the high cost of autoclaves and pressure handling systems needed to carry out pressure oxidation can result in high operating costs.

2.3.7 Ultra fine grinding as pretreatment method

Ultra-fine grinding may be used when liberation of gold particles from the surrounding mineral matrix is the primary refractory characteristic of the ore. This method makes the ore amenable to gold extraction by cyanide solution. Ores vary greatly regarding the fineness of grinding required to release their gold for leaching or for amalgamation with mercury. Thus, when the ore is intimately associated with pyretic minerals, ultra fine grinding is required to expose it for recovery process. Sometimes, an ore must be ground to minus 200mesh in size before an economic gold recovery can be affected.

Very fine particles will result in a decrease in the rate of dissolution of gold from ores containing cyanicides as there is an increase in the rate of competition for reagents during leaching. In such cases, the optimum particle size is a compromise between gold extraction and cyanide consumption. Besides the factors above, grinding is a major energy consumer and thus increases operating costs as only a small fraction of the energy is used in the actual size reduction. Also, fines are difficult to handle during transportation and take long to settle during solution purification.

All in all, most pre-treating methods have their advantages and disadvantages, but a more favourable method should be economically viable, environmentally friendly and easy to apply and manage.

2.3.8 Other leaching methods

Pyrite and the Thiourea Leaching System

In the thiourea leaching system, pyrite's effect on gold recovery has been confirmed through a series of experiments. The decomposition efficiency of thiourea decreases by 40%, and the recovery efficiency of gold increases by 56% after the removal of sulfide by roasting. This suggests

that the presence of pyrite in the leaching system can have adverse impact on the efficiency of gold recovery.

Ammonium Alcohol Polyvinyl Phosphate (AAPP) and Pyrite

The role of ammonium alcohol polyvinyl phosphate (AAPP) during gold leaching in ammoniacal thiosulfate solutions has also been investigated. AAPP has been found to alleviate the negative effects of pyrite by adsorbing on the surface of the pyrite and blocking the contact between the pyrite and thiosulfate anions. This suggests that AAPP can effectively stabilize the thiosulfate in the solution and facilitate the gold leaching in the presence of pyrite.

In conclusion, the role of pyrite in gold leaching is complex and multifaceted. While it can negatively impact the leaching process by catalyzing the decomposition of leaching agents and reducing gold dissolution, these effects can be mitigated using additives like AAPP.

2.3.9 Effects of Mechanical Activation on Recovery Rates

Mechanical activation is a process that involves the use of mechanical energy to alter the physical and chemical properties of material. This process has been found to significantly influence the recovery rates of various metals, including gold, copper, and uranium. The intensification of leaching processes using the activation phenomenon makes it possible to increase the recovery of the metals. The mechanical activation process can enhance the extraction efficiency of these metals to certain degrees. This is primarily due to the mechanical activation that can alter the surface properties of the materials, thereby improving their reactivity. Mechanical activation alters the microstructural properties of a particle via application of shear forces which subsequently induce potential energy into the crystal lattice of the particle. It's this potential energy, which when it becomes available, increases reactivity during a chemical reaction. The effects of mechanical activation time, roasting temperature, leaching temperature, solid-to-liquid ratio, particle size, and acid concentration on leaching efficiency were investigated. The results demonstrated that mechanical activation significantly decreased the optimum roasting temperature and increased the leaching efficiency.

Microstructure morphology and elemental analyses of the raw materials and leaching residue were also investigated using scanning electron microscopy and energy-dispersive X-ray spectroscopy.

These analyses provided insights into the changes in the material's structure and composition due to mechanical activation, further elucidating its effects on recovery rates.

In conclusion, mechanical activation plays a crucial role in enhancing the recovery rates of various metals. By altering the physical and chemical properties of materials, mechanical activation can improve their reactivity and extraction efficiency. However, the effects of mechanical activation can vary depending on several factors, including the type of material and the specific parameters of the activation process.

2.4 Role of metallurgical test works

Metallurgical testing is required to optimize feed material by determining the ore grade and the leachability for gold ores. Various investigations are made on the ore from its characterisation, grindability, gravity concentration and leaching test work.

2.4.1 Bottle roll test works

Bottle roll tests are usually conducted on reverse circulation agitated around 25-30 rpm, and most good roller drives have variable speed on them to allow a wide range of adjustment. Typical adsorption takes place depending upon the adsorption tank and normally the time ranges from 10 minutes to 4hours. The bottle roll tests are conducted by pulping the coarse ore with water to achieve 40 to 50 weight percent solids (The pulp density greatly influences the efficiency of the mass transfer in the leaching and adsorption processes). It is also considered that the CN-Au reaction requires oxygen to be present, so if initial recoveries are poor, a larger 1-liter bottle is used as it has a larger volume of air and hence more oxygen. Another common method for increasing amount of oxygen during adsorption is to increase the dosage of Hydrogen Peroxide (H_2O_2) within a solution (Shrithammavut 2008).

2.5 Factors Affecting Leaching Efficiency

Enhancing gold leaching involves optimizing various factors to increase gold dissolution rates and overall recovery. Key strategies include improving reagent efficiency, optimizing leaching conditions (like pH, temperature, and oxygen levels), and implementing pre-treatment methods to expose gold more effectively to the leach solution

2.5.1 Particle Size

Coarse particles lead to poor leaching since most of the gold locked in the gangue is not exposed to the action of cyanide. In contrast, finer particles lead to better leaching due to the exposure of gold particles to the action of cyanide. There is a presence of materials that affect the leaching process within the gold-bearing ore matrix. Gold is associated with other minerals which react with cyanide (cyanocides) such as copper which consumes cyanide, arsenic which consumes oxygen, oils and greases which hinder the adsorption of gold onto carbon by blocking the pores (Kendos et al. 1995).

2.5.2 Dissolved Oxygen

Dissolved oxygen is crucial for gold leaching as it acts as the oxidant in the cyanide leaching process, allowing gold to dissolve into a solution. The optimal dissolved oxygen concentration typically ranges between 8 and 20 ppm. Insufficient oxygen slows down the reaction, while excessive oxygen can lead to cyanide oxidation, reducing efficiency. The higher the amount of dissolved oxygen, the higher the leaching kinetics and vice versa (Brison 2002).

Lead nitrate

Lead nitrate as an oxidant is often added to the slurry to alleviate the negative effect of sulphides present, enhancing gold recovery and lower cyanide consumption. It has been suggested that in a cyanide solution, lead reacts with gold to form $AuPb_2$, $AuPb_3$ and metallic lead. This accelerates the gold dissolution. Lead nitrate can activate the surface of a passivated gold particle, prevent a passivation film forming on the gold surface and precipitate the soluble sulfides. It has been demonstrated that the effect of lead ions in the form of lead nitrate, lead sulfide and lead sulfite, which reacts with gold accelerates gold dissolution and that the sulfide minerals, which are distributed in the solution get precipitated by excess lead as PbS. Lead ions may also react with gold sulfide film which makes the passive gold particles become active again in the absence of sulfide formation. (Deschênes et al 2000).

Hydrogen peroxide

Research and industrial applications over the years have shown that the dissolution rate of gold is directly related to the concentration of dissolved oxygen. Oxygen from air has previously been used, however, it has been found that this was not sufficient. Therefore, considerable effort has been put into the improvement of the important oxidant supply, mainly by using advanced techniques for gas dispersion or substituting compressed air with pure oxygen. The idea of the controlled addition of an effective liquid oxidant could overcome this problem, thus the use of hydrogen peroxide has been introduced. Different mineralogical compositions need different amounts of oxygen, for example, pyrite-rich ore has a higher oxygen demand than ore containing albite, quartz, and chlorite as the main minerals. The application of liquid (H_2O_2) ensures the fast and homogenous distribution of the active oxygen in the pulp. Furthermore, the oxidant dosage can be controlled automatically according to the oxygen level in the pulp, resulting in very economical oxidant consumption. (Deschênes et al 2000).

2.5.3 pH

pH (more than 12) leads to a slurry being more alkaline, more viscous, and forms scales on gold particles, which affects CN ions by lessening its chance to make contact with gold particles while too low a pH (less than9.5) allows the formation and discharge of poisonous HCN gas from the slurry to the atmosphere. This reduces the cyanide available for reaction with gold particles. So, the optimal pH for gold leaching is 10.5-11.5 (Habashi 1999).

2.5.4 Residence Time

Residence time is the time the slurry is in contact with the cyanide in the leach tanks. Residence time will vary depending on the size of the tanks and the flow rate to the tank (Nanthakumar 2007).

2.5.5 Cyanide Concentration

The gold leaching rate varies in proportion to the cyanide concentration. Many studies have presented that gold extraction increases with increasing cyanide concentration. There must be sufficient free cyanide ions in the solution to dissolve all the gold particles otherwise it will be lost to the tailings. The recommended cyanide concentration is 0.06% NaCN, equivalent to 600ppmbut it can be raised to 1200ppm depending on the ore mineralogy and feed grade. The rate of gold dissolution grows linearly with increasing cyanide concentration until a maximum is attained

beyond which the increase in cyanide has no effect. In practice, a balance is sought so that there is sufficient cyanide to leach the gold. Excessive cyanide usage should be prevented to prevent an increase in production cost and reduce environmental problems. (Shrithammavut 2008).

In agitation leaching, cyanide consumption typically varies for free milling ores from about 0.25 to 0.75 kg/t. If there are cyanide-consuming minerals, then cyanide consumption varies from about 1 to 2 kg/t. In heap leaching, cyanide consumption is lower than in agitation leaching. Cyanide consumption typically ranges from 0.1 to 0.5kg/t for non-refractory ores. Intensive cyanidation leaching consumes the most cyanide of these three leaching techniques. Cyanide consumption in this intensive leaching varies from 5 to 25 kg/t *(Adams 2005)*.

2.5.6 Slurry Density

Leaching is normally performed at slurry densities between 35%- 50% of solids to maximize mass transfer phenomena (Mass transport of reacting species through the solution–solid boundary layer, to the solid surface). This depends on factors such as the specific gravity of solids, particles, and the presence of clay minerals that affect slurry viscosity. The rate of gold cyanide adsorption decreases with increasing slurry density. However, if the slurry density gets too low, the carbon particles may not stay in suspension and sink to the bottom of the tanks. The high slurry density can reduce the reagent consumption since the slurry concentration can be obtained by a smaller volume of solution per unit mass of material (Reagent consumption is minimized by maximizing slurry density) (Li 2006).

2.5.7 Mineralogy

The mineralogical composition of an ore plays a significant role in its leaching response. The presence of certain minerals can either enhance or hinder the leaching process. For instance, sulfide minerals often trap or occlude valuable metal particles, making it difficult for the leach solution to complex with the metal. This occlusion hinders the extraction process, necessitating more complex and intensive treatment methods.

In the context of uranium leaching, the mineralogical heterogeneity of the ore significantly impacts the leaching process. A study by Yang et al. (2021) found that the reactive leaching of uranium is most sensitive to the spatial distribution of the mineralogical properties of the uranium deposit.

The extent of the uranium plume was found to be highly dependent on mineralogical heterogeneity, and the effect of uranium grade heterogeneity was found to be important to improve the accurate capture of uncertainty.

Similarly, in the leaching of uraninite with dilute sulfuric acid from low-grade ores, the reactivity of gangue minerals was found to be a cause of concern. Gangue minerals reactivity can lead to the release of various anions from gangue or part of the mineral acid molecule (leachants) on the dissolution of uraninite, which can affect the purity of the leach liquor (Madakkaruppan et al., 2015).

Mineralogy is analyzed by methods such as Bulk Mineral Analysis (BMA) which is performed by the linear intercept method, where the electron beam is raftered at a predefined point spacing along several lines per field and covering the entire polished section at any given magnification. This measurement provides a robust data set for determining bulk mineralogy and modal proportions. Since the entire block is scanned, an extremely high statistical population is produced. In addition, the random alignment of the particles ensures appropriate sampling.

Quantitative modal mineralogy is obtained from the BMA measurement mode. The Bulk Mineral Analysis (BMA) measurement mode identifies the minerals present and determines their relative proportions in mass percentage. Qualitative XRD results were employed to assist with the bulk modal results. Bright Phase Search (BPS) is a two-dimensional bright phase search methodology in which a bright mineral area of every particle is measured. Bright Phase Search (BPS) is based on the particle mapping analysis (PMA) measurement mode. The BPS only analyses a pre-set sub-population of the particles present. It is based on the premise that the phases of primary interest (i.e., target phases) have a higher backscattered electron brightness than the bulk of the gangue phases. This enables each block to be scanned for particles containing the target phase, and only those of interest are fully analyzed. As the entire block is scanned, this also produces the highest possible statistical population for the trace phase. The information obtained from a BPS measurement is the same as that from the PMA measurement but relates only to the subpopulation analyzed. This is a particularly good analysis method for determining losses of sulphides and precious metal phases to silicate-rich tails. A predefined number of particles are mapped at a point spacing to spatially resolve and describe selected phases using BSE and/or compositional criteria.

2.6 Alternative Leaching Reagents and Their Efficacy

Leaching has traditionally relied on the use of cyanide. However, due to environmental concerns and regulations, there has been a growing interest in identifying alternative leaching reagents.

2.6.1 Non-Cyanide Leaching Reagents

A variety of non-cyanide leaching reagents have been explored for gold recovery from electric and electronic equipment waste. Among these, aqua regia and iodine/iodide leaching systems have shown promising results, yielding 100% of gold while demonstrating very fast leaching rates. The iodine leaching system was particularly favored due to its high selectivity for precious metals and reduced environmental impact.

2.6.2 JinchanTM Gold Leaching Reagent

JinchanTM gold leaching reagent has been identified as an alternative to conventional sodium cyanide. A comparative study found that while sodium cyanide has higher efficiency and effectiveness in leaching gold from mineral ores, JinchanTM is more environmentally friendly. However, the study also noted that JinchanTM requires a longer leaching time to achieve the desired plant recovery.

2.6.3 Glycine-Based Leaching System

Glycine, an amino acid, has been identified as an alternative leaching agent that can form stable complexes with gold and silver in alkaline solutions. This glycine-based leaching system has shown promise in enhancing the extraction of these precious metals.

A variety of alternative leaching reagents have been explored for their efficacy in extracting valuable metals from ores. These include non-cyanide reagents, JinchanTM gold leaching reagent, and glycine-based systems. While these alternatives show promise, further research is needed to optimize their use and to fully understand their potential environmental impacts.

2.7 Case Studies

There are several leaching methods available with their unique advantages and disadvantages. These methods include cyanide leaching, thiosulfate leaching, and chloride leaching. The selection of a leaching method is influenced by various factors, including the type of ore, the metal being extracted, and environmental considerations. Cyanide leaching is a common method used for gold extraction. However, it is highly toxic and poses significant environmental risks. Despite these challenges, cyanide leaching remains popular due to its high efficiency and effectiveness.

Case Study 1: Gold Tailings in Central China. A study conducted in central China investigated the heavy metal pollution caused by cyanide gold leaching. The research focused on the Yueliangbao gold tailings area, where gold was extracted by cyanidation. The study found that the concentrations of heavy metals such as copper (Cu), lead (Pb), zinc (Zn), and molybdenum (Mo) in the soils of the tailings pond were higher than those in the local background. The study concluded that the tailings pond could become a source of Cu pollution in the surrounding environment, thus endangering environmental safety and human health.

Case Study 2: Small Scale Miners in Tanzania. Another case study focused on small-scale miners in Tanzania. The study found that sodium cyanide leaching technology was used to recover gold from tailings. The research revealed that with a particle size of 180 μm, the cyanide dosage in the range of 700-1000 ppm, and a retention time of 72 hrs, a gold concentration of 2.45 ppm was achieved, which represented 42% of the gold recovery in the sample.

Case Study 3: Zortman-Landusky Gold Mine. The Zortman-Landusky gold mine is another case study of the environmental risks of cyanide heap-leach gold mining. This case study highlights the impacts which these operations can have on communities, water, and cultural resources and emphasizes the need for more sustainable and environmentally friendly extraction methods.

CHAPTER 3: METHODOLOGY

Enhancing gold leaching involves optimizing various factors to increase gold dissolution rates and overall recovery. Key strategies include improving reagent efficiency, optimizing leaching conditions (like pH, temperature, and oxygen levels), and implementing pre-treatment methods to expose gold more effectively to the leach solution

3.1 Sampling and submission of sample to the laboratory

In this study, pyrite gold refractory ore samples investigated were sampled and obtained from North Mara Gold Mine, a combined underground and open pit gold mine located in Tarime district of the Mara region in northern part of Tanzania. Upon receipt of the pyrite gold refractory ore samples from the Mine, their conditions were examined by the quality manager to verify if there were any contaminations. The damaged and contaminated samples were discarded while the undamaged and uncontaminated samples were thereafter labeled for easy reference and subsequently used in experimental work.

3.2 Sample preparation

Each of the received pyrite gold refractory ore samples was weighed and a bulk weight of 267.2 kg was recorded before being dried in the oven for 24 hours at 105℃ to determine the moisture content. The samples were thoroughly homogenized using a Rotary splitter and then split into 1kg sub-samples for metallurgical test-work.

3.3 Characterization

3.3.1 Head grade analysis

A 0.5kg sample was pulverised to 80% passing 75um and separated by riffler splitter to form 10 portions of 50g for Au Fire assay. (The analysis was done on 50g aliquots using the Fire assay method with AAS finish (LM-02). Wet digestion and ICP-OES were used for multi element analysis.

3.3.2 Multi element analysis

Arsenic and silver were analyzed using Aqua regia digestion with AAS finish (LM-01) and other aliquots were analyzed for other elements by Carbon and Sulphur Machine, UV-Vis-Machine, MP-AES, ICP SCAN and AAS analysis.

3.3.3 Mineralogy

Some of the representative aliquots of the samples were split for X-ray Diffraction (XRD), bulk modal mineralogy and gold deportment using automated scanning electron microscopy analysis (AutoSEM) using TESCAN Integrated Mineral Analyzer (TIMA). The XRD aliquots were pulverized, and the TIMA aliquots were used as received to prepare polished sections. In the AutoSEM Analysis, the prepared polished sections were analysed by TIMA using the Bulk Modal Analysis as the measurement mode to determine the samples bulk modal mineralogy and Bright Phase Search method for gold deportment.

3.3.4 Gold deportment

The methodology for the gold deportment analysis consists of the independent quantification of each form & carrier of gold from both leach and floatation perspectives, using a comprehensive mineralogical and analytical approach. It involves assaying by XRD to determine the general mineralogy, microscopy to identify and characterize gold minerals by type, grain size and association, SEM/EDX to determine the composition of gold grains, and SIMS to quantify sub-microscopic gold.

The QA/QC of the gold deportment analysis was ascertained by:

(i) Monitoring the accuracy of gold assaying using hidden standards.

(ii) Determining the submicroscopic gold by two independent methods.

(iii) The deviation of the mineralogically accounted gold from the assayed head grades.

A bright phase search measurement was set up to specifically search for gold grains. This measurement works by mapping grains with high backscattered electron (BSE) intensity and the

particles hosting these grains. A total of 25 normal-mounted polished blocks were prepared for each sample.

3.4 Grind establishment

This procedure consists of grinding 1 kg ore sample at incremental times using a laboratory ball mill. The ground samples are sized using a nest of screens (250, 180, 150, 106, 90, 75 and 53 μm) and the information is used to determine, by interpolation, the grinding times required to produce specific P_{80}'.

Grind Test Procedure

Stage 1

i. Thoroughly clean out the ball mill.

ii. Place 1.0 kg of ore (minus 2.360 mm) into a ball mill.

iii. Mill the material dry

iv. Milling is performed for **5, 10, 15 and 20** minutes.

v. When grinding is completed for the requisite time, the mill is stopped, and the lid removed.

vi. The ball mill ball charge is offloaded onto a screen located above the slurry collection bucket.

vii. The adhering solids from the mill and the balls are washed into the bucket.

viii. The slurry is pressure filtered, and the filter cake is dried in an oven at 105°C overnight.

ix. The dry filter cake is weighed, and the mass is recorded.

x. The filter cake is broken by passing it through a 1.18 cm screen.

xi. The solids are thoroughly blended.

xii. 100 g of solids are riffled out for size analysis and placed in a labelled plastic bag.

xiii. The 100 g sample is wet screened on a 53 µm screen

xiv. Dry the oversize plus <53 µm and weigh.

xv. The oversize +53 µm is dry screened on the following sieves: 500, 250, 180, 150, 106, 75, and 53 µm.

xvi. The mass of each size fraction (including any extra - 53 µm) is recorded and the size fractions are kept in discretely labelled plastic sample bags.

xvii. The size distribution of the sample is calculated and plotted using Excel spreadsheet and the 80% passing size is determined.

xviii. The Required Grinding Time to Achieve Target P_{80} is determined.

Four-point milling curve is used to determine the time required for milling to achieve the target particle size of 80% passing various screen sizes.

3.5 Gravity test-work

Gravity concentration tests were carried out using a Knelson concentrator. Each test used a 5 kg of gold ore material, ground to approximately 80% passing 150 microns. This grind is essentially the same as that determined as optimum for mineral liberation and typically used for gravity concentration test work such as Amenability Test using Knelson/Falcon for Gravity Recoverable Gold (GRG).

3.6 CIL leaching test work

CIL also known as carbon in leach is a hydrometallurgical technique used to extract gold from pyrite ore sample. CIL test works were conducted in an intensive leach reactor from which a high-grade gold concentrates and tails were obtained. Gravity and intensive leach reactor tailings were combined and mixed homogeneously to form a composite sample. About 2000 g of this sample was collected and mixed with 2168 ml of tap water. The test procedure was as follows:

i. The mass of empty, dry, vat is measured.

ii. 1.5 kg of P80 <106 microns of composite sample is placed into a six-litre, mechanically agitated leach vat.

iii. The gross mass of the vat and solids is measured.

iv. Tap Water is added to establish a slurry density of 50% solids (w/w).

v. The tare mass of the bottle is subtracted from the gross mass of the bottle + slurry to determine the slurry mass.

vi. The slurry % solids w/w is calculated as follows:

$$\%\ \text{Solids w/w} = \left(\frac{\text{Dry mass of Solids}}{\text{Slurry mass}}\right) \times 100$$

vii. Hydrated lime (60% CaO) is added to each slurry to establish a pH of approximately 10.5.

viii. pH of 10.2 is maintained up to the end of test work.

ix. Solid cyanide is added to the slurry to establish an initial nominal cyanide solution strength of 0.03%.

x. Oxygen gas is injected or sparged to achieve an oxygen level of 18-20ppm.

xi. 15 g/l of activated carbon of 1 mm diameter is added into the pulp/slurry.

xii. Leaching is resumed at the agitation intensity of 2400 rev/min

xiii. At intervals of 2, 4, 8, 12 and 24 hours during the leach, 100 ml of pulp was taken from the reactor by using a measuring cylinder. The pulp is filtered to supply pregnant liquor for Au analysis. 10 ml of the pregnant liquor solution is used for cyanide titration with silver nitrate and if required further lime and cyanide can be added to maintain the desired pH (10.5) and cyanide solution strength (0.011%).

xiv. At the termination of the experiment after 24 hrs, pH, oxygen and cyanide levels are determined, and a final solution sample is taken for Au analysis.

xv. The carbon is removed via screening and placed in an oven overnight to dry.

xvi. The residual slurry is filtered, washed and dried to provide solid leach residue and submitted for Au analysis. Further, to recover adsorbed Au from activated carbon aqueous sodium cyanide solution was used at higher temperature, i.e., 130℃ , to ensure carbon are eluted to recover Au present.

Facilities used

ITEM	QUANTITY	LOCATION
Erlenmeyer flask, 250 ml	10	Metallurgical section
Beaker	8	
Bottle, 3.5 liter	16	
Ph. meter, OHAUS	2	
Mass balance	2	
Spatula	4	
Sodium cyanide, 25 kgs	1	
Activated carbon, 25 kgs	1	
Hydrogen peroxide, 2.5 liters	1	
Caustic soda, 25 kgs	1	
Hydrated lime, 25 kgs	1	
Nesch Mintek CIL Pilot	2	
Filter paper, 0.45 mm	100 pcs	
Drying trays	25	
Drying oven	1	
Furnace	1	Fire assay section
Laboratory ball mill	1	

Canadian standard sieve, 3.35-53 µm	@1	Metallurgical section
Blushes	5	
Sample bags	100pcs	
Centrifuge	1	
Knelson concentrator	1	
ROTAP Sieve shaker	1	
Retort stand	1	
Measuring cylinder, 100 mls	2	
Pipette	1	
Burette	1	
MP AES	1	AAS Room
AAS	1	
Test tubes	16	
Hot plates	1	Digestion section
Stopwatch	1	Metallurgical section

CHAPTER 4: RESULTS, DISCUSSION AND COST ANALYSIS

This section details the results and discussion of all the experimental work done as described in the methodology section.

4.1 Samples

Figure 2 shows as received samples of the pyrite refractory gold ore. A total of 15 bulk rock samples weighing 267.2 kg were received to study the pyrite refractory gold ore leaching response and recovery. The samples were labelled for ease of reference and preliminary inspections on samples were done based on checklists to assess the sample contamination and damage. Preliminary inspection results of the samples received indicated that the samples were neither contaminated nor damaged. Therefore, the samples were of good quality and could be directly used in subsequent experimental tests.



Figure 2: Received pyrite refractory gold ore samples pictorial view.

4.2 Sample preparation

Weight and moisture content of the ore samples are among the parameters that can affect the overall leaching response. Thus, for effective and optimized refractory gold ore recovery, these parameters must be measured and controlled as highlighted in literature (Zhou et al, 2019; MoistTech, 2024).The measured weight and calculated moisture content of the dried samples are presented in Table 1. The samples' moisture content is slightly constant because they were evenly packed and dried in an oven under constant conditions, i.e., for 24 hours at 105℃. The average moisture content of the samples was approximately 8.56% which is similar to moisture content of gold ores as highlighted also by Costa et al. (2013) and Lage et al. (2018). The samples were thoroughly homogenized using a Rotary splitter and then split into 1kg sub-samples for metallurgical test-work.

Table 1: Weight and moisture content of the ore samples

Sample Type	Sample ID	Weight of wet samples (kg)	Weight dried samples (kg)	Weight deviation (kg)	Moisture content (%)
Rock	1	17.4	16.1	1.3	7.47
	2	21.4	19.7	1.7	7.94
	3	17.2	15.8	1.4	8.14
	4	16.9	15.4	1.5	8.88
	5	14.9	13.8	1.1	7.38
	6	13.1	12.2	0.9	6.87
	7	23.6	21.7	1.9	8.05
	8	20.4	18.8	1.6	7.84
	9	14.3	13.0	1.3	9.09

	10	16.7	15.1	1.6	9.58
	11	18.5	16.7	1.8	9.73
	12	20.8	19.1	1.7	8.17
	13	22.6	20.7	1.9	8.41
	14	14.2	12.6	1.6	11.27
	15	15.2	13.9	1.3	8.55
	Total	**267.2**	**244.6**	**22.6**	**-**
	Average	**17.84**	**16.32**	**1.52**	**8.56**

4.3 Characterization of ore

4.3.1 Head grade analysis

Head grade analysis measured using the gold fire assay method with AAS finish (LM-02) results are presented in Table 2. As shown in Table 2, the average gold content of the ore sample was approximately equal to 2.26 g/t. This indicated that the sample had enough gold that is recoverable using the leaching process. Similar gold content of ores that were processed by leaching have also been reported by Lyman (2021).

Table 2: Head grade average fire assay analysis results

Sample Type	Sample ID	Element	Unit	Value
Rock	Composite	Gold, Au	g/t	2.25
		Gold, Au duplicate	g/t	2.38
		Gold, Au triplicate	g/t	2.15

		Average Au	g/t	2.26

4.3.2 Multi element analysis

Table 3 presents the multi element analysis results of the ore samples. Arsenic and silver were analyzed via Aqua regia digestion with AAS finish (LM-01) and were observed to be equal to 923.03 and 0.97 mg/kg, respectively. Other minor elements were analyzed via Carbon and Sulphur Machine, UV-Vis-Machine, MP-AES, ICP SCAN and AAS analysis and small traces of these were also detected as depicted in Table 3. Thus, due to the low amount of the other minor elements, their respective effects on gold leaching process could therefore be neglected.

Table 3: Multi-element analysis results

Sample Type	Sample ID	Analysis type	Element	Unit	Value
Rock	Composite	Carbon speciation	C (Total)	%	0.25
			C (Inorganic)	%	0.22
			C (Organic)	%	0.03
		Sulphur speciation	S (Total)	%	0.87
			S (Sulphide)	%	0.62
			S (Sulphate)	%	0.25
		Multi-element	Ag	mg/kg	0.97
			As	mg/kg	923.03
			B	mg/kg	3.94
			Ba	mg/kg	1295.86

			Be	mg/kg	1.90
			Bi	mg/kg	0.38
			Cd	mg/kg	0.17
			Co	mg/kg	14.68
			Cr	mg/kg	121.32
			Cs	mg/kg	2.33
			Cu	mg/kg	24.33
			Ga	mg/kg	18.84
			Ge	mg/kg	1.40
			Hf	mg/kg	6.49
			Hg	mg/kg	0.02
			Ho	mg/kg	0.27
			In	mg/kg	0.03
			Li	mg/kg	25.26
			Mn	mg/kg	506.86
			Mo	mg/kg	7.84
			Nb	mg/kg	8.83
			Ni	mg/kg	64.94
			Pb	mg/kg	19.25

			Sb	mg/kg	3.40
			Sc	mg/kg	6.15
			Se	mg/kg	0.15
			Sn	mg/kg	1.93
			Sr	mg/kg	188.29
			Ta	mg/kg	0.47
			Te	mg/kg	0.23
			Th	mg/kg	2.60
			Tl	mg/kg	3.55
			V	mg/kg	88.03
			Y	mg/kg	4.45
			Zn	mg/kg	57.63
			Zr	mg/kg	238.71

4.3.3 Mineralogy

Mineralogical composition or mineral association of the ore is one of the main factors that must be considered when investigating gold leaching process. Gold Mineral association percentage refers to the proportion of Au species that occur and share grain boundaries with identified minerals within the ore sample. It quantifies fraction of Au species that is found either within certain minerals, i.e., quartz, sulfides and many others or has free surface. Free surface refers to the perimeter of the particle that is exposed and does not share a grain boundary with any mineral. Table 4 shows the quantitative bulk modal mineralogical results of the ore samples obtained by

using TESCAN Integrated Mineral Analyzer (TIMA). TIMA calculates mineral associations based on the number of pixels at the contact of the phase of interest with the other phases. Each phase of interest pixel is inspected whereby the program looks at 4 adjacent pixels and records which phase the pixels are assigned to. The pixel counts are then normalized to show the percentage mineralogical association. Mineral association data is derived from shared boundaries amongst the identified mineral grains. The higher the associated percentage, the greater the degree of boundary-sharing between mineral species. Therefore, from Table 4, it can be seen that the gold ore mineralization tends to be closely associated with quartz, hessite, and pyrite, and less associated with feldspar, other sulphides, chlorite and Illite. The results also show that approximately 18.22% of the free surface association was also recorded.

Table 4: The mineral associations (%) of the gold

Mineral	**Chemical Formula**	**Gold Association (%)**
Hessite	Ag_2Te	17.35
Pyrite	FeS_2	12.12
Pyrrhotite	FeS	0.00
Other Sulphides	-	3.65
Quartz	SiO_2	34.38
Mica	$KAl_2(Si_3Al)O_{10}(OH,F)_2$	0.10
Feldspar	$(K,Na)AlSi_3O_8$	9.17
Ferroan Dolomite	$(Fe,Mg)CO_3$	0.10
Apatite	$Ca_5(PO_4)_3(OH,F,Cl)$	0.00
Illite	$(K,H_3O)(Al,Mg,Fe)_2(Si,Al)_4O_{10}[(OH)_2,(H_2O)]$	1.18

Chlorite	$(Mg,Fe)_5Al(Si_3Al)O_{10}(OH)_8$	3.10
Rutile	TiO_2	0.15
Oxide/Hydroxide	-	0.23
Calcite	$CaCO_3$	0.20
Other	-	0.08
Free Surface	-	18.22
Total		**100.0**

4.3.4 Gold deportment

Gold Species and Grain Count

The gold deportment and gold grain count results are presented in Table 5. Twenty-five normal-mounted polished blocks were analyzed, with a total of 102 gold-bearing grains found in the North Mara sample. The gold-bearing mineralization was located in the sample occurring as two types, native gold (primarily Au with trace Ag) and the less abundant, petzite. Semi-quantitative EDS (energy-dispersive X-ray spectroscopy) was conducted on the individual grains to confirm the minerals identified (Figure 3 and Figure 4).The native gold phase is host to the majority of the gold in the sample at 95.32 %, and petzite hosts lesser amounts of gold at 4.68%. Since the petzite occurs in relatively minor amounts, all the Au-bearing minerals were grouped together for the characterization (exposure, GSD etc.).

Table 5: Grain counts located in the sample

Sample Type	Sample ID	Mineral	Au Deportment (%)	Number of grains
Rock	Composite	Native gold	95.32	66

		Petzite	4.68	36

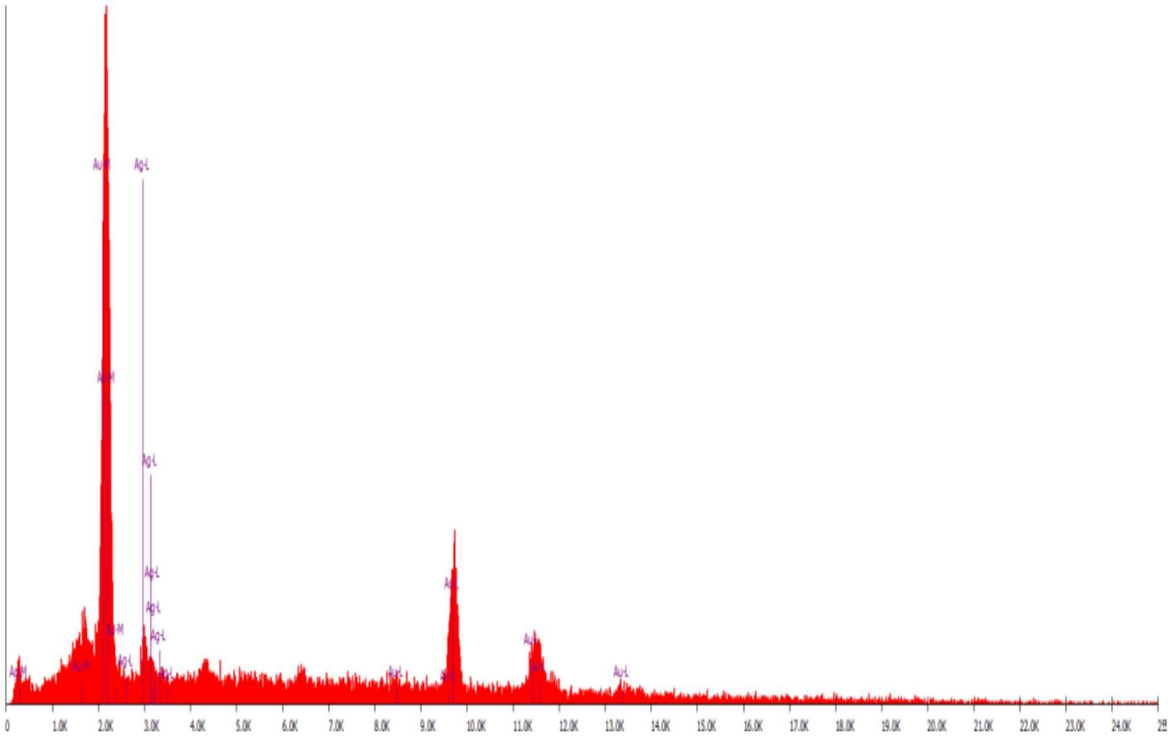


Figure 3: EDS of the native gold phase present in the sample.

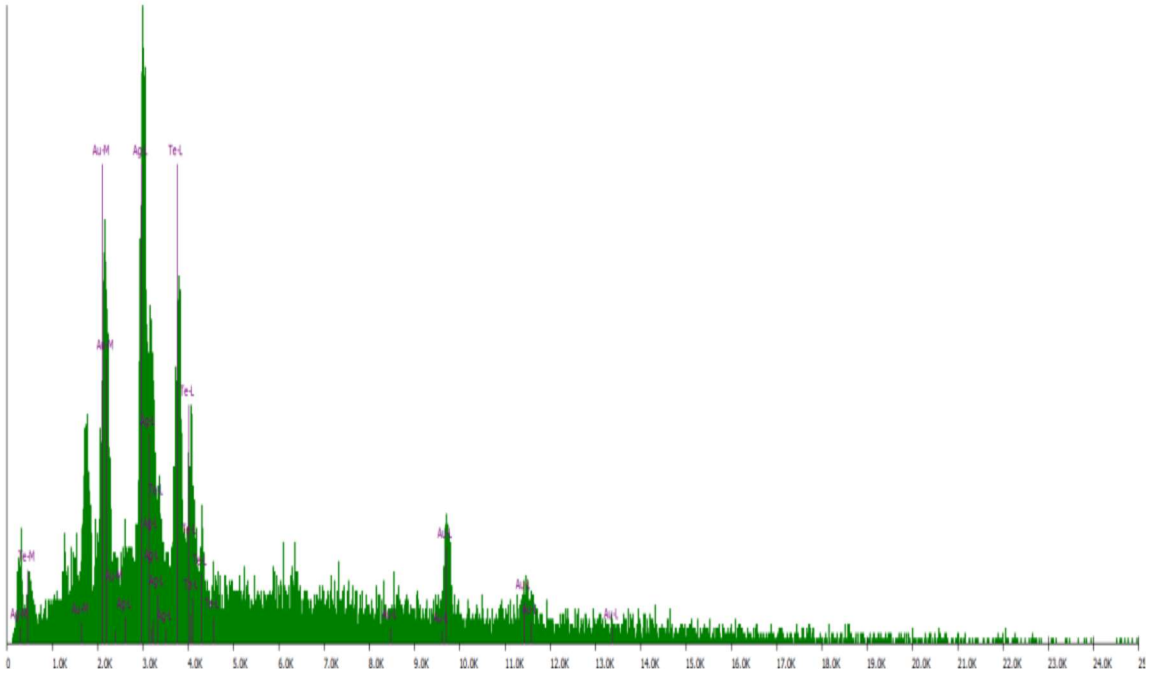


Figure 4: EDS of the petzite phase present in the sample.

Gold Grain Size Distribution (GSD)

Grain sizes are expressed in equivalent circle diameter (ECD), which is defined as the diameter of a circle with the same area as the measured grain. Note that the grain size data presented have been obtained from measurements on sectioned grains (i.e., in two dimensions only). Although stereological corrections have been applied to the data, the grain sizes reported are not comparable to true 3-D grain sizes. Comparison between samples, using the TIMA particle size data is, however, possible. The gold grain size distribution is illustrated in Figure 5. Gold has a variable grain size distribution, with 25.4 mass % of the gold less than 7 μm, but notable coarse content is observed as well. Approximately 35.2 mass % of the gold is in the 35-40 μm size range.

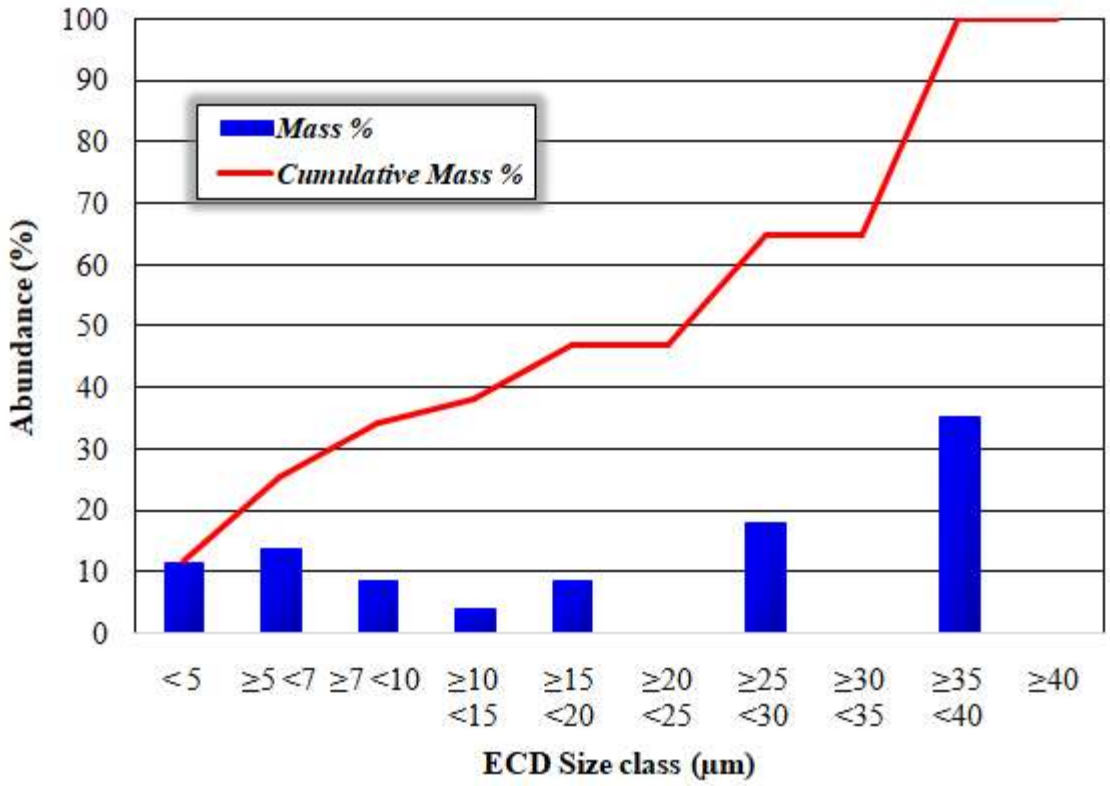


Figure 5: Grain size distribution (discrete and cumulative) of the native gold.

4.4 Grind establishment

The amount of accessible and recoverable refractory gold via the leaching process depends strongly on the particle size of the ore sample being leached. Ultrafine particles and very course particles decrease the gold dissolution rate, hence an optimal size is vital to ensure a cost effective and efficient leaching process. Particle size of the ore sample decreases with increase in the milling time. This consequently results in an increase in cumulative passing a certain size as shown in Figure 6. The figure shows a plot of cumulative passing percentage of ground sample against nominal sieve sizes for various milling times of 5, 10, 15 and 20 mins. An upward shift of plots was observed when milling time was increased from 5 to 20 min. This is expected because the particle size decreases with increase in milling time, leading to more material passing the same size as milling time increases. Similar results have also been reported by Abelhaffez (2016) and Egan *et al.* (2016).

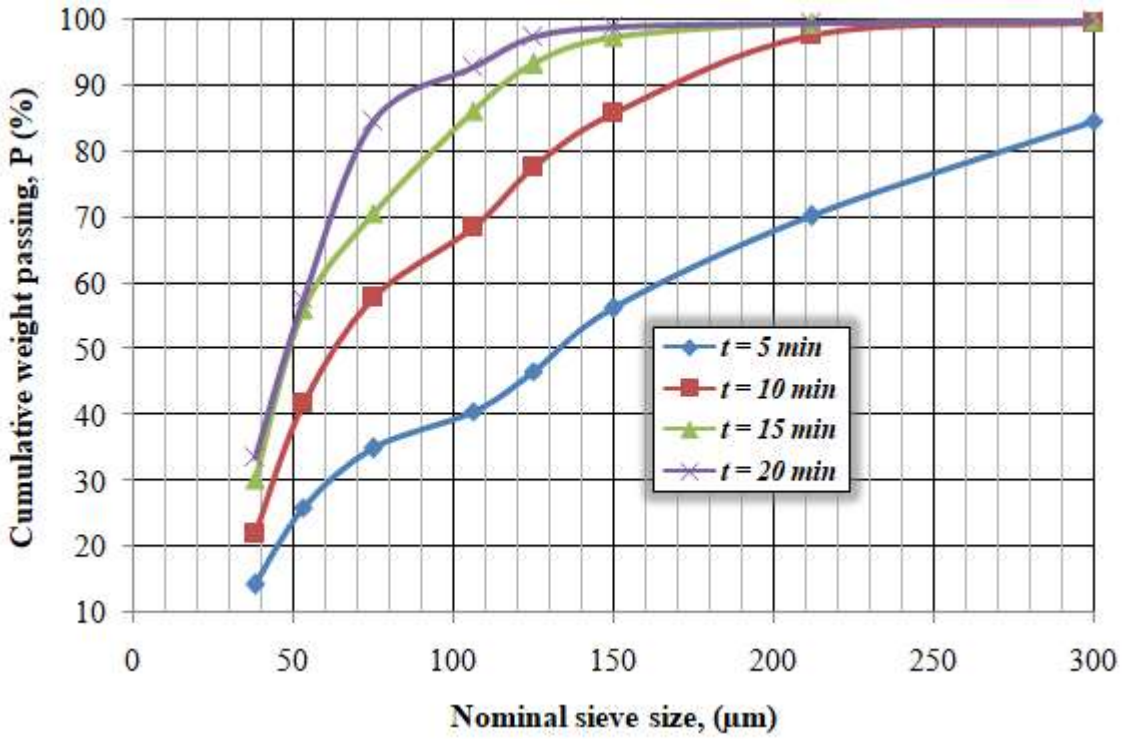


Figure 6: % Passing curve of the ore sample at 5, 10, 15 and 20 mins.

A four-point milling curve to determine by extrapolation, the time for milling the sample to P80 <75 µm was obtained from the results obtained in Figure 6 and is illustrated in Figure 7. It is observed that increase in milling time from 5 to 20 min resulted to a decrease in P80 grinding sizes from 270 to 70µm. The decrease in P80 milling sizes with increase in milling time is expected because the grindability of the sample increases with increase in milling time. Furthermore, the optimal milling time for P80 <75 µm and P80 150 µm was found to be 18.4 and 9 min respectively. The P80 <150 µm grind time was used for subsequent analysis in this study because it is the milling time that results in optimum mineral liberation for the particle sizes used for gravity concentration test work such as Amenability Test Knelson/Falcon for Gravity Recoverable Gold (GRG) (McGrath, Eksteen and Bode 2018).

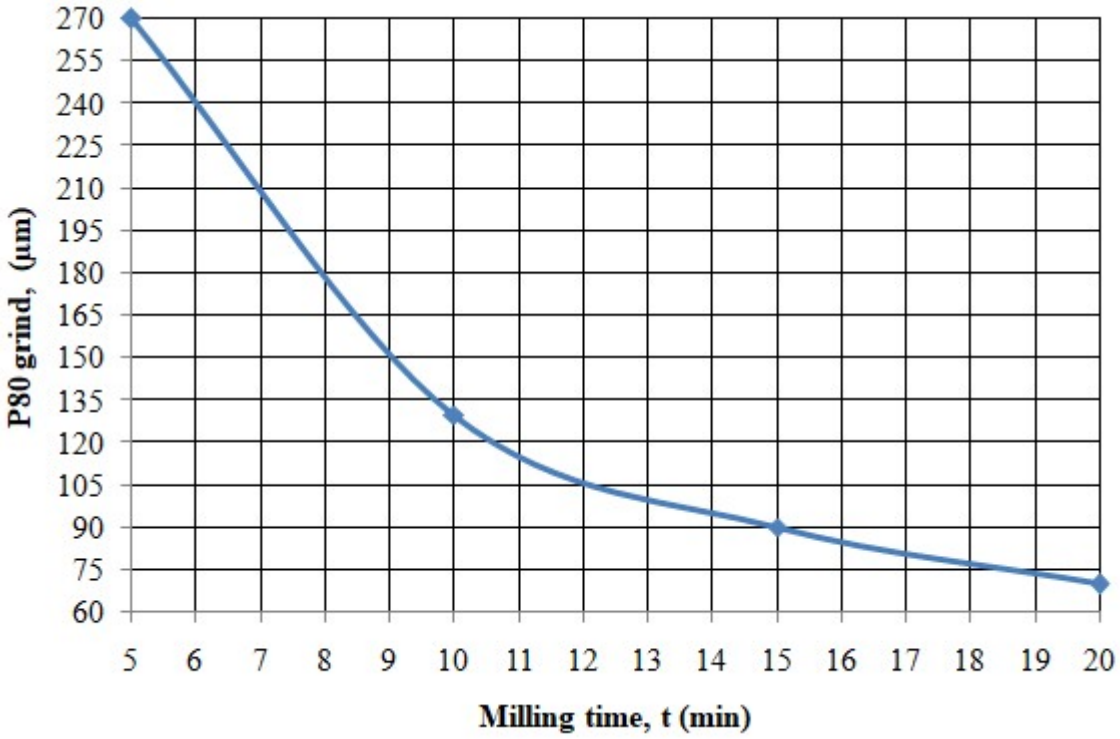


Figure 7: Four-point milling curve.

4.5 Gravity determination

Gravity concentration tests for the P80 <150 µm grind were carried out using a Knelson concentrator with a water pressure of 2 psi, a pulp density that ranges from 25% to 38% and a 2400 rpm concentrator speed based on standard methods for Knelson specifications. The gravity test work results obtained are presented in Table 6. It is observed from Table 6, that the decrease in pulp density from 38% to 25% resulted in an increase in the gravity concentrate mass pull from 121.65g to 126.14g indicating that at lower pulp density the gold bearing particles migrate much easier to the concentrate slots. The gravity concentrates grades also show a significant increase i.e. from 36.54 g/t to 39.71 g/t as the pulp density decreases from 38% to 25%, which suggests that at lower pulp density the concentrate is less contaminated. This observation is because increase in pulp density increases the pulp viscosity which hinders material transfer effectiveness between the particles and leaching solution during leaching processes. This consequently results in a decrease in the amount of gravity concentrate obtained and decreases the overall gold mass pull as pulp density increases.

Table 6: Gravity test work results

	Mass (g)		Gold grade (g/t)			Gold recovery (%)	
Pulp density (%)	**Gravity concentrates**	**Gravity tails**	**Gravity concentrates**	**Gravity tails**	**Total**	**Gravity concentrates**	**Gravity tails**
38	121.65	9878.35	36.54	1.59	2.02	22.09	77.91
33	122.92	9877.08	37.23	1.68	2.11	21.65	78.35
29	124.87	9875.13	38.73	1.74	2.20	22.00	78.00
25	126.14	9873.86	39.71	1.88	2.35	21.28	78.72

4.6 CIL leaching amenability for the gravity tailings

4.6.1 Effect of Leaching time on gold head assay in an aerated bottle roll leach result

Figure 8 shows the plot of gold assay value obtained against leaching time in an aerated bottle roll leach test-work conducted for a P80 <150 µm grind at a fixed concentration of 20000 ppm for cyanide concentration, 20 ppm for hydrogen peroxide, 2 kg/t for lead nitrate and a 2400 rpm concentrator speed based on standard methods for leach batch reactor specifications. For all pulp densities gold assay was observed to increase rapidly with the bottle roll leaching time at the beginning of the leaching tests up-to a critical value which occurred at about 20 hrs, i.e., the maximum gold recovery assay value. An increase in leaching time from 0 to 20 hrs resulted in an increase in gold assay attributed to the high gold dissolution rate at the beginning and higher residence time at the tail end of the tests. However, a slight decrease in gold assay for all pulp densities is seen at time > 20 hrs, this may indicate that some passivation may be occurring beyond the leaching time of 20 hrs. Further, from Figure 8 it is observed that the gold assay profiles tends to shift down with the increase in pulp densities from 25% to 38% indicating a decrease in the gold recovery efficiency, this is expected due to the fact that increase in pulp density increases the

viscosity of solution, consequently leading to inefficient material transfer during leaching operations. Similar observations have also been reported by Harjanto *et al.* (2019), Yanbo *et al.* (2021) and Hai *et al.* (2024).

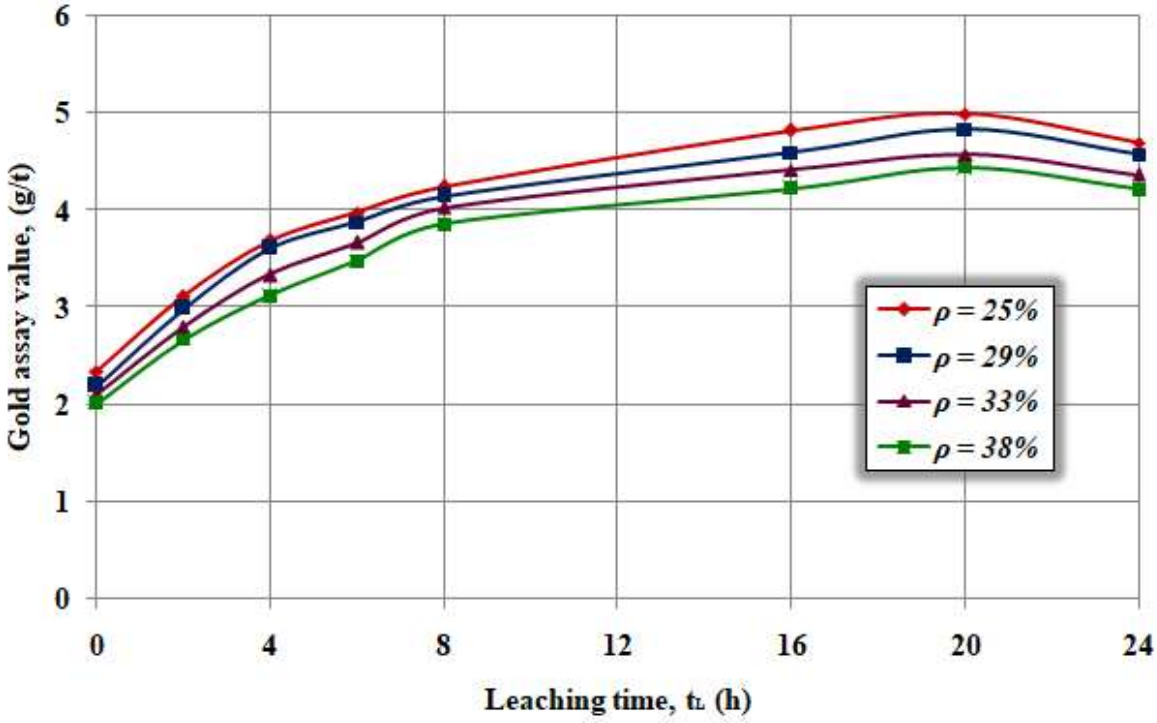


Figure 8: Effect of leaching time on head assay value in an aerated bottle roll leach test for the P80 <150 µm grind.

4.6.2 Effect of leaching time on gold recovery in CIL leaching

The effect of leaching time on gold extraction and recovery for both pre-leach and direct-leach CIL processes is shown in Figure 9. Pre-leach CIL is a process which involves pre-conditioning or reduction of materials via oxygenation before adding lixiviant i.e., cyanide to aid overall leaching process. On the other hand, in direct CIL a lixiviant is added directly and pre-condition stage is omitted as leaching process and material transfer commences directly. Both processes were conducted under a fixed pulp density of 48% and concentration of 20000 ppm for cyanide concentration, 20 ppm for hydrogen peroxide and 2 kg/t for lead nitrate based on standard methods for leach batch reactor specifications. Despite both processes being conducted under constant

parametric conditions, it was observed that for leaching time, i.e., t < 16 hrs, direct leaching had higher and better gold recovery efficiency compared to the pre-leach process. The reason for this observation is because in direct leach, direct cyanidation was conducted at the beginning of the process whereas, for the pre-leach, cyanide was added after 4 hrs and leaching for this process as reported also by Hai *et al.* (2024) was achieved via material transfer and reduction as a result of oxygenation process. Conversely, for higher leaching time, i.e., t ≥ 16 hrs, gold recovery efficiency was observed to be more or less constant indicating that maximum gold recovery under those conditions had been achieved. A sharp increase in gold recovery from 0 to 2 hrs is attributed to high leaching kinetics at the beginning of the leaching process when the well liberated mineral grains are reacting. Similar recovery profiles have also been reported by Parga, Valenzuela and Cepeda (2007).

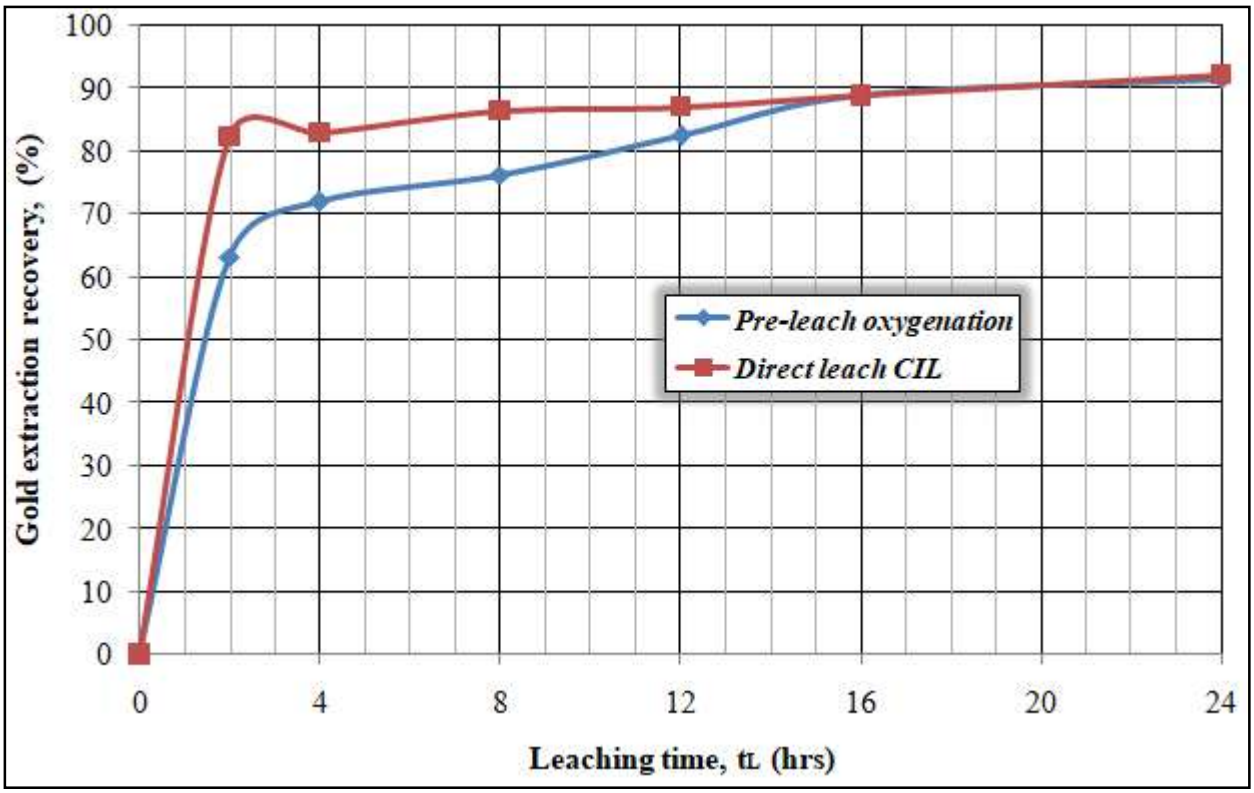


Figure 9: Effect of leaching time on gold extraction recovery in pre-leach and direct leach CIL processes for the P80 <150 µm grind.

4.6.3 Effect of particle sizes on gold recovery in Direct Leaching

Gold extraction recovery via the leaching process is significantly affected by the particle size or the P80 grind of the sample. The effect of P80 grind size on gold recovery at different leaching times is depicted in Figure 10. Increase in leaching time from 0 to 24 hrs for all particle grind sizes resulted in an increase in gold recovery until a maximum leaching point is reached, i.e., a condition at which maximum recovery is attained. Similar recovery profiles as shown in Figure 10 was observed across all grinds. Further, it is observed that at leaching time t <16 hrs, a remarkable gold recovery is achieved when P80 <150 µm compared to other particle grind sizes. Fine particles will result in a decrease in the rate of dissolution of gold from ores containing cyanicides as there is an increase in the rate of competition for reagents during leaching . This was reported also by Celep *et al.* (2016), McGrath *et al.* (2018) and Yin *et al.* (2020), that P80 <150 µm is the optimal grind size used in leaching and cyanidation process for the gold recovery. A promising gold recovery greater than 80% was attained regardless of particle sizes from P80 <53 µm to P80 <150 µm at higher leaching time, i.e. t ≥ 16 hrs, This is mainly because of increased residence time which causes slight improvements in overall gold recovery.

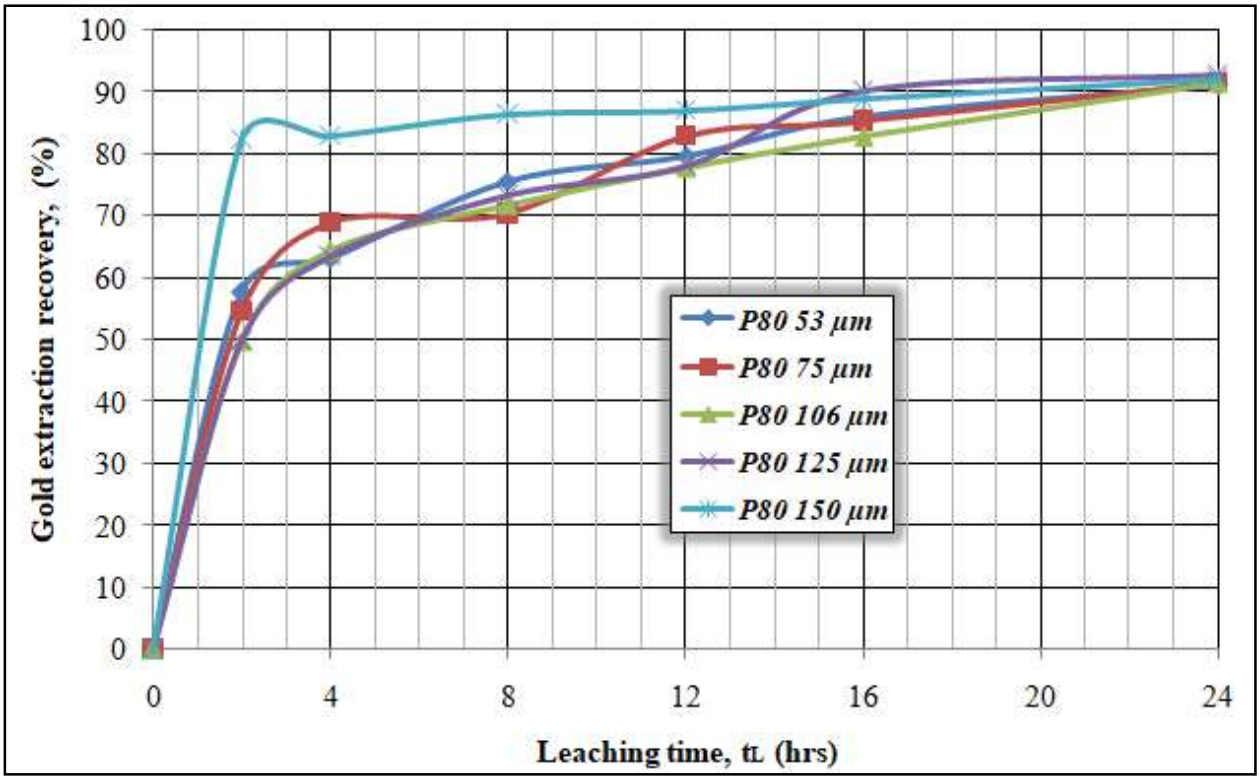


Figure 10: *Effect of P80 grind particle sizes on gold recovery.*

4.6.4 Effect of dissolved oxygen on gold recovery in CIL leaching

Dissolved oxygen (DO) as highlighted by Shrithammavut (2008), makes a significant impact on the overall gold extraction recovery via the leaching process, this is because DO facilitates mineral oxidation within the ore sample, consequently, resulting in an increase in gold dissolution and a more effective cyanide leaching. Depending on the initial gold recovery, an increase in the amount of DO at constant pulp density, and cyanide-gold mass transfer and adsorption efficiency may result in either an increase or decrease in gold recovery efficiency. The effect of DO on gold recovery for an increasing leaching time from 0 to 24 hrs for the P80 <150 μm grind is shown in Figure 11. In Figure 11, it is observed that the gold recovery is enhanced when DO is increased from the range of 10-15 ppm to 15-20 ppm. However, when DO was increased beyond 15-20 ppm, the gold recovery decreased by approximately 30%, this is because higher DO levels increase cyanide decomposition rate leading to decrease in cyanide content responsible for leaching operation and hence resulting in a decrease in leaching efficiency and gold recovery. Therefore, these results indicate the need to monitor and maintain DO at a range of 15-20 ppm for optimal gold recovery.

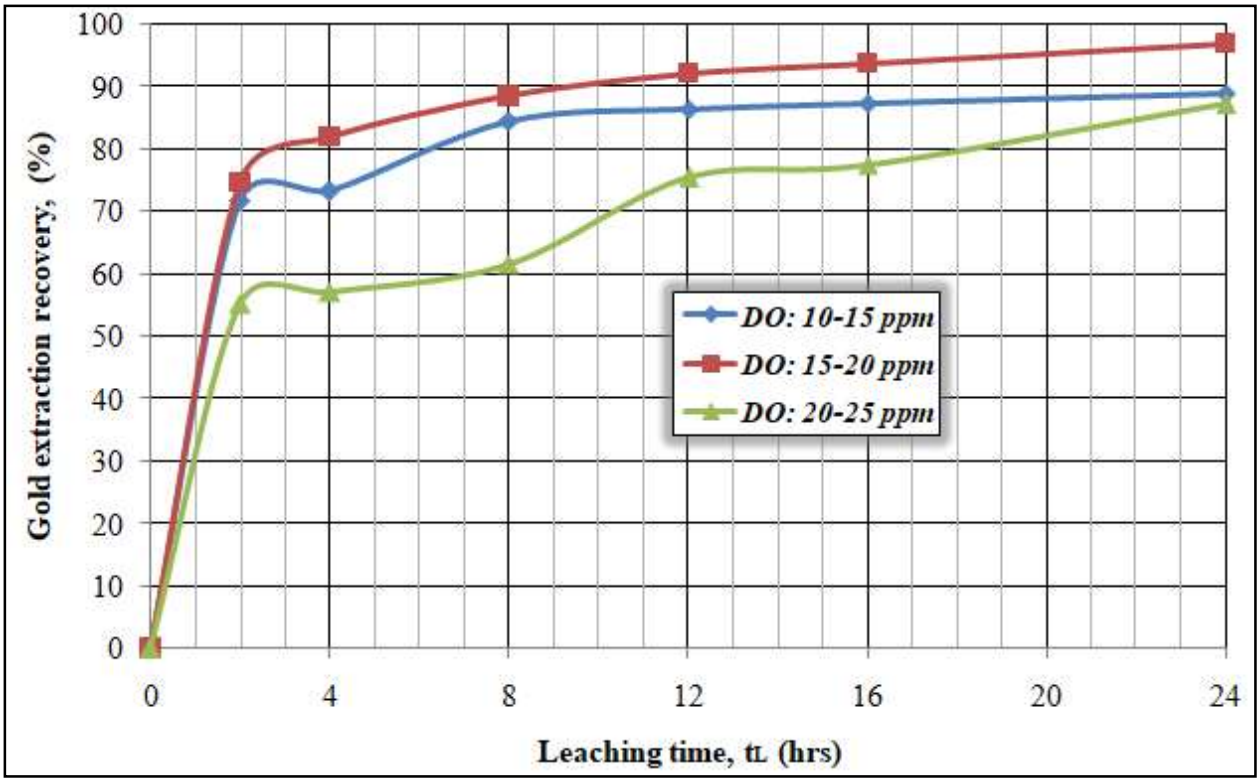


Figure 11: Effect of dissolved oxygen on gold recovery for the P80 <150 µm grind.

4.6.5 Effect of cyanide concentration on gold recovery in CIL leaching

Gold recovery is directly affected by the concentration and amount of cyanide used during the leaching process. The cyanide amount as highlighted in many works of literature including, but not limited to Azizi and Ghaedrahmati (2015), and Putri *et al.* (2023), suggests that cyanide concentration may either enhance or hinder the overall gold recovery. The effect of cyanide concentration on gold recovery is illustrated in Figure 12 where it was observed that the overall gold recovery efficiency increases rapidly at the beginning of the leaching process ($t < 16$ hrs) and slightly decreases at a higher leaching time ($t \geq 16$ hrs). This is due to extremely higher gold and silver concentrations at the beginning of the leaching process which reacts with cyanide quickly and these species decrease in concentration as the leaching time progresses.

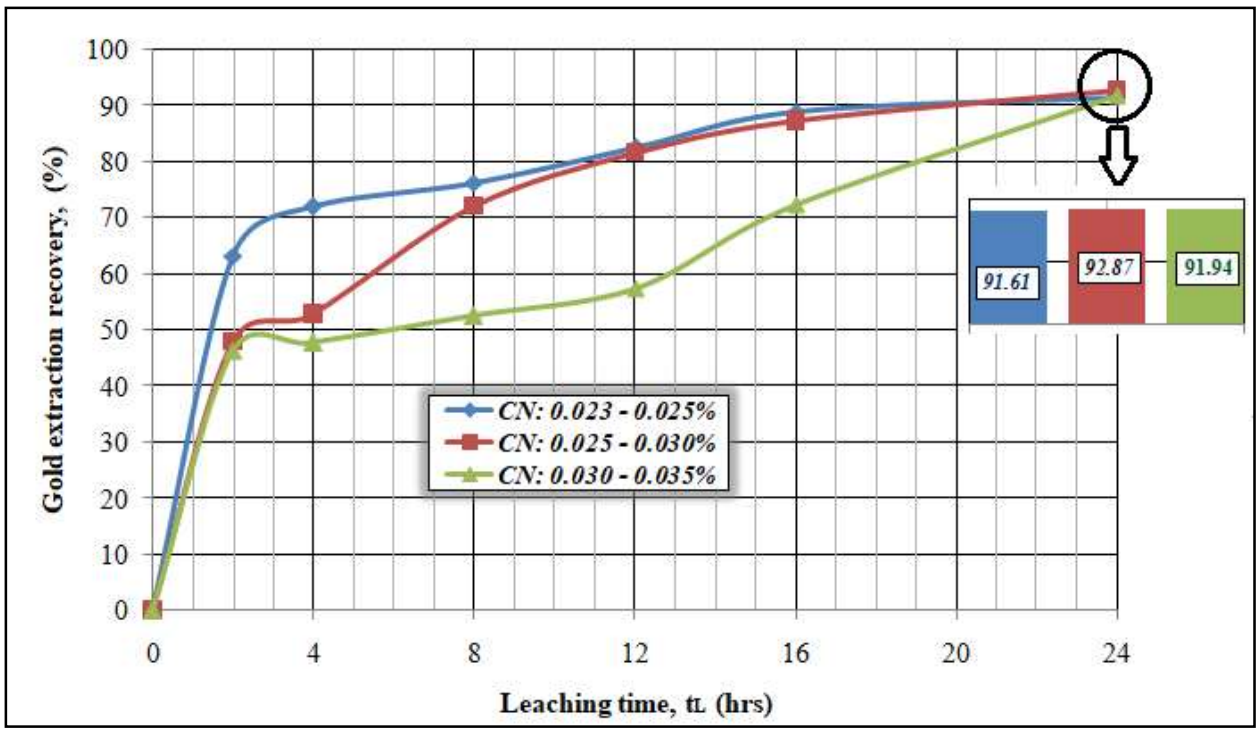


Figure 12: *Effect of cyanide concentration on gold extraction recovery.*

In the leaching period less than 20 hrs, an increase in the cyanide concentration from 0.023 - 0.025% to 0.030 - 0.035% results in a marked decrease in the gold recovery, indicating that higher leaching agent concentration does not necessarily increase the recovery. However, after the whole 24 hours the medium cyanide concentration 0.025-0.030% resulted in the highest gold extraction recovery efficiency of 92.87%, and the highest cyanide concentration 0.030 - 0.035% resulted in the gold extraction recovery efficiency of 91.94% and the lowest cyanide concentration 0.030 - 0.035% resulted in the lowest gold extraction recovery efficiency of 91.61%. This was also reported by Putri *et al.* (2023) and indicates the maximum cyanide concentration point for optimal gold recovery. In the leaching period less than 20 hrs, highest recovery was observed when cyanide concentration was lowest (0.023-0.025%) indicating a better CN-Au reaction and extraction rate.

4.7 Cost Analysis

The operating cost for a gold treatment plant can vary widely based on several factors. These factors include the plant's efficiency, scale of operations, local cost structure, and the specific technologies employed. Other costs to consider include processing, G&A, trucking, and all-in sustaining costs.

The table below shows a calculation for a net operation revenue for one ton of ore treated under the optimum conditions defined from the testwork.

Operating Revenue		
Gold Price	*$/oz*	3200
Grade	*g/t*	2.26
Plant Recovery	*%*	91.6
Conversion factor from oz to g		31.10348
Mining Cost	*$/t*	3.38
Processing cost (Including sustaining Capital)	*$/t*	21.83
General & Administration (Including sustaining Capital)	*$/t*	14.66
Total cost per ton treated	*$/t*	39.87
Revenue per ton treated	*$/t*	213
Net operation revenue per ton treated	*$/t*	**173**

$$Total\ cost\ per\ ton\ treated = Mining\ cost + Processing\ cost + G\&A\ cost$$

$$Revenue\ per\ ton\ treated = \frac{Grade\ \left(\frac{g}{t}\right) * Recovery(\%) * Gold\ price}{100 * Conversion\ factor}$$

$$Net\ operation\ revenue\ per\ ton\ treated = Revenue\ per\ ton\ treated - Total\ cost\ per\ ton\ treated$$

The net operation revenue is positive,173 $ per ton of ore treated, which shows that the treatment route is economically viable.

CHAPTER 5: CONCLUSIONS AND RECOMMENDATIONS

5.1. Conclusions

An assessment of factors affecting the leaching response and recovery of gold from pyrite refractory gold ores has been done in this study. The pyrite gold refractory ore samples investigated in this study were obtained from a combined underground and open pit North Mara Gold Mine located in Tarime district of the Mara region in northern part of Tanzania. A total of 15 bulk rock samples from the North Mara Gold mine weighing 267.2 kg with an average moisture content of 8.56% obtained after oven drying the samples for 24 hours at 105℃ were used in this study. The areas investigated in the research include ore characterization, grindability, leaching time, particle size, dissolved oxygen and cyanide concentrations.

Based on gold fire assay method with AAS finish, the rock samples had an average head gold assay content of approximately 2.26 g/t and low content of minor elements rendering them suitable to be directly used in the recovery of gold from refractory gold ore and leaching process. Further, mineralogical association results showed that the gold mineralization of the rock samples is closely associated with quartz, hessite and pyrite and less associated with feldspar, other sulphides, chlorite and Illite. The rock samples had an approximately 18.22% free surface association and the gold-bearing mineralization located in the sample are of two types, that is, the native gold (primarily Au with Ag traces) and the less abundant, petzite. The native gold phase is host to most of the gold in the sample at 95.32 %, and petzite hosts lesser amounts of gold at 4.68%.

From gold grain size distribution analysis, the gold samples were observed to have a variable grain size distribution, with 25.4 mass% of the gold less than 7 μm, but approximately 35.2 mass% of the gold is in the 35-40 μm particle size range. Additionally, based on the grinding establishment results, increasing in milling time from 5 to 20 mins resulted into decrease in P80 grinding sizes from 270 to 70 μm. A four-point milling curve showed that the optimal milling time for P80 <75 μm and P80 <150 μm was 18.4 and 9 min, respectively. The gravity test work conducted via Knelson concentrator for the P80 <150 μm grind showed that the decrease in pulp density from 38% to 25% resulted in an increase in the gravity concentrate mass pull from 121.65g to 126.14g indicating that at lower pulp density the gold bearing particles migrate much easier to the concentrate slots. The gravity concentrate grades also showed a significant increase i.e. from 36.54

g/t to 39.71 g/t as the pulp density decreases from 38% to 25%, which suggests that at lower pulp density the concentrate is less contaminated.

Leaching tests showed that the overall gold recovery via the leaching process is affected by several parameters including, pulp density, particle size distribution, leaching time, dissolved oxygen and cyanide concentration. Lower pulp density resulted in higher gold concentrate mass pull and higher concentrate grades in a Knelson concentrator. A grind of P80 <150 µm showed the best leaching results indicating that it is the optimum grind for effective liberation for the ore under investigation. An increase in the cyanide concentration from 0.023 - 0.025% to 0.030 - 0.035% resulted in a decrease in the gold recovery, indicating that higher leaching agent concentration does not necessarily increase the recovery considering the cyanide concentration range tested. High cyanide concentrations can lead to the formation of cyanide complexes with various metals, some of which may not be economically recovered. This can reduce the amount of free cyanide available to dissolve gold, complicate the downstream recovery processes and increase operational cost.

Increase in dissolved oxygen (DO) from the range of 10-15 ppm to 15-20 ppm resulted in an increase in gold recovery, however, further increase in DO beyond 15-20 ppm led to approximately 30% decrease in gold recovery efficiency. This is because higher DO levels increase cyanide decomposition rate causing a decrease in cyanide content responsible for leaching operation and hence a decrease in leaching efficiency and gold recovery.

Finally, the cost analysis shows that the net operation revenue is positive,173 $ per ton of ore treated at the current gold price, which indicates that the treatment route is economically viable.

5.2 Recommendations

Generally, based on the results obtained in this research, it is recommended that for a cost-effective, optimal leaching and higher gold recovery, i.e., greater than 80%. The leaching process of the pyrite gold refractory ore samples from North Mara Gold Mine should be conducted for about 24 hrs, for rock sample milled to P80 <150 µm grind and concentrations that ranges from 15 to 20 ppm for dissolved oxygen and from 0.023 to 0.025% for cyanide.

This study scope didn't cover the flotation of the ore coupled with ultra fine grinding and pressure oxidation of the flotation concentrate in an autoclave or roasting as a treatment route to further

improve the overall recovery as gold grain size distribution analysis showed that 25.4% mass of the gold is less than 7 μm and can only be liberated and exposed for leaching through intensive milling.

The future research will have the potential to focus on exploring the financial viability of the flotation-ultra fine grinding-pressure oxidation under autoclave or roasting option knowing the current gold price trend that favors high-cost treatment routes.

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APPENDIX

AutoSEM Definitions

The TESCAN Integrated Mineral Analyzer (TIMA) is a scanning electron microscopy (SEM)-based automated mineralogy solution for the mining and minerals processing industries. However, its ability for rapid and automated analysis of the chemical composition and textures of almost every solid material makes it useful in many other applications like petrology, mineralogy, palaeontology, archaeology, the oil & gas industry, environmental sciences, recycling, forensic sciences etc. TIMA measures mineral or phase abundance, size-by-size liberation, mineral associations, and grain size automatically on multiple samples of grain mounts, thin sections, or polished sections. The TIMA was used in combination with X-ray diffraction (XRD) and geochemical analyses to identify and quantify the mineralogical characteristics of the samples.

• **Area %:** Particles and grains are exposed at the surface of a polished section as two-dimensional cross-sections. Any quantification of mineral characteristics is based on measurements, in pixels, of the exposed areas.

• **Mass (mineral):** If a statistical number of mineral grains is measured, the area of each mineral can be converted into mass by taking the SG of each mineral into account. Mass is generally displayed in two formats, namely absolute and fraction %. Absolute mass is governed by the mass flow of fractions (e.g., mass % of HLS fractions).

• **Association:** Association refers to adjacency. Two minerals are "associated" if a pixel of one of the minerals occurs adjacent to a pixel of the other mineral. In this report, the association considers both vertically and horizontally adjacent pixels.

• **Association Mineral %:** The number of pixels of a mineral type adjacent to the mineral of interest expressed as a percentage.

• **Grain and particle**: A grain consists of a single mineral type, whereas several grains make up a particle. In the case of a liberated grain, the terms grain and particle are equivalent.

In conclusion, the mineralogy of an ore significantly influences its leaching response. Understanding the mineralogical composition of an ore and how it interacts with the leaching

solution is crucial for optimizing the leaching process and maximizing metal recovery. Further research is needed to better understand these interactions and develop more efficient leaching methods.