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The Application of Process Mineralogy to Improve Gold Extraction from Wits Tailings

A dissertation submitted to the school of Engineering and the Built Environment at the University of Witwatersrand, Johannesburg, in fulfilment of the Master of Science in Engineering degree requirements.

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Submitted to

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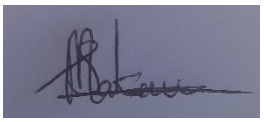
August 2022

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ABSTRACT

The extraction of gold from secondary resources, such as historical Witwatersrand (Wits) tailings, has been instrumental in extending the life of the South African gold industry. For a considerable period, Wits gold tailings were a liability to mining operations due to the cost of tailings dam management. If tailings are not adequately managed, they can have long-term negative effects on the environment and human health and safety. Pollutants from effluent and dust emissions can be hazardous to humans, animals, and plants. However, due to a decline in ore grades, the growing drive for zero waste and improvements in processing technologies, tailings retreatment is now a sustainable source of revenue for gold operations in South Africa.

Although the Wits gold tailings are characterised by low gold content, there is an expectation that residual gold can be efficiently and economically extracted, given the improvements in gold extraction technologies. However, low gold extraction efficiencies have typically been observed, with typical plant operations operating at 40-50% recoveries. Poor liberation is a common postulation for the causes of low gold extraction and slow leaching kinetics. As a result, local Wits tailings toll treatment plants apply ultra-fine grinding (P80 -20 µm) to partially liberate the gold and achieve economical gold extraction. However, an alternative hydrometallurgy approach that does not come with the capital and process implication of fine grinding can prove to be more economical. The study aimed to establish the cause of reduced gold extraction in a Wits tailings concentrate, followed by the establishment of a hydrometallurgy process route that improves gold extraction. The research objective was achieved through a process mineralogy study of a typical Wits pyrite concentrate sourced from DRDGold's ERGO flotation plant. The data from the process mineralogy study was then interpreted to develop a test work program that aimed to improve gold extraction from Wits pyrite concentrate.

The results of the study showed that gold in Wits tailings appears predominantly as native gold, which can easily leach with cyanide, with a negligible amount of electrum. The locking and reactive gangue minerals in the form of iron sulphides pyrite and pyrrhotite were noted to restrict cyanide access to the gold particle. Furthermore, these minerals tend to partially dissolve during cyanidation in an alkaline medium. Three options were identified to improve gold extraction, including the passivation of iron-sulphide mineral facades using a combination of pre-oxidation, lead nitrate addition to improve leaching kinetics and reduce cyanide consumption, and leaching at elevated dissolved O_2 concentrations with excess cyanide addition. The application of pre-oxidation (lime + air) with increased dissolved oxygen (DO) and cyanide (CN^-) increased gold extraction to 77% at a decreased retention time of 18 hours, which is 17% higher than gold extraction achieved during cyanidation of the same resource at DRDGold's ERGO plant. The addition of lead nitrate did not improve gold extraction but was beneficial in reducing cyanide consumption by 0.13 kg/t.

PUBLICATIONS AND PRESENTATIONS

Conference

S. Makamu and S. Ndlovu 2021, "A desktop study to understand the fundamental reasons for reduced gold extraction from gold tailings containing electrum", World Gold Hybrid Conference, 157-168.

Journal

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DEDICATION

Dedicated to

God and my family: Troy and Nokukhanya

Because you love and support me

Because you are my biggest cheerleaders

I can achieve my wildest dreams.

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GLOSSARY OF TERMS

Acid mine drainage ("AMD")	A type of waste that is produced when a mineral which contains sulphide is exposed to acid-generating microorganisms, oxygen and water
Ag	The chemical symbol for silver
Ag_2S	The chemical symbol for silver sulphide
Arsenopyrite	An iron arsenic sulphide (FeAsS)
Fire Assay	A chemical analysis that determines the proportion of certain elements (metallic and or non-metallic) in a sample
Au	The chemical symbol for gold
CAPEX	The amount of money an organisation or corporation spends to acquire, maintain, or upgrade its fixed assets, such as land, buildings, and equipment
Chalcopyrite	Minerals that contain an abundance of copper in the presence of iron and sulphide and are considered the most abundant copper ore mineral
CIL	Carbon-in-Leach is a technique used for the recovery of gold where the gold is exposed to carbon during cyanide leaching
CIP	Carbon-in-Pulp, a technique used for recovery of gold where the gold is exposed to carbon after cyanide leaching is complete
CN	Cyanide
CN	Direct cyanidation without carbon addition
Cu	The chemical symbol for copper
Cyanidation	The dissolution of gold from its ore deposit into solution using an alkaline weak cyanide solution
DO	Dissolved oxygen
Eh	The redox potential of a solution reflecting the activity of electrons
EPMA	Electron probe micro-analyser, used for in situ chemical analysis of small volumes of solid samples
Exposure Feed	The surface area fraction of the particle that is exposed, quantifies the amount of free-milling gold or gold that is accessible to the lixiviants for leaching
Extraction	An indication of the volume (in percentage) of valuable material obtained during the leaching of an ore deposit $\frac{Gold_{Feed}-Gold_{Residue}}{Gold_{Feed}} * 100$
Fe	The chemical symbol for iron
Feed	The original unaltered material used to supply a process or technique
Filtration	The process of separating usually valuable solid material from a liquid
Fire assay	An analytical technique used to measure the total gold contained in a solid material of a known mass. The technique comprises heating and the addition of a fluxing agent to fuse

	and collect the metal of interest (gold and precious metals). A spectroscopic method was used to analyse the collected metal of interest by weight
Flotation	A method of extracting precious minerals from their host rocks involves causing some mineral particles to attach to bubbles and float, while other particles sink. This allows the valuable minerals to be concentrated and separated from the host rock
g	gram
g/t	gram per tonne
Gangue	Valueless minerals that can be found with valuable minerals within an ore deposit
Grade	The mass or concentration of metal per volume of ore
Heap leaching	A gold extraction technology where ore is stacked into a heap on a waterproof pad and permeates a solution of lixiviants through the ore which is mixed with leaching reagents such as cyanide. The gold in the pregnant leach solution is recovered by carbon absorption or precipitation and the barren lixiviants are recycled back into the heap
ICP-OES	Inductively Coupled Plasma Optical Emission spectroscopy measures the concentration of elements in a sample by transferring electrons from a ground state to an excited state
Iron-sulphide minerals	Refers to the minerals Fe_3S_4 and Fe_7S_8, which are magnetic minerals, which contain iron and sulphur and are magnetic. They can host invisible gold within their matrix
kg	kilogram
kg/t	kilogram per tonne
Leach	The process of dissolving a metal of interest from its solid form to a solution state using lixiviants known to solubilise the said metal
Liberation	The mineral's fraction of the host particle's area. Physical separation processes like gravity concentration including liberation
mg/L	milligram per litre
Mill	A technology used to crush and ground ore to liberate metals of commercial value, making them amenable to physical and chemical treatment to recover the valuable metals or minerals
Ore	A naturally occurring collection of minerals at a specified location which may be unearthed for revenue purposes, from which some parts may be cost-effectively separated
OPEX	A recurring cost associated with the operation of a product, enterprise, or system
P80	80% passing
$Pb(NO_3)_2$	Lead Nitrate
pH	The scale used to specify how acidic a solution is and reflects the activity of H^+
PLS	Pregnant leach solution

Preg- robbing	The process of extracting gold from pregnant leach solutions by adsorbing gold-cyanide complexes onto various components of the ore
Pyrite	A naturally occurring iron disulphide mineral chemical formula FeS_2 (iron (II) disulphide)
Pyrrhotite	A naturally occurring iron-sulphide mineral chemical Fe(1-x)S (x = 0 to 0.2). It is a solid-state chemistry variant of FeS_2. Pyrrhotite is magnetic and its colour is similar to pyrite and is weakly magnetic
Reef	A metal-containing mineral deposit (gold) within a geographical zone
Refractory	The resistance to a process. In the context of gold processing, refers to a mineral that is unaffected by common hydrometallurgical processes for the extraction of gold
Residue	The material remaining after a process, commonly leaching, has been applied to the feed
S°	The chemical symbol for elemental sulphur
S^{-2}	The chemical symbol for sulphide
SCN	Thiocyanate
SO_4	The chemical symbol for sulphate
Tailings	The residue that remnants after all metals or minerals of economic value have been removed from the ore during processing
TEB	Triethylborane
Total S	The chemical symbol for total sulphur
UFG	Ultra-fine grinding
Wits	Wits (Wits)
XRD	X-Ray Powder Diffraction is an investigative method used to identify mineral phases
XRF	X-Ray Fluorescence, is a quantitative investigative technique used to determine metals in high
Zn	The chemical symbol for zinc

CHAPTER 1

1. INTRODUCTION

1.1 Contextual

The extraction of gold from secondary resources, such as historical Witwatersrand (Wits) tailings, has been vital in extending the life of the South African gold industry. For a considerable period, Wits gold tailings have been a liability to mining operations and a source of pollution for environmentalists. However, due to a decline in ore grades, the growing drive for zero waste and improvements in technology, tailings retreatment is now a sustainable source of revenue for gold operations in South Africa, as gold extraction technologies have improved.

Gold tailings in the South African Witwatersrand basin are noted to contain an average of 0.5 g/t of gold (Maseko 2017; Janse van Rensburg 2016). The presence of soluble gold in the Wits tailings could be due to the inefficiency of the original process used and possible changes in the structure of the gangue minerals due to their storage conditions that foster gold extractions of previously locked gold. However, most plants that treat Wits tailings have observed lower-than-expected gold recoveries, with actual plant operations reporting 40–50% recoveries (Janse van Rensburg 2016; Staden et al. 2020; Pillay et al. 2017; Teimouri et al. 2020). Direct cyanidation leach tests done under laboratory conditions on several tailings samples from different locations in the Wits area have, on the other hand, indicated that up to 60% gold is extractable from this material (Janse van Rensburg 2016; Mbayo, Simonsen, and Ndlovu 2019).

Widespread studies have been conducted on the mineralogy of gold ores from the Wits deposits, resulting in a significant understanding of the gold extraction process. However, inadequate studies have been done on the impact of mineral structure on the gold extraction from Wits tailings. Poor liberation is commonly considered the cause of low gold extractions and slow-leaching kinetics. For instance, a pyritic gold concentrate at a local tailings toll treatment plant requires ultra-fine grinding (P80 – 20 µm) to partially liberate the gold and improve gold extraction (Staden et al. 2020). Similar observations were made on a tailings sample from the West Rand, which showed that ultra-fine grinding increased gold extraction by 20% (Pillay et al. 2017). The gold tailings of Wits gold processing plants are already fine,

meaning further milling would be expensive if only a 20% improvement in extraction is achieved. Additionally, the substantial increase in the surface capacity of the sulphide minerals created by further milling amplifies all preg-borrowing effects and promotes the formation of thiocyanate (SCN^-) by free cyanide, resulting in high reagent consumption. Therefore, an alternative hydrometallurgy approach that does not come with the capital and process implications of fine grinding is required to increase gold extraction economically.

South African gold processing plants were designed to process whole ores in the early years. The Witwatersrand Gold Basin has overall mineralogy that is uniform in nature; however, the origin and construction of the dumps, and their storage and atmospheric conditions became variable over the years (Rensburg 2016). Tailings are typically stored in the open atmosphere where some bacteria are present. Such storage approaches allow bacteria to penetrate through the dumps leading to the oxidation of the pyrite minerals contained in the dumps over time (Lloyd 2007). Because of their sulphide content, pyrrhotite and pyrite can dissolve when exposed to acidic conditions to generate Fe^{2+} and S, which in turn can be oxidised to ferric and sulphate ions. The bacteria catalyse the oxidation reactions and the formation of ferric and sulphate (Belzile et al. 2004) in a process called bioleaching. The bacterial oxidation of pyrite and pyrrhotite occurs via two different mechanisms referred to as the indirect contact and direct contact mechanisms, of which the indirect is more acceptable (Silverman 1967). According to the indirect contact mechanism, the first step is the chemical oxidation of pyrrhotite or pyrite to generate ferrous (Fe^{2+}). This is then followed by bacterial oxidation of Fe^{2+} to ferric (Fe^{3+}) and then the reactions become self-sustaining. Consequently, the ferric iron oxidises additional pyrrhotite and pyrite creating a more acidic system. The further oxidation of Fe^{2+} during pyrite oxidation can also lead to the precipitation of ferric hydroxide ($FeOH_2$) in highly acidic conditions. These reactions continue to occur and are attributable to the high content of ferric ions in the solution, controlled by the low pH (Belzile et al. 2004).

The mineralogy of Witwatersrand tailings is characterised by the presence of iron-sulphide minerals, which generally increase in abundance with depth (). The mineralogical composition of the tailings differs from that of the whole ore due to the processing methods and storage conditions. Unlike in Wits whole ores, gold in Wits tailings pyrite concentrate is associated with partially soluble iron-sulphide minerals. The change in mineralogy is attributed to the leaching process used to extract gold from the whole ore, which generated the tailings, and the

subsequent storage conditions. Weathering and potential bio-oxidation processes that result from the exposure of gold mine tailings generally liberate gold from iron sulphides, such as pyrite, arsenopyrite and pyrrhotite (Barcelos et al. 2020; Xu and Wang 2015). The presence of these minerals necessitates upfront pre-treatment of the ore and tailings, optimised leaching conditions such as increased dissolved oxygen levels, excess "free" cyanide concentration, and lime addition to maintain pH to compensate for the soluble gangue minerals that may compete with gold during cyanidation (Breuer, Jeffrey, and Hewitt 2008; Rees 2000).

The Wits Gold Basin comprises several reefs, as shown in Table 1.1. Historical analysis of these reefs revealed that silver was present in all the reefs (Table 1.2). The silver content from most of the reefs was less than 25%, classifying the gold in the Wits Gold Basin as native gold rather than electrum (>25% silver content).

Table 1.1. Major producing reefs of the Witwatersrand Gold Basin (Janisch 1986)

Goldfields	Central Rand Group Formation	Major Producing Reefs
Orange Free State	Bird	Basal Reef, Leader Reef
	Kimberly	A and B Reef
		Dominion Reef
Klerksdorp	Bird	Ventersdorp Contact, Reef, Black Reef,
West Wits Line	Main	Carbon Leader, Main Reef
	Kimberly	Kimberly Reef, South Reef
	Elsburg	Elsburg Reefs, Ventersdorp contact, Reef
West Rand	Main	Main Reef, South Reef
	Bird	Bird Reef, White Reef, Monarch Reef
Central Rand	Elsburg	Elsburg Reef
	Main	North Reef, Main Reef, Main Reef Leader, South Reef
East Rand and Heidelberg	Main	Main Reef Leader
	Kimberly	Kimberly Reef, May Reef, Black Reef
South Rand Evander	Kimberly	Kimberly Reef

Table 1.2. Content of gold and silver grains in Witwatersrand Gold Basin reefs (Janisch 1986)

	Gold	Silver
Dominion	80-87	1-15
Government (Babrasco)	77	16
Main (City Deep)	82-94	4-12
Main Reef Leader (Geduld)	66-70	27-32
Carbon Leader (Blyvooruitzicht)	83-93	8-10
Vaal (Hartebeestfontein)	78-86	9-11
Basal (Loraine)	78-90	9-12
Monarch (Randfontein)	91-99	1-12
Elsburg (Loraine)	72-97	8-11
VCR (Vaal Reefs)	67-93	7-12
Black Reef (Geduld)	81	20

The dissolution of gold in an alkaline cyanide medium is known to result in silver dissolving at half the rate of gold (Habashi 1967). Therefore, tailings from gold-silver reefs may contain a higher silver-to-gold ratio than that of the whole ores, which may affect the leachability of the tailings when using a conventional cyanidation circuit (Khalid, Larachi, and Adnot 2018; Senanayake 2006; Senanayake 2008).

Gold-silver ores that are processed in an alkaline cyanide medium are typically designed for ores with low silver content (House and Marsden 2009). However, it is expected that the Wits tailings currently being processed in South African CIL/CIP plants are complex and contain a higher amount of silver than whole ores due to the reasons explained earlier. The expected higher ratio of silver to gold in these tailings can influence gold extraction during cyanidation if the plant is not designed for such conditions. Marsden and House (2009) described the leaching of gold and the behaviour of silver where the silver-to-gold ratio is significantly high. In a scenario where an ore body has a high volume of silver and the CIP circuit is not designed to account for the increased silver, the silver may decrease process kinetics and gold yield (House and Marsden 2009).

However, Ritchie and Jeffrey (2000) found that silver (5%) in the solid phase could enhance the kinetics of the gold oxidation half-reaction and the oxygen reduction half-reaction using a

rotating electrochemical lechatelierite mineral microbalance. They found that through the breaking of the AuCN coating, the Au-oxidation half-reaction is enhanced, while the O_2-reduction half-reaction is enhanced by a bimetallic corrosion mechanism when oxygen is favourably reduced at the active silver sites. This effect was found to significantly enhance gold leaching rates (Ritchie and Jeffrey 2000). Likewise, Wadsworth and Zhu (2003) determined through their research using a pure gold rotating disk electrode that similar effects of accelerated gold leaching were found with solid silver, although not to the same extent as with dissolved silver ions in the solution phase. However, it is important to note that both tests were conducted using a pure gold-silver alloy with no side reactions from other minerals present in gold ores (Wadsworth and Zhu 2003).

Deschênes et al. (2009) have observed that native gold and electrum from Wits pyrite tailings can easily be leached with cyanide. However, the presence of reactive gangue minerals or locked particles may result in poor extraction and high reagent consumption. In the leaching process of an orebody containing soluble gold, silver, base metals, and sulphides, the following scenarios are possible:

- Iron-sulphide minerals and base metals in the ore body may dissolve and become readily available in solution to consume lime, oxygen, and cyanide.
- Silver may leach together with gold.
- A reaction can occur between the leached silver and sulphides in the solution, resulting in the formation of a silver sulphide layer.

To maximise gold recovery, it is crucial to optimise the leaching conditions, which requires a thorough understanding of the mineralogical properties of the tailings. These properties include the size of gold particles, reagent consumers, preg-robbing minerals, mineral liberation, and association with gold grains (Chivers et al. 1998; Zhou et al. 2004).

This study critically characterises a typical Wits pyrite concentrate obtained from the DRG ERGO tailing retreatment plant using metallurgical, chemical, and mineralogical techniques. The study collates the results with the available literature. It is important to note that the ERGO flotation process involves the flotation of pyrite, and thus the sample used in the study represents only a potentially recoverable pyrite fraction of Wits tailings that were previously

treated at the DRD ERGO tailings retreatment plant. Therefore, the study's limitation is that the sample is not a representation of typical Wits tails.

The study uses a mineralogical characterisation and hydrometallurgical test work approach to confirm the hypothesis and establish a technique to improve the dissolution of gold from Wits tailings. The dissertation presents the results of gold mineral composition analysis, mineral association, and gold occurrence. The information is then used to investigate the cause of low gold extraction and develop suitable pre-treatment options and leaching conditions to address the adverse effects of silver and iron-sulphide minerals during cyanidation.

1.2 Problem statement

Passivation, inhibition, and a retarding effect can occur during the cyanidation of gold-containing material due to various factors such as gold particle size, liberation, reagent consuming, and preg-robbing minerals (Rabieh et al. 2017). Although several theories exist regarding the cause of poor gold extraction from the ERGO pyrite concentrate, none have been significantly proven, leading to limited information on the gold leaching behaviour of the concentrate and the reasons for low recovery.

Iron-sulphide minerals, which are concentrated in a pyrite concentrate, can affect gold extraction during cyanidation by locking up gold if not oxidised or partially dissolved to form sulphides and sulphates (Problems 2015). When the iron-sulphide minerals are partially soluble or oxidised, they dissolve and become readily available in solution to react with other elements, forming passivation layers. In the presence of sulphide ions, gold-silver alloys and base metals can dissolve easily (Zhou et al. 2004). Therefore, a detailed characterisation of the tailings material is critical to understanding the limitations that may arise during the leaching process and subsequently developing a process route that can lead to the enhanced dissolution of gold.

1.3 Research objectives

The objective of this study is to investigate the cause of reduced gold extraction observed during cyanide leaching of a Wits pyrite concentrate from ERGO and to develop pre-treatment options and processing conditions that can help maximise gold leaching. The study aims to achieve this objective through the following:

- Process mineralogy of the pyrite concentrate to establish the cause of reduced gold extraction.
- Improvement of gold extraction from Wits pyrite concentrate based on the mineralogical findings.

1.4 Research questions

Gold is typically linked with silver; however, the silver content, iron-sulphide mineral grade, and their association determine the impact – either negative or positive – on gold dissolution. This proposed research aims to address the following questions:

1. What is the content and association of silver and gangue minerals (iron sulphides) in Wits tailings?
2. What is the reactivity and role of the gangue minerals during the dissolution of gold from the Wits tailings?
3. Can pre-treating the sample by reacting the iron-sulphide minerals with lime and oxygen improve gold extraction during cyanidation?
4. Can the addition of lead nitrate during cyanidation reduce reagent consumption and improve gold dissolution kinetics?

Hypothesis

The association and reactivity of iron-sulphide minerals and/or silver limit the accessibility of cyanide to gold particles, which causes a reduction in gold extraction during the cyanidation of Wits pyrite concentrate. While Wits gold occurs as native and electrum and can be easily leached with cyanide, the presence of reactive gangue material associated with gold and the silver content of electrum > 25% can result in poor extraction and high reagent consumption.

This project challenges the conventional wisdom that ultra-fine grinding (UFG) is necessary to achieve improved gold recovery in Wits tailings. Since the tailings produced by cyanidation are already fine, UGF may not be required to enhance gold extraction. Silver and iron-sulphide minerals are often present in Wits pyrite ores; however, the grades of these minerals and their associations determine their effect on gold extraction, whether negative or positive. To identify the reasons for the inefficiency of the gold extraction process, process mineralogy methods were employed, as gold deportment is one of the determining factors of gold extraction efficiency.

The process mineralogy analysis revealed and characterised the gold-bearing particles, and identified the type of gold, particle size, and manner of occurrence, providing an in-depth understanding of the mineralogical occurrence of gold. The mineralogical and chemical characterisation of the tailings facilitated the selection of a pathway to increase gold extraction efficiency during the tailings reprocessing. Consequently, tests, including pre-oxidation cyanidation, were performed to identify the most beneficial environment for improving gold extraction. The investigation aimed to establish a relationship between the test parameters and the rate of gold disintegration.

1.5 Thesis outline

The thesis is structured as follows:

Chapter 2 presents the theoretical background on the formation of gold tailings, extractive metallurgy of gold, and the chemistry of cyanidation. The chapter focuses on gold cyanidation, mineralogy, refractory ores, and factors that affect gold extraction. It highlights process mineralogy literature that is relevant to the study, such as the content of additional valuable and detrimental minerals, gold ore grade, ore structure, gold particle size distribution, mineral nature, and liberation. Additionally, the chapter examines the impact of electrum and sulphide minerals on gold extraction, as well as the history and current Wits tailings retreatment projects.

Chapter 3 describes the primary experimental work conducted to achieve the research objectives. It outlines the sampling technique, preparation, and tools used to characterise the Wits pyrite tailings. Furthermore, this chapter covers the cyanidation gold extraction improvement test work that was carried out. The chapter provides essential background information to comprehend the outcomes of the experimental work.

Chapters 4 and 5 present the results and discussions of the experimental work. The chapters explore the mineralogical characterisation process and determine the root cause of poor gold extraction from Wits pyrite tailings, based on literature and findings from the experimental test work.

Chapter 6 concludes the study and suggests a process flow sheet for enhanced gold extraction in a CIP circuit. The chapter also recommends additional studies to further improve gold extraction from Wits pyrite tailings.

CHAPTER 2

2. LITERATURE REVIEW

2.1 Introduction

In South Africa, gold mines have progressively become deeper to access mineral resources, resulting in reduced virgin gold ore production and excess processing capacity at many gold processing plants due to the high costs associated with mining at greater depths (Janse van Rensburg 2016). To address this, several gold processing plants in South Africa have initiated tailings treatment projects to supplement the feed supply for economic viability. While some plants, such as DRD Gold, are processing gold tailings as a secondary source for financial benefits, others, such as Basadi Ba Afrika, are reprocessing gold tailings for environmental reasons. However, these operations that treat Wits deposits face challenges in optimising the extraction of gold from these tailings.

The previous chapter provided an introduction and background to the study. This chapter provides a historical overview of tailings, the emergence of tailings reprocessing, and the challenges faced by reprocessing companies in the industry. The literature review also discusses the fundamental principles of gold extraction, with a particular emphasis on cyanidation and process mineralogy tools that can be utilised to establish cyanidation conditions for improved gold extraction from a Wits pyrite concentrate.

2.2 Process mineralogy as a tool for characterisation

In order to comprehend the gold ore processing requirements, it is crucial to describe the key mineralogical characteristics of the ore, typically in quantitative and qualitative terms (Marsden and House 2006). Various methods have been developed and applied to examine gold forensically. A conventional process mineralogical investigation of gold ore may comprise some, if not all, of the following objectives (Zhou et al. 2004):

1. Determining the gold and silver content through fire assay and analysis of base metals, carbon, and sulphur species.
2. Establishing the total visible gold, which includes identifying and localising all gold minerals, their size distribution, composition, liberation properties, and mineral interactions (Çelik et al. 2018; Zhou et al. 2004).

3. Determining the fractions of liberated gold associated with sulphides and oxides/silicates, and gold associated with carbonaceous material.
4. Measuring sub-microscopic gold in sulphide and other minerals.
5. Determining the amount, nature, and distribution of any additional valuable minerals (such as silver and copper) and detrimental minerals (such as talc, serpentine, graphite, cyanicides, oxygen consumers, and water-soluble minerals).
   Evaluation of the preg-robbing capability of carbonaceous components and gangue minerals (Zhou et al. 2004).

To determine the compositional characteristics of a resource, statistically reliable mass samples must be taken from locations in the ore body that represent the entire deposit. Sampling is a crucial step in conducting a process mineralogy study to establish the gold processing flowsheet, but it is particularly vital when dealing with ore or concentrate that contains coarse gold particles or a high concentration of gold-bearing sulphides. The uneven spatial distribution of coarse gold often referred to as the "nugget effect", and the higher density of sulphide minerals account for this phenomenon. Samples that are not representative introduce an element of error into the gold assays and the mineralogical gold balance. It is necessary to use advanced instrumental techniques capable of analysing the samples' textural properties for characterisation. Therefore, comprehensive process mineralogical research is essential for gold ore processing, which has the potential to be beneficial in terms of cost reduction and increased efficiency in the process (Zhou et al. 2004).

Conventional and advanced mineralogical techniques, as demonstrated in Figure 2-1, were employed to ascertain the silver and iron-sulphide mineral content and association in Wits tailings, thereby establishing their reactivity and role during cyanidation. The use of these tools also enabled the determination of the occurrence of silver and gangue minerals and their influence on gold dissolution efficiencies during alkaline cyanide leaching of Wits pyrite concentrate.

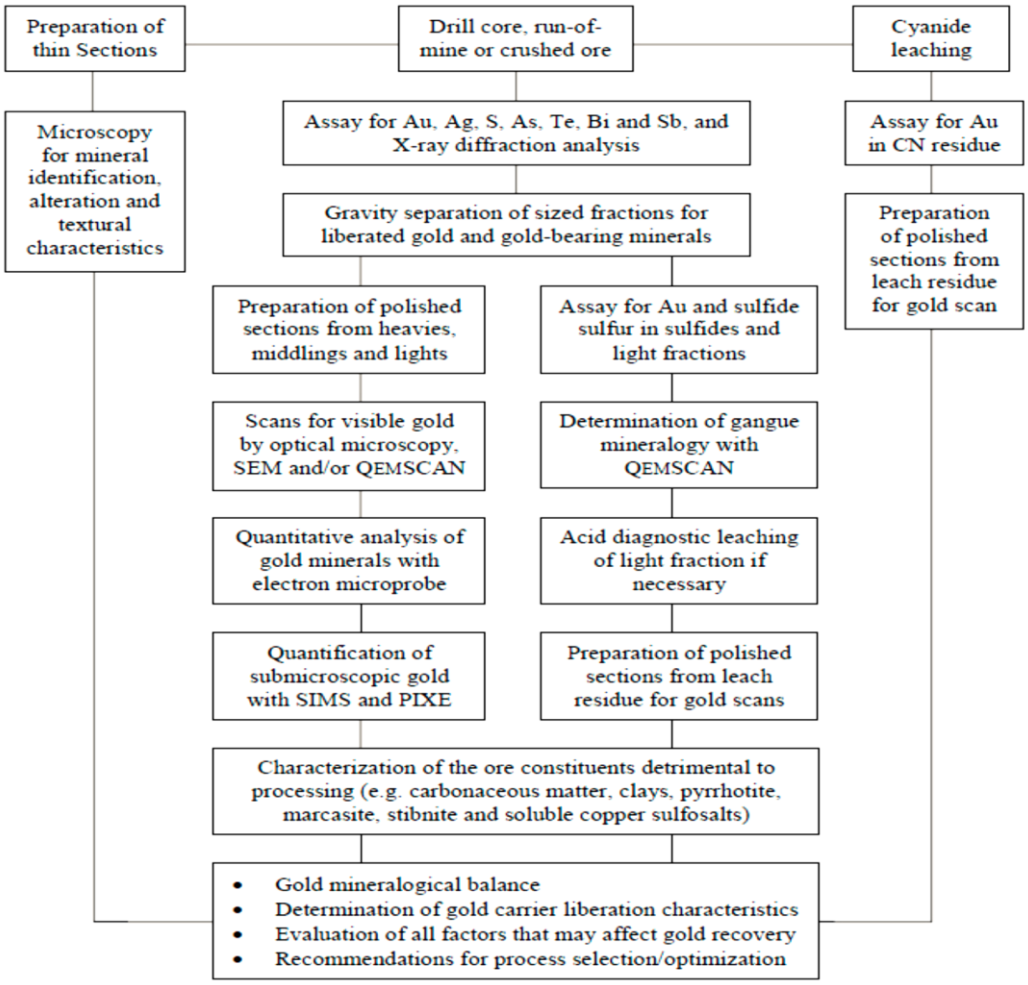


Figure 2-1. All-purpose investigative technique for gold studies (Zhou and Cabri 2004)

2.3 Gold mineralogy and classification

The average gold content in the earth's crust is 0.005 g/t. Due to the low concentration of gold in primary rocks, comminution procedures for ore feed must typically involve an upgrade of 3000-4000 times the initial concentration to achieve commercial concentrations (Chivers et al. 1998; Wai, Eugene, and Mujumdar 2009). Gold exhibits a high affinity for metals but a low affinity for oxygen and sulphur; thus, it is typically found in the metallic or sulphidic phases that develop after magma cools rather than the silicates that form earlier (Sousa et al. 2018; Wai, Eugene, and Mujumdar 2009).

Gold, unlike most other elements, exists in a variety of mineral forms that make it amenable to selective extraction from mineral combinations within many distinct mineral associations in an ore matrix. Understanding the distribution and mineral association of gold is essential in establishing an efficient processing regime. Gold is frequently alloyed with silver at

concentrations of up to 15%, but it is most commonly found in its elemental form (Harris 1990). Other elements found in gold alloys include electrum, tellurium, selenium, bismuth, mercury, copper, iron, rhodium, and platinum. Typically associated with gold are oxides, silicates, carbonates, sulphates, and sulphides (Venter, Chryssoulis, and Mulpeter 2004).

Gold ores are classified into two primary categories: "free-milling" and refractory ores. "Free-milling" ores are those where more than 90% of the gold is extractable by the conventional cyanidation method when the ore is milled to a size of 80% < 75 µm, without excessively high reagent consumptions (Marsden and House 2006; Vaughan 2004; Zhou et al. 2004). "Free-milling" gold is recovered through gravity concentration and amalgamation, and the processing requirements are not substantially different depending on the gangue mineral composition. In contrast, refractory gold ores result in poor gold recoveries (<90%) or require extensive additional reagent usage or sophisticated pre-treatment procedures to achieve adequate gold recoveries (Swash 1988; Zhou et al. 2004).

2.4 Refractory gold ores

The gold production industry has faced a recent depletion of free-milling ores and an increase in more challenging "refractory sulphide ores". The causes of refractory gold deposits can be attributed to several factors, including the following (Wai, Eugene, and Mujumdar 2009):

- Gold is extremely finely dispersed in ore or a solid solution sulphide matrix.
- Gold is found in association with tellurides.
- Gold values are present in the sulphides of basic metals such as lead, copper, and zinc.
- Gold may be found in carbonaceous ores.
- Silver is finely disseminated in the gangue matrix, predominantly formed of quartz, or complexed with manganese oxides.

Fine grinding of the ore can achieve the liberation of macroscopic gold particles from their encapsulation of sulphide minerals. However, some sulphidic gold ores cannot be crushed finely enough to expose the sub-microscopic gold particles (Wai, Eugene, and Mujumdar 2009). When this occurs, gold is referred to as "invisible gold", which is typically found in arsenopyrite or arsenic-rich pyrite (Harris 1990).

Direct contact between the cyanide solution and the gold particles is necessary for cyanide to dissolve gold (Wai, Eugene, and Mujumdar 2009). Early studies on the process of dissolving gold in a cyanide solution in the presence of sulphide minerals found that the inclusion of heavy metals, such as copper, iron, and zinc, significantly increased the amount of cyanide and oxygen required for the process (Karimi et al. 2010; Kim and Ghahreman 2019). Furthermore, the sulphide component has been shown to have a strong impact on the gold leaching kinetics (Deschênes et al. 1998). The leaching characteristic of gold in the presence of sulphide minerals is highly dependent on the solubility of the sulphides and the concentration of oxygen in the solution (Kondos, Deschênes, and Morrison 1995; Liu and Yen 1995; Rees 2000).

"Extraction of gold from refractory ores" using standard hydrometallurgical processing techniques is often enhanced by oxidative pre-treatment of the ore (Rees 2000). The available methods for oxidation can be either hydrometallurgical or pyrometallurgical (Afenya 1991). Pyrometallurgical oxidation of sulphidic and carbonaceous ores by roasting, which has been widely practised for decades, is well established and demonstrated. However, the increasing stringency of emissions legislation and environmental protection controls has amplified the difficulty and cost of roasting. This has made it even more important to understand the metallurgical problems associated with refractory gold ores and develop economically viable extraction techniques.

Although cyanide can dissolve gold in carbonaceous ores, the active carbon in the ore rapidly adsorbs it back. To overcome this challenge and preferentially absorb gold, the treatment of carbonaceous gold ores typically involves chlorination, which deactivates the carbon, roasting, which burns away the carbon, and carbon-in-leach (CIL). This process introduces high-activity carbon that can be easily separated from the leach slurry after loading.

In the treatment of sulphidic gold minerals, the primary objective of pre-treatment is to remove a sufficient amount of sulphide to ensure that a small fraction of all gold particles become immediately accessible. This makes the gold particles vulnerable to cyanidation. The pre-treatment methods available today produce water-soluble sulphates or sulphur dioxide through sulphur oxidation. The sulphur oxidation process aims to eliminate any sulphur-related factors that could result in increased consumption of cyanide (Wai, Eugene, and Mujumdar 2009). The primary sulphur oxidation processes include the following:

- ***Pressure oxidation*:** This method employs oxygen, heat, and pressure within a liquid medium to effect sulphur oxidation through a well-designed chemical process. Reactors are typically carried out in high-pressure autoclaves, and reactor-operating conditions may be adjusted to be either alkaline or acidic depending on the process requirements. The mass transfer needs to determine the required overpressure, and excessive overpressures tend to increase reaction rates and reduce oxidation duration (Wai, Eugene, and Mujumdar 2009).
- ***Roasting*:** Roasting involves heating the ore/concentrate below the fusion point in the presence of air, oxygen, water vapour, carbon, sulphur, or chlorine to produce the desired chemical and/or physical changes that will enhance downstream processing. The process was once a standard technique for sulphide oxidation, but strict environmental regulations have limited its use in recent years. Gas scrubbing systems used in modern roasting operations are usually complex and produce sulphuric acid as a by-product (Wai, Eugene, and Mujumdar 2009). However, the use of roasting as a pre-treatment method remains an environmental and safety concern, particularly for high-arsenic-containing gold ores, owing to the production of highly toxic arsenic trioxide gas (Strauss et al. 2021).
- ***Bio-oxidation*:** Bio-oxidation is a more eco-friendly alternative pre-treatment approach to roasting refractory concentrates. In this process, acidophilic chemolithotrophic bacteria remove the sulphur component of the ore. The most common method of biologically oxidising sulphide minerals involves using Thiobacillus Ferrooxidans at optimal pH and temperature, which are pH 2 and 30 degrees Celsius. This method's advantages include the use of air rather than oxygen and the maintenance of standard atmospheric operating conditions. However, bacterial oxidation is slow, and as a result, residence times are considerably longer (Wai, Eugene, and Mujumdar 2009).
- ***Chemical oxidation*:** A technique that involves using strong acids such as nitric acid at normal atmospheric pressure and temperature has also seen limited application (Wai, Eugene, and Mujumdar 2009).

2.5 Gangue minerals and their impact on the gold extraction process

The most effective route for gold extraction processing is closely related to the inherent mineralogical features of the gold-processed ore. The mineral assemblage determines the performance of all chemical and physical processes involved in gold extraction (Sousa et al. 2018; Coetzee et al. 2011). Therefore, precise characterisation of the mineralogical nature of the ore is crucial, including the characterisation of the precious metal phases (gold deportment) and gangue minerals.

During cyanidation, newly formed precipitates can encapsulate gold grains, limiting their exposure to the cyanide solution. The possibility of the formation of these so-called "passivation rims" is linked to the presence of antimony/arsenic sulphide and pyrrhotite in the ore (Azizi, Petre, and Larachi 2012). Passivation primarily affects gold recovery during flotation and cyanidation. The passivation layer can take the form of iron oxides or hydroxides (such as limonite or goethite) during oxidation, disintegration, and precipitation processes (Zhou et al. 2004). Additionally, a reaction can occur between sulphide ions and gold to form indissoluble aurous sulphide coatings (Liu and Yen 1995). Silver may occur as an individual mineral or alloyed with gold as electrum or kustelite. Silver-rich minerals, such as electrum and kustelite, quickly become stained in the presence of oxygen and sulphide ions, creating a silver sulphide layer that is one to two microns thick, limiting the accessibility of the cyanide solution to the mineral (Zhou et al. 2004).

Gold ores also tend to contain minerals that react in a cyanide solution and negatively affect the rate or extent of gold dissolution by consuming oxygen and cyanide. Iron sulphides, arsenic, antimony, copper, zinc, and tellurium are the most common cyanide-consuming minerals (Lee 2007).

Gangue minerals in the extractive metallurgy of gold are also known to cause the following problems during processing:

- Quartz and silicate minerals are essentially inert. However, under strongly acidic conditions, these minerals can dissolve to some extent, forming a jelly-like silicate that may cover exposed gold surfaces, making solid-liquid separation of the slurry challenging. Chlorite is an example of such a mineral; it is soluble in strong acid conditions and forms a slurry residue that is difficult to filter.

- Carbonates consume a considerable amount of acid due to their ease of dissolution in sulphuric acid. For instance, the presence of 2% limestone in an ore necessitates the use of 20 kg H_2SO_4 per tonne of ore. The sulphates generated due to carbonate decomposition may cause downstream difficulties such as higher slurry viscosities, scale build-up in pipelines and reactors, and coatings on exposed gold surfaces (Schwartz 1944; Zhou et al. 2004).
- Adsorption of dissolved gold from the pregnant leach solution may occur in the presence of carbonaceous matter, pyrophyllite, and shale in ores, reducing gold extraction (Zhou and Cabri 2004).

2.6 Extractive metallurgy of gold

In ancient times, gold was easily obtainable in its native state as nuggets through panning, manual sorting, and other direct mining practices. In the early 19th century, a surge in demand for gold led to a rush for the precious metal, and comprehensive knowledge of mining and metallurgical methods for gold recovery began to develop (Marsden and House 2006). Gravity concentration and amalgamation became the most developed and commonly used gold recovery methods (Sousa et al. 2018). These techniques treated a wide range of gold ores on a larger scale.

Over the years, there have been many advances in gravity concentration and amalgamation. However, these methods are not suitable for recovering fine gold and gold associated with sulphide minerals (Whiteman, Lotter, and Amos 2016). These shortcomings prompted the exploration of more efficient hydrometallurgical processes for gold recovery. Figure 2-2 depicts a basic flow diagram for the gold extraction process from its ore.

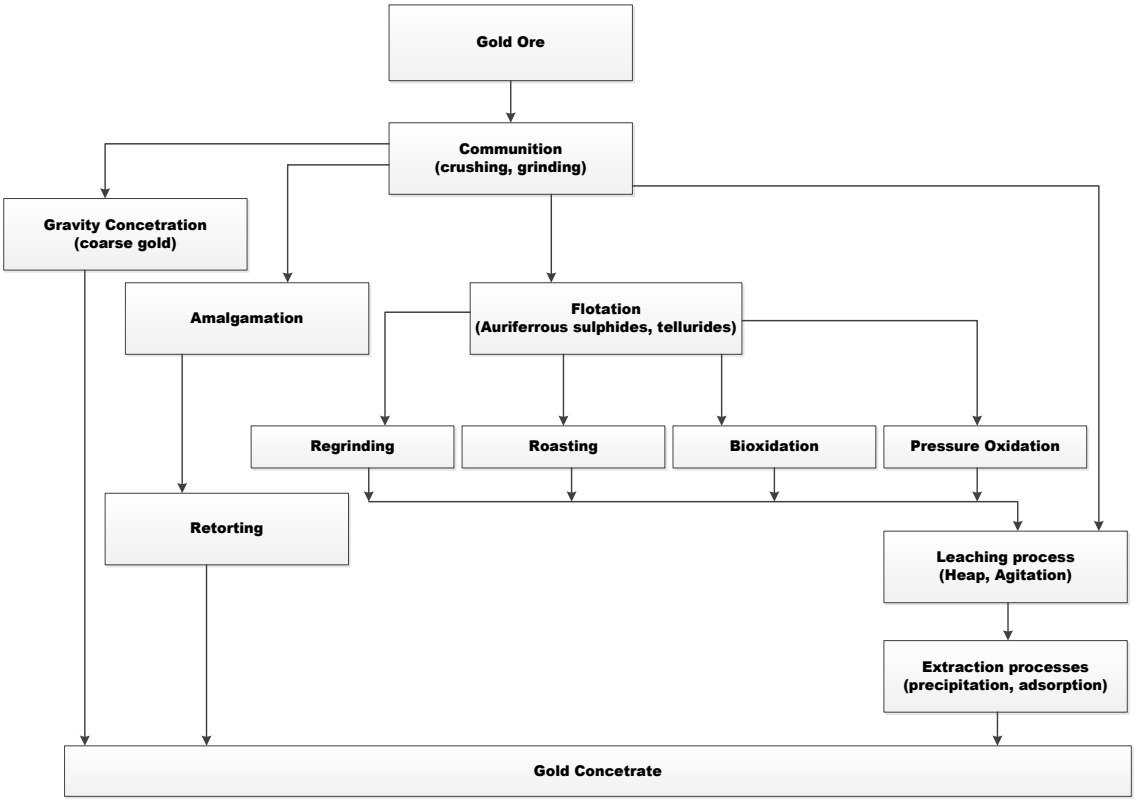


Figure 2-2. Basic flow diagram for gold recovery (Tran 1992)

The main objective of gold leaching is the recovery of gold from the rest of the ore by dissolution in an aqueous solution. To achieve acceptable leaching rates, this process requires both a complexing agent and an oxidant (Liu and Yen 1995). In gold extraction, only a limited number of ligands form complexes with sufficient stability for use in the extraction process, and cyanide is one of these. Cyanide is a universal, low-cost, and well-understood gold complexant that forms a stable gold complex.

2.7 Cyanidation effectiveness in dissolving gold

The most common oxidant in cyanide leaching is oxygen from the air, which makes the process even more desirable (Sandra Ellis and Senanayake 2004).

The dissolution of gold in aerated cyanide solutions and the role of oxygen in the chemical mechanism were first studied by Elsner (Nicol, Fleming, and Paul 1992; Tran 1992) and described according to the equilibria shown in Equation (2-1):

$$4Au + 8CN^- + O_2 + 2H_2O \leftrightarrow 4Au(CN)_2^- + 4OH^- \quad \textbf{(2-1)}$$

While the stoichiometric equation correctly represents the reaction, it does not fully explain the cathodic and anodic reactions involved in gold dissolution (Cathro and Koch 1964). The overall reactions for gold dissolution in an aerated, alkaline cyanide solution involve both anodic (Equation (2-2)) and cathodic (Equation (2-3)) half-reactions occurring on the gold surface.

$$2Au + 4CN^- \leftrightarrow 2Au(CN)_2^- + 2e^- \quad \textbf{(2-2)}$$

$$2O_2 + 2H_2O + 2e^- \leftrightarrow H_2O_2 + 2OH^- \quad \textbf{(2-3)}$$

The reactions involved in cyanidation occur across three phases: solid (metallic gold), liquid (water, cyanide ions), and gas (oxygen). The diffusion of cyanide ions and gaseous oxygen from the solution to the metallic gold surface controls the rate of cyanidation. The anodic reaction is complemented by the reduction of oxygen in the cathodic area. However, the cathodic reaction is not as straightforward as presented in Equation (2-4), as the mechanism of oxygen reduction involves several parallel reactions. Hydrogen peroxide is formed as an intermediate product, which subsequently decomposes to form oxygen and water, as shown below.

$$H_2O_2 \leftrightarrow O_2 + 2H_2O \quad \textbf{(2-4)}$$

$$2\mathrm{Au} + 4\mathrm{CN}^- + O_2 + 2\mathrm{H_2O} \leftrightarrow 2\mathrm{Au(CN)_2^-} + \mathrm{H_2O_2} + 2\mathrm{OH}^- \quad \textbf{(2-5)}$$

Based on the reaction equilibria outlined above, it is clear that the dissolution of “one mole of gold” requires “half a mole of oxygen and two moles of cyanide”, which depends on the efficacy of hydrogen peroxide reduction. The primary reaction is a two-electron process (Marsden and House 2006). The regulation of cyanide concentration in solution is relatively straightforward and can be achieved by adding either a solid cyanide salt or a concentrated cyanide solution (Wadsworth et al. 2000). Conversely, controlling the oxidant concentration, specifically dissolved oxygen, is challenging due to the low solubility of oxygen in water under atmospheric conditions (Rees 2000). Therefore, the maximum rate of gold dissolution for processes that employ air to supply oxygen in solution is dictated by the temperature and pressure conditions under which the process operates (Wadsworth and Zhu 2003).

Cyanidation has enabled the exploitation of lower-grade gold ores and the recovery of fine gold grains from hard rock. As cyanidation has evolved, extracting precious metal values from cyanide-pregnant leach solutions (PLS) has also received considerable attention (Marsden and House 2006). Techniques for concentrating gold from PLS include zinc cementation and electrowinning (Kongolo and Mwema 1998). The integration of the cyanidation process with a zinc cementation process enhanced gold recoveries from approximately 70% to 95% and revitalised the declining gold industry due to the decrease in gold grade (Marsden and House 2006). The effectiveness of the zinc cementation process was further improved by the use of powdered zinc, degasification, and the addition of minor amounts of soluble lead salts to solutions before cementation (Kongolo and Mwema 1998; Marsden and House 2006). Although cyanidation is an efficient and cost-effective process for most ore bodies, it is not without limitations that depend on the mineralogical characteristics of the ore.

2.8 The history of Wits tailings

In mining, tailings refer to the waste or by-products produced as a result of mining activities. As illustrated in the flowchart below, once valuable metals have been extracted from the mined ore, any remaining valuable material and gangue minerals are discarded as tailings, together with the water and chemicals used during processing (Adiansyah et al. 2015). The process flowsheet used to treat whole ores, resulting in the production of tailings, involves various stages, including grinding, sieving, and flotation (Figure 2-3).

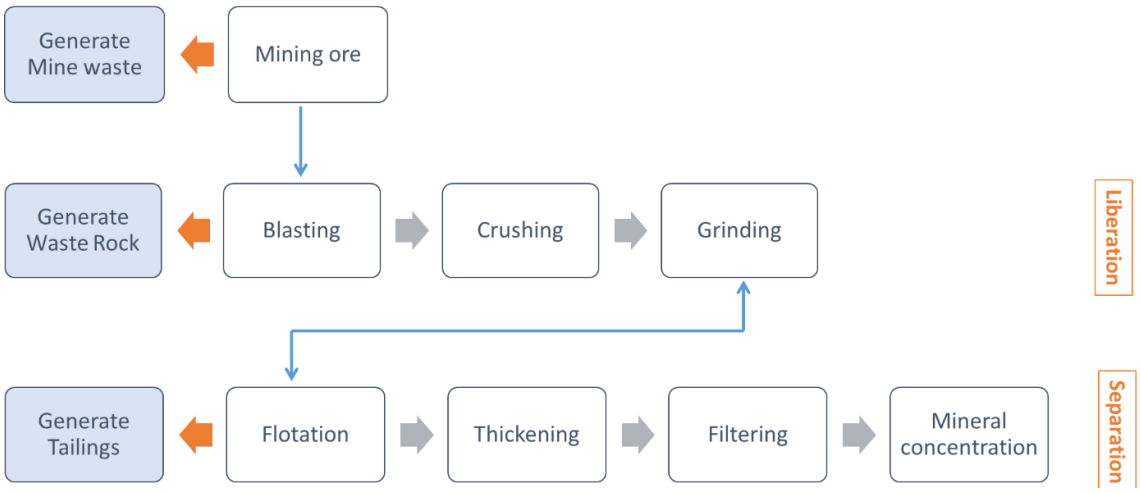


Figure 2-3. Mineral extraction from mining to tailings (Coetzee et al. 2011)

Tailings are typically separated from most of the water using pressurised filter systems or gravity. However, if the grinding of ores generates extremely fine particles, separating tailings from water can be challenging (Wang et al. 2014). The water that is recovered is recycled in the extraction process, while the tailings, in the form of a slurry, are conveyed through pipelines from the processing site to nearby storage facilities known as tailing dams. The objective of constructing tailing dams is to safely store and contain tailings and isolate them from the natural environment (Xu and Wang 2015).

The number of tailings generated is dependent on the ore's "grade". For example, if the ore's gold content is 1.0%, assuming a 100% efficient process, 99.0% of the total ore would be deposited as tailings. Tailings dams are designed to include fine-grained material and the long-term costs of tailings disposal are influenced by the mechanical stability and environmental reliability of the dam (Festin et al. 2019). Tailings are typically in slurry form and hydraulically deposited in the disposal area. The degree of thickening and the deposition method affect the engineering characteristics of tailings, as they are stored on the surface within retaining structures or piles.

Johannesburg has a rich history of gold mining that has resulted in the surface deposition of a large number of gold residues across the Witwatersrand region. These dumps comprise rock, sand, slime, mud, and sludge, and many of them still contain varying amounts of gold. In 2006, Mintek estimated that 40% of the estimated 438 Mt of mine tailings available on the surface in Johannesburg were being targeted for reprocessing (Maseko 2017). Similar gold mining operations have been ongoing for several decades, albeit on a smaller scale, in Giyani, the KOSH (Klerksdorp, Orkney, Stilfontein, Hartebeesfontein) area, the Free State goldfield, and around Barberton, the Kalgold operations, and Evander (Auchterlonie et al. 2017; Janse van Rensburg 2016; Maseko 2017). This implies that there are substantial amounts of gold tailings in South Africa. Establishing an economically viable cyanidation stage for extracting gold from these tailings would accelerate the reprocessing of these gold resources, thus extending the life of gold mining in the country.

Preliminary studies conducted on samples from West, East, Free State, and Central Gauteng mine dumps indicated that approximately 60% of gold (samples containing an average of between 0.3 g/t – 0.7 g/t gold) can be recovered via direct cyanidation. The tailings had a low

concentration of uranium (< 40 g/t), meeting environmental requirements. Sulphide recovery was as high as 94% during flotation, reducing the potential of acid mine drainage (Janse van Rensburg 2016).

Gold tailings are often considered waste after undergoing hydrometallurgical processes, but they still contain small amounts of valuable materials. Historically, reprocessing lower-grade ores and tailings was not a priority due to the availability of high-grade ore, which made this approach uneconomical. However, the increasing scarcity of high-grade ores, coupled with advancements in extractive technologies, has led to a growing interest in the reprocessing of tailings. Extractive technologies such as milling in raffinate, pre-oxidation, Carbon-In-Leach (CIL), Resin-In-Pulp (RIL), and Carbon-In-Pulp (CIP) have enabled the extraction of low concentrations of minerals from tailings and minerals from low-grade ores in a relatively economical manner (Lottermoser 2011).

Heap leaching has traditionally been applied as an efficient process to recover minerals from low-grade ores and re-process tailings. Studies have shown that reprocessing waste materials through heap leaching, for mineral recovery, requires significantly lower energy, reduced operational costs, and reduced environmental impact than initial hydrometallurgical extractions from virgin ore (Corder, Golev, and Giurco 2015). Although reprocessing tailings displays efficient and sustainable practices, the primary driver is usually economic gain (Phiri et al. 2007). The feasibility of reprocessing a tailings dump depends on factors such as processing proficiency, the measure of extraction and accompanying operating expenses, product price, and the gold grade of the tailings (Phiri et al. 2007).

2.8.1 Wits tailings retreatment projects

The key players in the dumps retreatment initiative include DRDGold, ERGO, and Sibanye Gold. Sibanye-Stillwater, a globally recognised precious metals mining company, has established its West Rand Tailings Retreatment Project (WRTRP) in collaboration with DRDGold (Phiri et al. 2007). In a collective assessment of tailings dams, the company estimates that there are 6.2 Moz of gold and 97.2 Mlb of uranium yet to be recovered (Sascha-Lee Solomonds 2018). Other companies that have entered the tailings reprocessing industry include Pan African Resources, a South African enterprise, and Metals X Limited, an Australian company. The Pan African Resources “Elikhulu” tailing retreatment plant is

projected to retreat tailings at a rate of 1000 kilotons/month to recover approximately 688,000 ounces of gold from historic tailings dams over the project's life (Corder, Golev, and Giurco 2015).

West Rand Tailings Retreatment Project (WRTRP)

Sibanye Gold has established the WRTRP at its Kloof, Driefontein, and Cooke operations. The deposit contains considerable gold grades, and a treatment facility for tailings is already in place. The tailings resource of West Rand Sibanye comprises 715 Mt of tailings containing 6 Moz of recoverable gold. The anticipated initial capital expenditure for this surface tailings reprocessing project was R9.6 billion. The feed comes from the Cooke 4 south tailings dam, the Cooke tailings dam, and the Driefontein 3 and 5 dams. Assuming an initial treatment rate of 1.4 Mt per month, its anticipated working life is 18 years, with 1.3 Moz of gold recovered by flotation at a central processing facility located between Sibanye's existing Kloof 1 and 4 operations (Maseko 2017).

Figure 2-4 illustrates the process flowsheet for the WRTP, indicating that Sibanye recovers both the uranium and gold from the tailings. The gold in the tailings undergoes a two-stage flotation circuit, polished by ultra-fine grinding to liberate the fine gold locked in gangue minerals, and then leached by cyanidation. Ultra-fine grinding is a CAPEX-intensive, high-energy-requiring method, and gold recovery improvements are estimated between 10-20% (Auchterlonie et al. 2017). The additional revenue resulting from the marginal increase in gold recovery should justify the costs associated with CAPEX and the higher cyanide and lime consumption resulting from the liberation of the gangue (Auchterlonie et al. 2017).

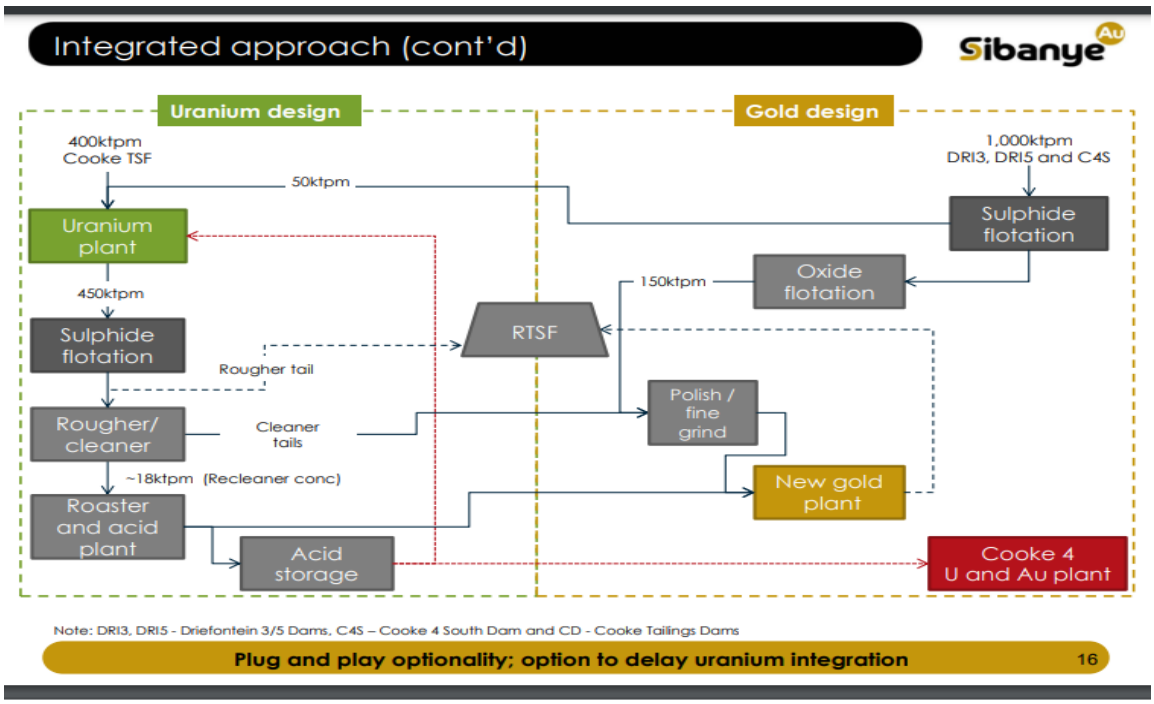


Figure 2-4. Sibanye WRTP flowsheet (Maseko 2017)

ERGO plant

The ERGO plant, situated in Brakpan, processes material from various sources, primarily the Elsburg tailings complex and the L/29 landfill. DRDGOLD's board identified the need to increase gold recovery as one of the primary objectives for the ERGO project. A custom-built test work program and pilot plant demonstrated that the CIL method was ineffective in recovering approximately 40% of the gold contained in pyrite particles. A dual-process approach was discovered involving the reintroduction of flotation and the addition of stage fine milling to the flotation concentrate. By January 2014, the novel circuits were operational, and the first gold bar was produced by the end of the month (Maseko 2017). The ERGO flowchart is depicted in Figure 2-5. During flotation, the plant produces a mass pull of 2.5%, with a rougher gold recovery of 20%, and the CIL circuit achieves a gold recovery of 55% (Staden et al. 2020).

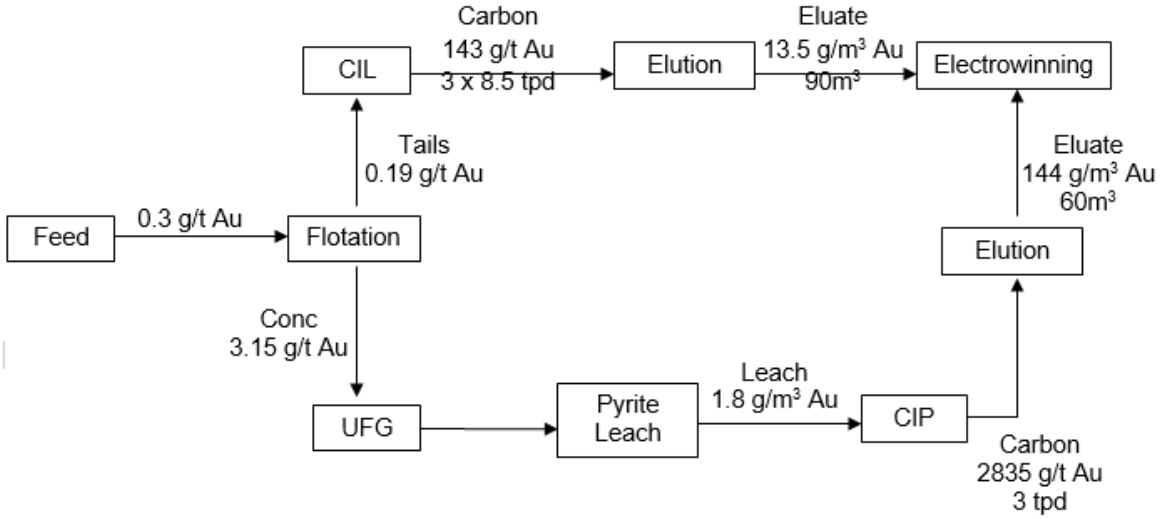


Figure 2-5. ERGO flow diagram (TWP The Basil Read Group 2012)

The ERGO plant has employed flotation as part of its process flowsheet since the 1970s, but when DRDGold took over in 2007, the flotation plant was shut down. It was later reintroduced in 2012 and became operational again in 2014, resulting in a gold extraction rate of 55% in the CIL circuit (Staden et al. 2020). The flotation circuit begins by feeding the tailings into conditioning tanks where xanthate (SNPX) and Senkol are added as co-collectors for the flotation of pyrite. The flotation concentrate is then taken for regrinding to expose more of the gold, which in turn is expected to lead to increased gold extraction in the CIP circuit. However, while the liberation of the gold was reportedly improved, the potential improvement in recovery was deemed unjustifiable given the capital and operating expenses required for ultra-fine grinding (UFG) (Andrianandraina et al. 2022).

Furthermore, although flotation collected much of the gold content into a small concentrate, a significant proportion of the gold remained in the flotation tail. To address this, the ERGO plant needed to treat both the concentrate and the flotation tail. In August 2019, the flotation plant was once again shut down, and no negative effect was observed on gold recovery (55%) (Andrianandraina et al. 2022). The aim of this project and associated studies is to improve gold recovery without the high capital and operating costs associated with UFG. A hydrometallurgical method resulting from this shutdown is being explored as a means of achieving this goal. UFG is known to consume a significant amount of energy, up to 150 kWh/t or $15-$30 per tonne, and generates ultra-fine particles that can pose a health risk for workers (Andrianandraina et al. 2022).

2.8.2 Wits tailings leaching behaviour

Cyanidation is a well-established hydrometallurgical process for gold extraction in the mineral processing industry. However, the presence of iron-sulphide minerals such as pyrite and pyrrhotite makes the optimisation of the cyanidation process more challenging, as these minerals can influence the rate of gold dissolution during cyanidation. This can result in high reagent consumption or restrict cyanide access to the gold particle, causing lower gold extraction efficiencies (Mbayo, Simonsen, and Ndlovu 2019). The Elsner equation, described in Equations (2-1) to (2-5), explains the gold leaching process by cyanidation, which involves the dissolution of gold when combined with oxygen and cyanide, while maintaining the pH of cyanidation at pH > 10 (Yannopoulos 1991). Oxygen, cyanide concentrations and temperature are essential factors that influence the gold dissolution rate (Nicol, Fleming, and Paul 1992). Electrochemical studies have indicated that the optimal molar ratio of $[CN^-]/[O_2]$ required for the oxidation of gold at the anode (Equation (2-2)) and the reduction of oxygen at the cathode (Equation (2-3)) is approximately 12 (Sandra Ellis and Senanayake 2004). However, excess oxygen is required due to the consumption of oxygen by ions such as Fe^{2+}, S^{-2}, and HS^- released from iron-sulphide minerals during cyanidation. The cyanide ion (CN^-) interacts readily with iron-sulphide minerals, causing them to release their constituent ions during the leaching process (Bartlett and Davis 1958).

The mineral associations of gold refer to the minerals that occur in close contact with gold grains. The mineral associations can have an impact on gold recovery during processing. For instance, iron-sulphide minerals that are associated with gold can contribute to the inefficiencies of gold extraction during cyanidation, as they leach simultaneously with gold. Reagent consumers, such as hydrosulphide (pre-oxidised iron-sulphide minerals), can also compete with gold for cyanide to dissolve gold alloys, affecting gold extraction (Tshilombo and Sandenbergh 2001; Jeffrey and Ritchie 2000). Hydrosulphides further oxidise in the presence of oxygen to form thiosulphate and sulphite ions. For pure gold not associated with other minerals, oxygen reduction and gold oxidation typically occur at the gold surface. In contrast, with sulphide ores, the gold and sulphide mineral surface acts as the cathode (Khalid 2017).

Soluble iron-sulphide minerals are recognised as consuming a considerable amount of cyanide and oxygen during the gold extraction process, resulting in the formation of $Fe(OH)_3$ and SCN^- as products, as shown in Equation (6) (Hedley and Tabachnick 1968; House 1992; Nagy, Mrkusic, and McCulloch 1966).

$$2Fe_{1-x} + 2CN^- + 1.5(1-x)O_2 + 3(1-x)H_2O \leftrightarrow 2(1-x)Fe(OH)_3 + 2SCN^- \quad \textbf{(6)}$$

The efficiency of gold extraction rates during cyanidation can be reduced due to the consumption of essential reagents by soluble iron-sulphide minerals. Additionally, passive layers created on the surface of the gold may also contribute to the poor outcomes of the cyanidation process (Karimi et al. 2010; Tran 1992). The leaching of iron-sulphide minerals during cyanidation leads to the formation of cyano-complexes that consume cyanide, thereby forming protective or passive layers on the gold surface. This phenomenon is illustrated in Equation (7) (Soltani et al. 2020).

$$S^{2-} + CN^- + 0.5O_2 + H_2O \rightarrow CNS^- + 2OH^- \quad \textbf{(7)}$$

Various studies have demonstrated that pre-treatment of gold iron-sulphide ores is essential to achieve efficient gold extraction using cyanidation. Pre-treatment oxidises sulphide minerals associated with gold, eliminating their interference in the cyanidation leaching process. Oxidation, roasting and ultra-fine grinding are commonly used pre-treatment methods.

Optimisation of lixiviant gold-extracting capability is dependent on the pH due to the need for a complexing agent. The pH of a solution determines which species the complexing agent generates and, subsequently, the formation of the gold complex. During cyanidation, the pH of the solution and the cation that affects the redox potential (Eh) of the solution dictate the several cyanide complexes that can develop. There are three categories of cyanide complexes; free cyanide (CN_{free}), including hydrocyanic acid (HCN) and the cyanide anion (CN^-); weak acid dissociable (CN_{WAD}) cyanide, and strong acid dissociable (CN_{SAD}) cyanide. Cyanide produces salts with alkali earth cations and ionic complexes of varying strengths with several metal cations. The breakdown of certain metal-cyanide complexes can liberate CN_{Free} and metal ions. CN_{SAD} is formed under extremely acidic conditions when cyanide complexes metals such as Fe, Co, Ag and Au. At very low Eh and predominately alkaline pH values, cyanide ions (CN^-) are stable, making conditions favourable for the formation of the aurous cyanide complex ($Au(CN)_2^-$).

The aurous cyanide complex ($Au(CN)_2^-$) is the only stable gold complex during gold cyanidation. The aqueous chemistry of gold in cyanide solution is illustrated in Figure 2-6. Aurous-cyanide ions ($Au(CN)_2^-$) are present throughout the entire pH range, while the auric ions (Au^+) are limited to pH less than 2. Elemental gold (Au°) is stable at relatively low redox potential (Eh) and covers the entire pH range.

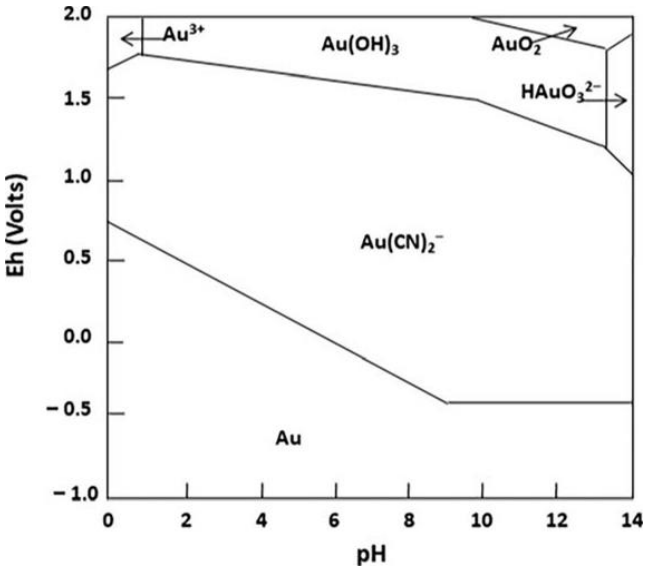


Figure 2-6. Eh-pH diagram of the Au-CN-H, 0 system at 25°C and 1 atm

A mineralogical characterisation study conducted by Mintek in 2016 aimed to guide new tailings reprocessing projects for Wits tailings. Various samples were collected from Central Gauteng, East Rand, West Rand, and Free State dumps and were evaluated for their metallurgical processing amenability and environmental impact. The study revealed that the majority of Wits tails contained sulphides, as presented in Table 2.1. The hypothesis is that either electrum, iron-sulphide minerals or both may cause poor gold extractions, and the objective is to investigate which of the two is at play during the cyanidation of Wits pyrite tailings. The presence of sulphides in the historic gold tailings samples collected in the Wits area supports the possibility of sulphur-containing gangue minerals playing a role in the observed lower gold recovery.

Table 2.1. Average sulphide concentration of various Wits tailings (Janse van Rensburg 2016)

		"East Rand"	**"Central Gauteng"**	**"West Rand"**	**"Free State"**
Au	g/t	0.27	0.32	0.35	0.41
Uranium	g/t	19	Not detected	59	65
Sulphides	%	0.41	0.25	0.50	0.90
Sulphates	%	0.50	0.01	0.30	0.40
Total S	%	0.90	0.28	0.80	1.05
Total C	%	0.31	0.14	0.26	0.06

The presence of soluble gold in Wits tailings indicates that the original process was inefficient rather than the ore being refractory. However, the presence of gangue minerals such as pyrite, pyrrhotite, and arsenopyrite can also contribute to the gold's immobilisation (Zhou et al. 2004).

Data obtained from a study of various Wits tailings, as shown in Table 2.2, indicated that gold extractions ranged from 58 to 63%, indicating the presence of cyanide-soluble gold. This suggests that Wits tailings are suitable for reprocessing, and that gold extraction can be improved by pre-treating the refractory minerals, particularly for the East and West Rand dumps.

Table 2.2. Leaching performance of Wits tailings (Janse van Rensburg 2016)

		"East Rand"	**"Central Gauteng"**	**"West Rand"**	**"Free State"**
Grind size of dumps as is		80% passing 53 µm	76% passing 75 µm	83% passing 106 µm	80% passing 106 µm
Gold extraction via direct cyanidation	%	60	61	58	63
Ave gold residue after direct cyanidation	g/t	0.18	0.19	0.16	0.25
Gold recovery via flotation @ 20% mass pull	%	73	65	65	70
Cyanide consumption	kg/t	0.5	0.9	1.1	0.9
Lime consumption	kg/t	2	2	4	3
Preg-robbing	%	0	3.5	4	5
Gold refractory	%	13	4.5	13	3
Gold locked in silicates	%	25	31	29	29

Note. Information sourced from Guidelines for Retreatment of SA Gold Tailings: MINTEK's Learnings (Janse van Rensburg 2016)

2.9 Effect of electrum on gold dissolution kinetics

A study using a rotating electrochemical quartz crystal microbalance, found that incorporating small amounts of silver in the solid phase enhanced the kinetics of the gold oxidation half-

reaction and the oxygen reduction half-reaction, thereby resulting in significantly improved gold leaching rates. The presence of silver disrupts the AuCN film, enhancing the gold oxidation half-reaction while preferentially reducing oxygen at the active silver sites (Anderson *et al.*, 2014).

Another study conducted on a rotating gold disc investigated the effect of silver content on gold dissolution rates in the presence of various sulphide minerals. Rees (2000) reported that the leaching of gold and silver depends strongly on both the solubility of the sulphides and the oxygen concentration in the solution. The study found that the maximum gold dissolution rate occurred when the gold contained 0.6% silver, while further increases in silver content resulted in a negative effect on gold dissolution. Both studies showed that trace amounts of silver in the solid phase enhance the gold oxidation half-reaction and the oxygen reduction half-reaction. In addition to this, dissolved silver ions in the solution phase can alleviate the depressing effect of the dissolved cyanide, improving the mass transfer of cyanide from the solution to a solid phase.

In cases where reactive gangue material is associated with or locks gold particles and the silver content in the gold particle is high (between 20% and 40%), extraction can be poor and reagent consumption high. If there is a high concentration of silver (more than 10 g/t), meaning electrum-type gold is present, the CIP procedure for extracting gold may require certain adjustments. Silver dissolution occurs at a slower rate than gold during cyanidation because the common silver minerals contained in nature are less soluble in cyanide solution than gold. Electrum is refractory due to the presence of sulphide and sulfosalt minerals, which are the main cause of poor gold extraction in electrum. Oxidation of the sulphide ion is required to release the ionic silver to promote the access of cyanide to the gold particle. Oxidation also reduces the concentration of sulphide ions available to react with the dissolved silver to form silver sulphides, which can coat the gold surface, causing poor gold extractions (Khalid 2017; Kim and Ghahreman 2019; Liu and Yen 1995).

2.10 Influence of sulphide minerals on the solubility of gold

A comprehensive understanding of the gold-containing minerals can provide insights into the extent and type of gold entrapment in associated mineral phases. This knowledge can predict and explain the leaching behaviour of Wits pyrite concentrate during cyanidation. The Wits

gold is commonly associated with sulphide minerals such as pyrite, pyrrhotite, arsenopyrite, and chalcopyrite. These sulphide minerals can partially dissolve during cyanidation, making them readily available in solution to react with gold-silver alloys and reagents (Zhou et al. 2004). Pyrite remains unreactive under mildly oxidising cyanide-leaching conditions (Marsden et al. 2006) unless it is partially oxidised due to tailings storage conditions (Dai, Simons, and Breuer 2012). Pyrrhotite, on the other hand, is more stable under reducing atmospheres than pyrite and tends to oxidise and dissolve during cyanide leaching (Klimpel 1999).

When electrum contains a high concentration of silver (more than 10 g/t), silver sulphide forms due to a reaction between silver and sulphides, which limits cyanide access to the gold portion of the electrum particles, decreasing gold extraction via reduced dissolution kinetics. Figure 2-7 illustrates this concept. The presence of soluble sulphides hinders the gold leaching rate, leading to the formation of a protective layer of Ag_2S.

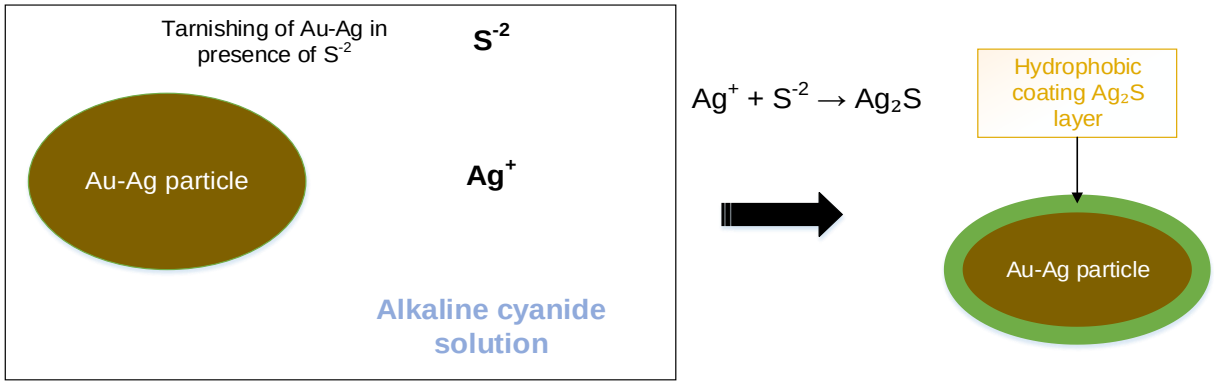


Figure 2-7. Mechanism of the formation of a silver sulphide layer

The degree of dissolution of gold and silver is contingent on the stability constants of the competing complexes that result from the concentration of sulphide, copper, and iron in the solution (Senanayake 2008). The formation of a complex between a metal ion and a ligand is a substitution reaction. In cyanide, aqueous solutions, metal ions exist as gold, iron, sulphide, and base metal ions, while lignin exists as cyanide. Ideally, a ligand should bind only to the target metal ion, but achieving such selectivity using cyanide in the presence of soluble iron-sulphide minerals is highly challenging. During the formation of bonds, the stability constant is used to determine the intensity of interaction between reagents that make up the final product (Kumari 1941; Senanayake 2004).

In an alkaline cyanide solution, iron complexes with cyanide more rapidly than gold and silver, leading to the formation of more stable iron-cyanide complexes, as shown in Table 2.3 (Khalid 2017; Marsden et al. 2006). Copper, on the other hand, complexes with cyanide to form cyanocuprate ions in an aqueous solution, but these complexes only form under excess cyanide conditions. The presence of high levels of iron in the solution reduces the extraction of the valuable metal by consuming the cyanide necessary for the formation of the gold-cyanide complex, thereby slowing down the process (retarding effect).

Table 2.3. Stability constants of gold and competing metal-cyanide complexes

Reaction	Stability constant Log K (at 25°C, I=0)
$Fe^{2+} + 6CN^- = Fe(CN)_6^{4-}$	45.6
$Fe^{3+} + 6CN^- = Fe(CN)_6^{3-}$	52.6
$2Fe^{2+} + 6CN^- = Fe_2(CN)_6^0$	57.0
$Au^+ + 2CN^- = Au(CN)_2^-$	37.0
$Ag^+ + 2CN^- = Ag(CN)_2^-$	20.4
$Cu^+ + CN^- = CuCN$	-9.21
$Cu^+ + 2CN^- = Cu(CN)_2^-$	-20
$Cu^+ + 3CN^- = Cu(CN)_3^{2-}$	5.3
$Cu^+ + 4CN^- = Cu(CN)_4^{3-}$	1.5

Sulphide, which is formed by the reaction of cyanide with pyrrhotite (Equations (8) and (9)) or oxidised pyrite and arsenopyrite, can undergo a further reaction with oxygen to produce sulphate, as stated in Equation (10) (Zia et al. 2020). The resulting sulphate then binds to the gold surface layer, leading to the passivation of the gold particle (Deschênes et al. 1998).

$$Fe_5S_6 + NaCN = NaCNS + 5\,FeS \quad \textbf{(8)}$$

$$FeS + 6CN^- + 2O_2 \rightarrow FeCN_6^{4-} + SO_4^{2-} \quad \textbf{(9)}$$

$$2FeS + 12CN^- + 5O_2 + 2H_2O \leftrightarrow 2Fe(CN)_6^{4-} + 2SO_4^{2+} + 4OH^- \quad \textbf{(10)}$$

According to the process mineralogy study, Wits gold occurs as native gold and electrum and is amenable to cyanide leaching. However, the reactive gangue minerals associated with the gold particles contribute to high reagent consumption. Pyrite and pyrrhotite, the most cyanide and oxygen-consuming iron-sulphide minerals, are particularly problematic. Pyrrhotite

consumes cyanide by reducing the pH, resulting in the formation of thiocyanate and sulphate (Anderson, Dunne, and Uhrie 2014). Thus, pre-treatment of pyrrhotite with alkaline or acidic media reduces the amount of iron and sulphide reporting to the CIP circuit. Moreover, the reduced gold dissolution from the Wits pyrite tailings may be due to the following factors:

- Simultaneous leaching of gold, iron, and sulphur species in the cyanide solution leads to a high concentration of sulphide (S^{2-}), iron (Fe^{2+} and Fe^{3+}), copper, and zinc, which react with cyanide, lime, and oxygen, thereby retarding the kinetics of gold leaching.
- The dissolution of pyrite, pyrrhotite, and arsenopyrite further decreases gold extraction by complexing the cyanide required for the formation of the gold-cyanide complex (inhibition effect).
- The reaction of sulphur to form sulphate on the gold surface directly hinders the access of oxygen and cyanide to the gold particle (passivation effect).

2.11 Conclusion

Various factors, including the mineralogical occurrence of gold, can influence the reprocessing of tailings dumps using different flowsheet configurations, as discussed in the literature review. The efficiency of gold extraction from a Wits pyrite concentrate via cyanidation requires a comprehensive understanding of the process mineralogy. Hence, the process mineralogy should dictate the efficacy of gold extraction, and not all tailings dumps are amenable to reprocessing due to unfavourable gold grades and occurrence, or the presence of gangue minerals.

Inefficiencies in gold extraction, can result from various factors, such as; the presence of very finely scattered gold in a matrix of ore or sulphide solid solution, gold coupled with tellurides, gold found in the sulphides of basic metals such as lead, copper, and zinc, and carbonaceous ores or silver finely scattered in the matrix of gangue predominantly made of quartz or complexed with manganese oxides. These characteristics classify gold deposits into two primary categories: “free-milling” and refractory ores. “Free-milling” ores are those from which more than 90% of gold can be recovered by the conservative cyanidation method when the ore is milled to 80% 75-µm particle size without excessive reagent use.

Gold extraction from refractory ores is typically improved through oxidative pre-treatment of the ore by pyrometallurgical or high-temperature processes. While these processes have been

extensively exploited for decades, the strict safety, emissions, and environmental protection regulations have made them more challenging and expensive. Consequently, it is critical to know the metallurgical limit of gold tailings to find economically feasible extraction methods.

The efficiency of the gold extraction-processing route is directly related to the mineralogy of the gold ore processed, as the mineral composition influences the efficacy of all chemical and physical gold extraction procedures. Therefore, it is essential to characterise the mineralogical nature of the ore to be treated, including the precious metal phases (gold deportment) and gangue minerals. Factors such as gangue minerals, oxygen and cyanide consumers, and passivation in the ore tend to influence the extractive metallurgy and are recognised as the primary cause of poor gold extractions in some gold ores.

CHAPTER 3

3. METHODOLOGY

3.1 Introduction

Passivation, inhibition, or retardation during the cyanidation of gold-containing material can be caused by factors such as gold particle size and liberation, reagent consumption, and preg-robbing minerals (Azizi, Petre, and Larachi 2012; Bas, Ghali, and Choi 2017; Zheng et al. 1995). To investigate which of these mechanisms reduces gold extractions, process mineralogy techniques were applied to evaluate the likely causes of low gold extraction efficiency. Mineralogy serves as the basis for research, as gold deportment is one factor that controls the extent of gold extraction. Gold deportment analyses were conducted to locate and describe gold-containing particles, and determine gold speciation, grain size, and mode of occurrence. The resulting output provided a profound understanding of the mineralogical occurrence of gold. Process mineralogy also facilitated the selection of a route for Wits tailings reprocessing that would enhance gold extraction efficiency. Consequently, numerous experiments were performed to determine the most favourable conditions for increasing gold extraction and identifying a relationship between test conditions and gold extraction from solids.

The preceding chapter reviewed literature relevant to the study. This chapter outlines the research methodology employed to determine the cause of reduced gold extraction observed during cyanide leaching of Wits pyrite tailings and establish improved CIP conditions.

The study's aim was achieved through the following activities:

1. Determination of the content and association of silver and iron-sulphide (separate Ag minerals and Au/Ag alloy) in Wits pyrite tailings.
2. Evaluation of the behaviour of gold, silver, and sulphide minerals during cyanidation.
3. Evaluation of pre-treatment of reactive iron-sulphide minerals with lime and oxygen to improve gold dissolution during cyanidation.
4. Addition of lead nitrate during cyanidation to improve gold dissolution kinetics.

3.2 Sample preparation for chemical characterisation: Wits pyrite concentrate

The pyrite concentrate sample used in the study was obtained from the ERGO tailings retreatment plant located in Johannesburg, South Africa. ERGO's primary objective is to recover and process surface tailings from the Elsburg, Driefontein, Far East Rand, and Far West Rand tailings complex and their surrounding dumps. The sample represents a concentrate of Wits tailings produced throughout the history of gold mining in South Africa, spanning from Springs to the Vaal River. In 2018, ERGO conducted a sub-sampling campaign of pyrite concentrate produced in the ERGO flotation plant for over ten weeks. The ERGO flotation process entails conditioning the tailings in conditioning tanks with the collector xanthate (SNPX) and co-collector Senkol for pyrite flotation. Sub-sampling was done by collecting the concentrate product at 24-hour intervals and extracting one litre of the flotation concentrate each time to create a composite sample for the study.

3.3 Sample preparation for mineralogy characterisation: Knelson concentrate

Directly characterising a Wits pyrite concentrate is challenging due to the low gold grade. As a result, pre-concentration was necessary for this study. The pyrite concentrate had a gold grade of 3.01 g/t, with only four gold grains found in a one-kilogram sample. Obtaining statistically significant results with polished sections proved costly and inadequate to address the quality constraint. Therefore, the Wits pyrite concentrate was upgraded using the Knelson Gravity separation technique. A 40 kg sample was processed through the Knelson MD-3 laboratory concentrator at 60 g centrifugal force and a fixed fluidising water flowrate of approximately 2.5 L/minute, then panned and submitted as a final concentrate for mineralogical characterisation.

3.4 Process mineralogy

3.4.1 Content of gold, silver, and gangue minerals

To determine the content of metals of interest, such as gold, silver, and base metals, various techniques were applied to analyse the Wits pyrite concentrate, as shown in Table 3.1. The table includes a list of analyses used to determine the sample's content and the limit of determination for each method.

Table 3.1. Elemental characterisation methods

Technique	Description	Analyte List	Limit
Fire assay	Destructive, precious metal analysis of the composition of an ore, metal or alloy	Total Au	0.001 g/t
ICP-OES	Detection of the chemical element of ores and slags. Ore is fused then followed by dissolution in HCl/HNO_3	Mg, Al, Si, Cr, Ca, Ti, V, Mn, Fe, Co, Ni, Cu, Zn and Pb	0.05% m/m, - 40% m/m
ICP-MS	Detection of metals and several non-metals in liquid samples at deficient concentrations	(Ag - HCl digestion)	0.1 mg/kg – 500 mg/kg
Combustion	Total sulphur, elemental sulphur, sulphates and sulphides	Combustion, after extraction	0.2% m/m

**m/m: Mass of element by mass of the sample*

3.4.2 Association of gold to silver and gangue minerals

The Mineral Liberation Analyser (MLA) was employed to acquire data regarding the identification of gold minerals, the silver content of gold grains, and the liberation and association of gold in the Wits pyrite-Knelson concentrate. The MLA process characterises discrete gold minerals by:

- Identifying gold minerals.
- Determining the relative proportions of silver content in gold grains.
- Locating all gold minerals and establishing their liberation, utilising a Quantitative Evaluation of Materials by Scanning Electron Microscopy (QEMSCAN). It also predicts the fraction of free and locked gold, which translates to maximum leachable gold (by cyanidation).

Energy Dispersive Spectroscopy (EDS) was utilised to establish the silver content of gold grains and predict the likelihood of silver tarnishing (formation of silver sulphide layer on the gold particle) during cyanidation of the Wits pyrite concentrate. Table 3.2 shows the characterisation methods employed and their respective descriptions.

Table 3.2. Mineralogy analysis

Technique	Description	Analyte list	Determination limit
Automated Scanning Electron Microscope (AutoSEM)	AutoSEM: Semi-quantitative gold scan, mineral classification and surface morphological analysis (Zhou et al. 2004)	Au minerals and surface morphological study	Semi-quantitative
Electron Probe Microanalysis	Compositional analysis of minerals	Gold alloys, silver sulphide	0.1% EDX 0.02% WDX

3.4.3 Gangue minerals limitation on the dissolution of gold during cyanidation

The diagnostic leaching technique is a valuable metallurgical tool used to determine the gold distribution in a sample, and it was employed in this study to establish the gold deportment in the pyrite concentrate. The technique involves the selective and sequential dissolution of minerals in a specific media, thereby releasing the gold associated with or locked in these minerals from the least stable mineral to the more stable minerals further down. The diagnostic leach tests consisted of five stages, as outlined in Table 3.3.

Table 3.3. Typical diagnostic leach conditions (Janse van Rensburg 2016)

Stage	Leach conditions	Mineral of interest
Step 1: Direct cyanidation on Feed	30% solids, 5 kg/t NaCN, pH 11 adjusted with NaOH, 24-hour leach	Free-milling gold
Step 2: Carbon-in-Leach (CIL) on Feed	30% solids, 5 kg/t NaCN, 25 g/L virgins activated carbon, pH 11.5 adjusted with NaOH, 24-hour leach	Preg-robbing gold was identified by differences observed when comparing direct cyanidation with CIL results
Step 3: Hydrochloric acid leach (HCl) followed by CIL on Stage 2 residue	Use 30% solids, 2M HCl for 1 hour at 70°C, followed by CIL at conditions used for step 2	Gold that was associated with HCl digestible minerals (Pyrrhotite)
Step 4: Nitric acid leach (HNO_3) followed by CIL on Stage 3 residue	Use 30% solids, 50%(v/v) HNO_3 for 1 hour at 70°C followed by CIL at conditions used for step 2	Gold that was associated with HNO_3 digestible minerals (Pyrite and arsenopyrite)
Step 5: Roasting on Stage 4 residue	Duration: 1 hour. Temperature 900°C followed by CIL at conditions as used for step 2	Gold associated with carbonaceous material
Step 6: Calculated value assuming the remaining gold in the sample is locked in silica gangue material	Calculated	Locked in silicates

Note. Information sourced from Guidelines for Retreatment of SA Gold Tailings: MINTEK's Learnings (Janse van Rensburg 2016)

3.5 Establishment of baseline gold extraction using ERGO conditions

The Wits pyrite concentrate was subjected to cyanidation using DRD ERGO CIP conditions with increased NaCN addition of 2 kg/t and a solids: liquid mass ratio of 33% to compensate for the dissolution of competing elements. The retention times were varied to establish the leaching kinetics and efficiency. The leach duration was extended from 24 to 72 hours to evaluate the maximum extractable gold with extended times and to determine the gold dissolution rate. The baseline 72-hour cyanidation test was used to evaluate the gold content in the residue over time and confirm the slow dissolution of gold from the deposit. Table 3.4 provides details of the baseline tests performed using the ERGO CIP cyanidation condition.

Table 3.4. ERGO CIP conditions

Parameter	ERGO		Laboratory	
Solids: Liquid mass ratio	43	%	33	%
pH	10.5		10.5	
NaCN addition	0.66	kg/t	2	kg/t
Carbon	0	g/L Pulp	0	g/L Pulp
Temperature (˚C)	ambient		ambient	
Retention time	22	hours	1, 2, 3, 4, 6, 8, 10, 12, 18, 24, 36, 48, 60 and 72	hours

3.6 Gold extraction improvement through the application of process mineralogy

Based on the chemical, mineralogical, and metallurgical characterisation findings, test conditions were established to improve gold extraction from the Wits pyrite concentrate. The results from the process mineralogy study and baseline cyanidation kinetic tests indicated that poor gold dissolution was due to the following reasons:

- Simultaneous leaching of gold and iron-sulphur species in cyanide solution leading to a high concentration of sulphide (S^{2-}), iron (Fe^{2+} and Fe^{3+}), and base metals that react with cyanide, lime, and oxygen, thus slowing down gold leaching kinetics.
- Dissolution of pyrite, pyrrhotite, and arsenopyrite, resulting in the formation of complexes with cyanide required for the gold-cyanide complex formation, thus reducing gold extraction (inhibition effect).

The results of the process mineralogy directed the leaching parameters to maximise gold extraction by limiting gangue minerals. The first part of the testing program, shown in Figure 3-1, investigated the effect of pre-oxidation of the pyrrhotite and pyrite associated with and locking the gold-on-gold extraction and reagent consumption. Next, the effect of lead nitrate addition on reducing cyanide use without impairing gold extraction was examined. Finally, the effect of pre-oxidation on the kinetics of gold dissolution was determined by reducing the retention time.

The investigation began with the assessment of pre-oxidation parameters while keeping the cyanide parameters constant for the first four tests. The pH of the slurry was adjusted to 11 using hydrated lime powder, and the bottle was rolled for variable times (1, 3, and 6 hours) while maintaining the target pH and controlling the dissolved oxygen (DO) using either compressed air or pure industrial oxygen as the oxidant. The use of compressed air instead of pure industrial oxygen as an oxidant was investigated to reduce the operating expenses (OPEX) of the flowsheet without compromising the extent of oxidation, which could lead to reduced gold extraction. The retention time of the pre-oxidation process was also investigated, and it was found that three hours were sufficient for pre-oxidation, as increasing the time beyond three hours did not improve gold extraction.

The investigation then proceeded to examine the effect of lead nitrate ($Pb(NO_3)_2$) addition on improving gold dissolution kinetics and reducing cyanide consumption. It was observed that $Pb(NO_3)_2$ only reduced cyanide consumption and did not improve gold dissolution. Additionally, investigations were carried out with reduced NaCN additions of 1.5 kg/t and 1 kg/t to determine whether the pre-oxidation stage affected the reactivity of competing elements. This was done to establish whether NaCN was still the controlling reagent in gold extraction after the soluble iron-sulphide minerals had reacted with lime and oxygen. The leach retention time was also investigated by reducing it to 8 and 18 hours to determine the pre-oxidation benefits on gold dissolution kinetics, to reduce the OPEX of the flowsheet.

After each leach test, the slurry was filtered, and the filter cake was washed by re-pulping it once at a solid-to-liquid ratio of 1:2 (as per 33% solids in the initial slurry) and plug washing it once at the same ratio using tap water. To ensure that all entrained gold was washed from the solid before submission for analysis, the washed filter cake was made homogenous for analysis

by drying it in an oven at 60°C for 24 hours before submission. Entrained gold can lead to poor accountabilities on the gold mass balance.

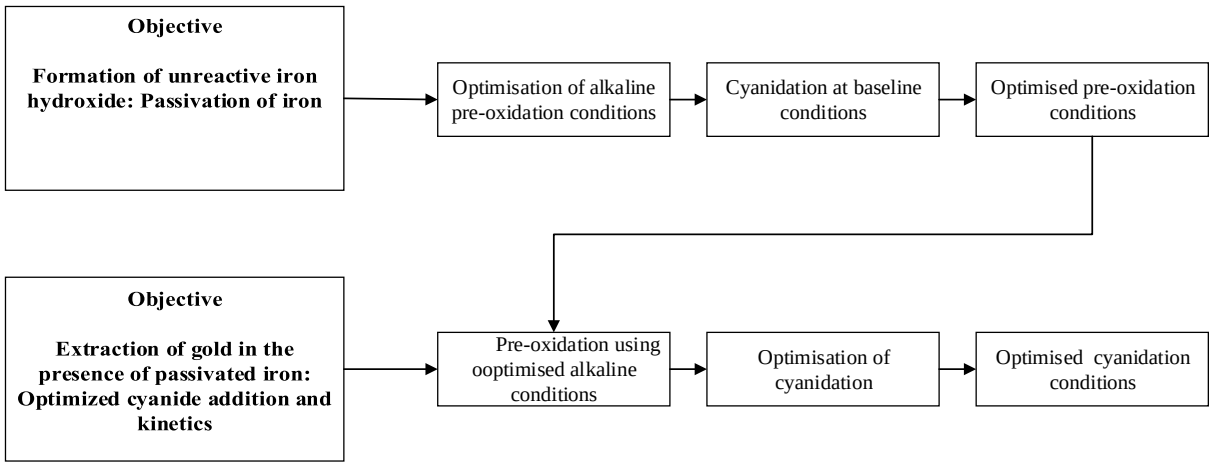


Figure 3-1. Test work program to improve gold extraction

The detailed test work conditions are shown in Table 3.5 and all cyanidation products of the test work program were analysed as described in Table 3.1 and Table 3.2.

Table 3.5. Alkaline pre-oxidation and cyanidation conditions for improved gold extraction

	Objective	Pre-oxidation and cyanidation conditions
Test 1	Alkaline O_2 pre-ox with consideration of retention time and oxidant	Pre-Ox: Compressed Air for 1 Hr @ pH 11.5 (Adjusted with lime) followed by CN @ 2 kg/t NaCN for 24 Hrs
Test 2		Pre-Ox: O_2 for 1 Hr @ pH 11.5 (Adjusted with lime) followed by CN @ 2 kg/t NaCN for 24 Hrs
Test 3		Pre-Ox: Compressed Air 6 Hrs @ pH 11.5 (Adjusted with lime) followed by CN @ 2 kg/t NaCN for 24 Hrs
Test 4		Pre-Ox: Compressed Air for 3 Hrs @ pH 11.5 (Adjusted with lime) followed by CN @ 2 kg/t NaCN for 24 Hrs
Test 5	To disrupt the "natural "passivation layer (hydroxide/AuCN) on gold surfaces, improving both extraction and dissolution kinetics	Pre-Ox: Compressed Air for 3 Hrs @ pH 11.5 (Adjusted with lime) followed by CN @ 2 kg/t NaCN + $Pb(NO)_3$ for 24 Hrs
Test 6		Pre-Ox: Compressed Air for 3 Hrs @ pH 11.5 (Adjusted with lime) followed by CN @ 2 kg/t NaCN + $Pb(NO)_3$ for 24 Hrs
Test 7	To reduce cyanide addition to cut reagent costs	Pre-Ox: Compressed Air for 3 Hrs @ pH 11.5 (Adjusted with lime) followed by CN @ 1.0 kg/t NaCN for 24 Hrs
Test 8		Pre-Ox: Compressed Air for 3 Hrs @ pH 11.5 (Adjusted with lime) followed by CN @ 1.5 kg/t NaCN for 24 Hrs
Test 9	To reduce retention time to cut energy costs	Pre-Ox: Compressed Air for 3 Hrs @ pH 11.5 (Adjusted with lime) followed by CN @ 2 kg/t NaCN for 8 Hrs
Test 10		Pre-Ox: Compressed Air for 3 Hrs @ pH 11.5 (Adjusted with lime) followed by CN @ 2 kg/t NaCN for 18 Hrs

CHAPTER 4

4. PROCESS MINERALOGY OF WITS PYRITE CONCENTRATE

4.1 Introduction

A pyrite concentrate was produced from the tailings material at the ERGO plant, belonging to DRD GOLD. As previously mentioned in the introduction, the sample was a concentrate produced through the flotation of free gold and gold associated with sulphides; therefore, very few of the gold locked in silicates were represented in the sample. The sample is a representation of typical Wits pyrite concentrate, which is a potentially recoverable fraction of Wits tailings previously treated at the DRD ERGO. Flotation is typically used to reject gangue minerals not associated with gold and reduce the size of the extraction plant that would be required, as well as the concentration of the leaching reagents (Stange 1999). Understanding the mineralogy of the pyrite concentrate would result in identifying suitable and efficient cyanidation conditions for maximum gold extraction without the application of ultra-fine grinding (UFG).

The flotation circuit at ERGO consisted of three identical streams, and each stream comprised two banks of cells, each with primary, secondary, and tertiary flotation cell stages. The material was fed into conditioning tanks from the reclamation sites where the collector, sodium normal propyl xanthate (SNPX), and co-collector Senkol A (alkyl dithiophosphate) were added to float pyrite.

The thickener overflows were recycled to the conditioning tanks to dilute the feed to 1.3. The conditioned slurry was then pumped into a flotation distribution box for each of the three streams, where it was split into the two banks (north and south) after the addition of frother, Senfroth, which is TEB based, with a chemical formula of 1,1,3-triethyoxy butane . Each bank consisted of 18 flotation cells. Each flotation cell had a unique mechanism to agitate the slurry and disperse air into the cell to produce air bubbles for the flotation process. A schematic diagram for the flotation cells and a process flow diagram for one of the flotation streams are given in Figure 4-1 and Figure 4-2, respectively.

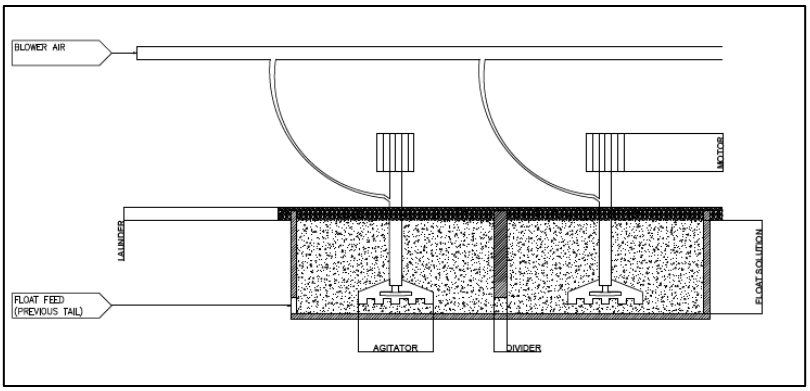


Figure 4-1. Flotation cells schematics

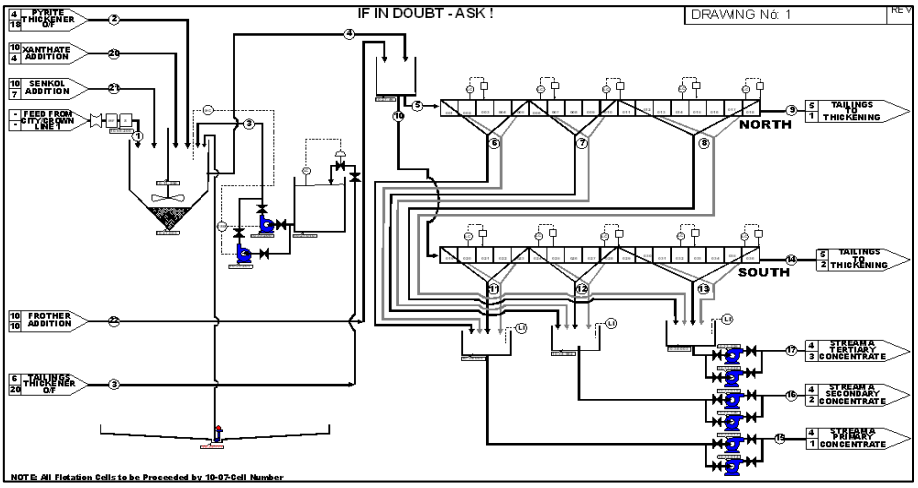


Figure 4-2. Flotation stream PFD

The aforementioned process ensured that the conditioned sulphide minerals became hydrophobic and attached to air bubbles, forming a froth that flowed over the sides of the cells as concentrate into launders. Primary, secondary, and tertiary concentrates were produced, and they were subsequently sent to the pyrite thickeners, where the sample for this test work program was collected.

4.2 Content of gold, silver, and gangue elements in a Wits pyrite concentrate

To establish the content of the metals of interest, such as gold, silver, base metals, and sulphur species, the flotation concentrate sample was analysed using various techniques described in Table 3.1. A combination of analytical and metallurgical tools was used to quantify the target mineralogy, gangue mineralogy, deleterious minerals and element distribution, mineral-mineral association, and degree of liberation. As shown in Table 4.1, the Wits pyrite concentrate contained 3.01 g/t gold and 0.94 g/t silver. The sample was analysed eight times for gold (standard deviation of 0.24) and silver (standard deviation of 0.69) to eliminate the

error of the nugget effect. The nugget effect is a sampling error when there are large, valuable gold particles in a given sample. Two gold analysis outliers (3.22 and 3.58) were identified as the nugget effect. The standard deviation was less than one in both instances, proving good quality analysis.

Table 4.1. Total gold and silver analysis

	Number of analyses (n)	Unit	Au	Ag
Wits Pyrite Concentrate	1	g/t	3.22	0.71
Wits Pyrite Concentrate	2	g/t	2.91	0.69
Wits Pyrite Concentrate	3	g/t	2.91	0.76
Wits Pyrite Concentrate	4	g/t	2.85	0.80
Wits Pyrite Concentrate	5	g/t	2.88	0.68
Wits Pyrite Concentrate	6	g/t	2.94	0.71
Wits Pyrite Concentrate	7	g/t	2.81	1.70
Wits Pyrite Concentrate	8	g/t	3.58	1.50
Average grade n=8		g/t	3.01	0.94

In addition to precious metals, the sample contained aluminium, iron, silica, copper, nickel, titanium, chromium, and sulphur species. Table 4.2 shows that V, Co, Ca, Mg, Mn, Pb, and Zn were detected at less than 0.05%. Although these base metals may occur in low concentrations, they can still affect gold dissolution and leach simultaneously with gold and silver during cyanidation (Oraby, Eksteen, and Tanda 2017; Yliniemi et al. 2018; Wu, Ahn, and Lee 2021). The chemical analysis indicated the presence of non-oxidised sulphide minerals in the tails. These are believed to be iron-sulphide minerals with a small content of base metal sulphides containing Ni, Co and Cu. In comparison to the gold grade, these metals are more predominant and soluble during cyanidation, and therefore these ions in solution affect the solubility of gold. Moreover, the sample contained carbon, of which 0.76% is organic carbon, which can have a pre-robbing effect during cyanidation.

Table 4.2. Bulk chemical analysis

Au	**g/t**	3.01	Ni	**%**	0.11
Ag	g/t	0.94	Pb	%	<0.05
Co	g/t	428	Si	%	26.6
Al	%	4.34	Ti	%	0.22
Ca	%	0.31	V	%	<0.05
Cr	%	0.08	Zn	%	<0.05
Cu	%	0.08	Elemental S	%	<0.05
Fe	%	13.7	Total sulphur	%	12.4
Mg	%	0.47	Sulphate SO_4	%	0.94
Mn	%	<0.05	Sulphide S	%	11.7
Organic carbon	%	0.76	Total carbon	%	3.41

4.3 Association of gold to silver and gangue minerals in a Wits pyrite concentrate

The nature of gold was determined by identifying and locating all gold minerals and their liberation, using Quantitative Evaluation of Materials by Scanning Electron Microscopy (QEMSCAN). To locate and identify gold and electrum, a specific mineral search (SMS) was implemented. This allowed the QEMSCAN to analyse mineral phases within a defined Backscattered Electron (BSE) range of 176, which was specifically designed to contain high-density minerals of interest. To include silver-bearing phases in the selected BSE range of the SMS search, an acanthite (Ag_2S) standard was analysed; the EDS is shown in Figure 4-3, the BSE of this phase, was well inside the SMS range of 176. Despite this, no free silver-containing phases were found.

Automatic image analysis of Backscattered Electron (BSE) and Energy Dispersive X-ray (EDX) signals from a Scanning Electron Microscope (SEM) were employed to generate digital images in which each pixel corresponds to the mineral species in the appropriate region under an electron beam. Species Identification Protocol (SIP) development was used to accomplish mineral processing (classification and identification). SIP is a library of analysed phases classified as specific minerals according to their BSE and chemical composition compared to that within a user-developed, reference mineral library (Warlo et al. 2019).

In this study, only one electrum gold grain was identified, representing 2% (mass by mass) of the total gold grains. The Ag grade in the particle is variable as it occurs within a gold phase

and not on its own with no clear boundaries to indicate the phase change. Therefore, it was not accurately analysed by EDS. However, the SIP used to identify the phases is robust and accurate in identifying all electrum particles, even in minor quantities. The QEMSCAN descriptions and quantitative data were produced from two-dimensional (x, y) measurements of the polished section surfaces.

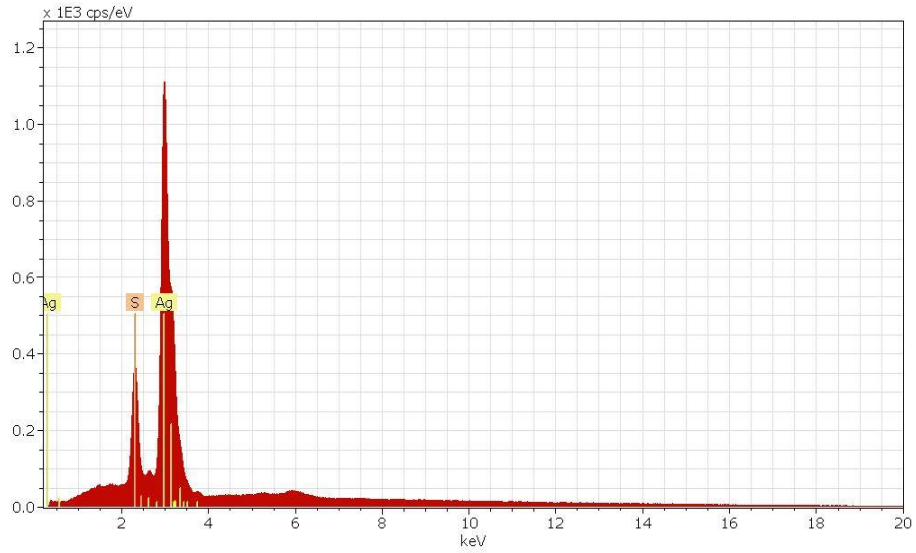


Figure 4-3. EDS Spectrum of the acanthite standard

Figure 4-4 illustrates the gold-bearing phases in the sample in terms of area and mass percentage. The sample contained two types of gold phases, namely electrum (2% of the gold content) and native gold (98%). These are natural gold phases found in Wits ores (Janisch 1986).

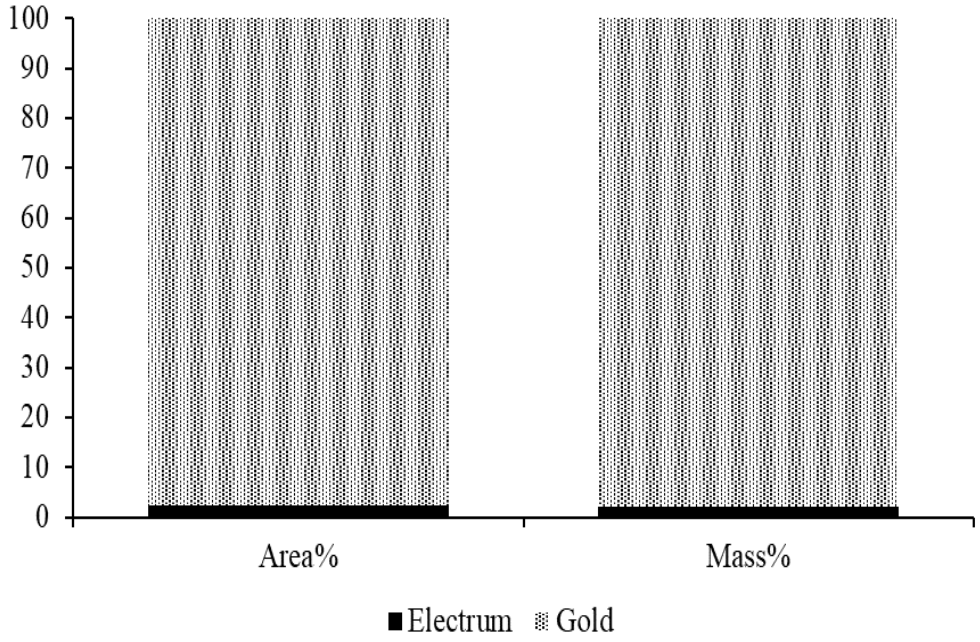


Figure 4-4. Wits pyrite gold occurrence

A single electrum grain was identified in the sample, with an Ag content of 22.3% and was associated with gold. Most of the grains characterised had low silver concentrations, and significant numbers of gold grains were present. An Energy Dispersive Spectroscopy (EDS) was used to determine the silver content of the gold grains and confirmed that silver cannot tarnish the gold surface, as there is less than 25% silver in all gold grains. Tarnishing occurs due to the oxidation of electrum when it reacts with sulphur, water and oxygen. Subsequently, corrosion takes place on the surface of the electrum, leading to the formation of a silver sulphide layer on the gold surface (Khalid, Larachi, and Adnot 2018; Liu et al. 2013).

The Mineral Liberation Analyser (MLA) found a total of 33 grains, of which 32 (97%) were identified as native gold grains (Table 4.3), and only one grain was identified as an electrum grain. The silver concentration of the native gold grains was less than 9% and 12% of the grains had no silver content. The gold fineness values (purity of gold grains in parts per 1000) shown in Table 4.3 ranged between 900 and 1000 ppt, similar to those of the Archaean deposit. Archaean deposits typically have a high and consistent average fineness and a narrow fineness range (Morrison, Rose, and Jaireth 1991).

Table 4.3. The silver content of the gold grains

Grain No.	Ag Content %	Fineness ppt	Grain No.	Ag Content %	Fineness ppt
1	6.5	938	17	5.4	948
2	6.3	940	18	0	1000
3	7.9	925	19	5.9	943
4	2.6	974	20	5.0	951
5	5.7	945	21	5.4	948
6	0	1000	22	11	899
7	5.9	943	23	4.0	961
8	1.9	981	24	8.2	923
9	5.7	945	25	6.3	940
10	6.7	936	26	7.0	933
11	10	907	27	0	1000
12	5.9	943	28	2.8	972
13	8.1	924	29	1.3	987
14	0	1000	30	4.3	958
15	11	902	31	7.0	933
16	1.9	981	32	6.1	941

4.4 Characterisation of gold-associated sulphide minerals in Wits pyrite concentrate

Photographs of each gold grain were taken to determine their association with iron-sulphide minerals. Figure 4-5 displays the different gangue mineral associations with gold, where gold was either liberated (Figure 4-5-A, B, K and L), associated (Figure 4-5-J and I), or locked (Figure 4-5-C, D, E, F, G and H) within pyrite, pyrrhotite, and arsenopyrite. Pyrite and pyrrhotite were the primary gold-bearing minerals, and thus, pre-treatment of these iron-sulphide minerals would expose the gold particles and improve the efficiency of cyanidation (Rabieh et al. 2017; Teague, Van Deventer, and Swaminathan 1999).

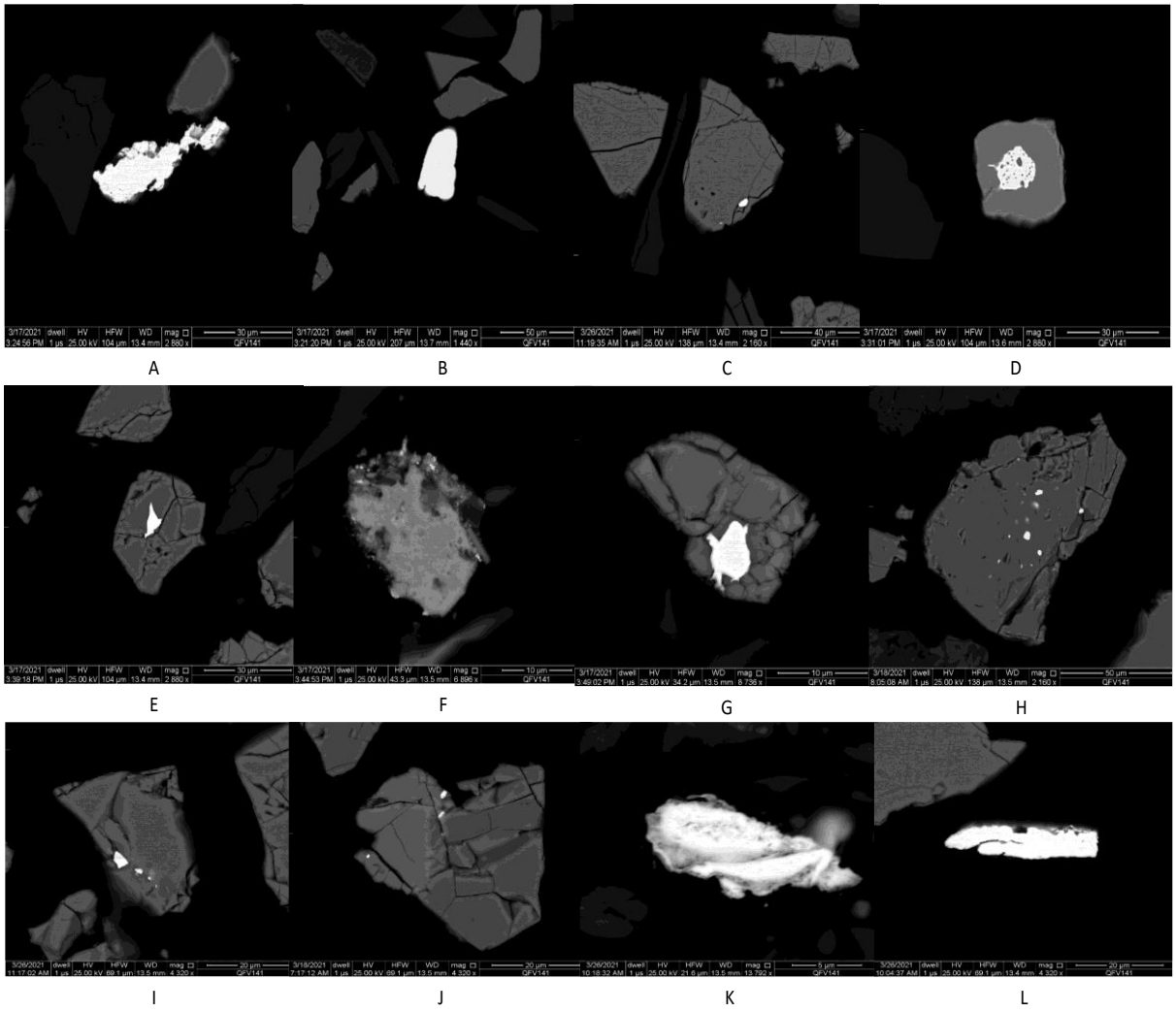


Figure 4-5. Photomicrographs of Wits pyrite concentrate gold grains and associated gangue

A	Liberated electrum	G	Gold locked within pyrrhotite
B	Liberated gold	H	Gold locked within pyrite and pyrrhotite
C	Gold locked within pyrite	I	Gold associated with pyrite
D	Gold locked within arsenopyrite	J	Gold associated with chalcopyrite and pyrite
E	Gold locked within pyrite	K	Liberated gold
F	Gold locked within a Cl-bearing silicate	L	Liberated gold

Table 4.4 presents the mineralogy of bulk samples and the mineralogical distribution of gold in the sample, as analysed by the MLA. The results indicate that 33% of the native gold grains are fully liberated and all electrum particles are fully liberated. Furthermore, 33% of the native gold is leachable by cyanide. The majority of the remaining gold (64%) was found to be associated with iron-sulphide minerals, while only a small fraction (3%) was associated with silicates, as shown in Table 4.4.

Table 4.4. Bulk mineral and gold association analysis (Liu and Beaudoin 2021; Reyes et al. 2018)

Mineral phase	Formula	Content %	Mineral type	Gold associated %	Electrum associated %
Liberated Gold	Au		Free milling	33	100
Quartz	SiO2	45	Silica	2	0
Mica	KAl2(AlSi3O10)(F,OH)2	17			
Chlorite	(Mg,Fe)5(Si3Al)O10(OH)8	6.7			
Gypsum	CaSO4·2H2O	1.9	Sulphate	0	0
Feldspar	KAlSi3O8	5.3	Aluminium tectosilicate	0	0
Pyrophyllite	Al2Si4O10(OH)2	1.9	Phyllosilicate	1	0
Pyrite	FeS2	16	Iron-sulphide	38	0
Pyrrhotite	FeS	4.3		21	0
Arsenopyrite	FeAsS	2.1		5	0

The elemental composition of each gangue mineral was determined and presented in Table 4.5 and then compared to the bulk chemical analysis presented in Table 4.6. The mineralogy and elemental data for each element were found to be similar.

Table 4.5. Mineral composition from XRD analysis (%)

Compound Name	Al	Ca	Fe	K	Mg	Mn	Si	S
Chlorite	6.2	0.0	6.4	9.0	14.0	0.0	19	0.0
Gypsum	0.0	23	0.0	0.0	0.0	0.0	0.0	19
Mica	20	0.0	0.0	9.8	0.0	0.0	21	0.0
Feldspar	9.7	0.0	0.0	14	0.0	0.0	30	0.0
Pyrite	0.0	0.0	46	0.0	0.0	0.0	0.0	54
Pyrophyllite	15	0.0	0.0	0.0	0.0	0.0	31	0.0
Pyrrhotite	0.0	0.0	62	0.0	0.0	0.0	0.0	38
Quartz	0.0	0.0	0.0	0.0	0.0	0.0	47	0.0
Arsenopyrite	0.0	0.0	34	0.0	0.0	0.0	0.0	0.0

Table 4.6. Mineral chemical reconsolidation (%)

Compound Name	Mineral Chemical Reconciliation					
Chlorite	Al	Ca	Fe	Mg	Si	S
Gypsum	0.54	0.00	1.96	0.37	0.85	0.00
Mica	0.00	0.44	0.00	0.00	0.00	0.35
Feldspar	3.39	0.00	0.00	0.00	3.53	0.00
Pyrite	0.51	0.00	0.00	0.00	1.60	0.00
Pyrophyllite	0.00	0.00	7.59	0.00	0.00	8.71
Pyrrhotite	0.28	0.00	0.00	0.00	0.58	0.00
Quartz	0.00	0.00	2.65	0.00	0.00	1.60
Arsenopyrite	0.00	0.00	0.00	0.00	21.1	0.00
Chlorite	0.00	0.00	0.71	0.00	0.00	0.41
Mineralogy Total	4.47	0.44	12.9	0.37	27.6	11.1
Chemical Total	4.43	0.31	13.7	0.47	26.6	12.4

According to the process mineralogy study, gold exists in the sample primarily as native gold, which can be easily leached by cyanide if exposed, with a silver content of less than 20% (Henley, Clarke, and Sauter 2001). However, gold particles that are locked and associated with iron-sulphide minerals may cause high reagent consumption, leading to the depletion of cyanide and oxygen in the pulp due to their reaction with the constituents of the pulp, such as sulphides, iron, and base metals generated during cyanidation (Abdel and Aloub 2018). These

elements compete with gold for cyanide, slowing down the process of gold dissolution (Bas, Ghali, and Choi 2017). The liberation of the various size fractions of discrete gold particles by surface is shown in Figure 4-6. The liberation distribution can be classified according to Table 4.7, which presents the abundance and gold grade distribution of gold phases in a population of posterity particles that vary in size. The gold surface liberation data showed that all 53 µm gold particles are locked, while all 10 µm gold particles are liberated, indicating that the product of UFG should be 100% -10 µm to achieve 100% gold extraction during cyanidation. The liberated gold particle surfaces of the combined sample were found to be 48%. The concentrate also contained low middlings, which are considered partially locked and can be liberated by the application of UFG (Coetzee et al. 2011). The majority of the gold particles were either smaller than 10 µm or between 50 or 60 µm. additionally, fine particles smaller than 1 micron were present in significant amounts (Karimi et al. 2010). The distribution of gold particle sizes could make it difficult to dissociate the gold particles by ultra-fine grinding (Wang et al. 2019).

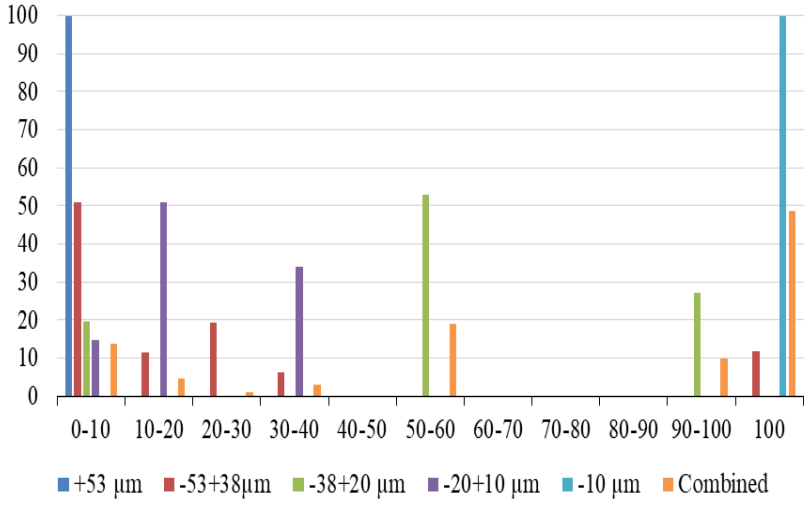


Figure 4-6. Discrete gold grain liberation by surface

Table 4.7. Liberation classification (Sousa et al. 2018)

Locked	<= 10%	High Middling's	<= 60%
	<= 20%		<= 70%
	<= 30%		<= 80%
Low Middling's	<= 40%	Liberated	<= 90%
	<= 50%		< 100%

The dissolution of gold during cyanidation is facilitated by decreasing particle size due to the increased contact surface area, which prevents surface passivation of the gold grains. This size-dependent effect suggests that separate treatments may be required for fine and coarse particles (Harris 1990; Xu and Wang 2015; Zhou and Cabri 2004). The gold particles in the Wits pyrite concentrate all report to the +25-µm fraction, with 86% passing 212 µm, as shown in Table 4.8. The tailings contain a range of fine and coarse gold particle sizes, with the majority of the gold being less than 75 µm in size but greater than 53 µm. Invisible gold is found within unaltered pyrite and arsenopyrite domains, while visible gold is associated with a later-stage alteration event that produced chemically modified alteration rims on these minerals.

Table 4.8. Particle size assay

Screen size	**Mass**	**Retained**	**Cumulative passing**	**Au**
µm	g	%	%	g/t
425	0.78	0.08	100	3.01
212	140	14.0	86	3.17
106	235	23.5	62	2.71
75	295	29.5	33	2.99
53	265	26.5	6	2.83
38	51.0	5.10	1	3.23
25	11.8	1.18	0	-
Total	1000	100		0
Head grade				3.01
Calculated head grade				3.07

The Grain Size Distribution (GSD) analysis of the Wits pyrite concentrate is presented in Figure 4-7, showing a range of gold-bearing minerals from macroscopic to sub-microscopic. The Wits pyrite concentrate contains a considerable amount of microscopic gold (ranging from 100 to 1 µm), which is suitable for cyanidation (Tourtelot and Riley 1973). Gold in the Wits pyrite concentrate occurs as fine-grained inclusions (visible gold), chemically bonded (sub-microscopic gold), or both in pyrite and arsenopyrite. Acceptable gold extraction can only be achieved by applying a pre-oxidation step before cyanidation. Fine grinding is not expected to increase extraction significantly as the majority of the gold particles in the Wits pyrite concentrate are already at a grind size suitable for cyanidation (100 to 1 µm) (Tourtelot and Riley 1973).

In terms of free surface liberation, the gold is moderately liberated, with 58% of the gold occurring in the liberated fraction. The auriferous phase is predominantly associated with pyrite and pyrrhotite, as shown in Figure 4-5. Therefore, fine grinding will liberate other competing elements, leading to high reagent consumption and uneconomical outcomes.

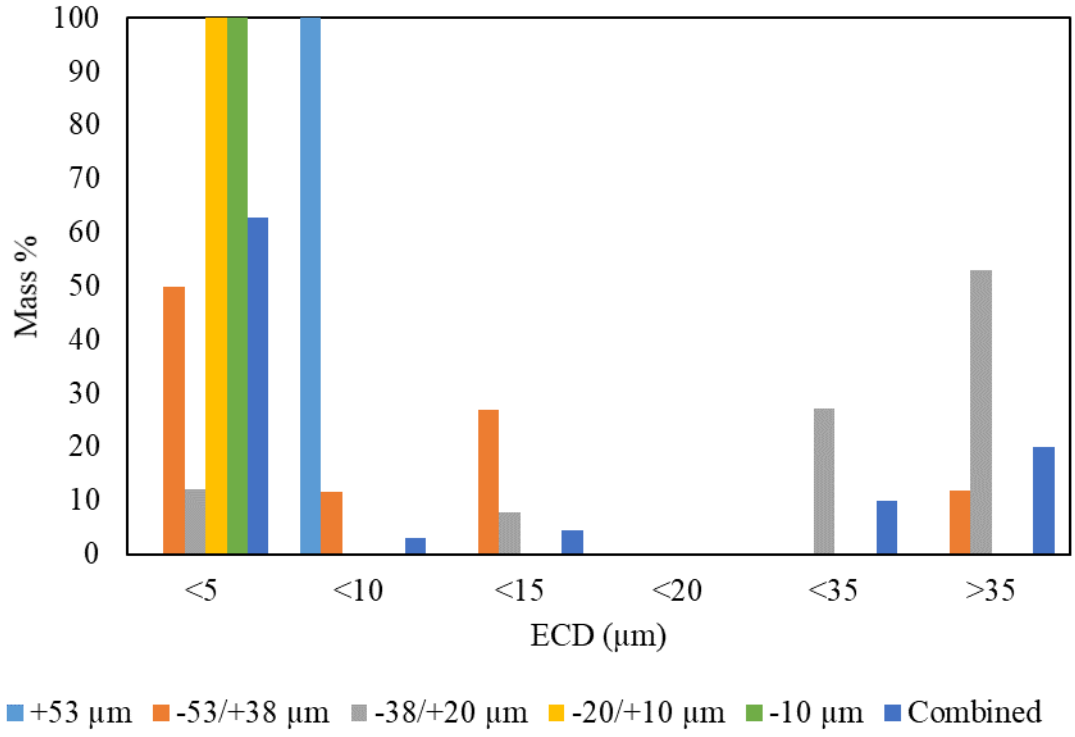


Figure 4-7. Grain size distribution (GSD)

3.6.1 Reactivity of iron-sulphide minerals in a Wits pyrite concentrate

To investigate the reactivity of iron-sulphide minerals and their impact on gold dissolution, a two-stage leaching test was conducted using the conditions presented in Table 3.4. The residues were analysed for pyrite, pyrrhotite, and arsenopyrite. Table 4.9 illustrates the mass balance of the two-stage leach. Results indicated that 41% of the gold was leached in the first stage of leaching, and an additional 65% was leached in the second stage. Gold leaching was enhanced when additional cyanide solution was added to the residue of stage 1, resulting in an overall gold extraction of 83%. The addition of fresh cyanide shifted the equilibrium, and more free cyanide was available to react with gold, extending gold extraction. Once the surface gold was depleted, the addition of fresh cyanide caused the leaching of the faster-leaching minerals closer to the centre of the large ore particles. This improvement is similar to the increase in

PGM extraction observed when two-stage heap leaching was applied to Platreef flotation concentrates (Mwase, Petersen, and Eksteen 2012). The detailed mass balance and mineralogy analysis are presented in sections 9.5 and 9.6.

Table 4.9. Two-stage leach mass balance

Sample ID	IN Head				OUT Residue		OUT Filtrate			Accountability	Au Extraction (Residue)
	Mass Feed	Au	Au	Mass of residue	Au	Au	Mass filtrate	Au	Au	Au	Au
	g	g/t	mg	g	g/t	mg	g	mg/L	mg	%	%
Stage 1	**5000**	**3.01**	15.1	4950	2.24	6.50	7900	1.13	8.89	102%	41%
Stage 2	**2000**	**2.24**	4.48	1967	1.30	2.60	5188	0.30	1.56	93%	65%

The findings demonstrated that the iron-sulphide minerals, particularly pyrrhotite, are soluble during cyanidation. The dissolution of gold and silver depends on the stability constants of the competing complexes, and the presence of high levels of iron in the solution can lead to a retarding effect on the formation of the gold-cyanide complex by consuming the required cyanide. However, the addition of fresh cyanide can alleviate the retarding effect.

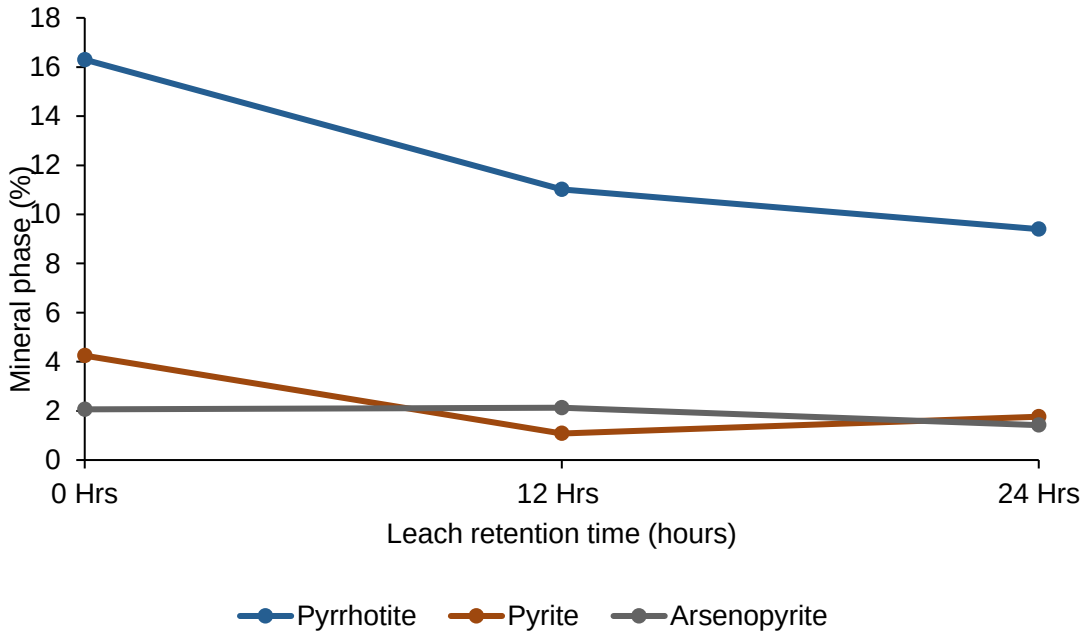


Figure 4-8. Iron-sulphide solubility

4.5 Gangue minerals limitation on the dissolution of gold during cyanidation

The results of the diagnostic leach test illustrated in Figure 4-9 indicate that intensive cyanidation (10 kg NaCN per tonne of ore) was able to extract 80% of the gold, which is 22% more than the 58% liberated gold found in the MLA analysis. This suggests that the Wits pyrite concentrate is refractory as the diagnostic intensive cyanidation stage resulted in less than 90% even with extensive additional NaCN usage. The observation indicates that high cyanide concentration caused the partial dissolution of iron-sulphide minerals, thereby dissolving 80% of the gold, which was liberated and associated or locked with the gangue minerals. This finding further confirms that cyanide is the limiting reagent during the cyanidation of the Wits pyrite concentrate. A further 15.6% of gold can be extracted with rigorous chemical treatment of the iron-sulphide minerals pyrrhotite (5.1%), pyrite, and arsenopyrite (10.5%), with less than 1% gold loss due to pre-robbing. Iron-sulphide minerals tend to slow down gold dissolution kinetics; therefore, chemical pre-treatment of these minerals would improve the dissolution rate of gold. The detailed mass balance is presented in section 9.7.

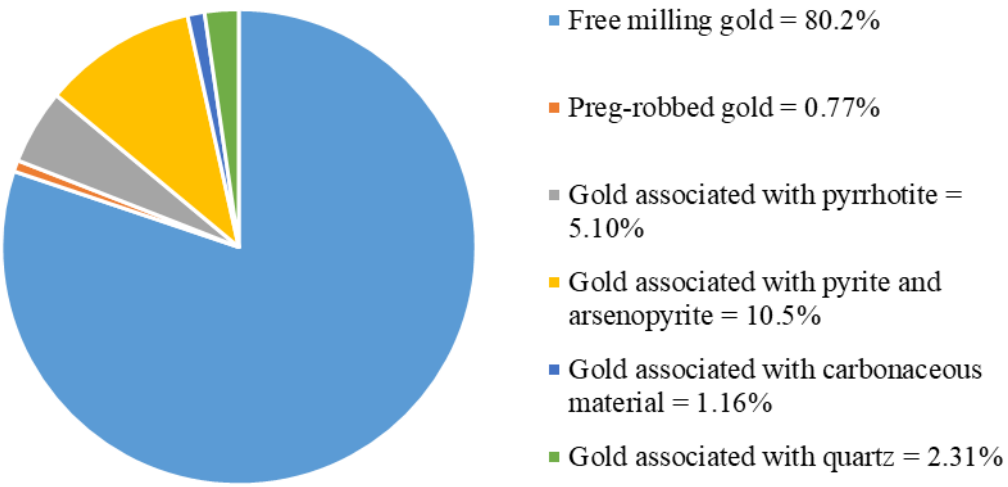


Figure 4-9. Gold deportment

According to the diagnostic leach results, 80% of the gold in the Wits pyrite concentrate can be extracted by adding 10 kg/t of NaCN, with less than 1% of the gold being preg-robbed. Of the remaining gold, 5.1% and 10.5% are associated with pyrrhotite and pyrite/arsenopyrite, respectively. These findings suggest that 97.7% of the gold can be leached with access to NaCN, if activated carbon is added to overcome pre-robbing (0.77%). Additionally, 15.6% of the gold is locked in iron-sulphide minerals that require oxidation, while 1.16% is locked in

carbonaceous material that requires roasting for liberation and 2.31% is locked in quartz, requiring UFG.

4.6 Conclusion

Gold in the Wits pyrite concentrate predominantly occurs as native gold, with a small average grain size (5 µm), which facilitates easy leaching with cyanide. The native gold grains also contain a low silver content, with less than 9% of the identified grains classified as electrum. Electrum accounts for a minor component (2 mass percentage) of the total precious metal phase, indicating that it is not the cause of poor gold extraction. Rather, it is the gold particles associated with reactive gangue minerals, namely iron sulphides pyrite and pyrrhotite, which cause poor gold extraction. During cyanidation, these minerals can partially dissolve in an alkaline medium, or restrict cyanide access to gold particles. The dissolution of iron and sulphide minerals associated with gold, caused by the attack of cyanide, further competes with the formation of the gold-cyanide complex, thereby retarding gold extraction. Therefore, pre-treating the pyrrhotite and pyrite is necessary to reduce their adverse effects on gold dissolution. Based on the association and locking of gold by iron-sulphide minerals, two options were identified to improve gold extraction:

1. Increase cyanide addition to compensate for the leachable base metals and iron sulphides that compete with gold through undesired reactions.
2. Dissolve or passivate the iron-sulphide mineral surfaces using a combination of pre-oxidation with lime, oxygen, and lead nitrate conditioning to ensure the dissolution of all exposed gold.
3. Decrease the solids: liquid mass ratio to compensate for the overload of base metals and sulphate that can form from the oxidation of iron-sulphide minerals.

CHAPTER 5

5. BASELINE GOLD EXTRACTION EFFICIENCIES USING ERGO CONDITIONS

5.1 Introduction

The DRDGold ERGO cyanidation plant conditions were used to leach the Wits pyrite concentrate to evaluate and benchmark the gold extraction efficiencies observed on the plant in 2018. It was established that gold and silver leached simultaneously. The extraction of gold gradually increased to 71% after 24 hours, which is the current CIL retention time at DRD, and reached a maximum of 78% after 60 hours of leaching. In comparison, DRDGold achieved 55% gold extraction at plant conditions, while laboratory tests achieved 71% leaching at the same 24-hour retention time due to the addition of 2 kg/t NaCN instead of 0.66 kg/t and the reduction of solids: liquid mass ratio.

Extending the retention time to 60 hours achieved 78% gold extraction, gaining an additional 7% with 36 hours of leaching time. However, this would require a high capital investment in terms of the addition of CIP tanks, and longer retention time would result in gold being locked in the CIP circuit for extended times, reducing the feed flow rate to the elution plant. To reduce the time required to leach 78% of the available gold from a Wits pyrite concentrate, it is suggested to introduce pyrite and pyrrhotite pre-oxidation, which breaks down the gangue minerals, allowing cyanide faster access to the gold particle. Pre-oxidation not only improves dissolution kinetics but also increases cyanide and lime consumption, as more elements such as base metals and sulphate will be available in the solution to react with cyanide and lime, justifying the increase in NaCN addition and reduction in solids: liquid mass ratio from 43% to 33%.

During the first 4 hours of cyanidation, the lime consumption was high, with low gold extraction, indicating that the iron-sulphide minerals used up the lime added. Cyanide consumption remained above 0.5 kg/t after 3 hours despite the slow dissolution of gold. During the cyanidation of the Wits pyrite concentrate, both cyanide consumption and lime addition increased with longer cyanidation retention time.

5.2 Baseline gold extraction and reagent consumption

The Wits pyrite concentrate was subjected to cyanidation using the DRD ERGO plant condition, with an increased NaCN addition of 2 kg/t and a low pulp density for improved mass transfer, as shown in Table 3.4. This was done to evaluate the baseline cyanidation performance at an extended leaching time of 72 hours. The mass balance of the kinetic tests is presented in Table 5.1. During the first 4 hours, gold extraction remained below 50%, but it gradually increased to over 60% and ultimately reached 78% after 72 hours of cyanidation. The extraction of gold increased gradually during the leach duration, with a gold extraction of 71% after 24 hours, which is the current CIP retention time at DRD ERGO. Notably, ERGO reported 55% gold extraction from the same sample in their CIP circuit when using 0.66 kg/t cyanide (Staden et al. 2020). Gold extraction increased to a maximum of 78% after 60 hours and remained stable. A longer retention time would require a high capital investment in terms of the addition of CIP tanks. Furthermore, a longer retention time would lock gold in the CIP process for longer, reducing the feed flow rate to the elution plant.

8. During the first 4 hours of cyanidation, the lime consumption was high, with low gold extraction, indicating that the iron-sulphide minerals used up the added lime. The introduction of chemical oxidation of pyrite and pyrrhotite is suggested to reduce the time required to leach 78% of the available gold from a Wits pyrite concentrate, as pre-oxidation breaks down the gangue minerals, allowing cyanide faster access to the gold particle. The gold accountability was well within the range of ±10%. The detailed mass balance, measured parameters, and reagent consumption of the tests are shown in APPENDIX B, section 10.1.

Table 5.1. Gold mass balance of ERGO conditions baseline kinetic tests

Retention time	In head		Out residue		Out carbon		Accountability	Au extraction (residue)
	Au	Au	Au	Au	Au	Au	Au	Au
	g/t	mg	g/t	mg	g/t	mg	%	%
0	3.01		3.01					0.00
1	3.01	1.51	2.38	1.19	12.1	0.32	100	20.9
2	3.01	1.51	2.31	1.16	13.7	0.36	101	23.3
3	3.01	1.51	2.25	1.12	17.4	0.46	105	25.3
4	3.01	3.01	1.90	1.90	24.3	1.26	105	36.9
6	3.01	3.01	1.04	1.04	38.1	1.98	100	65.4
8	3.01	3.01	1.00	1.00	39.1	2.03	101	66.8
10	3.01	3.01	0.96	0.96	39.5	2.05	100	68.1
12	3.01	3.01	0.92	0.92	40.1	2.09	100	69.4
18	3.01	3.01	0.86	0.86	41.9	2.18	101	71.4
24	3.01	3.01	0.86	0.86	43.2	2.24	103	71.4
36	3.01	3.01	0.82	0.82	44.8	2.33	105	72.8
48	3.01	3.01	0.76	0.76	45.2	2.35	103	74.8
60	3.01	3.01	0.65	0.65	45.7	2.38	101	78.4
72	3.01	3.01	0.64	0.64	47.2	2.45	103	78.7

Figure 5-1 depicts the gold extraction kinetics, which reveals a slow start to gold dissolution followed by a sharp increase after 4 hours. In order to achieve adequate gold dissolution from the sample, the leaching efficiencies, including Au extraction, residual gold, and reagent consumptions, as well as parameters such as redox potential and dissolved oxygen, were interpreted to identify favourable leaching conditions.

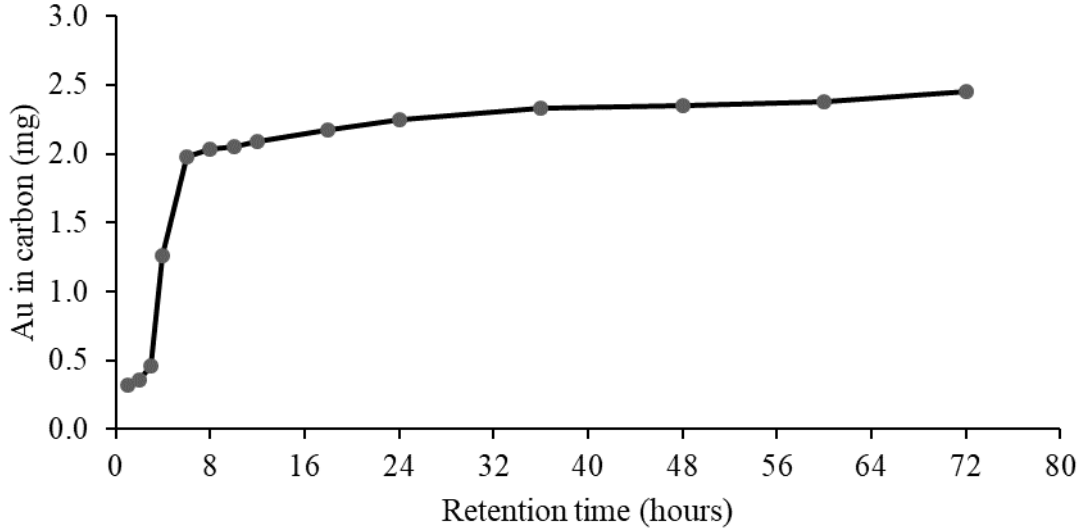


Figure 5-1. Baseline carbon loading kinetics

In Figure 5-2, the gold extraction achieved is compared to the consumption of lime and cyanide. The results show that gold extraction remained below 50% during the first 4 hours, increased to 71% at 24 hours, and ended at 78% after 60 hours of cyanidation. During the first 4 hours of cyanidation, the lime consumption was high, and gold extraction was low, indicating that the iron-sulphide minerals consumed the added lime. However, cyanide consumption remained steady between 0.4 and 0.6 kg/t during the 72 hours, suggesting that longer retention times did not consume more cyanide since all leachable gold was leached in 24 hours. Studies have shown that pyrite is unreactive under mildly oxidising cyanide-leaching conditions unless partially oxidised due to the tailings storage conditions. This suggests that the additional gold dissolution is more likely from pyrrhotite than pyrite. Pyrrhotite is more stable under reducing conditions than pyrite and tends to oxidise and dissolve during cyanide leaching.

During the cyanidation of the Wits pyrite concentrate with increased NaCN (2 kg/t) addition and reduced solid: liquid mass ratio (1:2), cyanide and lime consumption remained consistent after 4 hours of leaching. This suggests that enough cyanide was available to the gold particle for dissolution, and increasing the mass transfer improves the gold dissolution kinetics.

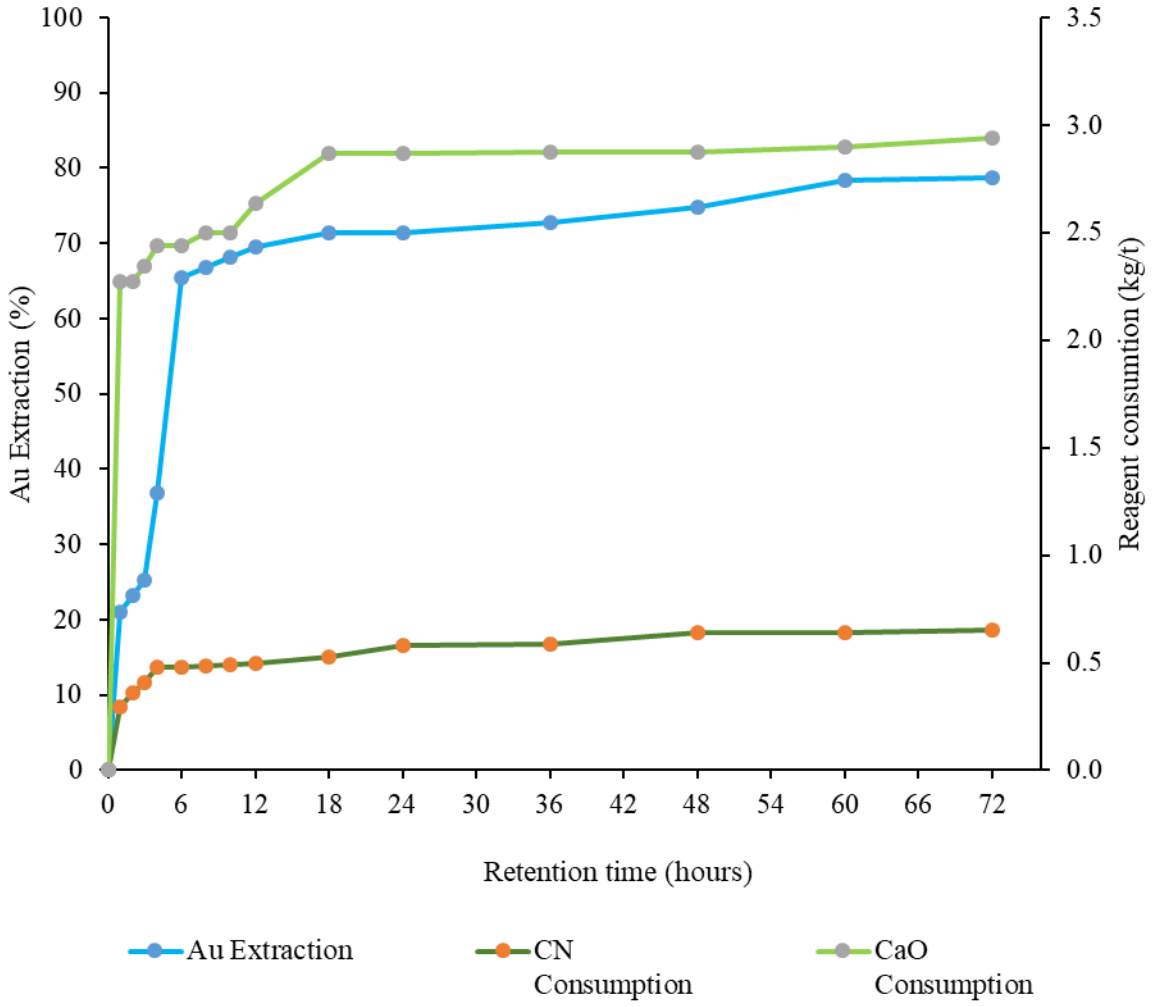


Figure 5-2. Baseline gold extraction vs reagent consumption

The cyanidation of gold ores containing iron-sulphide minerals leads to significant consumption of oxygen, resulting in dissolved oxygen (DO) starvation during the initial stages of the leaching process. As demonstrated in Figure 5-3, the DO level decreased to nearly zero within the first 4 hours due to the iron-sulphide minerals consuming all available oxygen. During this period, the rate of gold extraction was 18%, and the oxidation-reduction potential (ORP) was highly negative, indicating that the conditions were reductive rather than oxidative (Korolev et al. 2018; Kucernak 2007; Nunan et al. 2017). However, after 4 hours, the DO level increased to above 30 mg/L, leading to an increase in the gold extraction rate (up to 60%) and a less negative ORP, indicating that the conditions were now oxidative and more favourable for gold extraction. This highlights the importance of maintaining adequate DO levels during cyanidation to maximise gold recovery.

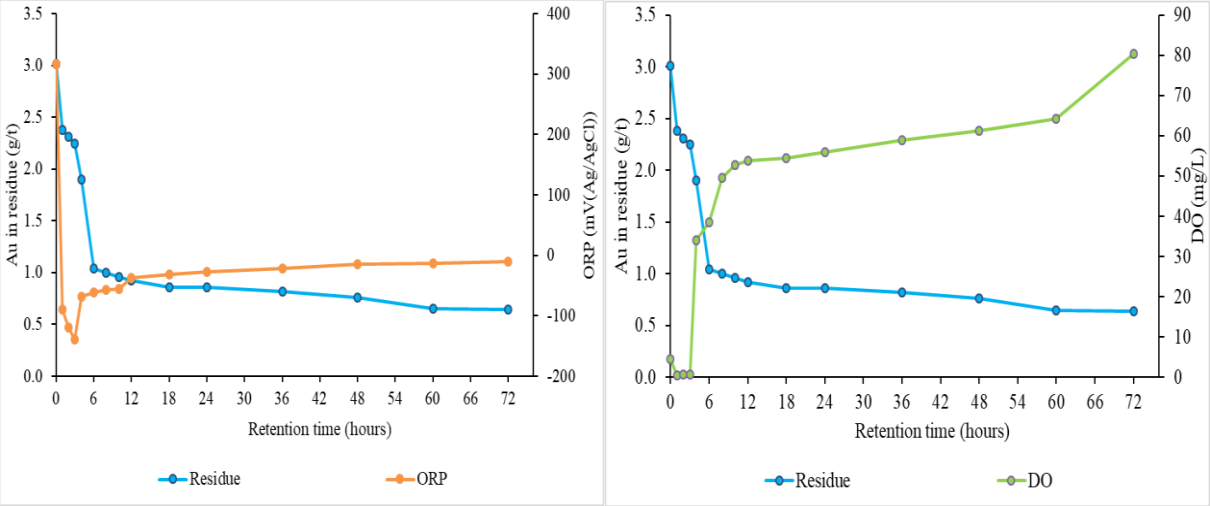


Figure 5-3. Dissolved oxygen + ORP versus gold dissolution

5.3 Base metal dissolution of the Wits pyrite concentrate

In gold sources, base metals often need to be partially removed from the ore before cyanidation, as they can negatively influence the process. As depicted in Figure 5-4, aluminium, iron, silica, copper, magnesium, nickel and titanium were present in high concentrations of above 2% in the residue, remaining constant over the 72-hour reaction period. The presence of soluble iron in iron-containing ores can result in excessive cyanide and lime consumption during cyanidation. To address this issue, pre-oxidation is recommended before the cyanidation of iron-containing gold ores to make the pyrite and pyrrhotite soluble. Although cyanidation of iron-sulphide-containing ore is technically feasible, pre-oxidation can improve the solubility of pyrite and pyrrhotite, making them more water-soluble. (Larrabure and Rodríguez-Reyes 2021).

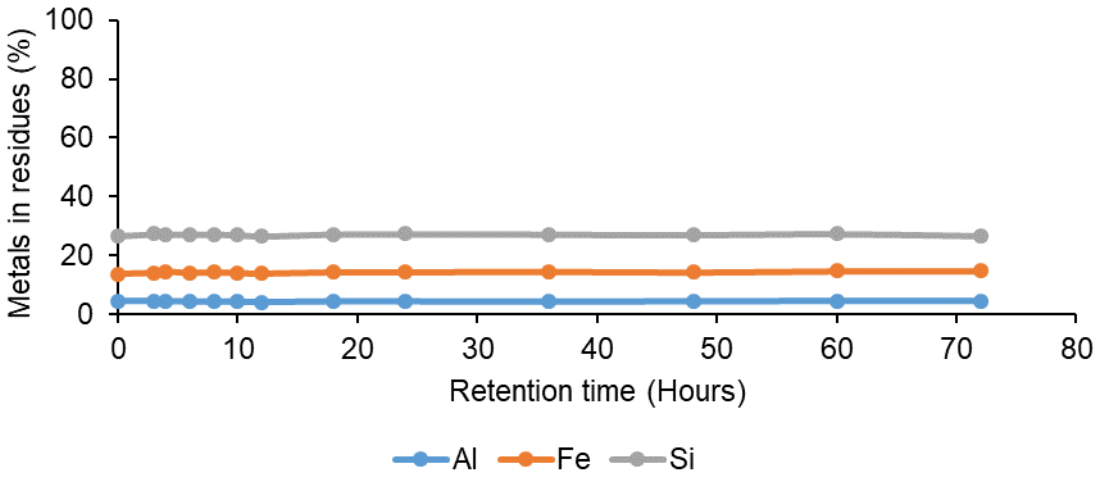


Figure 5-4. Aluminium, Iron, and Silica in baseline cyanidation residues

Iron reacts with cyanide to form both soluble and insoluble cyanide complexes, and this reaction preferentially occurs with iron rather than gold, depending on the pH and Eh values of the solution, as depicted in Figure 5-5. During cyanidation at pH 11 and Eh below zero, the iron content in the residue remained constant because it was oxidised to form insoluble $Fe(OH)_3$. The decomposition of iron-sulphide minerals in alkaline cyanide solutions depends on the type of sulphide mineral present (pyrite being the least stable, marcasite and pyrrhotite being the most soluble), the particle size, and the pH and redox potential of the cyanide solution (Guo, Peng, and Espinosa-Gomez 2014).

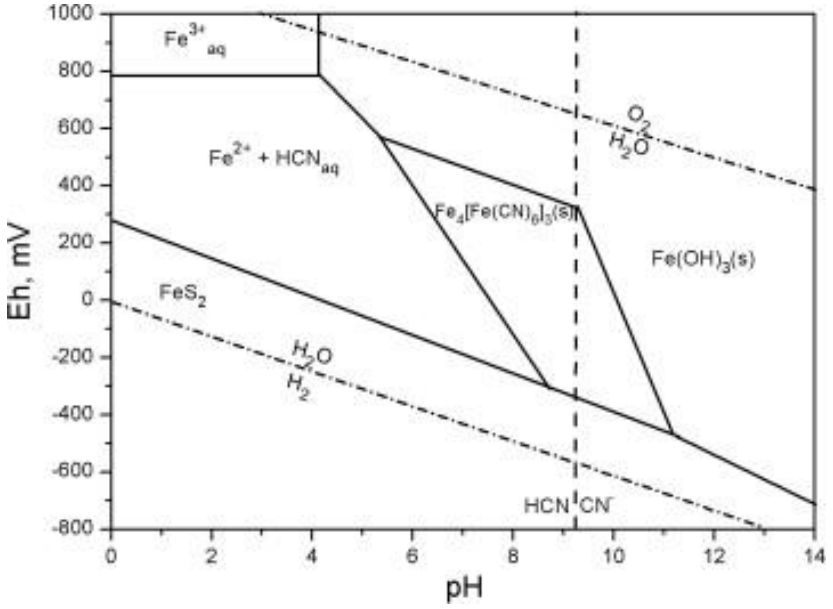


Figure 5-5. Eh-pH stability diagram for Fe–S–CN–H2O system ([Fe] = 5.0 × 10−5 M, [S] = 3.0 × 10−4 M, [CN] = 6.0 × 10−4 M) (Guo, Peng, and Espinosa-Gomez 2014)

Figure 5-6 indicates that copper, magnesium, nickel, and titanium were present in the residues at concentrations below 1%. During the first 30 minutes of leaching, there was a slight decrease in the concentration of titanium, nickel, and copper in the residue, indicating that some of these base metals leached simultaneously with gold. The content of copper and nickel in the residue continued to decrease and only stabilised after 1.5 hours of leaching. Copper and zinc can occur in gold ores as oxides, sulphides, silicates, carbonates, or as intrinsic metals (Cui et al. 2015; Larrabure and Rodríguez-Reyes 2021). Except for chalcopyrite, most copper minerals are moderate to highly soluble in cyanide solutions, leading to high oxygen and cyanide consumption due to their cyanicidal nature.

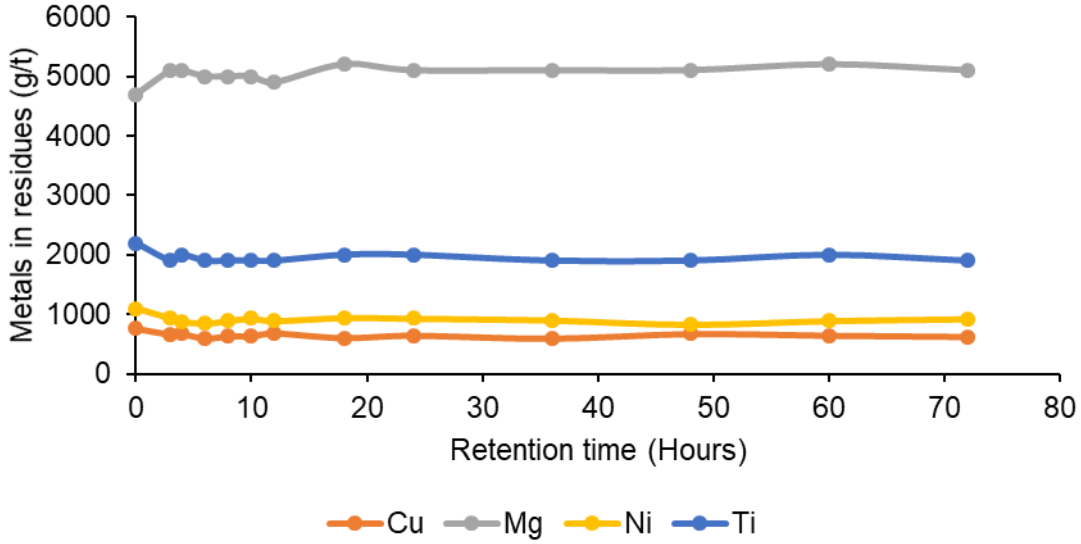


Figure 5-6. Copper, Magnesium, Nickel, and Titanium in baseline cyanidation residues

Iron sulphides have been identified as hosts of dispersed gold and silver particles, but they are refractory in nature. The physical enclosure of gold particles within iron-sulphide minerals results in poor gold dissolution due to their fine distribution. This physical enclosure causes difficulty in the dissolution of gold particles. Moreover, high pyrite content can lead to gold passivation and hinder the kinetics of gold dissolution as a result of the breakdown of iron and sulphur species during cyanidation, as pyrite is a sulphide mineral (Larrabure and Rodríguez-Reyes 2021; Teague, Van Deventer, and Swaminathan 1999).

Iron-sulphide minerals are known to host dispersed gold and silver particles and are considered refractory in nature. During cyanidation, iron-sulphide minerals cause the following negative outcomes:

- Passivation of the gold surfaces occurs due to the creation of silver sulphide (Dai and Jeffrey 2006; Larrabure and Rodríguez-Reyes 2021).
- Negative galvanic effects occur from the interaction of the gold anodic and sulphide mineral cathodic reactions, which impede gold dissolution.
- Encapsulation of the gold particles.
- Formation of elemental sulphur also contributes to the passivation of gold surfaces, which leads to reduced formation of the gold-cyanide complex and hence, poor gold extraction (Aghamirian and Yen 2005; Azizi et al. 2010).

These refractory and preg-robbing effects make direct cyanidation alone inefficient in extracting gold from Wits pyrite concentrate. As shown in Figure 5-7, all sulphur species remain in the residue and cause poor gold dissolution. The total sulphide and sulphur increase in the first 30 minutes of the leach, which is typically the time when only lime is present in the solution. This makes conditions favourable for iron-sulphide pre-oxidation (Larrabure and Rodríguez-Reyes 2021).

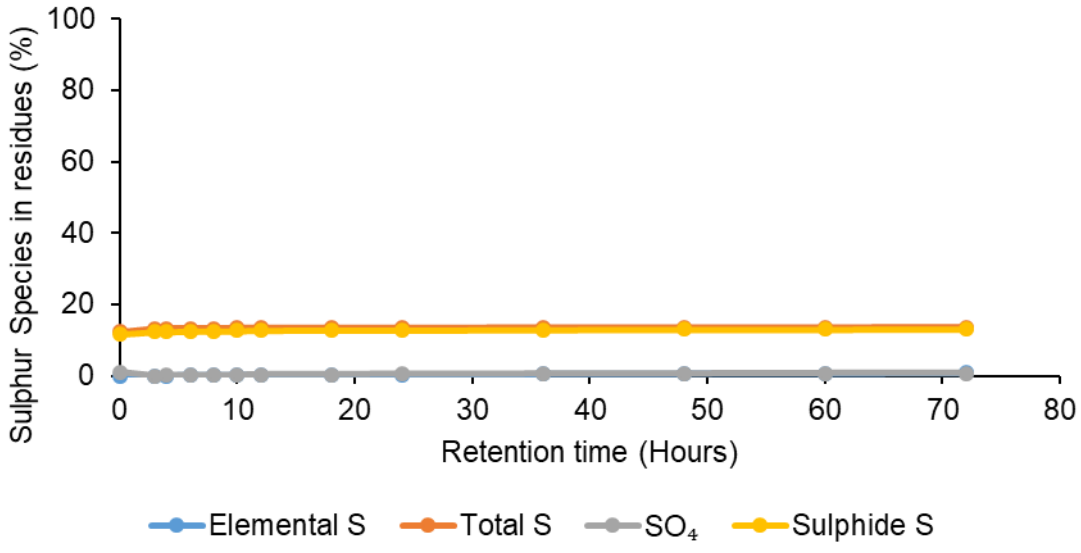


Figure 5-7. Sulphur species in baseline cyanidation residues

XRD analyses were conducted on the feed and kinetic leach residue samples to determine the mineral composition of the residues over time and identify the minerals that have an adverse

effect on the leaching process. The Bruker D8 Advance powder diffractometer, with a Lynxeye detector, was used to determine the bulk modal mineralogy of the samples. The samples were subjected to a step size of 0.02 degrees 2Θ and a counting time of 3 seconds per step. Phases were initially identified using Bruker Eva software followed by Quantitative XRD analysis using the DIFFRAC.SUITE TOPAS software package, based on Rietveld Refinement and the Fundamental Parameters Approach (FPA). The detection limit of Quantitative XRD is between 1 to 3 mass %, which enables the determination of specific amounts of different minerals present. Arsenopyrite was detected, but it was below the detection limit and was therefore not included in the analysis.

The bulk modal mineralogical composition of the kinetic sample, identified using a diffractogram (Figure 5-8), is typical of the Wits reefs. The samples were dominated by quartz, mica, and chlorite, as well as pyrite and lesser pyrrhotite and arsenopyrite (Nengovhela, Yibas, and Ogola 2006). Pyrite and pyrrhotite were closely related to gold and were the primary gold-bearing minerals (Table 4.4). Therefore, destroying these iron-sulphide minerals before cyanidation would free the gold particle associated with the mineral and positively affect the efficiency of cyanidation by allowing more cyanide to access the gold particles.

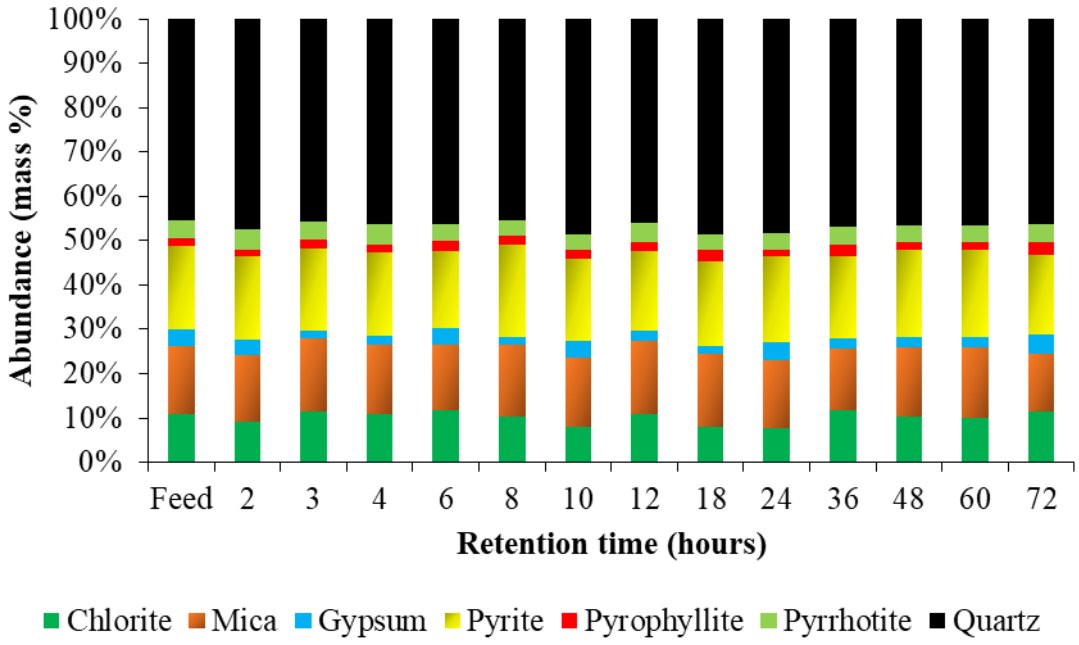


Figure 5-8. Modal mineralogy of the feed and kinetic baseline leach residue samples

The XRD analysis of the kinetic residue samples was used to determine the major mineral phases present, as the detection limit of the technique was 1%. The assay data was also used to verify the XRD results as much as possible by calculating mineral abundances. From the peaks at respective d-spacing, several minerals were identified in the samples' diffractogram (Figure 5-9). The bulk modal mineralogy of the kinetic samples remained relatively consistent, with no significant changes attributed to the leaching process, such as a reduction in the gold-locking minerals pyrite and pyrrhotite. However, some minor variations in mineralogy were present, which may be related to the detection limit of the technique and minor variations within the provided sample.

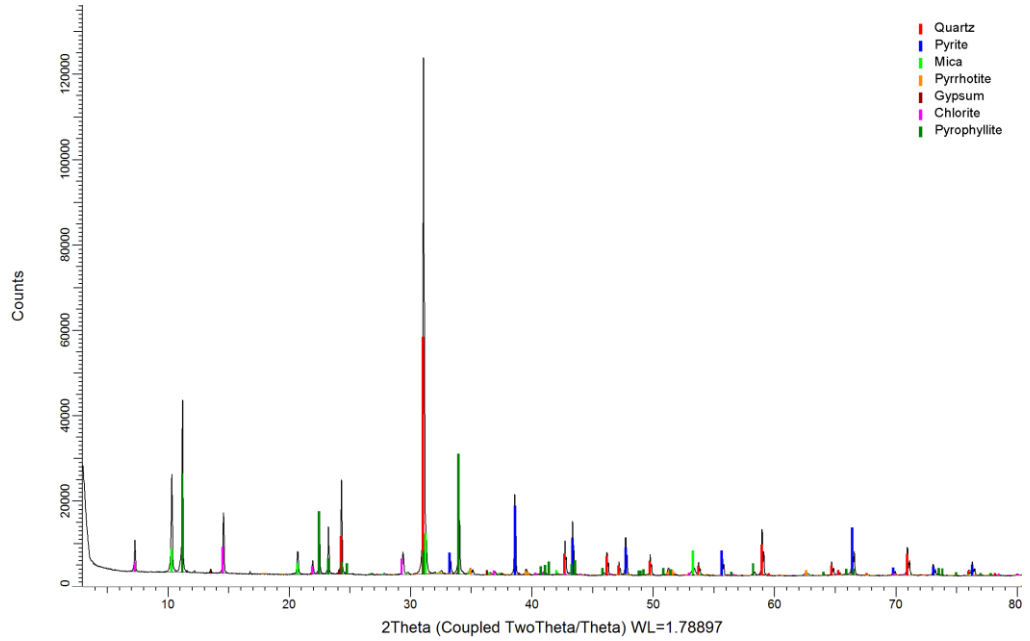


Figure 5-9. Typical X-ray diffractogram indicating the main mineral phases identified in the feed as well as subsequent kinetic baseline leach residue samples

5.4 Conclusion

An increased addition of cyanide (2 kg/t instead of 0.66 kg/t) was employed to offset the presence of base metals and sulphates in the solution caused by the simultaneous leaching of pyrrhotite with gold during cyanidation. This simultaneous leaching leads to high reagent consumption and constrains cyanide access to the gold particle. To counter the overload of base metals and sulphate formed by the attack of cyanide on pyrrhotite, the solids: liquid mass ratio was decreased (33% instead of 43%). This phenomenon is described in detail in Chapter 2, Equations (8) and (9).

CHAPTER 6

6. GOLD EXTRACTION IMPROVEMENT

6.1 Introduction

The chemical, mineralogical, and metallurgical characterisation findings were used to determine the test conditions necessary to improve gold extraction from the sample. The results of the process mineralogy study established that poor gold dissolution was due to the association and locking of gold by the iron-sulphide minerals. The simultaneous leaching of these minerals in cyanide solution resulted in a high concentration of sulphide (S^{2-}), iron (Fe^{2+} and Fe^{3+}) and base metals, which react with cyanide lime and oxygen, slowing down gold leaching kinetics.

Table 6.1 shows the recommended test work conditions to minimise the negative effects of pyrite and pyrrhotite and thus improve gold extraction efficiencies of a Wits pyrite concentrate. Pyrite and pyrrhotite are the main gangue minerals associated with gold in the Wits pyrite concentrate. Therefore, it is recommended to pre-oxidise the pyrrhotite and pyrite with lime to consume and reduce the amount of reactive iron and sulphide reporting to the leaching circuit. Pre-oxidation also generates ferric ions (Fe^{3+}), which are the primary oxidant during the oxidation of pyrite. This process requires increased dissolved oxygen levels and lime addition for maintaining pH. Pre-oxidation was carried out for 1, 3, and 6 hours to determine the effect of retention time on gold extraction. Figure 5-1 in section 5.2 indicated that gold dissolution was below 50% at 4 hours and 65% at 6 hours. The pre-oxidation retention time variables were based on the first six hours of cyanidation without pre-oxidation (Baseline kinetics) when lime and cyanide consumption was high and gold extraction was low. The soluble iron-sulphide minerals consumed the reagents, competing with gold. Therefore, pre-oxidation for 4 hours ensured that there was enough lime to react with the cyanides to minimise the concentration of cyanides reporting to the leaching circuit.

To assess whether pre-oxidation could reduce cyanide consumption while maintaining gold extraction, sodium cyanide additions of 1.5 kg/t and 1 kg/t were tested. The aim was to determine whether NaCN could control gold extraction when lime and oxygen reacted with soluble iron-sulphide minerals. The baseline kinetic studies (Section 5.2) showed that cyanide consumption remained consistent between 0.4 and 0.6 kg/t for 72 hours, indicating that

extended retention durations did not consume more cyanide as all leachable gold was leached within 24 hours. Free cyanide additions of 0.5 and 0.7 kg/t were selected for the evaluation, with cyanide being the limiting reagent. The sample required 0.7 to 1 kg/t of NaCN, which corresponds to the selected free cyanide additions.

9. To examine the benefits of pre-oxidation on gold dissolution kinetics and flowsheet operating expenses, the leach retention duration was reduced to 8 and 18 hours. The baseline gold extraction tests indicated that 71% gold was leached within 18 hours, equivalent to 71% gold leached in 24 hours. Therefore, a reduced retention time of 18 hours was tested. APPENDIX B sections 10.2 and 10.3 provide a detailed mass balance of all the tests.

6.2 Effects of pre-oxidation on gold extraction and reagent consumption

The effectiveness of alkaline pre-oxidation of iron-sulphide minerals in enhancing cyanidation has been established in previous research, as it dissolves the matrix of the locking mineral and exposes occluded gold to cyanide (Larrabure and Rodríguez-Reyes 2021). To investigate the impact of pre-oxidation on gold extraction, tests were conducted involving pre-oxidation followed by cyanidation. Pre-oxidation was performed at one, three, and six hours of residence time, with lime added to maintain pH above 11.5 to oxidise pyrite to thiosulphate, pyrrhotite to sulphate, and arsenopyrite to iron oxide. Table 6.1 describes the conditions for each test performed.

Tests 1 to 4 were conducted to assess the effect of using compressed air and 99% industrial oxygen as the oxidant and retention time (1, 3, and 6 hours) for pre-oxidation. The purity of the oxidant did not have any impact on gold extraction, as extraction rates remained constant regardless of the oxygen source. Reducing the pre-oxidation time from six hours to one hour resulted in less exposure of the lime and oxygen to the pyrrhotite, therefore reducing the consumption of these reagents. Moreover, the majority of the gangue minerals associated with gold were unlocked within three hours of pre-oxidation, and gold extraction was not adversely affected, indicating that pre-oxidation could be an effective means of decreasing lime consumption.

Table 6.1. Pre-oxidation and gold extraction laboratory tests

Baseline	Methodology	Objective
Test 1	Pre-Ox: Air 1 Hrs @ pH 11.5 (Adjusted with lime) followed by CN @ 2 kg/t NaCN for 24 Hrs	Alkaline O_2 pre-ox with consideration of retention time and oxidant
Test 2	Pre-Ox: O_2 1 Hrs @ pH 11.5 (Adjusted with lime) followed by CN @ 2 kg/t NaCN for 24 Hrs	
Test 3	Pre-Ox: Air 6 Hrs @ pH 11.5 (Adjusted with lime) followed by CN @ 2 kg/t NaCN for 24 Hrs	
Test 4	Pre-Ox: Air 3 Hrs @ pH 11.5 (Adjusted with lime) followed by CN @ 2 kg/t NaCN for 24 Hrs	

Figure 6-1 illustrates the gold extraction and reagent consumption for the pre-oxidation tests, and a detailed mass balance can be found in Appendix 10.2. Pre-oxidation proved effective in releasing gold by dissolving the matrix of the locking minerals, resulting in gold extractions ranging from 75% to 77%, depending on the pre-oxidation duration (Zia, Mohammadnejad, and Abdollahy 2020). Figure 6-1 shows a 6% increase in gold extraction compared to the baseline test when examining the results of the gold extraction improvement tests (Tests 1 to 4). Lime consumption in Tests 1 to 4 increased by an average of 49% compared to the baseline test, primarily due to the additional lime required for oxidation. Cyanide consumption also increased by 89% to ensure sufficient cyanide for the formation of the gold-cyanide complex in the presence of cyanicides. The best gold extraction results were achieved under Test 3 and 4 conditions, where compressed air was used as the source of oxygen, demonstrating that compressed air provided enough oxygen for pre-oxidation. Furthermore, the comparable gold extraction results from both tests demonstrated that three hours was sufficient to oxidise the maximum amount of available gold.

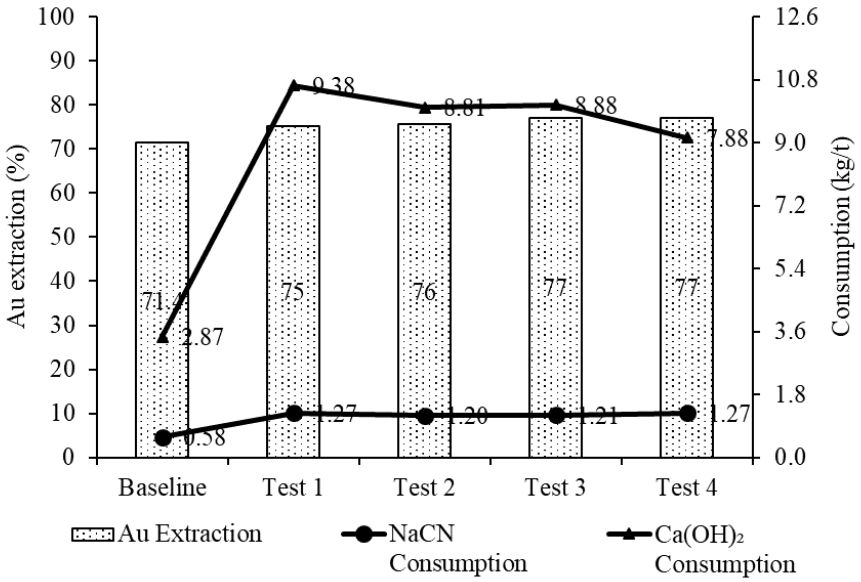


Figure 6-1. Pre-oxidation gold extraction and reagent consumptions

6.3 Effects of $Pb(NO_3)_2$ addition on gold extraction and reagent consumption

The effect of lead nitrate ($Pb(NO_3)_2$) addition on gold dissolution kinetics and cyanide consumption was investigated after determining the pre-oxidation parameters. The results showed that $Pb(NO_3)_2$ was only effective in reducing cyanide consumption and did not improve gold dissolution. Lead nitrate conditioning ensures that all exposed gold is leached, but in this case, no additional benefit in gold extraction was observed as excess NaCN was already added. Table 6.2 outlines the test conditions for each of the experiments conducted.

Table 6.2. $Pb(NO_3)_2$ and gold extraction laboratory tests

Baseline	Methodology	Objective
Test 5	Pre-Ox: Air 3 Hrs @ pH 11.5 followed by CN @ 2 kg/t NaCN + $Pb(NO_3)_2$for 24 Hrs	To disrupt the "natural "passivation layer (hydroxide/AuCN) on gold surfaces, improving both extraction and dissolution kinetics while reducing reagent consumption
Test 6	Pre-Ox: Air 3 Hrs @ pH 11.5 followed by CN @ 2 kg/t NaCN + $Pb(NO_3)_2$ for 24 Hrs	

Figure 6-2 and the detailed mass balance in Appendix 10.2 illustrate the gold extraction and reagent consumption for Tests 5 and 6. The gold extraction remained the same as in Test 4, despite the addition of lead nitrate in Test 5 (300 mg/L) and Test 6 (150 mg/L). The addition of lead nitrate decreased cyanide consumption by 0.13 kg/t. Lead nitrate is added to reduce cyanide consumption by reacting with sulphide gangue minerals (pyrite and pyrrhotite). The oxidised pyrite and pyrrhotite particles react with lead nitrate, forming a hydroxide layer on their surface, which reduces their reaction rate with cyanide, resulting in reduced cyanide consumption (Deschênes et al. 2002; Rabieh et al. 2017; Soltani et al. 2020).

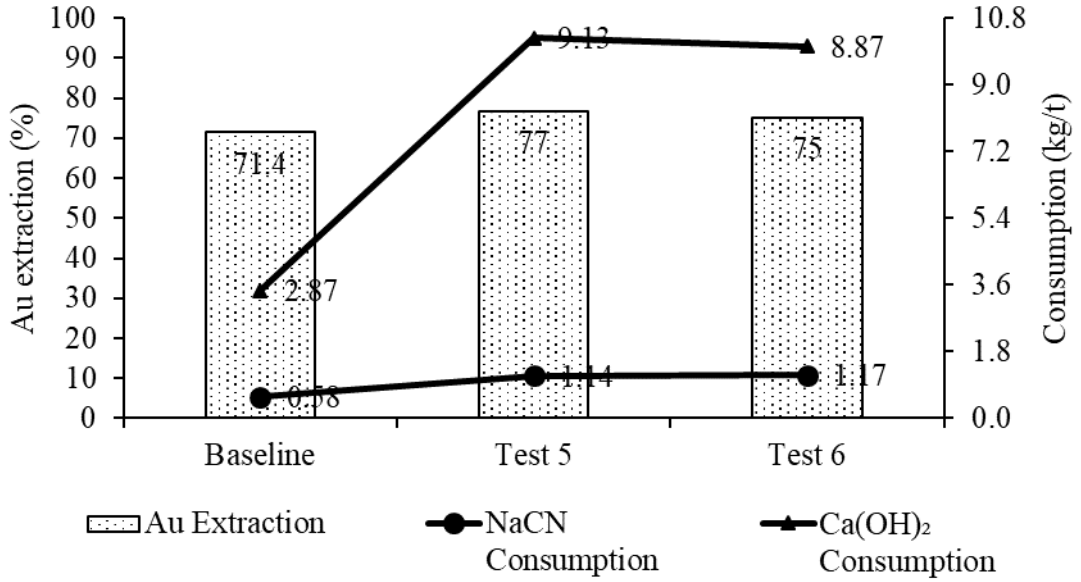


Figure 6-2. $Pb(NO_3)_2$ effect on gold extraction and reagent consumptions

6.4 Effects of cyanide addition reduction on gold extraction and reagent consumption

Alkaline pre-oxidation of iron-sulphide minerals is an effective method for improving cyanidation as it dissolves and alters the matrix of the locking mineral, thereby exposing the occluded gold to cyanide (Larrabure and Rodríguez-Reyes 2021). To observe the impact of

pre-oxidation on gold extraction, the effects of pre-oxidation on cyanide requirements were investigated. Table 6.3 provides details of the conditions for each of the tests performed. The study investigated whether pre-oxidation altered the reactivity of competing elements by testing reduced NaCN additions of 1.5 kg/t and 1 kg/t.

Table 6.3. Cyanide reduction effect on gold extraction laboratory tests

Baseline	Methodology	Objective
Test 7	Pre-Ox: Air 3 Hrs @ pH 11.5 followed by CN @ 1.5 kg/t NaCN for 24 Hrs	To reduce cyanide addition to cut reagent costs
Test 8	Pre-Ox: Air 3 Hrs @ pH 11.5 followed by CN @ 0.5 kg/t NaCN for 24 Hrs	

Pre-oxidation with lime and oxygen aims to pre-leach the sample, targeting the iron sulphide minerals that dissolve or lock gold during cyanidation. Cyanidation of iron sulphide minerals results in marginal solubility and unnecessary cyanide consumption in the CIP circuit. Pre-oxidation is aimed at reacting and breaking down the locking or associated iron sulphide minerals, thereby exposing the encapsulated gold. Pre-oxidation makes the iron sulphide minerals less reactive during cyanidation, reducing competition for free cyanide to extract leachable gold. This is one of the ways that pre-oxidation passivates the minerals (Karimi et al. 2010; Problems 2015; Rabieh et al. 2017; Teague, Van Deventer, and Swaminathan 1999).

Reducing cyanide concentrations (Tests 7 and 8) was detrimental to gold extraction, proving that cyanide is the limiting reagent during the cyanidation of a Wits pyrite concentrate. Reducing the retention time of cyanidation led to a reduction in gold extraction by 3% for an 8-hour leach and 2% for an 18-hour retention time (Tests 9 and 10). Shorter retention times result in lower operating expenses.

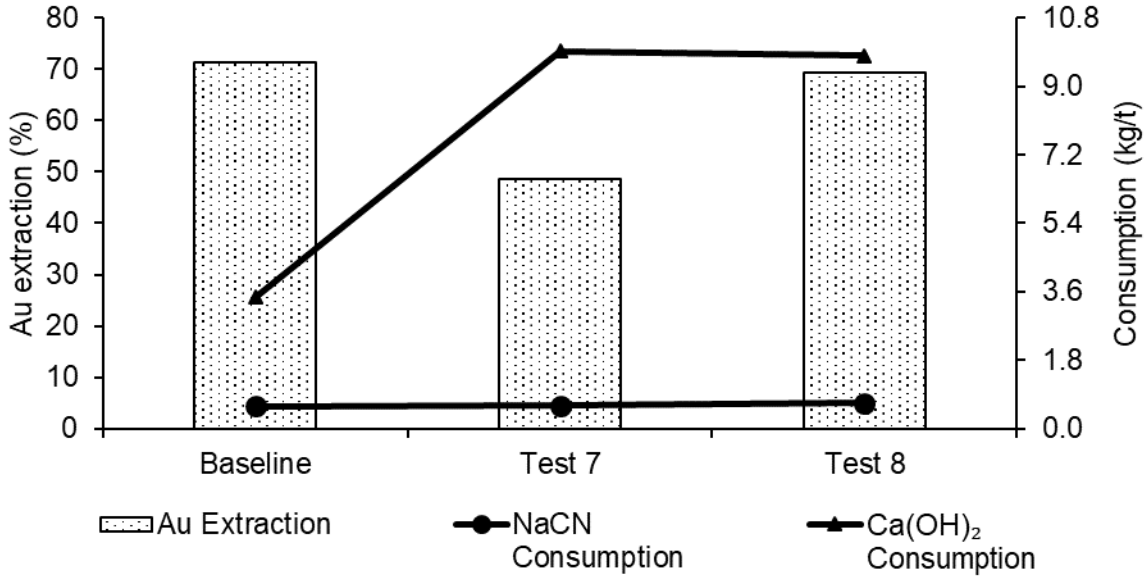


Figure 6-3. Cyanide reduction on gold extraction and reagent consumptions

6.5 Effects of retention time on gold extraction and reagent consumption

Alkaline pre-oxidation of iron-sulphide minerals can enhance the process of cyanidation by dissolving the matrix of the locking mineral and exposing the occluded gold to cyanide (Larrabure and Rodríguez-Reyes 2021). To investigate the improvement in gold extraction, pre-oxidation tests followed by cyanidation were conducted, where pre-oxidation was tested at residence times of one, three, and six hours, with lime addition to maintain a pH above 11.5 to oxidise the pyrite to thiosulphate, pyrrhotite to sulphate, and arsenopyrite to iron oxide. Table 6.1 describes the conditions for each of the tests performed.

Table 6.4. Retention time effect on gold extraction laboratory tests

Baseline	Methodology	Objective
Test 9	Pre-Ox: Air 3 Hrs @ pH 11.5 followed by CN @ 2 kg/t NaCN for 8 Hrs	To reduce retention time to cut energy costs
Test 10	Pre-Ox: Air 3 Hrs @ pH 11.5 followed by CN @ 0.5 kg/t NaCN for 18 Hrs	

The results of the gold extraction improvement tests are presented in Figure 6-4, and the detailed mass balance can be found in Appendix 10.2. It was found that reducing the cyanide concentrations (Tests 7 and 8) had a negative impact on gold extraction, demonstrating that cyanide is the limiting reagent during the cyanidation of a Wits pyrite concentrate.

Additionally, reducing the cyanidation retention time resulted in a 3% reduction in gold extraction for an 8-hour leach and a 2% reduction for an 18-hour retention time (Tests 9 and 10). A decrease in retention time could potentially lead to reduced OPEX costs.

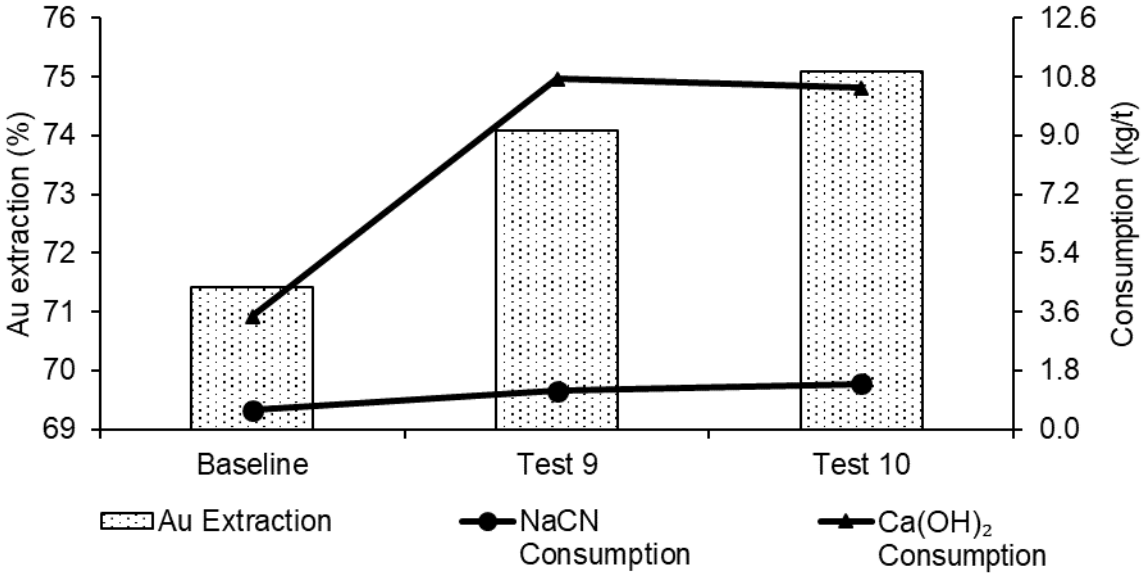


Figure 6-4. Pre-oxidation gold extraction and reagent consumptions

6.6 Base metal dissolution

Table 6.5 presents the results of the analysis of residues and pregnant leach solutions (PLS) of the gold extraction improvement tests for base metals and total sulphur. The analysis of residues indicates that after cyanidation of the pre-oxidised slurry, there was a decrease in the content of copper, iron, zinc and sulphides. This suggests that pre-oxidation facilitated the extraction of gold, copper, zinc, and sulphides by cyanide as it degraded some of the encapsulating minerals. As a result, these base metals were available in low concentrations and leached simultaneously with gold and silver during cyanidation (Oraby, Eksteen, and Tanda 2017; Yliniemi et al. 2018; Wu, Ahn, and Lee 2021).

The analysis of PLS in Figure 6-5 shows that these base metals and other metals that were below the detection limit in the residues are present in significant amounts in solution, particularly in Tests 3 and 4. In Tests 3 and 4, some of the iron-sulphide minerals were oxidised, resulting in higher gold extractions, since a portion of the gold locked in pyrite and pyrrhotite was liberated.

Table 6.5. ICP analysis of residues and PLS

Solids	**Al**	**Ca**	**Co**	**Cr**	**Cu**	**Fe**	**Mg**	**Mn**	**Ni**	**Pb**	**S**	**Si**	**Ti**	**V**	**Zn**
	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%
Baseline	4.32	0.30	0.05	0.05	0.072	14.4	0.52	0.05	0.085	0.05	12.4	27.2	0.2	0.05	0.05
Test 1	4.06	0.32	0.05	0.05	0.057	14.9	0.51	0.05	0.096	0.05	12.4	24.6	0.22	0.05	0.05
Test 2	4.11	0.29	0.05	0.05	0.058	14.8	0.52	0.05	0.095	0.05	12.1	24.6	0.21	0.05	0.05
Test 3	4.26	0.34	0.05	0.05	0.069	13.8	0.44	0.05	0.092	0.05	11.9	24.3	0.21	0.05	0.05
Test 4	3.84	0.21	0.05	0.05	0.058	13.5	0.56	0.05	0.086	0.05	11.5	21.3	0.21	0.05	0.05
Test 5	4.36	0.24	0.05	0.05	0.075	14.0	0.51	0.05	0.095	0.05	12.0	27.2	0.20	0.05	0.05
Test 6	4.33	0.20	0.05	0.05	0.082	14.1	0.51	0.05	0.094	0.05	11.5	27.1	0.20	0.05	0.05
Test 7	3.96	0.28	0.05	0.05	0.070	14.0	0.49	0.05	0.099	0.05	12.9	26.9	0.20	0.05	0.05
Test 8	3.84	0.29	0.05	0.05	0.072	14.3	0.48	0.05	0.093	0.05	11.4	26.5	0.20	0.05	0.05
Test 9	4.69	0.32	0.076	0.05	0.069	13.5	0.48	0.05	0.100	0.05	11.7	26.5	0.21	0.05	0.05
Test 10	4.62	0.26	0.07	0.05	0.064	13.9	0.48	0.05	0.095	0.05	11.2	26.8	0.21	0.05	0.05
Solutions	**Al**	**Ca**	**Co**	**Cr**	**Cu**	**Fe**	**Mg**	**Mn**	**Ni**	**Pb**	**S**	**Si**	**Ti**	**V**	**Zn**
	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
Baseline	2.30	835	8.41	2.00	35.2	11.6	2.30	2.00	32.2	2.00	1330	2.00	2.00	2.00	2.00
Test 1	2.00	1020	18.5	2.00	72.6	3.81	2.00	8.5	80.3	2.00	1350	4.48	2.00	2.00	15.9
Test 2	2.75	1010	16.5	2.00	72.7	2.28	2.00	2.51	74.2	2.00	1370	5.59	2.00	2.00	15.6
Test 3	2.00	753	24.1	2.00	92.8	408	3.49	3.22	92.1	2.00	2060	2.00	2.00	2.00	38.7
Test 4	2.00	738	23.1	2.00	99.8	407	3.67	3.16	90.3	2.00	2030	2.00	2.00	2.00	37.3
Test 5	2.60	837	16.7	2.00	84.0	2.00	2.95	2.00	91.6	2.00	1220	2.00	2.00	2.00	4.12
Test 6	2.60	843	18.5	2.00	76.5	2.00	4.36	2.00	98.8	2.53	1140	2.00	2.00	2.00	2.00
Test 7	2.00	872	12.8	2.00	19.9	2.00	2.00	2.00	57.7	2.00	889	2.00	2.00	2.00	2.00
Test 8	2.00	861	16.0	2.00	68.3	2.00	2.26	2.00	77.9	2.00	990	2.00	2.00	2.00	2.00
Test 9	2.00	902	12.1	2.00	68.6	3.23	2.00	2.00	48.4	2.00	1040	5.37	2.00	2.00	24.6
Test 10	2.00	864	24.1	2.00	81.1	2.17	2.00	2.00	108	2.00	1080	5.12	2.00	2.00	4.15

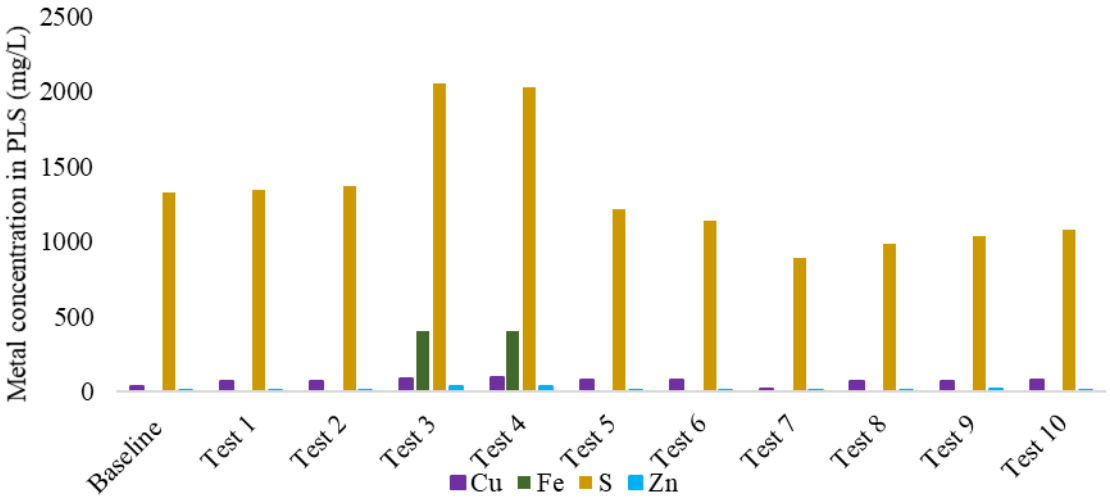


Figure 6-5. Cu, Fe, S, and Zn concentration of gold extraction improvement tests PLS

X-ray diffraction (XRD) analysis was conducted on the pre-oxidation residues of Tests 4, 9 and 10 to identify the changes in gangue minerals after pre-oxidation, and the results are presented in Table 6.6. The pre-oxidation residues were primarily composed of chlorite, mica, and quartz, with pyrite, gypsum, pyrophyllite and pyrrhotite occurring as trace minerals. The analysis revealed a decrease in pyrite (from 15% in Table 4.4 to less than 5%) and pyrrhotite (from 4.3% in Table 4.4 to below the detection limit) in the residues, which confirms the pre-oxidation of these minerals. The formation of anhydrite and basanite in the pre-oxidised residues is likely a by-product of the oxidation of pyrite and pyrrhotite with oxygen and $Ca(OH)_2$. Basanite is the intermediate phase in the transformation of gypsum to anhydrite, while anhydrite is the dehydrated form of gypsum. The precipitation of secondary sulphate minerals during pyrite oxidation further confirms that pyrite and pyrrhotite were indeed oxidised before cyanidation (Fan et al. 2022).

Table 6.6. Qualitative XRD analysis of pre-oxidation products (Test 4, 9, and 10)

Mineral	Formula	Test 4	Test 9	Tests 10
Quartz	SiO_2	Predominant	Predominant	Predominant
Mica	$[KAl_2(AlSi_3O_{10})(F, OH)_2]$	Minor	Minor	Minor
Pyrite	FeS_2	Trace	Trace	Trace
Basanite	$2CaSO_4 \bullet (H_2O)$	Trace	Trace	Trace
Anhydrite	$CaSO_4$	Trace	Trace	Trace
Pyrophyllite	$Al_2Si_4O_{10}(OH)_2$	Minor	Minor	Minor
Clinochlore	$(Mg; Fe^{2+})_5Al(Si_3Al)O_{10}(OH)_8$	Minor	Minor	Minor
*>50% predominant, 20-49 % major, 5-19% minor, <5% trace				

The tailings from Wits' historical mining activities are typically stored in the open air, where bacteria are present. Due to the outdated process used to produce historic Wits sand dumps, the dumps became porous, allowing bacteria to penetrate and catalyse the oxidation of the pyrite minerals present in the dumps. Using the QEMSCAN data, an image grid of the sulphide particles before and after oxidation was generated using the iDiscover software and the Injector pre-processor. Porosity tests were conducted on the Wits pyrite concentrate feed and the pre-oxidised Wits pyrite concentrate of Test 4 to compare the porosity of the sulphides before and after pre-oxidation, and the results are shown in Figure 6-6. The feed had a porosity of 0.7%, while the pre-oxidised sample had a notably lower porosity of approximately 0.2%. Porosity provides pathways for the solution to flow through the mineral and hence promotes pre-oxidation of the pyrite and pyrrhotite. The reduced porosity of the pre-oxidised sample was due to the breakage and reactivity of the sulphides along the pores during pre-oxidation, which broke down the large sulphide particles. The reduction in grain size from the feed to the residue, particularly in the 40-60 µm size fractions, further confirms this. The reduction in porosity and sulphide grain size in the pre-oxidised sample confirms the occurrence of pre-oxidation of the sulphides.

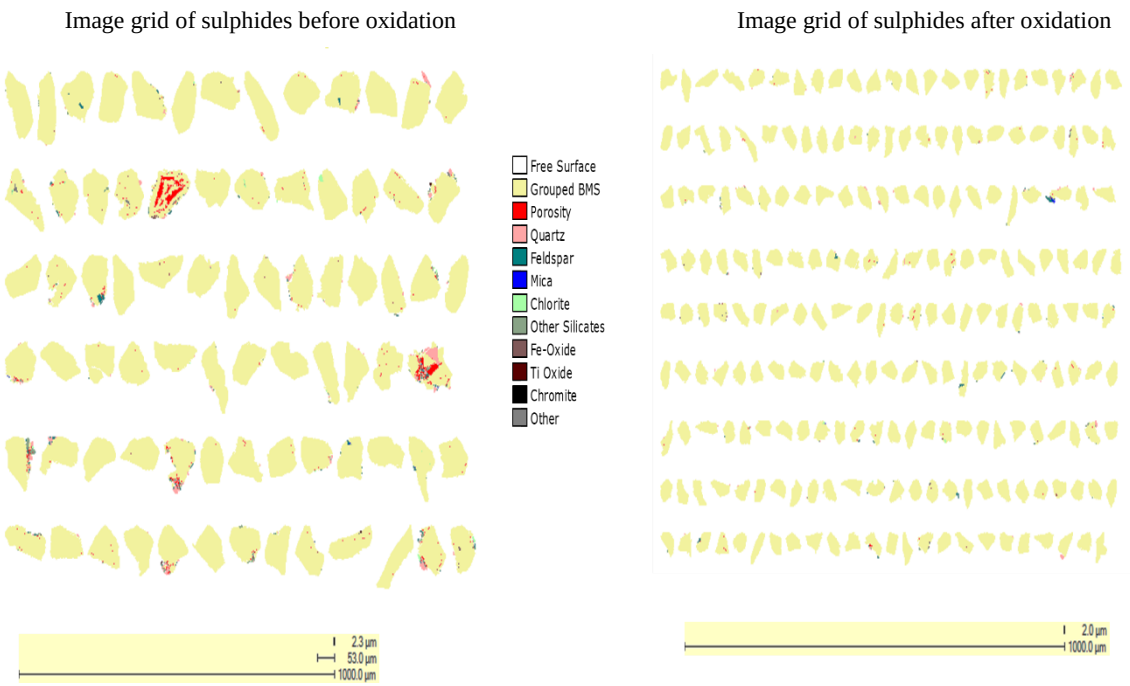


Figure 6-6. Comparison of porosity before and after oxidation

6.7 Comparison of ERGO plant and laboratory leaching efficiencies

In 2007, DRDGold acquired the ERGO plant, which had been incorporating flotation in the feed preparation before leaching and recovery since the 1970s. However, upon taking over, DRDGold initially discontinued the use of the flotation plant. Later in 2012, they reintroduced it, and by 2014, it was operational again. The company's approach was to float the gold associated with sulphide minerals and then regrind the flotation concentrate, which contained a considerable amount of gold in a smaller stream than the total plant feed. The objective was to expose more of the gold in the pyrite concentrate through regrinding, leading to greater gold recovery. Although regrinding improved the liberation of gold, the desired improvement in gold extraction was not fully realised due to the poor efficiency of the leaching circuit.

A desktop economic model was developed for a hypothetical ultra-fine grind (UFG) process and the established pre-oxidation process, following this methodology:

1 Production standards were determined, including feed rates, characteristics (grades), leach data (residence times, gold extraction, reagent consumptions), and operating years.

2 Mass balance information was computed, including mass flows, component flows, and volumetric flow rates.
3 Design criteria for each unit operation, such as residence time and solids concentration, were specified, and equipment sizes were determined.
4 Capital costs for important equipment components were computed using literature, and a breakdown of capital costs for the unit operation was determined, including tanks, agitators, pumps, etc.
5 Reagent costs were calculated based on experimental consumptions (e.g., kg/t cyanide usage) using public data or supplier information.
6 Indirect capital and operating costs were computed by multiplying direct costs by relevant benchmark factors based on prior experience or available data, expressed as a percentage of the direct cost.

Table 6.7 compares the production criteria at ERGO when UFG was applied with the production criteria without UFG, as established by the study. The data shows that the ERGO flowsheet with UFG extracts 5.5 kg less gold compared to the pre-oxidation flowsheet. To evaluate the effect of various parameters on gold recovery for the Wits pyrite concentrate, feed grade, solids percentage, oxygen pre-oxidation, cyanidation retention time, cyanide and lime addition was tested. The solids content of the CIP circuit at ERGO is 43%, while the study used a solids content of 33%. The pulp solids content can affect mixing during cyanidation, thereby influencing the mass transfer of the metals of interest. A high pulp solids content increases throughput but hinders mass transfer, whereas reducing it can improve mixing, thereby increasing gold extraction.

The laboratory cyanidation conditions established that the cyanide addition at ERGO (0.6 g/t) is 50% less than the cyanide consumption (1.14 g/t) observed in the study. The ore contains several components (aluminium, iron, silica, copper, nickel, and sulphur species) that consume cyanide through side reactions. Therefore, enough cyanide should be added to account for all possible side reactions. Moreover, cyanide is also lost by hydrolysis in the plant. Therefore, for effective leaching, an excess amount of cyanide (2 kg/t) must be added at the start of the leach. In the gold extraction improvement test, this addition resulted in 120 mg/L of residual cyanide

in the leach, which is the recommended minimum cyanide value in the last tank of a cyanidation circuit (Stange 1999).

Dissolved Oxygen (DO) is also a vital reagent for cyanidation, as the sample contains inorganic and organic components that consume oxygen. Reducing DO reduces the leaching kinetics and efficiency of cyanidation. ERGO's DO is 0.011 mg/L, while the study found that 7.74 mg/L of DO is required for efficient gold extraction. The slurry pH at ERGO is 10.5, while the study slurry pH was 11.0. Increasing the pH value is beneficial for gold extraction in sulphide ores, whereas pH value has no noticeable influence on some carbonaceous ores. Wits' tailings are predominantly sulphide ores and, therefore, require higher pH values (high CaO consumption) for maximum gold extraction.

During the study period, ERGO used CIL instead of CIP. In the CIL process, an adsorption plant to contact the slurry with carbon follows cyanidation. However, this process is not always appropriate for low-grade ores because large leaching tanks are required to provide sufficient residence time for cyanidation. Moreover, the low carbon concentrations in the reactors make the counter-current transfer of carbon more challenging in CIL circuits. Additionally, the concurrent leaching and adsorption technique employed in CIL further reduces the driving force for adsorption (Abdel and Aloub 2018; Cui et al. 2015). For cyanidation of Wits pyrite concentrate, CIP is preferable to CIL because all leachable gold would be in solution before the first carbon adsorption tank, creating a considerable kinetic driving force for adsorption. The flotation concentrate consisted only of iron-sulphide gangue minerals (pyrite, pyrrhotite and traces of arsenopyrite) since all the pre-robbers were rejected in the tailings. Furthermore, the diagnostic leach indicated that the deposit has a pre-robbing potential of only 0.77%, making CIP applicable in this case. With a CIL plant, the slurry first encounters activated carbon before sufficient gold is leached, thereby slowing down the rate of gold adsorption onto activated carbon (Fleming et al. 2011).

Table 6.7. ERGO production criteria vs study production criteria

	Units	CIP ERGO plant (With UFG) (Staden et al. 2020; TWP The Basil Read Group 2012)	CIP Laboratory (Without UFG)
Plant slurry throughput	tonnes/day	5581	5581
Leach plant solids throughput	tonnes/day	2400	1842
Leach operating schedule	hrs/day	21.7	21.7
Leach plant solids throughput	t/hr	110.6	84.9
Head Grade	g/tonne	3.01	3.01
Leach slurry solids	%	43	33
Leach solution flowrate	t/hr	147	170
Leach solution density	t/m^3	1.00	1.27
Leach solution flowrate	m^3/hr	147	134
Leach Efficiency	%	55	75
Leach recovery	g/hr	183	192
Leach solution grade	g/m^3	2.29	1.29
NaCN consumption	kg/t	0.58	1.14
CaO consumption	kg/t	2.87	7.88
Gold production	kg/month	119	125

The unit operation's day-to-day expenses are referred to as operating expenses, while its substantial, long-term spending is known as capital expenditures. The theoretical CAPEX per tonne of the UFG unit (Table 6.8) and pre-oxidation unit (Table 6.8) indicate that the pre-oxidation CAPEX is $299,338 lower than the UFG CAPEX is. It should be noted that currency fluctuations and steel prices are now highly unpredictable. On average, the UFG milling plant's cost will be at least three times that of the pre-oxidation process per tonne. However, if only 20% of the feed is processed in the UFG plant, the CAPEX may be reduced (Janse van Rensburg 2023).

Table 6.8. Theoretical UFG CAPEX

CAPEX	$	6,000,000
Throughput (tph)		20
CAPEX per tonne	$	300,000

Table 6.9. Theoretical pre-oxidation CAPEX

	100,000 tonnes (Clary et al. 2018)		1 tonne	
Direct costs	$	30,200,000	$	302
Indirect costs	$	9,700,000	$	97
EPCM	$	8,700,000	$	87
Contingency	$	15,300,000	$	153
Capital spares and first fill	$	2,300,000	$	23
Total	**$**	**66,200,000**	**$**	**662**

According to Table 6.10 and Table 6.11, the theoretical OPEX for the pre-oxidation unit is $54,795.62 less than that of the UFG unit. Despite the pre-oxidation unit's lower throughput, it still achieves higher gold production than the anticipated production of the UFG unit.

Table 6.10. Theoretical UFG OPEX (Ellis 2003)

Throughput	t/day	2400	
EPC	$/t	$	1.51
Maintenance labour	$/t	$	1.40
Pumps	$/t	$	1.13
Cyclone parts	$/t	$	1.47
Discs/liners	$/t	$	3.60
Media	$/t	$	5.37
Power	$/t	$	9.88
Concentrate handling	$/t	$	1.28
UFG Mills	$/t	$	24.36
Services	$/t	$	1.93
Total OPEX per day		$	124,629

Table 6.11. Theoretical pre-oxidation OPEX

Throughput	t/day	1842
Reagents (Table 6.7)	$/t	$ 3
Power (Clary et al. 2018)	$/t	$ 23
Maintenance labour (Clary et al. 2018)	$/t	$ 12
Total OPEX per day		**$ 69,832.91**

The proposed pre-oxidation unit operation is expected to generate an additional 19 kg of revenue per month. However, daily operating expenses for the unit, including electricity, oxygen, and lime, amount to around $69,832. Despite this disparity, the pre-oxidation process provides a significant margin for profit, even when considering conservative estimates.

To assess the economic performance and acceptability of the two unit operations, their finance-related transactions were analysed. The financial study reveals that pre-oxidation is a more viable investment than UFG. According to Table 6.12, the return on investment for the pre-oxidation-CIP circuit is 100% higher than that of the UFG-CIP circuit for Wits pyrite concentrate. At a gold head grade of 3.01 g/t and a processing rate of 5,581 tonnes per day, the pre-oxidation-CIP circuit can achieve an overall gold recovery of 77%, compared to 55% for the UFG-CIP circuit. With a gold price of $58,976.85 per kilogram, the pre-oxidation case is economically viable with a return on investment of 100%, while the UFG case is marginally viable with a return on investment of 0.16%. The pre-oxidation-CIP circuit requires a capital expenditure of 99% of the expected monthly gold revenue, while the UFG-CIP circuit's capital expenditure is much higher than the monthly gold revenue. Therefore, it would take more than a month to recover the capital expenditure for the UFG-CIP circuit. The payback period for the pre-oxidation-CIP circuit is less than two months.

Table 6.12. Financial analysis, comparison of pre-oxidation, and UFG

	Pre-oxidation + CIP Laboratory	**UFG + CIP ERGO**
Gold price (USD/kg)	$ 58,976.85	$ 58,976.85
Gold revenue per month	$ 7,356,568	$ 7,029,609
Solids throughput per month (kg)	55254	71998
OPEX per month	$ 2,094,987	$ 3,738,856
% OPEX	28%	53%
CAPEX	$ 7,315,666	$ 4,319,879,760
% CAPEX	99%	61453%
Payback period (Months)	1.39	1313
Profit	$ 5,261,580.18	$ 3,290,753
Return on investment (ROI)	101%	0.16%

6.8 Conclusion

The pre-oxidation process of pyrite and pyrrhotite (iron-sulphide minerals) resulted in the exposure of gold particles and had a positive impact on cyanidation efficiency. The gold particles were previously trapped by the iron-sulphide minerals, which resulted in limited access to cyanide, significant reagent consumption, and depletion of cyanide and oxygen in the pulp due to reactions with the contents of the pulp, including sulphides, iron, and base metals (Abdel and Aloub 2018). The two-stage addition of cyanide and the addition of excess cyanide extended the gold and iron-sulphide dissolution, revealing that cyanide is the limiting reagent during the cyanidation of the Wits pyrite concentrate.

The pre-oxidation process improved gold extraction without reducing cyanide consumption since the primary objective was to enhance gold extraction, which in turn also promoted the extraction of copper, iron, and zinc, which follow gold during cyanidation (Aghamirian and Yen 2005). Furthermore, the role of mineralogical analysis in developing the process route that would result in improved gold extraction needs to be evaluated.

Comparing the CAPEX and OPEX requirements, pre-oxidation is more economical than UFG for improving gold extraction from a Wits pyrite concentrate.

CHAPTER 7

7. CONCLUSION AND RECOMMENDATION

7.1 Conclusion

This study aimed to investigate the cause of reduced gold extraction during the cyanidation of a Wits pyrite concentrate from ERGO and develop an effective hydrometallurgical process for gold extraction. Process mineralogy was utilised to determine the reason for poor gold extraction, and a test work program was developed based on the findings of the process mineralogy study to improve gold extraction in the leaching circuit.

The study revealed that the Wits pyrite concentrate was dominated by native gold, which is cyanide-leachable. However, gold particles locked in reactive gangue minerals induced poor gold extraction, unlike electrum. During cyanidation, the locking and reactive gangue minerals, pyrite and pyrrhotite, could limit cyanide access to the gold particle and partially dissolve, leading to slow gold dissolution.

Pre-oxidation of pyrrhotite and pyrite with lime and oxygen decomposed the iron-sulphide minerals, unlocking the gold particles, and reducing the adverse effect of slow leaching kinetics during cyanidation. This resulted in achieving a gold extraction of 75% in 18 hours, compared to 78% in 72 hours without pre-oxidation. The addition of lead nitrate to the leach slurry was also effective in reducing cyanide consumption and achieving 77% gold extraction when adding 300 g of lead nitrate per tonne of ore.

Process mineralogy is a valuable tool that provides information for cost-effective and efficient processing, aiding decisions regarding resource evaluation and extraction process optimisation. The mineral composition of a gold resource determines the performance of all chemical and physical processes involved in gold extraction. It is, therefore, crucial to accurately characterise the gold deportment and gangue minerals in the resource to establish suitable treatment options. The study concluded that the CAPEX and OPEX requirements of pre-oxidation are more economical than UFG to improve the extraction of gold from a Wits pyrite concentrate.

Furthermore, the study highlights that the application of process mineralogy to gain knowledge of ores is vital in developing the most suitable flowsheet as per the metallurgical requirements of the resource. Failure to do so would limit the opportunity to develop the most appropriate flowsheet for efficient and cost-effective gold extraction.

7.2 Recommendation

The study employed pre-oxidation with lime and air as a pre-treatment step to partially dissolve the iron-sulphide minerals and reduce their negative impact on cyanidation. Other technologies such as ultra-fine grinding (UFG), pressure oxidation (POX), and biological oxidation (BIOX) are also available, but the CAPEX required for their implementation can be higher than that of pre-oxidation with lime and air. Further work is recommended to develop software that can choose the appropriate tailings pre-treatment option based on the process mineralogy of the tailings deposit. The model would receive and analyse process mineralogy data to provide suggestions for efficient tailings reprocessing flowsheets for maximum gold extraction. The outputs of this model will guide operations on the various pre-treatment and cyanidation conditions for tailings, thereby deciding on the mode of tailings pre-treatment and test work program for maximum gold extraction.

10. REFERENCES

Abdel, Co-supervisor Osama, and Karim Aloub. 2018. "Metallurgical Evaluation Of The Recovery Of Residual Gold From The Tailings Of Abidyia Artisanal Miners."

Adiansyah, Joni Safaat, Michele Rosano, Sue Vink, and Greg Keir. 2015. "A Framework for a Sustainable Approach to Mine Tailings Management: Disposal Strategies." Journal of Cleaner Production 108: 1050–62.

Afenya, P. M. 1991. "Treatment of Carbonaceous Refractory Gold Ores." Minerals Engineering 4 (7–11): 1043–55.

Aghamirian, M. M., and W. T. Yen. 2005. "A Study of Gold Anodic Behavior in the Presence of Various Ions and Sulfide Minerals in Cyanide Solution." Minerals Engineering 18 (1): 89–102.

Allard, Olivier, Mathieu Lopez, Isabelle Demers, and Lucie Coudert. 2022. "Gold Recovery from Sulfide Concentrates Produced by Environmental Desulfurization of Mine Tailings." Minerals 12 (8): 1–15.

Anderson, Corby G., Robert C. Dunne, and John L. Uhrie. 2014. "Mineral Processing and Extractive Metallurgy: 100 Years of Innovation." Mineral Processing and Extractive Metallurgy: 100 Years of Innovation.

Andrianandraina, S. H., J. Dionne, H. Darvishi-Alamdari, and J. F. Blais. 2022. "Effect of Grain Size on the Bacterial Oxidation of a Refractory Gold Sulfide Concentrate and Its Dissolution by Cyanidation." Minerals Engineering 176 (January): 107360. https://doi.org/10.1016/J.MINENG.2021.107360.

Anthony Kucernak. 2007. "Handbook of Electrochemistry." Handbook of Electrochemistry, 221–26.

Auchterlonie, Angus, Christoph Pawlik, Herman Cornelissen, Marian Lydall, and Sazini Makamu. 2017. "External Report 7641 an Assessment of Potential for the Recovery of Uranium From Tailings Resources." http://www.mintek.co.za.

Azizi, Abdelaaziz, Catalin Florin Petre, and Faïçal Larachi. 2012. "Leveraging Strategies to Increase Gold Cyanidation in the Presence of Sulfide Minerals - Packed-Bed Electrochemical Reactor Approach." Hydrometallurgy 111–112 (1): 73–81. https://doi.org/10.1016/j.hydromet.2011.10.005.

Azizi, Abdelaaziz, Catalin Florin Petre, Caroline Olsen, and Faïçal Larachi. 2010. "Electrochemical Behavior of Gold Cyanidation in the Presence of a Sulfide-Rich Industrial Ore versus Its Major Constitutive Sulfide Minerals." Hydrometallurgy 101 (3–4): 108–19. https://doi.org/10.1016/j.hydromet.2009.12.004.

Bartlett, Paul D, and Robert Earl Davis. 1958. "Reactions of Elemental Sulfur. II. The Reaction of Alkali Cyanides with Sulfur, and Some Single-Sulfur Transfer Reactions1." Journal of the American Chemical Society 80 (10): 2513–16. https://doi.org/10.1021/ja01543a037.

Bas, Ahmet Deniz, Edward Ghali, and Yeonuk Choi. 2017. "A Review on Electrochemical Dissolution and Passivation of Gold during Cyanidation in Presence of Sulphides and

Oxides." Hydrometallurgy 172 (October 2016): 30–44. https://doi.org/10.1016/j.hydromet.2017.06.021.

Belzile, Nelson, Yu Wei Chen, Mei Fang Cai, and Yuerong Li. 2004. "A Review on Pyrrhotite Oxidation." Journal of Geochemical Exploration 84 (2): 65–76. https://doi.org/10.1016/J.GEXPLO.2004.03.003.

Benzaazoua, M, H Bouzahzah, Y Taha, L Kormos, D Kabombo, F Lessard, B Bussière, I Demers, and M Kongolo. 2017. "Integrated Environmental Management of Pyrrhotite Tailings at Raglan Mine: Part 1 Challenges of Desulphurization Process and Reactivity Prediction." Journal of Cleaner Production 162: 86–95. https://doi.org/https://doi.org/10.1016/j.jclepro.2017.05.161.

Breuer, P. L., M. I. Jeffrey, and D. M. Hewitt. 2008. "Mechanisms of Sulfide Ion Oxidation during Cyanidation. Part I: The Effect of Lead(II) Ions." Minerals Engineering 21 (8): 579–86. https://doi.org/10.1016/j.mineng.2007.11.010.

Cathro, K. J., and D. F. A. Koch. 1964. "The Anodic Dissolution of Gold in Cyanide Solutions." Journal of The Electrochemical Society 111 (12): 1416. https://doi.org/10.1149/1.2426017.

Çelik, Ali, Halil Yaman, Servet Turan, Alpagut Kara, Ferhat Kara, Baojun Zhu, Xuanhui Qu, et al. 2018. "Catalysis of Organic Reactions." Journal of Materials Processing Technology 1 (1): 1–8.

Chivers, Peter T., J. P. Vaughan, J. D. Cashion, and L. J. Brown. 1998. "The Process Mineralogy of Gold: The Classification of Ore Types." Jom 2014-Janua (7): 381–428. https://doi.org/10.1023/A:1012674205425.

Clary, Rodney, Paul DiNuzzo, Thomas Hunter, and Saleem Varghese. 2018. "Making the Right Selection: A Comparative Analysis for the Treatment of Refractory Gold Concentrates," 1327–38. https://doi.org/10.1007/978-3-319-95022-8_108.

Coetzee, Louis L., Salomon J. Theron, Gavin J. Martin, Juan David Van Der Merwe, and Tracey A. Stanek. 2011. "Modern Gold Deportments and Its Application to Industry." Minerals Engineering 24 (6): 565–75. https://doi.org/10.1016/j.mineng.2010.09.001.

Corder, G D., A. Golev, and D. Giurco. 2015. "'Wealth from Metal Waste': Translating Global Knowledge on Industrial Ecology to Metals Recycling in Australia." Minerals Engineering 76: 2–9. https://doi.org/https://doi.org/10.1016/j.mineng.2014.11.004.

Cui, Peiqiang Peide, Shaopeng Wu, Yue Xiao, Miao Wan, Peiqiang Peide Cui, Chirjiv Kaur Anand, Véronique Bisaillon, et al. 2015. "The Effect of Operating Conditions on Density Stratification in a Batch Jig II: The Influence on Stratification Kinetics." Journal of Cleaner Production 108 (May): 107057. https://doi.org/10.1016/j.jclepro.2015.07.084.

Dai, X., and P. L. Breuer. 2013. "Leaching and Electrochemistry of Gold, Silver and Gold–Silver Alloys in Cyanide Solutions: Effect of Oxidant and Lead(II) Ions." Hydrometallurgy 133 (February): 139–48. https://doi.org/10.1016/j.hydromet.2013.01.002.

Dai, X., and M. I. Jeffrey. 2006. "The Effect of Sulfide Minerals on the Leaching of Gold in Aerated Cyanide Solutions." Hydrometallurgy 82 (3–4): 118–25.

https://doi.org/10.1016/j.hydromet.2006.03.005.

Dai, Xianwen, Andrew Simons, and Paul Breuer. 2012. "A Review of Copper Cyanide Recovery Technologies for the Cyanidation of Copper Containing Gold Ores." Minerals Engineering. https://doi.org/10.1016/j.mineng.2011.10.002.

Deschênes, G., J. Rajala, A. R. Pratt, H. Guo, M. Fulton, and S. Mortazavi. 2011. "Advances in the Cyanidation of Silver." Minerals and Metallurgical Processing 28 (1): 37–43. https://doi.org/10.1007/bf03402323.

Deschênes, G., C. Xia, M. Fulton, Louis J. Cabri, and J. Price. 2009. "Evaluation of Leaching Parameters for a Refractory Gold Ore Containing Aurostibite and Antimony Minerals: Part I - Central Zone." Minerals Engineering 22 (9–10): 799–808. https://doi.org/10.1016/j.mineng.2009.02.003.

Deschênes, G, D Hall, and M Fulton. 2002. "Cyanidation of a Pyrrhotite-Bearing Gold Ore." 34th Annual Meeting of the Canadian Mineral Processors 3 (January 2003): 353–61.

Deschênes, Guy, Marlaine Rousseau, Jean Tardif, and P. J.H. Prud'homme. 1998. "Effect of the Composition of Some Sulphide Minerals on Cyanidation and Use of Lead Nitrate and Oxygen to Alleviate Their Impact." Hydrometallurgy 50 (3): 205–21. https://doi.org/10.1016/S0304-386X(98)00055-3.

Ellis, S. 2003. "Ultra Fine Grinding - A Practical Alternative to Oxidative Treatment of Refractory Gold Ores." Proceedings Eighth Mill Operators' Conference, 11–17. https://pdfs.semanticscholar.org/7367/fbb93ff8890719d5b1aa743b1495b6783d3e.pdf.

Ellis, Sandra, and Gamini Senanayake. 2004. "The Effects of Dissolved Oxygen and Cyanide Dosage on Gold Extraction from a Pyrrhotite-Rich Ore." Hydrometallurgy 72 (1): 39–50. https://doi.org/https://doi.org/10.1016/S0304-386X(03)00131-2.

Fan, Rong, Gujie Qian, Yubiao Li, Michael D. Short, Russell C. Schumann, Miao Chen, Roger St C. Smart, and Andrea R. Gerson. 2022. "Evolution of Pyrite Oxidation from a 10-Year Kinetic Leach Study: Implications for Secondary Mineralisation in Acid Mine Drainage Control." Chemical Geology 588 (November 2021): 120653. https://doi.org/10.1016/j.chemgeo.2021.120653.

Festin, Emma Sandell, Mulualem Tigabu, Mutale N. Chileshe, Stephen Syampungani, and Per Christer Odén. 2019. "Progresses in Restoration of Post-Mining Landscape in Africa." Journal of Forestry Research 30 (2): 381–96. https://doi.org/10.1007/s11676-018-0621-x.

Fleming, C. A., A. Mezei, E. Bourricaudy, M. Canizares, and M. Ashbury. 2011. "Factors Influencing the Rate of Gold Cyanide Leaching and Adsorption on Activated Carbon, and Their Impact on the Design of CIL and CIP Circuits." Minerals Engineering 24 (6): 484–94. https://doi.org/10.1016/j.mineng.2011.03.021.

Goldfields. 2019. "Delivering Delivering Value for A." https://www.goldfields.com/reports/annual-report-2019/tailings-and-waste.php.

Guo, Bao, Yongjun Peng, and Rodolfo Espinosa-Gomez. 2014. "Cyanide Chemistry and Its Effect on Mineral Flotation." Minerals Engineering. Elsevier Ltd. https://doi.org/10.1016/j.mineng.2014.06.010.

Harris, D. C. 1990. "The Mineralogy of Gold and Its Relevance to Gold Recoveries." Mineralium Deposita 25 (1): 3–7. https://doi.org/10.1007/BF00205243.

Hedley, Norman, and H Tabachnick. 1968. "Mineral Dressing Notes No. 23." Chemistry of Cyanidation, American Cyanamid Company, New Jersey, USA.

Henley, K. J., N. C. Clarke, and P. Sauter. 2001. "Evaluation of a Diagnostic Leaching Technique for Gold in Native Gold and Gold±silver Tellurides." Minerals Engineering 14 (1): 1–12. https://doi.org/10.1016/S0892-6875(00)00156-4.

House, John Marsden and Iain, John Marsden, C. Iain. House, John Marsden and Iain House, John Marsden, and C. Iain. House. 2006. Chemistry of Gold Extraction. London: SME. https://doi.org/10.1179/030801892791925466.

Janisch, P. R. 1986. "GOLD IN SOUTH AFRICA." Journal of The South African Institute of Mining and Metallurgy 86 (8): 273–316.

Janse van Rensburg. 2023. "Re: Request for Information for Research Study."

Janse van Rensburg, Sonestie. 2016. "Guidelines for Retreatment of SA Gold Tailings: MINTEK's Learnings." Proceedings of the 23rd WasteCon Conference, no. October: 367–76. https://iwmsa.co.za/sites/default/files/downloads/56. Janse van Rensburg%2C S.pdf.

Jeffrey, Matthew I., and Ian M. Ritchie. 2000. "The Leaching of Gold in Cyanide Solutions in the Presence of Impurities II. The Effect of Silver." Journal of The Electrochemical Society 147 (9): 3272. https://iopscience.iop.org/article/10.1149/1.1393892.

Karimi, P., H. Abdollahi, A. Amini, M. Noaparast, S. Z. Shafaei, and F. Habashi. 2010. "Cyanidation of Gold Ores Containing Copper, Silver, Lead, Arsenic and Antimony." International Journal of Mineral Processing 95 (1–4): 68–77. https://doi.org/10.1016/j.minpro.2010.03.002.

Khalid, Muhammad. 2017. "Gold Cyanidation – Gold Associated with Silver Minerals Embedded within Base-Metal Sulphide Matrices."

Khalid, Muhammad, Faïçal Larachi, and Alain Adnot. 2018. "The Role of Silver Minerals on the Cyanidation of Gold Particles Embedded within Multi-Sulphidic Mineral Matrices." Canadian Journal of Chemical Engineering 96 (10): 2299–2307. https://doi.org/10.1002/cjce.23218.

Kim, Rina, and Ahmad Ghahreman. 2019. "The Effect of Ore Mineralogy on the Electrochemical Gold Dissolution Behavior in Various Cyanide and Oxygen Concentrations; Effect of Sulfidic Ores Containing Heavy Metals." Hydrometallurgy 184 (November 2018): 75–87. https://doi.org/10.1016/j.hydromet.2018.12.022.

Klimpel, R. R. 1999. "Industrial Experiences in the Evaluation of Various Flotation Reagent Schemes for the Recovery of Gold." Minerals and Metallurgical Processing 16 (1): 1–11. https://doi.org/10.1007/bf03402850.

Kondos, P. D., G. Deschênes, and R. M. Morrison. 1995. "Process Optimization Studies in Gold Cyanidation." Hydrometallurgy 39 (1–3): 235–50. https://doi.org/10.1016/0304-386X(95)00032-C.

Kongolo, K., and M. D. Mwema. 1998. "The Extractive Metallurgy of Gold." Hyperfine Interactions 111 (1–4): 281–89. https://doi.org/10.1007/978-1-4684-8425-0.

Korolev, Ivan, Pelin Altinkaya, Mika Haapalainen, Eero Kolehmainen, Kirsi Yliniemi, and Mari Lundström. 2022. "Electro-Hydrometallurgical Chloride Process for Selective Gold Recovery from Refractory Telluride Gold Ores: A Mini-Pilot Study." Chemical Engineering Journal 429 (February). https://doi.org/10.1016/j.cej.2021.132283.

Korolev, Ivan, Pelin Altınkaya, Petteri Halli, Pyry Mikko Hannula, Kirsi Yliniemi, and Mari Lundström. 2018. "Electrochemical Recovery of Minor Concentrations of Gold from Cyanide-Free Cupric Chloride Leaching Solutions." Journal of Cleaner Production 186 (June): 840–50. https://doi.org/10.1016/j.jclepro.2018.03.177.

Kuyucak, Nural, and Ata Akcil. 2013. "Cyanide and Removal Options from Effluents in Gold Mining and Metallurgical Processes." Minerals Engineering 50–51 (September): 13–29. https://doi.org/10.1016/j.mineng.2013.05.027.

Larrabure, Gonzalo, and Juan Carlos F. Rodríguez-Reyes. 2021. "A Review on the Negative Impact of Different Elements during Cyanidation of Gold and Silver from Refractory Ores and Strategies to Optimize the Leaching Process." Minerals Engineering 173 (September): 107194. https://doi.org/10.1016/j.mineng.2021.107194.

Lee, Youngmi. 2007. "Handbook of Electrochemistry." Handbook of Electrochemistry, no. 8: 249–53. http://www.sciencedirect.com/science/article/pii/B9780444519580500185.

Liu, G. Q., and W. T. Yen. 1995. "Effects of Sulphide Minerals and Dissolved Oxygen on the Gold and Silver Dissolution in Cyanide Solution." Minerals Engineering 8 (1–2): 111–23. https://doi.org/10.1016/0892-6875(94)00107-N.

Liu, Haiming, and Georges Beaudoin. 2021. "Geochemical Signatures in Native Gold Derived from Au-Bearing Ore Deposits." Ore Geology Reviews 132 (February): 104066. https://doi.org/10.1016/j.oregeorev.2021.104066.

Liu, Yun, Zhi Dang, Yin Xu, and Tianyuan Xu. 2013. "Pyrite Passivation by Triethylenetetramine: An Electrochemical Study." Journal of Analytical Methods in Chemistry 2013. https://doi.org/10.1155/2013/387124.

Lottermoser, Bernd G. 2011. "Recycling, Reuse and Rehabilitation of Mine Wastes." Elements 7 (6): 405–10.

Maseko, Thandazi. 2017. "INTERNAL REPORT Gold Dump Study Update." http://www.mintek.co.za.

Mbayo, Jean Jacques Kalombo, Henry Simonsen, and Sehliselo Ndlovu. 2019. "Improving the Gold Leaching Process of Refractory Ores Using the Jetleach Reactor." Minerals Engineering 134 (April): 300–308. https://doi.org/10.1016/j.mineng.2019.02.003.

Morrison, Gregg W, Williams J Rose, and Subhash Jaireth. 1991. "Geological and Geochemical Controls on the Silver Content (Fineness) of Gold in Gold-Silver Deposits." Ore Geology Reviews 6 (4): 333–64. https://doi.org/https://doi.org/10.1016/0169-1368(91)90009-V.

Mwase, James M., Jochen Petersen, and Jacques J. Eksteen. 2012. "Assessing a Two-Stage

Heap Leaching Process for Platreef Flotation Concentrate." Hydrometallurgy 129–130 (November): 74–81. https://doi.org/10.1016/j.hydromet.2012.09.007.

Nagy, I, P Mrkusic, and H W McCulloch. 1966. "Chemical Treatment of Refractory Gold Ores." National Institute for Metallurgy, Ran Burg–South Africa 124.

Nengovhela, A. C., B. Yibas, and J. S. Ogola. 2006. "Characterisation of Gold Tailings Dams of the Witwatersrand Basin with Reference to Their Acid Mine Drainage Potential, Johannesburg, South Africa." Water SA 32 (4): 499–506. https://doi.org/10.4314/wsa.v32i4.5290.

Nicol, M J, C A Fleming, and R L Paul. 1992. "The Chemistry of the Extraction of Gold." The Extractive Metallurgy of Gold, 831–905.

Nunan, Thiago Oliveira, Isabella Lima Viana, Guilherme C. Peixoto, Herbert Ernesto, Daniel Martin Verster, Jose Henrique Pereira, Jose M. Bonfatti, and Luiz Alberto Cesar Teixeira. 2017. "Improvements in Gold Ore Cyanidation by Pre-Oxidation with Hydrogen Peroxide." Minerals Engineering 108 (July): 67–70. https://doi.org/10.1016/j.mineng.2017.01.006.

Oraby, E. A., and J. J. Eksteen. 2016. "Gold Dissolution and Copper Suppression during Leaching of Copper – Gold Gravity Concentrates in Caustic Soda-Low Free Cyanide Solutions." Minerals Engineering 87 (March): 10–17. https://doi.org/10.1016/j.mineng.2015.08.006.

Oraby, E. A., J. J. Eksteen, and B. C. Tanda. 2017. "Gold and Copper Leaching from Gold-Copper Ores and Concentrates Using a Synergistic Lixiviant Mixture of Glycine and Cyanide." Hydrometallurgy 169 (May): 339–45. https://doi.org/10.1016/j.hydromet.2017.02.019.

Phalwane, Mpho, Sazini Makamu, and Kevin Pillay. 2017. "EXTERNAL REPORT 7599 IMPROVING THE PERFORMANCE OF ANGLOGOLD ASHANTI'S MWS EAST DAM TAIILINGS TREATMENT PLANT." http://www.mintek.co.za.

Phiri, C., A. Nahman, J. Hall, L. Godfrey, and S. Oelofse. 2007. "Mineral Waste The Required Governance Environment To Enable Reuse Final Report," no. May. https://doi.org/CSIR/NRE/PW/IR/2007/0080/C.

Pillay, Seshree, Olga Yahorava, and Musa Dlame. 2017. "EXTERNAL REPORT 7510 SIBANYE WEST RAND TAILINGS PROJECT." http://www.mintek.co.za.

Problems, Physicochemical. 2015. "Influence of Pre-Aeration on Cyanide Leaching of a Non-Refractory" 51 (2): 647–60.

Rabieh, A., J. J. Eksteen, B. Albijanic, G C Allan, J T Woodcock, M. N. Lehmann, S. O'Leary, et al. 2017. "The Effect of Grinding Chemistry on Cyanide Leaching of Gold in the Presence of Pyrrhotite." Hydrometallurgy 173 (April): 115–24. https://doi.org/10.1016/j.hydromet.2017.08.013.

Rees, Kevin Lee. 2000. "THE LEACHING AND ADSORPTION BEHAVIOUR OF GOLD ORES."

Reyes, F., Q. Lin, J. J. Cilliers, and S. J. Neethling. 2018. "Quantifying Mineral Liberation by

Particle Grade and Surface Exposure Using X-Ray MicroCT." Minerals Engineering 125 (August): 75–82. https://doi.org/10.1016/j.mineng.2018.05.028.

S. Kumari. 1941. "Stability Constants of Complexes." https://mltcollege.org/wp-content/uploads/2021/06/Stability-Constants.pdf.

Sascha-Lee Solomonds. 2018. "DRDGold: South Africa's Largest Tailings Project Kicks into First Gear." Mining Review Africa, 2018.

Schwartz, George M. 1944. "The Host Minerals of Native Gold." Economic Geology 39 (6): 371–411. https://doi.org/10.2113/gsecongeo.39.6.371.

Senanayake, G. 2004. "Gold Leaching in Non-Cyanide Lixiviant Systems: Critical Issues on Fundamentals and Applications." Minerals Engineering 17 (6): 785–801. https://doi.org/10.1016/j.mineng.2004.01.008.

Senanayake, G. 2008. "A Review of the Effects of Silver, Lead, Sulfide and Carbonaceous Matter on Gold Cyanidation and Mechanistic Interpretation." Hydrometallurgy 90 (1): 46–73. https://doi.org/10.1016/j.hydromet.2007.09.013.

Senanayake, Gamini. 2006. "The Cyanidation of Silver Metal: Review of Kinetics and Reaction Mechanism." Hydrometallurgy 81 (2): 75–85. https://doi.org/10.1016/j.hydromet.2005.12.001.

Silverman, M. P. 1967. "Mechanism of Bacterial Pyrite Oxidation." Journal of Bacteriology 94 (4): 1046–51. https://doi.org/10.1128/jb.94.4.1046-1051.1967.

Small, G., A. Michelmore, and S. Grano. 2003. "Size-Dependent Gold Deportment in the Products of Copper Flotation and Methods to Increase Gold Recovery." Journal of The South African Institute of Mining and Metallurgy 103 (9): 573–79.

Soltani, Faraz, Mehdi Marzban, Hossna Darabi, Mokhtar Aazami, and Mohsen Hemmati Chegeni. 2020. "Effect of Oxidative Pretreatment and Lead Nitrate Addition on the Cyanidation of Refractory Gold Ore." JOM 72 (2): 774–81. https://doi.org/10.1007/s11837-019-03859-0.

Sousa, Rui, Bethany Simons, Kathy Bru, Ana Botelho de Sousa, Gavyn Rollinson, Jens Andersen, Mirko Martin, and Mário Machado Leite. 2018. "Use of Mineral Liberation Quantitative Data to Assess Separation Efficiency in Mineral Processing – Some Case Studies." Minerals Engineering 127 (October): 134–42. https://doi.org/10.1016/j.mineng.2018.08.004.

Staden, Petrus Van, Sazini Makamu, Roxanne Mottay, Tebogo Moloane, Lesetsa Mabokela, Ryan Govender, and Carl Bergmann. 2020. "INTERNAL REPORT Adding Value to the ERGO Tailings Treatment Plant." http://www.mintek.co.za.

Stange, W. 1999. "The Process Design of Gold Leaching and Carbon-in-Pulp Circuits." Journal of The South African Institute of Mining and Metallurgy 99 (1): 13–25.

Strauss, Jakolien A., Volha Bazhko, Genarro Ventruti, Xuan Liguo, and Mario A. Gomez. 2021. "Arsenic Behavior during the Treatment of Refractory Gold Ores via POX: Characterization of Fe-AsO4-SO4 Precipitates." Hydrometallurgy 203 (August): 105616. https://doi.org/10.1016/j.hydromet.2021.105616.

Swash, P. M. 1988. "Mineralogical Investigation of Refractory Gold Ores and Their Beneficiation, With Special Reference To Arsenical Ores." Journal of The South African Institute of Mining and Metallurgy 88 (5): 173–80.

Teague, A. J., J. S.J. Van Deventer, and C. Swaminathan. 1999. "The Effect of Galvanic Interaction on the Behaviour of Free and Refractory Gold during Froth Flotation." International Journal of Mineral Processing 57 (4): 243–63. https://doi.org/10.1016/S0301-7516(99)00025-3.

Teimouri, S., J. H. Potgieter, G. S. Simate, L. van Dyk, and M. Dworzanowski. 2020. "Oxidative Leaching of Refractory Sulphidic Gold Tailings with an Ionic Liquid." Minerals Engineering 156 (September): 106484. https://doi.org/10.1016/j.mineng.2020.106484.

Teimouri, Samaneh. 2020. "Feasibility Study on the Use of Imidazolium-Based Ionic Liquids in the Extraction of Gold," 1–89.

Tourtelot, Harry A., and Leonard B. Riley. 1973. "Size and Shape of Gold and Platinum Grains." In Ores in Sediments, 307–19. Springer Berlin Heidelberg. https://doi.org/10.1007/978-3-642-65329-2_22.

Tran, Tam. 1992. "The Chemistry of Gold Extraction." International Materials Reviews 37 (1): 292–292. https://doi.org/10.1179/imr.1992.37.1.292.

Tshilombo, Alain F., and Roeløf F. Sandenbergh. 2001. "Electrochemical Study of the Effect of Lead and Sulphide Ions on the Dissolution Rate of Gold in Alkaline Cyanide Solutions." Hydrometallurgy 60 (1): 55–67. https://doi.org/10.1016/S0304-386X(00)00157-2.

TWP The Basil Read Group. 2012. "ERGO Process Discussion Memo (2)." TWP The Basil Read Group.

Vaughan, J. P. 2004. "The Process Mineralogy of Gold: The Classification of Ore Types." Jom 56 (7): 46–48. https://doi.org/10.1007/s11837-004-0092-8.

Venter, D., S. L. Chryssoulis, and T. Mulpeter. 2004. "Using Mineralogy to Optimize Gold Recovery by Direct Cyanidation." Jom 56 (8): 53–56. https://doi.org/10.1007/s11837-004-0183-6.

Wadsworth, M. E., X. Zhu, J. S. Thompson, and C. J. Pereira. 2000. "Gold Dissolution and Activation in Cyanide Solution: Kinetics and Mechanism." Hydrometallurgy 57 (1): 1–11. https://doi.org/10.1016/S0304-386X(00)00084-0.

Wadsworth, Milton E., and Ximeng Zhu. 2003. "Kinetics of Enhanced Gold Dissolution: Activation by Dissolved Silver." International Journal of Mineral Processing 72 (1–4): 301–10. https://doi.org/10.1016/S0301-7516(03)00106-6.

Wai, Wong, Leong Eugene, and Arun S Mujumdar. 2009. "Gold Extraction and Recovery Processes." Minerals, Metals and Materials Technology Center, 5–10.

Wang, Chen, David Harbottle, Qingxia Liu, and Zhenghe Xu. 2014. "Current State of Fine Mineral Tailings Treatment: A Critical Review on Theory and Practice." Minerals Engineering 58: 113–31. https://doi.org/10.1016/j.mineng.2014.01.018.

Wang, Xu, Wenqing Qin, Fen Jiao, Congren Yang, Yanfang Cui, Wei Li, Zhengquan Zhang, and Hao Song. 2019. "Mineralogy and Pretreatment of a Refractory Gold Deposit in Zambia." Minerals 9 (7). https://doi.org/10.3390/min9070406.

Warlo, Mathis, Christina Wanhainen, Glenn Bark, Alan R. Butcher, Iris McElroy, Dominique Brising, and Gavyn K. Rollinson. 2019. "Automated Quantitative Mineralogy Optimized for Simultaneous Detection of (Precious/Critical) Rare Metals and Base Metals in a Production-Focused Environment." Minerals 9 (7). https://doi.org/10.3390/min9070440.

Whiteman, E., N. O. Lotter, and S. R. Amos. 2016. "Process Mineralogy as a Predictive Tool for Flowsheet Design to Advance the Kamoa Project." Minerals Engineering 96–97: 185–93. https://doi.org/10.1016/j.mineng.2016.05.004.

Wu, Jiajia, Junmo Ahn, and Jaeheon Lee. 2021. "Gold Deportment and Leaching Study from a Pressure Oxidation Residue of Chalcopyrite Concentrate." Hydrometallurgy 201 (May). https://doi.org/10.1016/j.hydromet.2021.105583.

Xu, Baotian, and Yuying Wang. 2015. "Stability Analysis of the Lingshan Gold Mine Tailings Dam under Conditions of a Raised Dam Height." Bulletin of Engineering Geology and the Environment 74 (1): 151–61. https://doi.org/10.1007/s10064-014-0602-z.

Yannopoulos, J C. 1991. "Cyanidation of Gold Ores." In The Extractive Metallurgy of Gold, 141–70. Boston, MA: Springer US. https://doi.org/10.1007/978-1-4684-8425-0_8.

Yliniemi, Kirsi, Zulin Wang, Ivan Korolev, Pyry Hannula, Petteri Halli, and Mari Lundström. 2018. "Effect of Impurities in Precious Metal Recovery by Electrodeposition-Redox Replacement Method from Industrial Side-Streams and Process Streams." ECS Transactions 85 (4): 59–67. https://doi.org/10.1149/08504.0059ECST.

Youlton, K. L., J. A. Kinnaird, and B. J. Youlton. 2021. "Depositional Environment -The Original Control on Gold Processing." Journal of the Southern African Institute of Mining and Metallurgy 121 (6): 267–76. https://doi.org/10.17159/2411-9717/811/2021.

Zhang, Suchun. 2004. "Oxidation of Refractory Gold Concentrates and Simultaneous Dissolution of Gold in Aerated Alkaline Solutions." PhD Thesis.

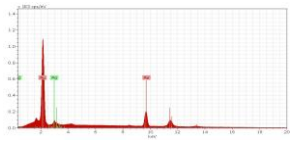
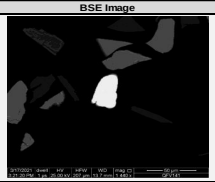
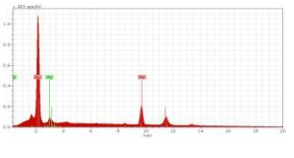
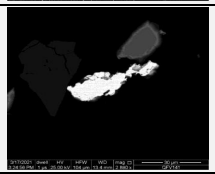
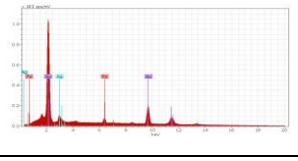
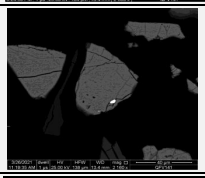
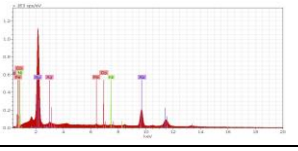
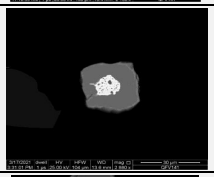
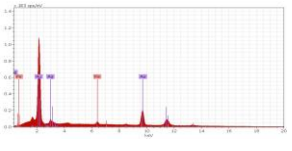
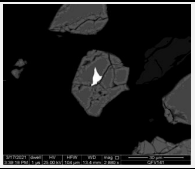
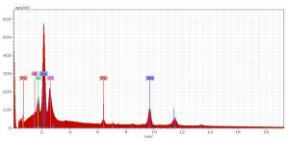
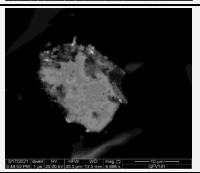
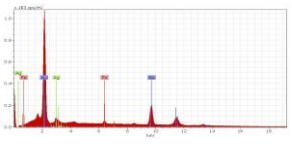
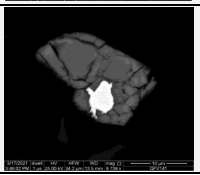
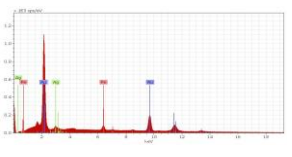
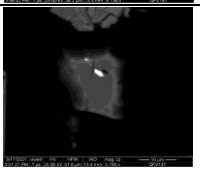
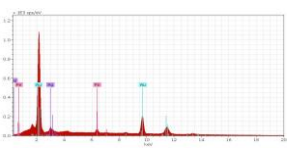
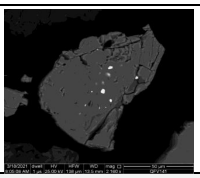
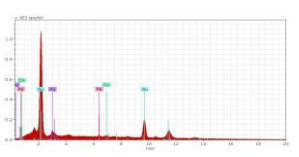
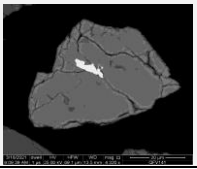
Zhou, Joe, Bruce Jago, Chris Martin, and S G S Lakefield. 2004. "Establishing the Process Mineralogy of Gold Ores."

Zhou, Joe Y., and Louis J. Cabri. 2004. "Gold Process Mineralogy: Objectives, Techniques, and Applications." JOM 56 (7): 49–52. https://doi.org/10.1007/s11837-004-0093-7.

Zia, Yasin, Sima Mohammadnejad, and Mahmoud Abdollahy. 2020. "Destabilisation of Gold Cyanide Complex by Sulphur Species: A Computational Perspective." Hydrometallurgy 197 (November). https://doi.org/10.1016/j.hydromet.2020.105459.

9. APPENDIX A

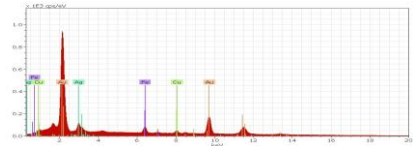
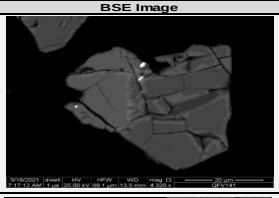
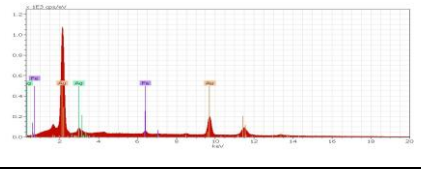
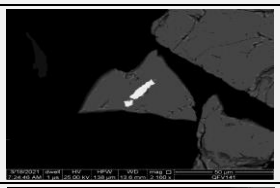
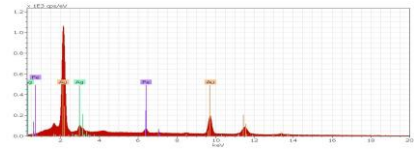
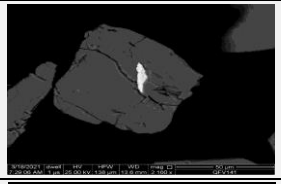
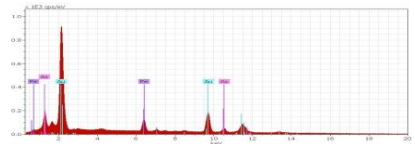
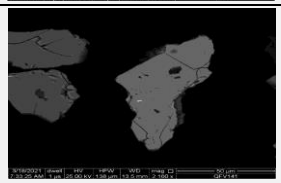
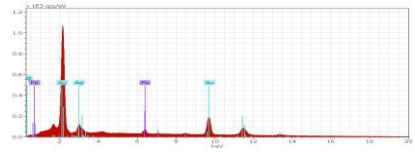
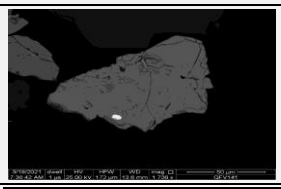
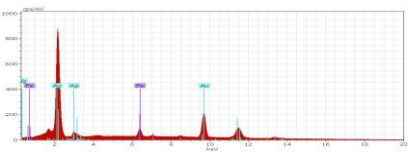
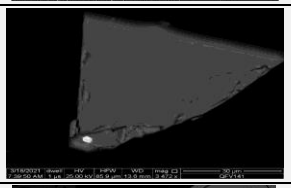
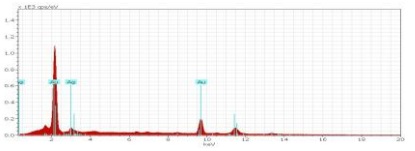
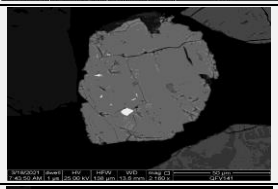
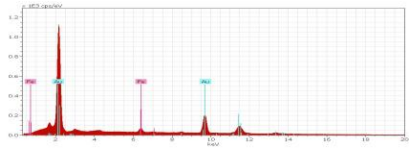
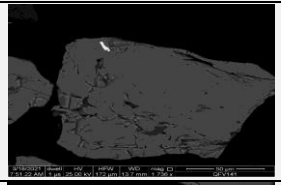
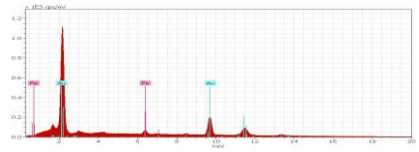
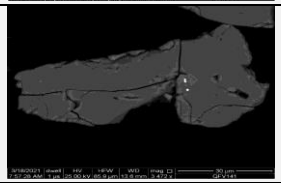
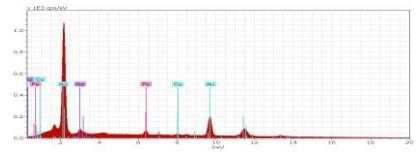
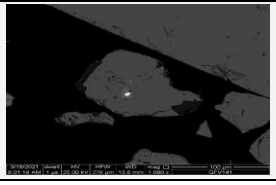
9.1 Process mineralogy

Grain#	Particle Map	EDS Spectra	BSE Image	Comment
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2				Liberated gold
3				Gold locked within pyrite
4				Gold locked within arsenopyrite
5				Gold locked within pyrite
6				Gold locked within a Cl-bearing silicate
7				Gold locked within pyrrhotite
8				Gold locked within pyrite
9				Gold locked within pyrite and pyrrhotite
10				Gold locked within arsenopyrite

*grain comments (locked/liberated) based on free surface liberation

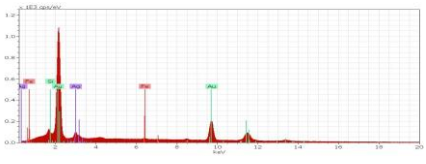
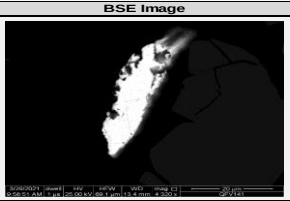
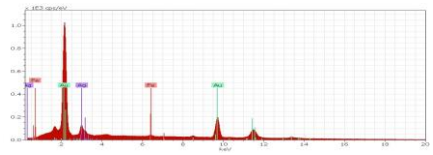
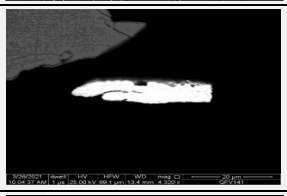
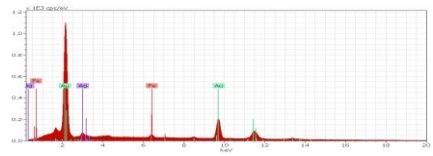
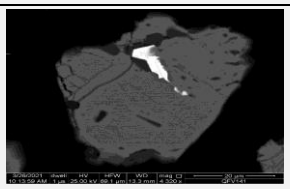
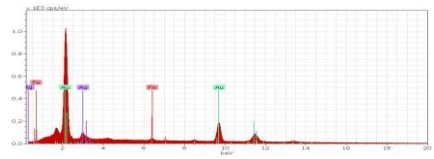
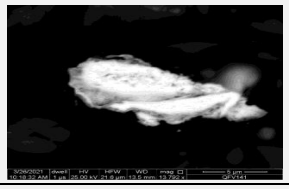
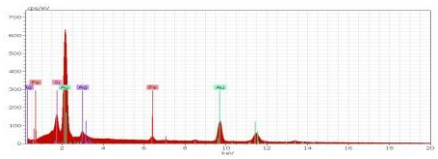
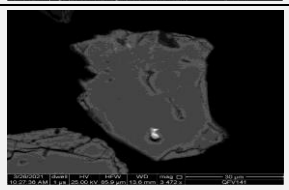
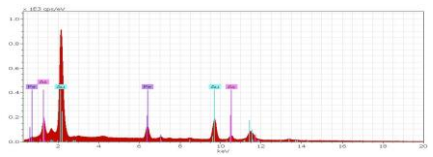
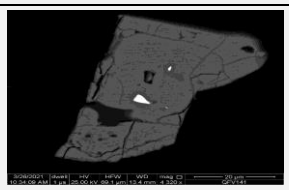
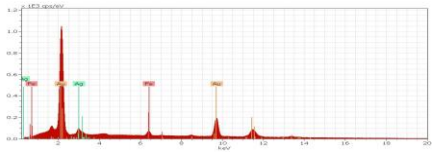
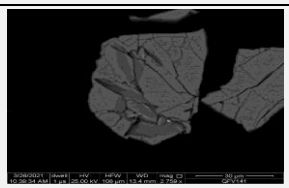
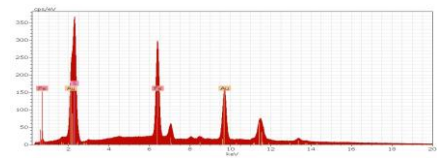
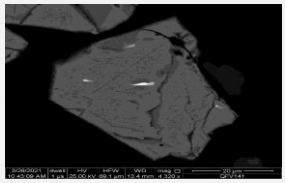
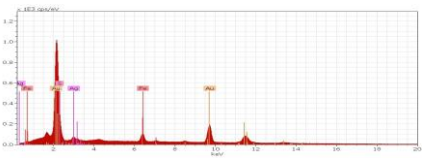
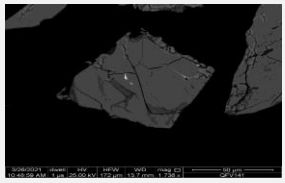
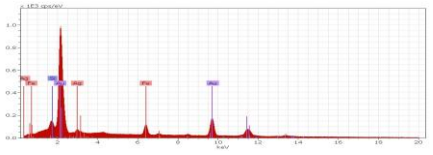
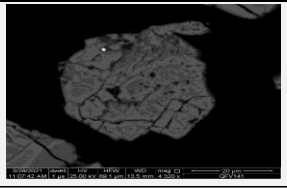
The Application of Process Mineralogy to Improve Gold Extraction from Wits Tailings

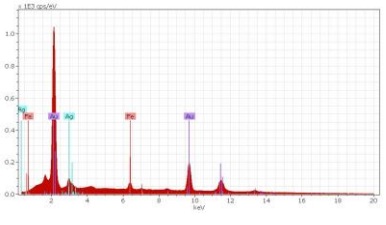
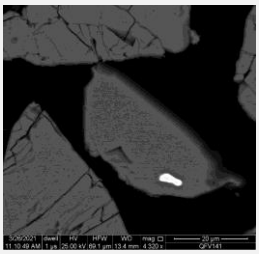
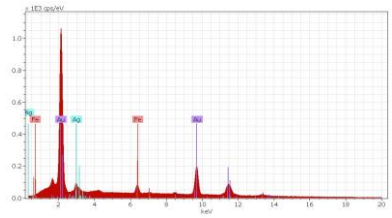
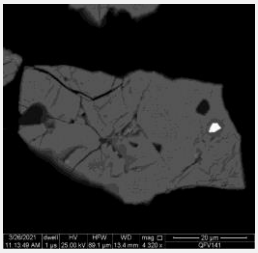
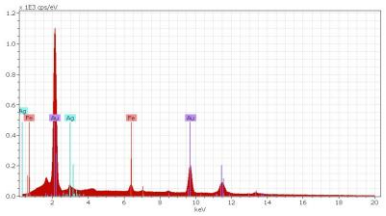
Sazini Makamu

Grain#	Particle Map	EDS Spectra	BSE Image	Comment
11				Gold associated with chalcopyrite and pyrite
12				Gold locked within pyrite
13				Gold locked with pyrite
14				Gold locked within arsenopyrite
15				Gold locked within pyrrhotite
16				Gold locked within pyrite and pyrrhotite
17				Gold locked within arsenopyrite
18				Gold locked within pyrite
19				Gold locked within pyrite
20				Gold locked within pyrite

The Application of Process Mineralogy to Improve Gold Extraction from Wits Tailings

Sazini Makamu

Grain#	Particle Map	EDS Spectra	BSE Image	Comment
21				Liberated Gold
22				Liberated gold
23				Gold locked within pyrite and pyrrhotite
24				Liberated gold
25				Gold locked within pyrite
26				Gold locked within pyrite
27				Gold locked within pyrite and pyrrhotite
28				Gold locked within pyrite
29				Gold locked in pyrite
30				Gold locked in pyrite and pyrrhotite

Grain#	Particle Map	Energy Dispersive X-Ray Spectrometer (EDS) Spectra	BSE Image	Comment
31				Gold locked within pyrite
32				Gold locked within pyrite
33				Gold associated with pyrite

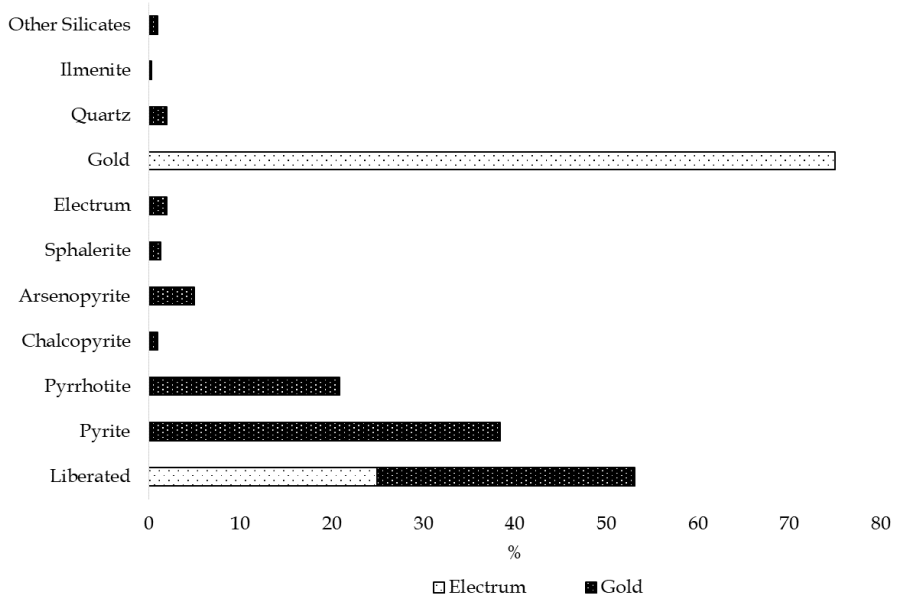
9.2. Liberation data

	Fraction		+53 µm	-53+38µm	-38+20 µm	-20+10 µm	-10 µm	Combined
Liberation by Free Surface (Discrete)	Locked	0-10	100.00	51.04	19.80	14.92	0.00	13.68
		10-20	0.00	11.64	0.00	51.05	0.00	4.63
		20-30	0.00	19.30	0.00	0.00	0.00	1.14
	Low Middling's	30-40	0.00	6.27	0.00	34.03	0.00	3.00
		40-50	0.00	0.00	0.00	0.00	0.00	0.00
	High Middling's	50-60	0.00	0.00	52.98	0.00	0.00	19.17
		60-70	0.00	0.00	0.00	0.00	0.00	0.00
		70-80	0.00	0.00	0.00	0.00	0.00	0.00
	Liberated	80-90	0.00	0.00	0.00	0.00	0.00	0.00
		90-100	0.00	0.00	27.22	0.00	0.00	9.85
		100	0.00	11.76	0.00	0.00	100.00	48.54

	Fraction		+53 µm	-53+38µm	-38+20 µm	-20+10 µm	-10 µm	Combined
Liberation by Area (Discrete)	Locked	0-10	100.00	56.41	12.03	65.97	0.00	15.13
		10-20	0.00	21.09	7.78	0.00	0.00	4.05
		20-30	0.00	4.48	0.00	0.00	0.00	0.26
	Low Middling's	30-40	0.00	0.00	0.00	34.03	0.00	2.63
		40-50	0.00	6.27	52.98	0.00	0.00	19.54
	High Middling's	50-60	0.00	0.00	0.00	0.00	0.00	0.00
		60-70	0.00	0.00	0.00	0.00	0.00	0.00
		70-80	0.00	0.00	0.00	0.00	0.00	0.00
	Liberated	80-90	0.00	0.00	0.00	0.00	0.00	0.00
		90-100	0.00	0.00	27.22	0.00	0.00	9.85
		100	0.00	11.76	0.00	0.00	100.00	48.54

Size Class (ECD-µm)	+53 µm			-53/+38 µm			-38/+20 µm			-20/+10 µm			-10 µm			Combined	
	Mass%	Area %	N	Mass%	Area %	N	Mass%	Area %	N	Mass%	Area %	N	Mass%	Area %	N	Mass%	N
<5	0.0	0.0	0	49.8	50.5	45	12.0	12.8	9	100.0	100.0	6	100.0	100.0	1	62.9	61
<10	100.0	100.0	31	11.6	12.1	3	0.0	0.0	0	0.0	0.0	0	0.0	0.0	0	3.0	34
<15	0.0	0.0	0	26.8	26.2	5	7.8	11.5	2	0.0	0.0	0	0.0	0.0	0	4.4	7
<20	0.0	0.0	0	0.0	0.0	0	0.0	0.0	0	0.0	0.0	0	0.0	0.0	0	0.0	0
<35	0.0	0.0	0	0.0	0.0	0	27.2	23.1	1	0.0	0.0	0	0.0	0.0	0	9.8	1
>35	0.0	0.0	0	11.8	11.2	1	53.0	52.6	1	0.0	0.0	0	0.0	0.0	0	19.9	2
Total	**100.0**	**100.0**	**31**	**100.0**	**100.0**	**54**	**100.0**	**100.0**	**13**	**100.0**	**100.0**	**6**	**100.0**	**100.0**	**1**	**100.0**	**105**

9.3 Characterisation of gold-associated sulphide minerals in Wits pyrite concentrate



9.4. Bulk modal mineralogy characterisation using X-Ray Diffraction (XRD)

Feed		
Compound Name	**Formula**	**Abundance**
Chlorite	$(Mg,Fe)_5(Si_3Al)O_{10}(OH)_8$	10.8
Mica	$KAl_2(AlSi_3O_{10})(F,OH)_2$	15.3
Gypsum	$CaSO_4 \cdot 2H_2O$	3.7
Pyrite	FeS_2	19.0
Pyrophyllite	$Al_2Si_4O_{10}(OH)_2$	1.8
Pyrrhotite	$Fe_{(1-x)}S$	4.1
Quartz	SiO_2	45.3
	Total	**100.0**

Mineral Chemical Reconciliation					
	Al	**Fe**	**Mg**	**Si**	**S**
	0.9	3.2	0.6	1.4	0.0
	3.1	0.0	0.0	3.2	0.0
	0.0	0.0	0.0	0.0	0.7
	0.0	8.8	0.0	0.0	10.1
	0.3	0.0	0.0	0.6	0.0
	0.0	2.6	0.0	0.0	1.5
	0.0	0.0	0.0	21.2	0.0
Total	**4.3**	**14.6**	**0.6**	**26.3**	**12.4**
Chemical Total	**4.4**	**13.7**	**0.5**	**26.6**	**12.4**

Mineral chemical compositions*						
Compound Name	**Al**	**Fe**	**K**	**Mg**	**Si**	**S**
Chlorite	8.1	29.4	0.0	5.5	12.7	0.0
Mica	20.3	0.0	9.8	0.0	21.1	0.0
Gypsum	0.0	0.0	0.0	0.0	0.0	18.6
Pyrite	0.0	46.6	0.0	0.0	0.0	53.5
Pyrophyllite	15.0	0.0	0.0	0.0	31.2	0.0
Pyrrhotite	0.0	62.3	0.0	0.0	0.0	37.7
Quartz	0.0	0.0	0.0	0.0	46.7	0.0

* based on theoretical compositions (http://webmineral.com/)

9.5. Logsheet of reactivity of iron-sulphide minerals during Cyanidation

Initial												
	Mass of empty reactor	Mass of ore	Mass of water	Volume of water	Mass of slurry+ reactor	Mass of slurry	Slurry Volume	pH	Eh	DO	NaCN target	NaCN addition
Stage 1	g	g	g	mL	g	g	mL		mV	mg/L	kg/t	g
	5288.22	5000	10200	10200	15150.00	9861.78	12000	11.46	189	2.98	2	10

Lime addition to control pH				
	Mass of container	Before container + lime	After container + lime	Mass of lime added
	g	g	g	g
Stage 1	7.76	32.74	4.65	28.09

Final												
	Mass of system	Mass of slurry	pH	Eh	DO	Mass of dry solids	Solid mass loss	Vol. filtrate	Mass filtrate	Density filtrate	Final free CN	NaCN Consumption
	g	g		mV	mg/L	g	%	mL	g	g/mL	mg/L	kg/t
Stage 1	20600.00	15311.78	10.64	-103.6	3.58	4950.00	1%	8000	7900.00	0.99	164	

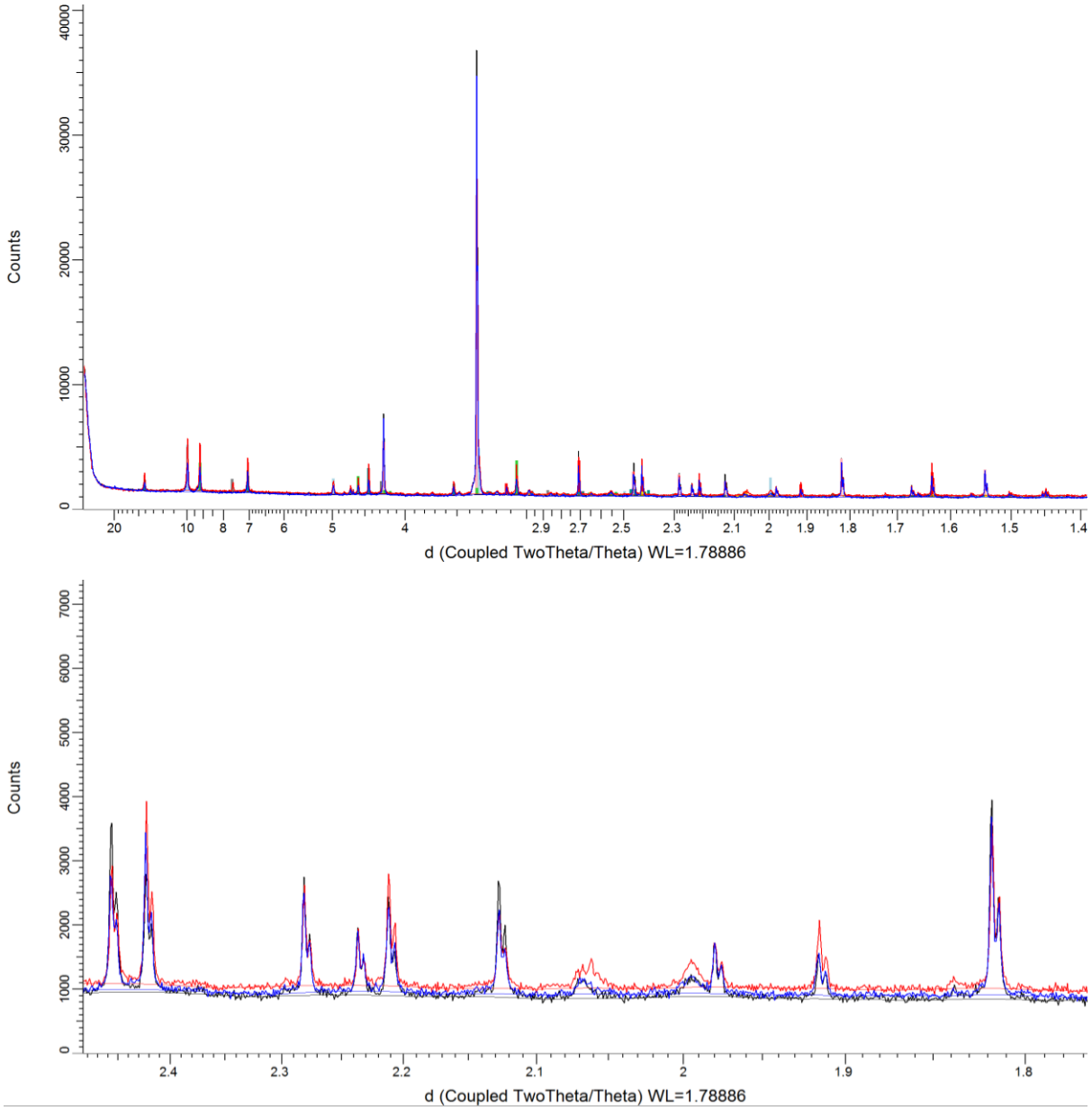
Initial												
	Mass of empty reactor	Mass of ore	Mass of water	Volume of water	Mass of slurry+ reactor	Mass of slurry	Slurry Volume	pH	Eh	DO	NaCN target	NaCN addition
Stage 2	g	g	g	mL	g	g	L		mV	mg/L	kg/t	g
	2744.73	2000	6100	61000	10800.00	8055.27	7000	11.65	-819	3.51	2	6

Lime addition to control pH				
	Mass of container	Before container + lime	After container + lime	Mass of lime added
	g	g	g	g
Stage 2	7.78	25.06	13.7	11.36

Final												
	Mass of system	Mass of slurry	pH	Eh	DO	Mass of dry solids	Solid mass loss	Vol. filtrate	Mass filtrate	Density filtrate	Final free CN	NaCN Consumption
	g	g		mV	mg/L	g	%	mL	g	g/mL	mg/L	kg/t
Stage 2	10800.00	8055.27	11.36	-117.6	2.76	1967.12	2%	5200	5187.78	1.00	435	

9.6. Reactivity of iron-sulphide minerals during cyanidation: QXRD analysis

Mineral Phase	Formula	0 Hrs	12 Hrs	24 Hrs
Quartz	SiO_2	45.1	50.7	51.7
Mica	$KAl_2(AlSi_3O_{10})(F,OH)_2$	16.7	17.4	16.5
Chlorite	$(Mg,Fe)_5(Si_3Al)O_{10}(OH)_8$	6.7	6.6	7.5
Gypsum	$CaSO_4 \cdot 2H_2O$	1.9	2.5	3.6
Feldspar	$KAlSi_3O_8$	5.3	6.7	6.5
Pyrophyllite	$Al_2Si_4O_{10}(OH)_2$	1.9	1.9	1.6
Pyrrhotite	FeS	16.3	11.0	9.4
Pyrite	FeS_2	4.3	1.1	1.8
Arsenopyrite	FeAsS	2.1	2.1	1.4


Counts
40000
30000
20000
10000
0
20
10
8
7
6
5
4
2.9
2.7
2.5
2.3
2.1
2
1.9
1.8
1.7
1.6
1.5
1.4
d (Coupled TwoTheta/Theta) WL=1.78886
Counts
7000
6000
5000
4000
3000
2000
1000
0
2.4
2.3
2.2
2.1
2
1.9
1.8
d (Coupled TwoTheta/Theta) WL=1.78886

9.7. Log sheet and mass balance of Deportment Of Gold In Various Mineral Phases

Date	11-06-2020							
Project name	ERGO FLOATATION CONCENTRATE							
Project number	HMR221003							
Test number/name	DIAGNOSTIC LEACH							
Measured parameter	CN/A	CIL/B	HCl/C	CIL/C	HNO_3/D	CIL/D	Roast/E	CIL/E
Ore Mass(g)	200	1600	1450	1222.25	1050	660	500	350
Solution Volume (g)	467	3733	2175.00	2851.92	1575.00	1540.00		816
L/S RATIO	30%	30%	40%	30%	40%	30%		30%
pH initial	5.17	5.18	0.76	3.97	-0.17	4.05		5.78
Eh initial (Ag/AgCl, Mv)	255	256	540	302	406.00	345		176
$Ca(OH)_2$ addition(g)	1.01	7.65		5.00		1.63		1.8
NaCN addition(g) 10 kg/t	2	16		12.20		6.60		3.5
Carbon addition (g) 20g/L		74		57.00		31.50		16.32
HCl concentration initial (%)			2.5 M					
HNO_3 concentration initial (%)					27.50			
Roast temperature (ºC)							850.00	
Running time (hrs.)	24	24	1.00	24.00	1.00	24.00	1.00	24
Wet cake (g)	274.18	2199.86	2337.1	1680.41	1115	956.04		584.38
Dry cake (g)	195.66	1593.53	1367.51	1205.63	807.39	647.97	485	344.23
Wet carbon (g)		98		89		66.5		16
Dry carbon (g)		72		51		28		14.5
Moisture loss (%)	28.64	27.56	41.49	28.25	27.59	32.22		41.09
Mass loss (%)	2.17	0.40	5.69	1.36	23.11	1.82	3.00	1.65
Filtrate (ml)	365	3010	1940	2288.78	760	1185		500
Filtrate (g)	362.14	2953.65	2027.17	2290	947.91	1151.61		497.69
Density	1.008	1.019	0.957	0.999	0.802	1.029		1.005
Evaporation (mL)	102	723	235	563.14	815	1529.67		805
pH final	10.7	10.76	-0.29	10.97	-0.28	10.33		11
Eh final (Ag/AgCl, Mv)	-169	-200	465	-194	509	-137		-152

Parameter		CN_A	CIL_B	CIL_C	CIL_D	CIL_E
Start solid	g	200	1600	1222.25	660	350
Start solution	g	467	3733	2851.92	1540	816
Final solution	g	362	2954	2027	1152	498
Final carbon	g	0	72	51	28	14.5
Final residue		196	1594	1206	648	344
Head grade Ave (N=3)	g/t	3.01	3.01	0.62	0.6	0.22
	mg Au	0.602	4.816	0.757795	0.396	0.077
Residue analysis	g/t	0.66	0.62	0.60	0.22	0.12
	mg Au	0.13	0.99	0.72	0.14	0.04
Carbon analysis	g/t	-	11.3	2.60	2.60	2.02
	mg Au	-	0.81	0.13	0.07	0.03
Filtrate analysis	mg/L	0.99	0.19	0	0	0
	mg Au	0.36	0.43	0.00	0.00	0.00
Au Extraction	%	78.55	79.49	4.54	64.00	46.35
Au accountability	%	81.01	46.31	112.96	54.38	91.69

12. APPENDIX B

10.1 Baseline Gold extraction improvement tests Mass balance, reagent consumption and measured parameters

Retention time	Mass of ore	Volume of water	Mass of carbon		IN head		Residue		Carbon		Accountability	Au extraction	NaCN consumption	CaO Consumption	DO	pH	ORP
	Feed	Solution	Output		Au	Au	Au	Au	Au	Au	Au	Au	NaCN	CaO	DO	pH	Eh
	g	mL	g		g/t	mg	g/t	mg	g/t	mg	%	%	kg/t	kg/t	mg/L		mV(Ag/AgCl)
0					**3.01**		**3.01**					**0.00**	**0**	**0**	**4.49**	**3.51**	**317**
1	**500**	**1000**	**26**		3.01	1.51	2.38	1.19	12.1	0.32	100	20.9	0.29	2.27	0.54	11.2	-90
2	**500**	**1000**	**26**		3.01	1.51	2.31	1.16	13.7	0.36	101	23.3	0.36	2.27	0.71	10.8	-120
3	**500**	**1000**	**26**		3.01	1.51	2.25	1.12	17.4	0.46	105	25.3	0.41	2.35	0.78	10.7	-139
4	**1000**	**2030**	**52**		3.01	3.01	1.90	1.90	24.3	1.26	105	36.9	0.48	2.44	34.1	10.6	-68
6	**1000**	**2030**	**52**		3.01	3.01	1.04	1.04	38.1	1.98	100	65.4	0.48	2.44	38.6	10.5	-62
8	**1000**	**2030**	**52**		3.01	3.01	1.00	1.00	41.1	2.14	104	66.8	0.49	2.50	49.6	10.5	-57
10	**1000**	**2030**	**52**		3.01	3.01	0.96	0.96	38.2	1.99	98	68.1	0.49	2.50	52.7	10.5	-56
12	**1000**	**2030**	**52**		3.01	3.01	0.92	0.92	40.1	2.09	100	69.4	0.50	2.64	53.8	10.4	-38
18	**1000**	**2030**	**52**		3.01	3.01	0.86	0.86	41.9	2.18	101	71.4	0.53	2.87	54.5	10.4	-32
24	**1000**	**2030**	**52**		3.01	3.01	0.86	0.86	43.2	2.24	103	71.4	0.58	2.87	55.9	10.4	-28
36	**1000**	**2030**	**52**		3.01	3.01	0.82	0.82	44.8	2.33	105	72.8	0.59	2.88	59.0	10.3	-22
48	**1000**	**2030**	**52**		3.01	3.01	0.76	0.76	45.2	2.35	103	74.8	0.64	2.88	61.3	10.3	-15
60	**1000**	**2030**	**52**		3.01	3.01	0.65	0.65	45.7	2.38	101	78.4	0.64	2.90	64.3	10.3	-14
72	**1000**	**2030**	**52**		3.01	3.01	0.64	0.64	47.2	2.45	103	78.7	0.65	2.94	80.3	9.80	-11

10.2 Gold extraction improvement tests Log sheet

Pre Ox										
	Test 1	**Test 2**	**Test 3**	**Test 4**	**Test 5**	**Test 6**	**Test 7**	**Test 8**	**Test 9**	**Test 10**
Solid%	33.0	33.0	33.0	33.0	33.0	33.0	33.0	33.0	33.0	33.0
Reactor Mass, g	541.8	554.3	574.3	561.2	558.7	561.2	492.3	555.6	555.1	549.2
Mass Solids, g	160.0	160.0	160.0	160.0	160.0	160.0	160.0	160.0	160.0	160.0
Mass water, g	324.8	324.8	324.8	324.8	324.8	324.8	324.8	324.8	324.8	324.8
Reactor mass + slurry	1026.6	1039.2	1059.4	1045.1	1048.4	1047.4	979.0	1042.5	1041.9	1033.2
Target pH.	11	11	11	11	11	11	11	11	11	11
DO/AIR control	Air	O_2	Air	Air	Air	Air	Air	Air	Air	OPT
Time, hours	1	1	6	3	3	3	3	3	3	3
DO After sparging	7.27	38.80	10.20	6.50	11.90	11.80	10.70	11.00	11.00	11.00
PH Natural	3.40	3.45	3.64	3.37	3.51	3.46	3.61	3.36	3.30	3.43
PH Initial	11.15	11.11	11.11	11.23	11.63	11.49	11.35	11.20	11.05	11.00
Eh initial	-40.00	-26.00			-19.00	-20.00	-64.00	-79.00	-70.00	-68.00
PH	9.74	10.49	8.47	8.84	9.40	9.15	9.50	9.29	8.36	7.47
EH	46.00	74.00	135.50	11.50	90.00	140.00	41.00	45	70	68
pH. adjusted	11.09	11.39	11.15	11.40	11.05	11.08	11.25	11	11	11
Eh adjusted	-69.00	13.00	-51.30	-88.00	-6.00	-78.00	-133.00	-100	-60	-85
NaCN, kg/t	2.00	2.00	2.00	2.00	2.00	2.00	1.00	1.5	2.00	2
NaCN, g	0.32	0.32	0.32	0.32	0.32	0.32	0.16	0	0.32	0
$Pb(NO_3)_2$, mg	0.00	0.00	0.00	0.00	96.00	48.00	0.00	0	0.00	0
Time, hours	24.00	24.00	24.00	24.00	24.00	24.00	24.00	24	8.00	18
Before container + lime	16.89	18.16	9.33	9.33	9.26	9.20	9.26	9.23	9.33	9.33
After container +	15.39	16.75	7.91	8.07	7.80	7.78	7.77	7.77	7.80	7.88
Mass of lime added, g	1.50	1.41	1.42	1.26	1.46	1.42	1.49	1.46	1.53	1.45
Cyanidation										
	Test 1	**Test 2**	**Test 3**	**Test 4**	**Test 5**	**Test 6**	**Test 7**	**Test 8**	**Test 9**	**Test 10**
Reactor mass + slurry final, g	1010.53	1023.24	1009.96	1045.88	998.25	995.20	978.96	1042.47	1023.65	989.53
PH final	10.76	10.72	10.99	11.13	9.87	9.72	10.50	9.96	10.33	10.28
Eh final	-64.00	-98.00	-144.00	-211.00	-69.00	-80.00	-31.00	-116.00	-173.00	-146.00
Wet cake mass, g	225.73	226.03	219.48	230.37	206.45	214.00	220.00	215.60	200.00	208.00
Container	212.12	221.18	230.53	212.14	214.09	232.90	230.53	212.14	649.05	631.90
Dry cake mass, g	155.95	155.94	151.84	155.57	150.24	152.82	154,68	151.36	158.00	140.94
filtrate	231.20	237.39	185.80	228.68	187.15	194.41	190,97	206.00	214.02	203,59
filtrate volume, ml	230.00	241.80	190.00	230.00	190.00	193.50	194.00	210.80	225.00	210.00
DO final	3.80	4.68	9.28	7.74	9.76	9.66	8.78	8.99	8.00	7.98
Free cyanide	127.00	214.00	189.00	127.00	80.00	63.00	7.00	250.00	160.00	44.00

10.3 Log sheet, Mass balance and measured parameters of Wits tailings pyrite concentrate gold improvement tests

Pre Ox										
	Test 1	**Test 2**	**Test 3**	**Test 4**	**Test 5**	**Test 6**	**Test 7**	**Test 8**	**Test 9**	**Test 10**
Solid%	33.0	33.0	33.0	33.0	33.0	33.0	33.0	33.0	33.0	33.0
Reactor Mass, g	541.8	554.3	574.3	561.2	558.7	561.2	492.3	555.6	555.1	549.2
Mass Solids, g	160.0	160.0	160.0	160.0	160.0	160.0	160.0	160.0	160.0	160.0
Mass water, g	324.8	324.8	324.8	324.8	324.8	324.8	324.8	324.8	324.8	324.8
Reactor mass + slurry	1026.6	1039.2	1059.4	1045.1	1048.4	1047.4	979.0	1042.5	1041.9	1033.2
Target pH.	11	11	11	11	11	11	11	11	11	11
DO/AIR control	Air	O_2	Air	Air	Air	Air	Air	Air	Air	OPT
Time, hours	1	1	6	3	3	3	3	3	3	3
DO After sparging	7.27	38.80	10.20	6.50	11.90	11.80	10.70	11.00	11.00	11.00
PH Natural	3.40	3.45	3.64	3.37	3.51	3.46	3.61	3.36	3.30	3.43
PH Initial	11.15	11.11	11.11	11.23	11.63	11.49	11.35	11.20	11.05	11.00
Eh initial	-40.00	-26.00			-19.00	-20.00	-64.00	-79.00	-70.00	-68.00
PH	9.74	10.49	8.47	8.84	9.40	9.15	9.50	9.29	8.36	7.47
EH	46.00	74.00	135.50	11.50	90.00	140.00	41.00	45	70	68
pH. adjusted	11.09	11.39	11.15	11.40	11.05	11.08	11.25	11	11	11
Eh adjusted	-69.00	13.00	-51.30	-88.00	-6.00	-78.00	-133.00	-100	-60	-85
NaCN, kg/t	2.00	2.00	2.00	2.00	2.00	2.00	1.00	1.5	2.00	2
NaCN, g	0.32	0.32	0.32	0.32	0.32	0.32	0.16	0	0.32	0
$Pb(NO_3)_2$, mg	0.00	0.00	0.00	0.00	96.00	48.00	0.00	0	0.00	0
Time, hours	24.00	24.00	24.00	24.00	24.00	24.00	24.00	24	8.00	18
Before container + lime	16.89	18.16	9.33	9.33	9.26	9.20	9.26	9.23	9.33	9.33
After container +	15.39	16.75	7.91	8.07	7.80	7.78	7.77	7.77	7.80	7.88
Mass of lime added, g	1.50	1.41	1.42	1.26	1.46	1.42	1.49	1.46	1.53	1.45
Cyanidation										
	Test 1	**Test 2**	**Test 3**	**Test 4**	**Test 5**	**Test 6**	**Test 7**	**Test 8**	**Test 9**	**Test 10**
Reactor mass + slurry final, g	1010.53	1023.24	1009.96	1045.88	998.25	995.20	978.96	1042.47	1023.65	989.53
PH final	10.76	10.72	10.99	11.13	9.87	9.72	10.50	9.96	10.33	10.28
Eh final	-64.00	-98.00	-144.00	-211.00	-69.00	-80.00	-31.00	-116.00	-173.00	-146.00
Wet cake mass, g	225.73	226.03	219.48	230.37	206.45	214.00	220.00	215.60	200.00	208.00
Container	212.12	221.18	230.53	212.14	214.09	232.90	230.53	212.14	649.05	631.90
Dry cake mass, g	155.95	155.94	151.84	155.57	150.24	152.82	154,68	151.36	158.00	140.94
filtrate	231.20	237.39	185.80	228.68	187.15	194.41	190,97	206.00	214.02	203,59
filtrate volume, ml	230.00	241.80	190.00	230.00	190.00	193.50	194.00	210.80	225.00	210.00
DO final	3.80	4.68	9.28	7.74	9.76	9.66	8.78	8.99	8.00	7.98
Free cyanide	127.00	214.00	189.00	127.00	80.00	63.00	7.00	250.00	160.00	44.00

Sample ID	Mass of ore	Volume of water	IN head		Residue		Solution		Accountability	Au extraction	NaCN consumption	$Ca(OH)_2$ Consumption	DO	pH	ORP
	Feed	Solution	Au	Au	Au	Au	Au	Au	Au	Au	NaCN	CaO	DO	pH	Eh
	g	mL	g/t	mg	g/t	mg	mg/L	mg	%	%	kg/t	kg/t	mg/L		mV(Ag/AgCl)
Baseline (24 hours)			**3.01**	**3.01**	**0.86**	**0.86**	**1.11**	**2.24**	**103**	**68**	**0.58**	**2.87**	**55.9**	**10.4**	**-28**
Test 1 AIR	160	325	3.01	0.48	0.75	0.12	1.12	0.36	**100**	**75**	1.27	9.38	3.80	10.76	-64
Test 2 O_2	160	325	3.01	0.48	0.74	0.12	1.13	0.37	**100**	**76**	1.20	8.81	4.68	10.72	-98
Test 3 6 hr AIR	160	325	3.01	0.48	0.69	0.11	1.23	0.40	**106**	**77**	1.21	8.88	9.28	10.99	-144
Test 4 3hr Air	160	325	3.01	0.48	0.69	0.11	1.29	0.42	**110**	**77**	1.27	7.88	7.74	11.13	-211
Test 5 96 mg Lead	160	325	3.01	0.48	0.70	0.11	1.21	0.39	**105**	**77**	1.14	9.13	9.76	9.87	-69
Test 6 48 Lead	160	325	3.01	0.48	0.75	0.12	1.18	0.38	**105**	**75**	1.17	8.87	9.66	9.72	-80
Test 7 1 kg/t NaCN	160	325	3.01	0.48	1.55	0.25	0.81	0.26	**106**	**49**	0.61	9.31	8.78	10.50	-31
Test 8 1.5 NaCN	160	325	3.01	0.48	0.92	0.15	0.80	0.26	**85**	**69**	0.68	9.13	8.99	9.96	-116
Test 9 8hr 3h Air	160	325	3.01	0.48	0.78	0.12	0.95	0.31	**90**	**74**	1.17	9.56	8.00	10.33	-173
Test 10 18 hr 3h Air	160	325	3.01	0.48	0.75	0.12	1.02	0.33	**94**	**75**	1.40	9.06	7.98	10.28	-146

10.4 Pre-oxidation OPEX and CAPEX calculation

Pre-oxidation tank CAPEX				
100000	100000 tonnes		1 tonne	
Direct costs	$	30,200,000	$	302
Indirect costs	$	9,700,000	$	97
EPCM	$	8,700,000	$	87
Contingency	$	15,300,000	$	153
Capital spares and first fills	$	2,300,000	$	23
Total	$	66,200,000	$	662

	Price per unit	**Unit (kg)**	**Consumption (kg/t)**	**Consumption per day (kg)**	**CaO Cost per kg**	**Cost per day**	**Cost per tonne**
CaO (90%)	$ 235	850	7.88	14513	$ 0.276	$ 4,013	$ 2.18
Compressed air	$ 0.30	1290	7.74	14256	$ 0.000	$ 3	$ 0.002
$Pb(NO_3)_2$	$ 2,450	1000	0.30	553	$ 2.450	$ 1,354	$ 0.735
Total							$ 2.92

10.5 Financial analysis formulas

		CIP Plant (With UFG)	**CIP Laboratory (Without UFG)**
Plant slurry throughput	tonnes/day	**=(G9+G6)*G5**	**=(H9+H6)*H5**
Leach plant solids throughput	tonnes/day	=G3*0.43	=H3*0.33
Leach operating schedule	hrs/day	21.7	24
Leach plant solids throughput	t/hr	110	=J2*0.33
Head Grade	g/tonne	3.01	3.01
Leach slurry solids	%	43	33
Leach solution flowrate	t/hr	146.6	=J2*0.66
Leach solution density	t/m^3	1	1.27
Leach solution flowrate	m^3/hr	=G9/G10	=H9/H10
Leach Efficiency	%	50	77
Leach recovery	g/hr	=(G7*G6)*(G12/100)	=(H7*H6)*(H12/100)
Leach solution grade	g/m^3	2.29	1.29
NaCN consumption	kg/t	0.58	1.14
CaO consumption	kg/t	2.87	7.88
Gold production	kg/month	=G13*G5*30/1000	=H13*H5*30/1000

Pre-oxidation tank OPEX		
Throughput	t/day	=H6*H5
Reagents	$/t	=O60
Power	$/t	23
Maintenance labour	$/t	12
Total OPEX per day		=SUM(G32:G34)*G31

		Pre-oxidation tank CAPEX
100000	100000 tonnes	1 tonne
Direct costs	30200000	=H39/$G$38
Indirect costs	9700000	=H40/$G$38
EPCM	8700000	=H41/$G$38
Contingency	15300000	=H42/$G$38
Capital spares and first fills	2300000	=H43/$G$38
Total	=SUM(H39:H43)	=SUM(I39:I43)

		UFG Mill OPEX
Throughput	t/day	=G6*G5
EPC	$/t	1.51
Maintenance labour	$/t	1.4
Pumps	$/t	1.13
Parts	$/t	1.47
Discs/liners	$/t	3.6
Media	$/t	5.37
Power	$/t	9.88
Concentrate handling	$/t	1.28
UFG Mills	$/t	24.36
Services	$/t	1.93
Total OPEX per day		=SUM(D32:D41)*D31

	UFG Mill CAPEX
CAPEX	=6*1000000
Throughput (tph)	20
CAPEX per tonne	=B60/20