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The Study of Alternative Leaching Processes on South African Gold Ores

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A Thesis submitted to the Faculty of Science, University of the Witwatersrand, Johannesburg, in fulfilment of the requirements for the degree of Doctor of Philosophy

15 August 2020, Johannesburg

Plagiarism Declaration

University of the Witwatersrand, Johannesburg

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Abstract

The increasing environmental concern surrounding the use of cyanide and the decreasing applicability of cyanidation to new deposits has led to extensive research on alternative gold extraction techniques. The three alternative techniques that were selected for testing (relative to cyanide) were chloride-hypochlorite, thiourea and thiosulphate. These techniques were evaluated using a number of ores including free milling Witwatersrand and Pilgrim's Rest samples as well as refractory Fairview, Barbrook and Consolidated Murchison samples. From several leach tests it was evident that chloride-hypochlorite and thiourea leaches are highly reactive, reagent consuming and have limited capacity for gold dissolution in the context of the samples of study. This study established that in addition to being reactive to sulphides (from the highly reactive phases such as pyrrhotite, to the more stable phases such as pyrite and chalcopyrite) they also effect phyllosilicates, a typically stable component in gold leaching. This degree of reactivity is significantly greater than suggested in literature and as such these lixiviants are even more impractical than previously thought. The thiosulphate leaching was significantly less reactive with gangue mineralogy and showed no interaction with even the most reactive sulphides, this is in stark contrast to the literature which consistently details the deleterious potential of sulphide phases. Furthermore, this confirms that any shortfalls with regards to limited leaching results is inherently related to the leaching system and related conditions and is not affected by the gangue mineralogy. Thus, thiosulphate leaching has far greater applicability to sulphide-bearing ores than previously thought. Despite the lack of reactivity with gangue minerals, the overall dissolution achieved using thiosulphate was limited without the addition of a stabilising agent to prevent thiosulphate degradation. The use of a stabilising agent showed major improvements in total dissolution, with a dissolution limit for all samples of approximately 70% of the total available gold. The addition of the stabilising agent glycine was found to have no influence on the non-reactive nature of thiosulphate leaching. This limited gold dissolution is likely to be due to reduced kinetics associated with insufficient reagent concentrations throughout leaching. Other relevant factors such as reagent cost and detoxification considerations still favour the well constrained and optimised method that is cyanidation. Significant work in optimisation of leach conditions and detoxification would be important before major changes in the gold industry could be expected.

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Glossary of Terms

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
BIF	Banded iron formation, an iron rich chemical sedimentary rock type
BIOX	Bacteria oxidation, the oxidation of sulphides as a pre-treatment for refractory ores causing the exposure of gold previously locked within sulphides
BK	An abbreviated term used in this work to refer to Barbrook
CIL	Carbon in Leach, a technique used for gold recovery where carbon is added to the leach tanks, leaching and subsequent adsorption take place in the same vessel
CIP	Carbon in Pulp, a technique used for recovery of gold where the gold is exposed to carbon after cyanide leaching is complete
CM	An abbreviated term used in this work to refer to Consolidated Murchison
Conc	Concentrate, the material of interest as produced by a concentrating process
Cu	The chemical symbol for copper
Eh	The redox potential of a solution this reflects the activity of electrons
EPMA	Electron probe micro-analyser, used for in situ chemical analysis of small volumes of solid samples
Exposure	The percentage of the grain that has a free, exposed surface. This quantifies the amount of gold exposed to lixiviants that would be available for leaching
Feed	The original unaltered material used to supply a process or technique
FV	An abbreviated term used in this work to refer to Fairview
Glycine	A simple amino acid with the formula $C_2H_5NO_2$
Head	The original unaltered material used to supply to a process or technique
HLS	Heavy liquid separation, a commonly used analytical procedure that allows for separation of materials based on their density.
ICP AES	Inductively Coupled Plasma- Atomic Emission Spectrometer, an atomic emission spectroscopy technique that uses an inductively coupled plasma to ionize the sample, this is used to detect a large range of elements
ICP MS	Inductively Coupled Plasma- Mass Spectrometry, a mass spectrometry technique that uses an inductively coupled plasma to ionize the sample, this is used to detect certain elements in low concentration
Liberation	The area percent of the mineral of interest, relative to the area of the total host particle. Liberation is relevant to physical separation processes such as gravity concentration
Lixiviant	A component used in hydrometallurgy to facilitate the extraction of the commodity of interest from its host mineral or ore
MLA	Mineral Liberation Analyser, an automated technique used to determine composition and texture of minerals within a solid sample
Ni	The chemical symbol for nickel
Ore	The material from which a valuable commodity or component could be economically extracted
Pb	The chemical symbol for lead
PG	An abbreviated term used in this work to refer to Pilgrim's Rest
pH	The scale used to specify how acidic a solution is, this reflects the activity of H^+

Preg-robbing	When the gold-cyanide complex is removed from solution by absorption on to some component within the ore
Refractory	The resistance to a process, in the context of gold processing refers to an ore that is resistant to recovery by common hydrometallurgical processes
Residue	The material remaining after a process, commonly leaching, has been applied to the feed
Sb	The chemical symbol for antimony
Schistosity	The planar arrangement of elongate minerals (such as micas) to form a foliation
Thiosulphate	An oxyanion of sulphur with the formula $S_2O^{2-}_3$
Thiourea	An organosulfur compound with the formula $SC(NH_2)_2$.
VCR	An abbreviated term used in this work to refer to the Ventersdorp Contact Reef
XRD	X-Ray Powder Diffraction, an analytical technique used for mineral phase identification and semi-quantification
XRF	X-Ray Fluorescence, an analytical technique used to determine high concentration sample chemistry using secondary X-rays emitted due to excitation.
Zn	The chemical symbol for zinc

Preface

This thesis was written in conjunction with 3 papers which are found in Appendix E, these works are in various stages of submission and review;

1) Formational Conditions as a Predictor of Gold Recovery, in review for the SAIMM Journal. This work is based on content found in Chapter 7 (Formational Observations) as well as Chapters 4-6 (Ore Characterisation).
2) A Comparison of Thiourea and Chloride-Hypochlorite Leaching and Consumption on South African Ores. This work is based on content found in Chapter 9 (Leaching Results), Chapter 10 (Residue Mineralogy) and Chapter 11 (Consumption).
3) Thiosulphate Leaching of South African Ores submitted to the Journal of Hydrometallurgy. This work is based on content from Chapter 9 (Leaching Results), Chapter 10 (Residue Mineralogy).

These papers were produced with the guidance and input of co-authors Prof. J. Kinnaird and Dr B. Youlton.

In addition to the papers above, this research has also produced two peer-review conference papers that have been presented and can also be found in Appendix E. Please note that conference paper one and paper one are identical as the paper was submitted to the SAIMM Journal after the successful attendance and presentation at the SAIMM Geometallurgy conference. The second conference paper was presented at the AusIMM World Gold Conference.

Chapter 1

Literature Review

1.1. Introduction

Gold has been an important commodity throughout much of human history due to both its highly appealing nature and its tendency to occur in its native form (Kongolo and Mwema, 1998; Reardon, 2011). Although the physical properties of gold include low tensile strength and hardness, which render it useless for both tools and weaponry, it was extensively recovered for decorative purposes and it is thought to have been one of the earliest metals mined in human history (Kirkemo et al., 2001). The earliest evidence of gold appreciation has been found in the form of gold flakes in Palaeolithic caves that were in use 40 000 BC. Additionally, evidence exists for gold mastery as early as 6 000 BC (Kongolo and Mwema, 1998) and it continues to be highly sought after to the present day. Examples of the influential role that gold plays in human history is illustrated in gold use by ancient civilisations, gold monetary standards, global gold rushes throughout much of modern history or even the increasing gold price in uncertain economic times (Beckmann et al., 2019). Gold also has some technological importance, as it is used for electronics, in a variety of devices including phones, cars and medical devices (Shishmanian, 2018).

Early gold mining involved the use of manual or gravity-based separation methods from gravels and sands in streambeds. Beyond these methods, little advancement took place until underground mining was initiated, in Russia, in the 1740s, where gravity concentration and additional recovery through amalgamation were used for gold extraction (Kongolo and Mwema, 1998). Subsequent modifications were made to existing amalgam extraction processes to incorporate cyanide. However, cyanide was used exclusively to clean the mercury surfaces. The processes used here were still ineffective for the treatment of low grade and fine-grained gold. The discovery of chlorine gas prompted its use as a treatment for gold ores, as determined by Plattner (Eidem and Lundevall, 1974). The utilisation of chlorine gas to extract gold was uneconomic for anything except high-grade deposits, which meant that the ores that could be feasibly exploited were limited by extraction technology for over 100 years. Thus, the patent filed in 1887 describing gold dissolution using cyanide by MacArthur and the Forrest brothers, hailed a new era in gold mining. Cyanidation was rapidly commercialised, first in New

Zealand (in 1888) and subsequently in a number of countries including South Africa, USA, Mexico and France (Reardon, 2011). The advent of cyanide globally is credited with reviving the failing Witwatersrand gold mining industry in 1890 when oxidised ore was no longer available (Fivaz,1988). Today, cyanidation is used to extract more than 90% of gold produced and is therefore a cornerstone of the gold industry, particularly with the increasing need for exploitation of low-grade/disseminated gold deposits (Aylmore, 2016a).

Despite the importance of cyanide to the gold industry, increasing environmental concerns have been voiced in past decades, owing to a number of well-publicised incidents involving cyanide in the mining industry (Donato et al., 2007; Laitos, 2012; Aylmore, 2016a). The increasing number of deposits that are not amenable to cyanidation, coupled with these restrictions has acted as a global initiative to establish economic and environmentally friendly alternatives (Aylmore, 2005; Aylmore, 2016a).

1.2. Environmental Concerns and Legislation

Cyanide refers to any chemical compound that contains a cyano – a carbon triple bond nitrogen (–C≡N) group. Common compounds containing this group include cyanide salts, such as sodium cyanide, potassium cyanide and hydrogen cyanide as well as organic cyanides such as acetonitrile. Cyanides is found in nature in a number of different plant sources, however it is produced on an industrial scale for extensive use in manufacturing as well as in gold extraction (Jaszczak et al., 2017). Exposure to the more toxic forms of cyanide (such as hydrogen, sodium and potassium cyanide) can occur through inhalation, ingestion or even absorption through eyes and to a lesser degree skin. Cyanide is acutely toxic in varying degrees in any of the aforementioned forms; however, hydrogen cyanide as a gas, is more readily available for inhalation, which is the most toxic form of exposure (Swanepoel, 2001). The LC50 (which refers to the lethal concentration of 50% of the tested population for inhalation) for HCN is 100-300 ppm, with death occurring within an hour- the precise time scale is highly dependent on the relative concentration. The oral LD50 (which refers to the lethal dosage for ingestion) is 1.52 mg/kg, while the dermal LD50 (the lethal dose due to skin contact) is significantly higher at 100 mg/kg (Gallagher and Manoil, 2001; Gracia and Shepherd, 2004). The effect of cyanide in the body, regardless of route of exposure, is the same. Cyanide, upon entering the bloodstream forms complexes with the iron atom in cytochrome oxidase. Cytochrome oxidase is an enzyme

found in bacteria, archaea and eukaryotes that is fundamental in providing oxygen with electrons that are needed to allow the mitochondria to produce what is known as the "molecular unit of energy"- ATP (adenosine triphosphate). As cyanide acts as an enzyme inhibitor to cytochrome oxidase, important tissues cannot receive energy in an appropriate form, causing cellular asphyxiation and death (Gallagher and Manoil, 2001; Gracia and Shepherd, 2004). Tissues that are most sensitive to this process include heart muscle and nerve cells, which leads to respiratory arrest and death. The mechanism described for cyanide toxicity applies to both humans and other animal species including mammals, reptiles, birds and fish etc.

Some symptoms from non-lethal dosages or initial symptoms of lethal dosages include headaches, dizziness, weakness, fatigue, convulsions, slowed heart rate and respiratory failure (Gallagher and Manoil, 2001; Gracia and Shepherd, 2004). The symptoms in a non-lethal dosage subside as the cyanide is detoxified within the body, predominately by way of thiocyanate production, which is then safely eliminated by the body. While thiocyanate is significantly less toxic, extended exposure can have long-term effects particularly with the thyroid (Vogel et al., 1981; Yen et al., 1995). Cyanides do not bio-accumulate and are not carcinogenic. Cyanides tend to remain in solution (as they are not retained in soils) where they rapidly complex with metals such as copper, iron and nickel. These complexes, unlike potassium and sodium cyanide do not rapidly degrade into more toxic forms and are therefore generally safe at a range of stable conditions. In addition, other processes such as volatilisation, microbial action and sedimentation also neutralise the toxic component of cyanides. Thus, despite the extreme toxicity of cyanide, its non-persistent behaviour in the environment makes this industrial chemical significantly less dangerous than generally perceived (Donato et al., 2007; Dash et al., 2009). The degree of fear surrounding cyanide could be considered unjustified, however a number of catastrophic, and well-publicised incidents have left the public reminded only of cyanide in its most highly toxic form. Fourteen major cyanide related incidents have occurred in the past 25 years, which were caused by tailings dams and pipeline failures as well as accidents during transportation (Kulig and Ballantyne, 1991; Hilson and Monhemius, 2006b).

The most memorable of these incidents is the Baia Mare Cyanide Disaster in Romania, where Esmeralda Exploration in conjunction with the Romanian Government – known as Aurul, were responsible for a major tailings spill. Aurul dealt with mining and processing waste issues by

shipping products to a nearby dam. However, on 30th January 2000, the dam ruptured releasing over 120 000 m^3 of toxic waste-bearing water. High cyanide concentrations in addition to heavy metals-bearing water moved rapidly into several important rivers such as the Somes, Tisza and Danube. This caused major ecological losses, not only in Romania but also Serbia and Hungry. Major loss of human life was avoided due to the rapid response of the Romanian Government (Korte et al., 2000; Soldán et al., 2001). As a result of Baia Mare, as well as other similar incidents, and their ensuing public concern, a number of countries have introduced varying degrees of legislation related to cyanide usage in the mining industry (Laitos, 2012).

In addition to catastrophic environmental incidents there are also concerns surrounding the effects of continuous use of large masses of cyanide in tailings dams, which even when safely enclosed can still pose a threat to local wildlife that utilise cyanide tailings as a water source. The advent of heap leaching in the 1970s, as an economically feasible method of extracting gold from low-grade deposits, through spraying or dripping cyanide solution over large bodies of ore increase the environment effects associated with cyanidation (Eisler and Wiemeyer, 2004; Hilson and Monhemius, 2006b).

Countries such as Germany, Hungary, the Czech Republic, Slovakia, and Costa Rica, as well as a number of provinces of Argentina and states in the USA have instituted blanket bans on the use of cyanide in gold extraction (Rodriguez and Macias, 2009). Other laws, such as those instituted by the European Union severely restrict the usage of cyanide (Szilvia, 2011), while even more countries, including South Africa, have systematically lowered acceptable levels of cyanide in discharge. These increasingly stringent laws make gold processing challenging and costly.

Alternatives to cyanide have been extensively investigated, and while there is no single, equivalent competitor to cyanide, there are a number of potential alternatives, that have significant advantages (Aylmore, 2005). This thesis evaluates some of these alternatives as solutions to cyanide using a variety of gold settings, spanning a spectrum from free milling to doubly refractory.

1.3. Extractive Metallurgy

Gold, a noble metal, by definition is resistant to chemical action; therefore, there are only a limited number of solvents capable of dissolving it (Moffa, 1977). Systems where gold dissolution is most efficient typically require an oxidant, capable of oxidising the gold, as well

as a reactant/complexing agent able to form a reasonably stable gold complex (Kongolo and Mwema, 1998). Gold, under soluble conditions can oxidise to two states; Au^{+1} and Au^{+3}, with Au^{3+} being significantly more stable. Therefore, a strong oxidant is commonly required, particularly with the more unstable leaching systems (Nam et al., 2008).

1.3.1. Cyanide

Chlorination and amalgamation were industry standards prior to the discovery of cyanide-based gold extraction over 100 years ago. Although the overall mechanism of cyanide and gold complexation was understood as early as 1844 when it was described by Elsner (Equation 1.1), it took a number of years before the efficiency of this process was determined and later patented by J. MacArthur in conjunction with the Forrest brothers, Robert and William in 1887. Gold recovery grades of greater than 90% were achieved using sodium (or potassium) cyanide in solution with ore. Despite minor refinements of the cyanide gold dissolution first described, the process has not been significantly altered or improved in this time. Advancements in gold extraction have come from improved equipment, pre-treatment processes such as flotation, recovery methods such as carbon in pulp (CIP) and new methods of implanting cyanidation such as heap leaching. In subsequent years the mechanism of this process has been better understood by Bodländer (1896) and later by Kameda (1949), Lund (1951), Deitz and Halpern (1953). This work has shown that the overall reaction mechanism (Equation 1.1) operates with intermediate steps (Equation 1.2 and Equation 1.3) and that the first step (Equation 1.2) is the most important, with only minor contributions from the second step (Equation 1.3) (Kongolo and Mwema, 1998).

$$4Au + 8NaCN + O_2 + 2H_2O \rightarrow 4AuNa(CN)_2 + 4NaOH \qquad \text{(Equation 1.1)}$$

$$2Au + 4CN^- + O_2 + 2H_2O \rightarrow 2Au(CN)_2^- + 2OH^- + H_2O_2 \qquad \text{(Equation 1.2)}$$

$$2Au + 4CN^- + H_2O_2 \rightarrow 2Au(CN)_2^- + 2OH^- \qquad \text{(Equation 1.3)}$$

Oxygen acts as an oxidising agent, while CN^- dissociates from sodium, and complexes with the gold (Habashi, 1967). Cyanide-gold complexes are significantly more stable than those yielded by alternative lixiviants, making it a useful tool in gold extraction (Chen et al., 1980; Aylmore, 2005; Aylmore, 2016a).

This reaction is maintained at a basic pH of approximately 10-11; as this lowers reduction potential and is the most efficient leaching condition for many ores (Kongolo and Mwema, 1998). Furthermore maintaining the pH between 10-11 is important for safety as HCN becomes the stable phase at more acidic conditions and the volatile nature of HCN makes this a safety concern as well as an economic one due to increased reagent losses (Eisler and Wiemeyer, 2004).

1.3.1.1. Ore Classifications

The cyanidation reaction typically occurs quite readily, which accounts for its widespread use, however the leaching ability is limited in the presence of interfering agents. The presence of these agents allows for the classification of gold ores, while the classification system may vary from author to author (Chryssoulis et al., 1987; Haque, 1987; Monhemius, 1987; Komnitsas and Pooley, 1989; Zhou and Cabri, 2004; Chryssoulis and McMullen, 2005; Li and Miller, 2006; Karimi et al., 2010) the overall concept is the same. The description given here has been strongly based on Vaughan, (2004). Three distinct gold classification types exist in this system (Figure 1-1); free milling, complex and refractory (in increasing order of processing difficulty).

Free milling ores encompass all ores that readily undergo gold extraction by cyanide, within these ores there are no factors which complicate or reduce extraction efficiency. A gold recovery of 90% or more defines free milling ores (La Brooy et al., 1994; Marsden and House, 2006).

Complex ores are the next most difficult to process, this term refers to ores which can undergo gold extraction with a high dissolution rate, but only under more intense leaching conditions but without the need for additional processing steps. Complex ores can be subdivided into two categories; preg-robbing and reactive. Preg-robbing ores contain material capable of adsorbing gold-cyanide complexes on to its surface during leaching. This material tends to be carbonaceous matter (specifically elemental and organic carbon), although clays are also thought to have a minor capacity for preg-robbing (Vaughan, 2004; Goodall et al., 2005; Miller et al., 2005). The effects of preg-robbing can be countered during leaching (to a limited degree) by 1) addition of kerosene to deactivate the carbonaceous material, 2) Carbon-In-Leach, where activated carbon is present throughout the leach, and can adsorb the gold onto its surface more rapidly than the competing preg-robbing component within the ore (La Brooy et al., 1994;

Vaughan, 2004). Reactive ore refers to any ore that contains material that reacts with reagents required for successful gold leaching. These reactive materials tend to be cyanicides (i.e. they react with and reduce the available concentration of cyanide), or oxygen consumers. Cyanicides are most apparent when considering polymetallic or sulphide-bearing ores. Sulphide-bearing ores have the potential to react with oxygen forming unsaturated sulphides capable of consuming cyanide to allow for the formation of thiocyanate. A number of base metals interact with cyanide to varying degrees, particularly deleterious elements to consider include copper, arsenic, antimony, zinc and mercury. While iron is commonly found in gold ores, the nature of cyanide-iron reactivity is not generally a major cause for concern in processing, as the most abundant iron mineral, pyrite, is unreactive in gold cyanide solutions. However, minerals such as pyrrhotite and marcasite, which are generally less abundant, are far more reactive. Additionally, iron oxides are capable of passivating the surface of gold. Arsenopyrite is also rather unreactive, while other arsenic-bearing as well as antimony-bearing minerals such as realgar and stibnite are major cyanide consumers. Mercury- and zinc-bearing ores are generally only moderately reactive, this in conjunction with the scarcity of these minerals in gold deposits makes these elements of limited concern. Copper tends to be significantly more problematic during gold leaching and generally the most concerning cyanide consumer found to occur in gold ores. Fortunately, chalcopyrite and enargite are not commonly reactive during cyanidation, however all other common copper minerals such as azurite, cuprite and malachite are highly reactive. In addition, the copper cyanide complexes are reported as free cyanide in the standard silver nitrate titration used to test for the concentration of cyanide, in order to maintain leach conditions. This could result in incomplete leaches due to falsely reported free cyanide concentrations. The failure of the silver nitrate titration to accurately account for the state of cyanide in the leach can occur with copper as well as several other base metal cyanide complexes. Other potential metals of concern include nickel and cobalt (Aylmore and Muir, 2001; Karimi et al., 2010). These ores can be dealt with by adding higher concentrations of reagents than would be required for standard cyanidation, although this may become economically unfeasible.

Refractory ores refer to deposits that cannot achieve acceptable gold dissolution using standard procedures. There are two types of refractory ores that contain either physically or chemically locked gold, both of which make the gold inaccessible to lixiviants. Physically locked gold occurs

when gold is present as fine inclusions within a mineral that is insoluble in the leach solution. Chemically locked gold is found in discrete gold minerals such as gold tellurides, aurostibite ($AuSb_2$), auricupride (Cu_3Au) and maldonite (Au_2Bi) (Harris, 1990; Vaughan, 2004). Chemically locked gold can also refer to gold present as solid solution within certain minerals, which means that atomic gold is present within the crystal lattices of its host mineral (Chryssoulis et al., 1987; Haque, 1987; Monhemius, 1987; Komnitsas and Pooley, 1989; Vaughan and Kyin, 2004). Arsenopyrite and to a lesser extent pyrite can act as host minerals to this solid solution gold (Vaughan, 2004; Vaughan and Kyin, 2004; Barker et al., 2009). In order to ensure efficient leaching for refractory ores, it is usually necessary to implement a pre-treatment step in the extraction process, examples of these pre-treatments include ultra-fine milling, pressure oxidation, biological oxidation (BIOX®) and in older designs, roasting. Pressure cyanidation, while not a pre-treatment, may also be an effective option (Davies and Paterson, 1986; Haque, 1987; Monhemius, 1987; Lindström et al., 1992).

A large percentage of free milling gold deposits have been extensively exploited, and therefore many of the remaining deposits available are increasingly complex or refractory. Furthermore, newly discovered gold deposits, in general, tend to be lower grade and more difficult to process (Davies and Paterson, 1986; Haque, 1987; Monhemius, 1987; Lindström et al., 1992).

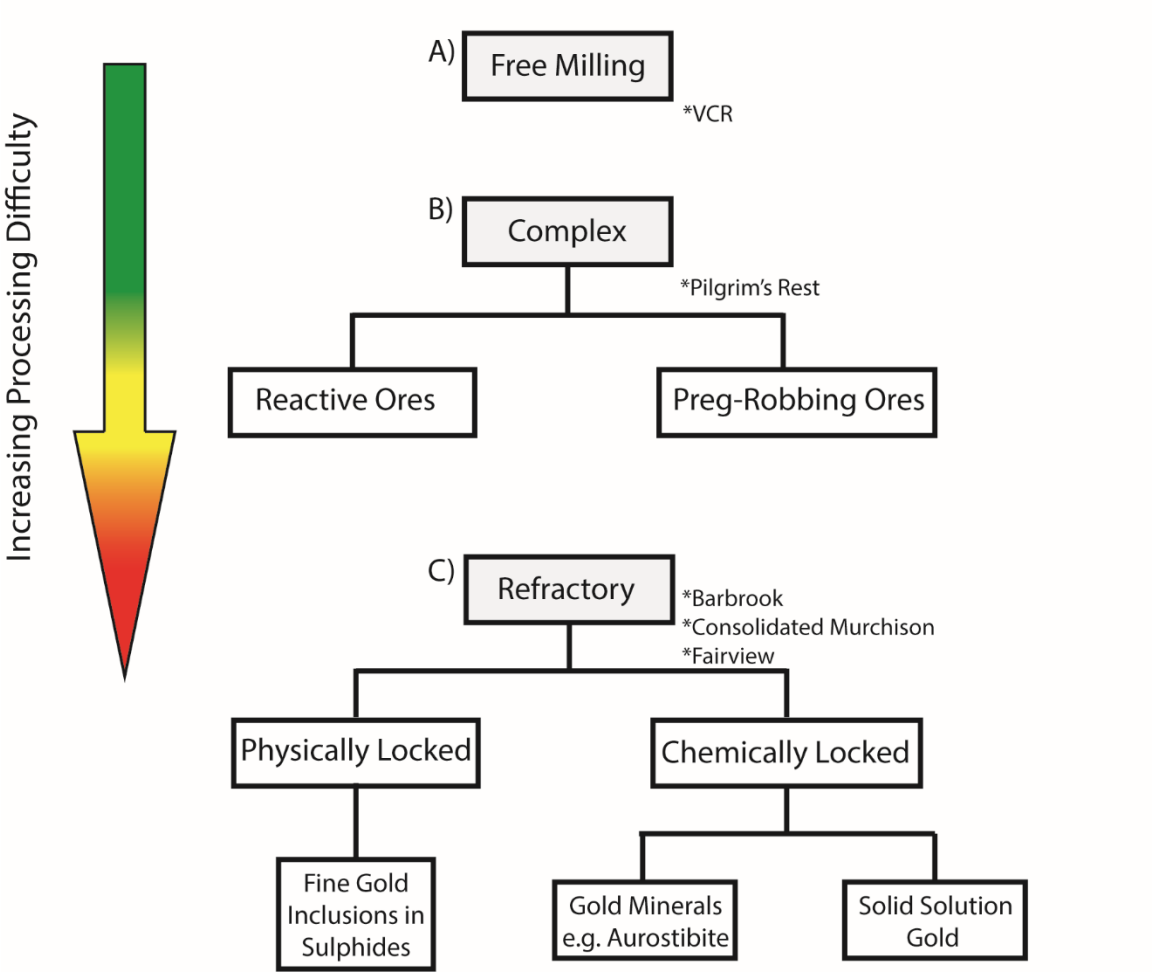


Figure 1-1 Three main types of gold deposit types exist, based on their response to recovery; 1) Free Milling, 2) Complex and 3) Refractory. While the deposits of study and their respective mineralogy will be outlined in subsequent chapters, the deposits have been grouped with their respective deposit types. Ventersdorp Contact Reef is free milling, Pilgrim's Rest is complex, Barbrook is refractory but also contains a complex component, Fairview and Consolidated Murchison are both refractory.

1.3.2. Alternative Lixiviants

Due to both increasing restrictions and scrutiny on cyanide usage around the world, along with the increasing number of refractory/complex gold ores, there has been extensive study of alternative lixiviants in recent years. While as many as 27 possible systems have been investigated, of these there are a limited number that show the potential to be financially and chemically feasible. However, systems of interest include (but are not limited to); thiosulphate, thiourea and chloride-hypochlorite. These are the primary lixiviants, along with cyanide, that will be evaluated in this research. It should be noted that the commercial application of some of these systems is still in the developmental stages, while the effective use of the others tends

to be proprietary information. All of these lixiviants form gold complexes that are stable over a significantly smaller pH-Eh space than the gold cyanide complex (Figure 1-2). This instability complicates the chemistry further and requires more significant controls on the leaching conditions, as well as higher reagent concentration than used in cyanidation. In addition, oxygen is generally an insufficient oxidiser for alternative leaching systems, due to slow kinetics and low mass transports. Instead of oxygen for leaching alternative reagents are required for these extractions. The need for increased reagent concentrations (relative to cyanide) necessitates greater emphasis on reagent recycling, to improve economics as well as reduce the environmental impact (Chen et al., 1980; Jeffrey, 2001; Baghalha, 2007; Aylmore, 2016b).

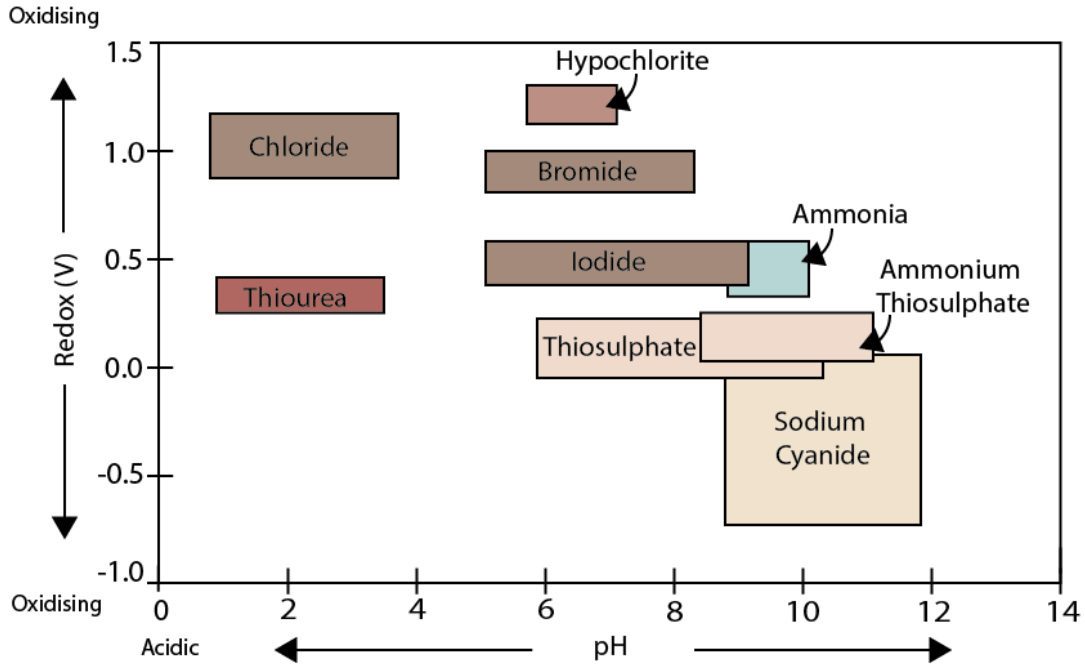


Figure 1-2 Stability fields (Redox and pH) of several gold lixiviants (After Aylmore, 2016a). Note that the sodium cyanide operating conditions are significantly larger than for any alternative lixiviant. Alternative lixiviants of interest to this study is the hypochlorite, thiourea and thiosulphate.

1.3.3. Chloride-Hypochlorite

The ability of chloride to extract gold has been known for over a century, and was utilised considerably in the gold industry prior to the discovery of cyanidation (Kongolo and Mwema, 1998). Chlorine has also been important in the pre-treatment processes in refractory or carbonaceous ores (Marsden and House, 2006). However, cyanide was a superior lixiviant in

terms of extraction efficiency as well as safety considerations and cost. Safety considerations with regards to chlorine-based extraction are strongly dependant on which of the chlorine extraction processes were selected. For example, the utilisation of chlorine gas is a major safety concern but was the extraction of choice for the Deetken or Meers processes. However, significantly safer options exist, and the use of chlorine in extractions is not prohibitive. Chlorine is used in PGE extraction which highlights the capacity for this reagent to be safely used in processing (Ferron, 2003). The chlorine-based extraction described in this work refers to chloride-hypochlorite, where chlorine is generated in-situ and occurs in the aqueous form, reducing safety concerns. For gold dissolution to take place in a chloride media, an oxidant is required. The oxidant used varies but at ambient temperature and pressure conditions, hypochlorite is thought to be the most effective reagent (Baghalha, 2007; Hasab et al., 2013; Hasab et al., 2014; Adams, 2016). The dissolution of gold in a chloride-hypochlorite leaching system operates by the following reaction mechanism (Equation 1.4) (Hasab et al., 2014);

$$2Au + 3HOCl + 3H^+ + 5Cl^- \rightarrow 2[AuCl_4]- + 3H_2O \qquad \text{(Equation 1.4)}$$

As with all leaching systems, chloride-hypochlorite has its own advantages and disadvantages; it has significantly faster kinetics than cyanidation, however the gold-chloride complex is unstable (Nam et al., 2008; Adams, 2016). Significant reagent consumption, in conjunction with high concentration requirements is a prohibitive factor in chloride usage. However in recent years much research has gone into processes that allow for the reagents to be recycled and significant improvements have been made (Aylmore, 2016a). Halide leaching is thought to be one of the less complex alternative leaching systems which, in addition to very rapid leaching rates relative to cyanide (given the correct conditions), is thought to be a major advantage (Jeffrey, 2001). The stability of $AuCl_4$ is strongly dependant on the solution pH as well as chloride and chlorine concentrations. The complex will also re-precipitate on contact with a reducing agent, and therefore the usefulness of this system will be limited with regards to sulphide-bearing ores (Adams, 2016). Silver-rich ores ore not amenable to chloride leaching due to the formation a passivating layer of insoluble silver chloride of the surface of silver grains (Sparrow and Woodcock, 1995).

Significant studies of this leaching system have been done by Yen et al., (1990) and Tran, (1992) to understand dissolution of gold strips, and leaching conditions have been optimised to this application. The stability of gold complexes can reach levels similar to cyanidation if the pH is

maintained below 3 and this is aided by correct redox conditions, high reagent concentration and elevated temperatures. While limited testing has been done on ores (Baghalha, 2007; Nam et al., 2008) more research is required to achieve optimal results (Hilson and Monhemius, 2006b). In addition, ores with significant reductants present, such as base metal sulphides, are not particularly amenable to this process as they reduce the stability of the gold complexes. Decreased gold complex stability may cause precipitation of gold. Oxidised ores are, therefore, better for this extractive technique, while pre-treated ores could also be amenable to this process (Tran, 1992; Sparrow and Woodcock, 1995). While chloride-complexed gold can be absorbed by activated carbon the mechanism of this absorption requires the gold chloride complex to be converted to metallic gold, this altered mechanism as well as reduced absorption of gold on to the surface of carbon in the presence of aqueous chlorides means that the recovery of chloride leached gold using activated carbon is not currently a feasible process and requires further study (Hilson and Monhemius, 2006b).

1.3.4. Thiourea

Thiourea leaching is one of the more recently discovered gold extraction methods. It was proposed by Plaskin and Kozhukhova in 1941, with subsequent work by Plaskin, (1960), Lodeishchikov et al. (1968), and later by Groenewald, (1976), Chen et al. (1980) and Deschenes and Ghali, (1988) that was aimed at gaining a better understanding of the leaching process and to improve gold dissolution. Factors such as reagent concentration, preferred oxidants, redox and pH conditions were studied to produce a feasible system capable of extracting gold from ore or concentrates (Lodeishchikov et al., 1968; Deschenes and Ghali, 1988; Murthy et al., 2003). The major delay in extensive research between the early work by Plaskin and that undertaken in the 1970s and 1980s is likely to have been due to much of the original work being published in Russian during a time of major political conflict. However, thiourea has been actively researched for a number of years. Unfortunately work done by Newmont at Carlin at a pilot plant level proved to have limited results, with insufficient recovery (Wan et al., 1995; Aylmore, 2016b). Important improvements that have been made include the discovery that certain acids hinder leaching, while sulphuric acid (as well as hydrochloric acid) is generally effective.

Interest has waned somewhat in recent years, as thiourea has been classified as a potential carcinogen in certain animal species (Aylmore, 2005), but in spite of this, research continues.

Thiourea dissolves in an acidic aqueous solution forming a species capable of complexing with gold. The gold-thiourea cation that is formed in leaching is seen in Equation 1.5 (Li and Miller, 2006; Aylmore, 2016b).

$$Au + 2NH_2CSNH_2 + Fe^{3+} \rightarrow [Au(NH_2CSNH_2)_2]^+ + Fe^{2+} \quad \text{(Equation 1.5)}$$

The limited stability of gold in the thiourea leach system is due to the fact that the stable form of gold - Au(III), oxidises thiourea, generating the less stable form, Au(I) (Groenewald, 1976). However, with an effective oxidant the extent of this process can be minimised. A number of oxidants, such as dissolved oxygen, hydrogen peroxide and formadine disulphide have been tested with limited results, however ferric iron has been shown to be the most efficacious (Schulze, 1984; Ubaldini et al., 1998; Hilson and Monhemius, 2006;). The concentration of the oxidising agents should be sufficient to oxidise approximately 50% of the thiourea to allow for accelerated leaching conditions (Zhang et al., 2001). This leaching system is very sensitive to both Eh and pH conditions, as thiourea is inherently unstable and readily decomposes to substances incapable of leaching gold (Groenewald, 1976; Li and Miller, 2006). Despite these disadvantages, leaching rates of thiourea have been found to be up to 10 times faster than cyanidation (Hilson and Monhemius, 2006; Li and Miller, 2006), and good leaching conditions can be maintained if the oxidising potential is between 0.42 and 0.45 V SHE, while higher oxidising potential leaching conditions can be seriously deleterious.

Rapid leaching rates can have major advantages for capital costs associated with plant construction, as leaching tanks could have significantly lower capacity requirements, as required capacity is proportional to leach time (Abbruzzese et al., 1995; Crundwell, 2005).

Ores associated with pyrite or chalcopyrite tend to have accelerated leaching, however the presence of these base metals (copper and iron, as well as lead and zinc) tend to complex with thiourea and therefore increase reagent consumption (Deschenes and Ghali, 1988; Van Deventer et al., 1990).The breakdown of sulphides in the ore is likely to produce sulphur or hydrogen sulphides, this is disadvantageous as it is thought to have a passivating effect on the gold, as well as reducing the stability of the gold complex (Sparrow and Woodcock, 1995). Pre-treatments such a bio-oxidation and acid washing, as well as leaching additives have been evaluated with varying degrees of success (Sparrow and Woodcock, 1995; Deng et al., 2001; Deng and Liao, 2002).

Gold-thiourea complexes are susceptible to the same preg-robbing components that affect cyanide. This susceptibility has the potential to be used for recovery, however, the adsorption of gold onto the carbon surface is hindered due to the presence of excessive sulphur in the system introduced by the breakdown of thiourea (Aylmore, 2016a). Alternative recovery methods have been considered, including electro-oxidation to recover gold ions, however further research is required in this area (Poisot-Díaz et al., 2008).

Thiourea leaching takes place in acidic conditions (Figure 1-2), and as such may be useful for refractory ores that have undergone oxidative pre-treatment, and can subsequently be leached without the neutralisation step that is required to attain the targeted pH of cyanide leaching (Ubaldini et al., 1998; Aylmore, 2016a).

1.3.5. Thiosulphate

Thiosulphate as a gold processing method has been considered as early as the 1900s, where it was used in conjunction with a chlorine-based roast pre-treatment, followed by the thiosulphate leaching. However only much later, in the 1970s, was the application of this system relative to copper-rich and sulphidic ores understood and patented by Berezowsky and collaborators (1978). Furthermore, the novel application of this process at atmospheric conditions and lower reagent concentrations, made this alternative extraction technique far more attractive (Berezowsky, 1979; Hiskey, 2000). Subsequent research has shown that the presence of catalytic copper in the ore increases the rate of dissolution, in contrast to the reduced results incurred with cyanide in the presence of copper (Ter-Arakelyan, 1984). Thiosulphate has been successfully developed for commercial use in heap leaching at the Goldstrike operation of Barrick in Nevada (Aylmore, 2016a). Copper significantly increases cyanide consumption, consequently increasing the cost of reagent (Kongolo and Mwema, 1998; Muir and Aylmore, 2004b). However, in the thiosulphate system copper is a necessary reagent and therefore its presence within the ore will reduce the amount/cost of reagent required. In addition, a preg-robbing component such as that found in the ore of the Goldstrike mine has no effect on recovery in thiosulphate conditions. More recently, CLEAN Mining has acquired a patent for thiosulphate extraction from the CSIRO, after 10 years of intensive study by the Australian parastatal (Heselev, 2019).

The thiosulphate leaching system is considerably more complicated than cyanide, as the Eh, pH and reagent concentration (thiosulphate, copper and ammonia) must be precisely controlled, not only to promote gold leaching but also to minimise the decomposition of thiosulphate, a metastable complex in aqueous leach conditions. The decomposition of thiosulphate is undesirable as it wastes reagent as well as passivating gold surfaces (Hilson and Monhemius, 2006). Despite these complications, the low toxicity, the capacity to treat complex ores and the competitive dissolution rates make thiosulphate the most promising alternative lixiviant currently being researched (Aylmore, 2016a; Aylmore, 2016b). Thiosulphate dissolution of gold is slow without an oxidant that is more efficient than oxygen, a catalyst is also required (Abbruzzese et al., 1995). Extensive studies (Langhans et al., 1992; Breuer and Jeffrey, 2000; Jeffrey et al., 2001; Muir and Aylmore, 2004b; Aylmore, 2005) have found that the thiosulphate – copper – ammonia system is able to achieve kinetics of a similar rate to that of cyanidation under specific reagent conditions. Elevated reagent concentrations increase leach kinetics but also increase the degradation rates. The thiosulphate can undergo oxidative decomposition if the conditions are not well controlled, which will negatively affect the leaching efficiency. The oxidative decomposition is particularly pronounced in the absence of copper. In addition to increasing reagent cost, the decomposition of thiosulphate is thought to produce species capable of passivating gold, such as tetrathionate and polythionate (Hilson and Monhemius, 2006).

The high redox conditions required for thiosulphate leaching leads to significant reactivity with sulphides, increasing reagent consumptions in sulphide-bearing ores. The extent of thiosulphate consumption can be controlled, while still maintaining acceptable gold extraction. The use of calcium thiosulphate rather than sodium thiosulphate also improves the stability of thiosulphate as well as contributing to maintaining a rapid leaching rate (Aylmore, 2016a; Aylmore, 2016b). Thiosulphate has been comprehensively researched for applications relating to carbonaceous and copper-gold deposits, as cyanide has limited suitability for these ores. Carbonaceous material is preg-robbing in a cyanide leaching system, while gold-thiosulphate complexes are unaffected by the presence of carbon (Muir and Aylmore, 2004).

The gold dissolution proceeds by the mechanism shown in Equation 1.6, which requires that copper (II)-ammonia-thiosulphate complexes on to the surface of the gold, as the gold and thiosulphate oxidise, while copper reduces. Equation 1.7 represents the regeneration of

copper, Cu(II) from Cu(I), as well as thiosulphate, which is fundamental for continuous and sustainable leaching (as used in Equation 1.6). Oxygen is responsible for the regeneration of catalytic copper.

$$Au + 5S_2O_3^{2-} + Cu(NH_3)_4^{2+} \rightarrow Au(S_2O_3)_2^{3-} + 4NH_3 + Cu(S_2O_3)_3^{5-} \quad \text{(Equation 1.6)}$$

$$4Cu(S_2O_3)_3^{5-} + 16NH_3 + O_2 + 2H_2O \rightarrow 4Cu(NH3)_4^{2+} + 4OH^- + 12S_2O_3^{2-} \quad \text{(Equation 1.7)}$$

Gold recovery using activated carbon is not a feasible process, in the same manner that preg-robbing carbon in the ore does not affect the leach. Therefore, other recovery processes must be considered; feasible alternatives include resins and precipitation processes. Precipitation options include metal powders such as Zn and Fe in conjunction with a reducing agent such as sodium tetrahydroborate. Strong base resins are the most amenable to gold recovery in the thiosulphate system, however the degradation products of thiosulphate reduce the selectivity of the resin. As an alternative to resin, it has been suggested that stoichiometric amounts of cyanide could be used to recover gold from thiosulphate and traditional methods could subsequently be utilised (Lulham and Lindsay, 1991). This option is only feasible if the usage of thiosulphate is from a purely economic perspective, where gold extraction using cyanide was ineffective and therefore an alternative was required. However, if usage was due to cyanide bans or restrictions, despite significantly reduced cyanide concentrations, the original complication will still exist.

The instability of non-cyanide lixiviants is a major drawback, as is evident in the thiosulphate-gold complex, which is an order of magnitude less stable than cyanide-complexes (Aylmore, 2016a). Significant research has gone into a variety of stabilising agents. Glycine (NH_2-CH_2-COOH), a simple amino acid, has several suggested uses for gold leaching; in conjunction with hydrogen peroxide as an alkaline leach (Oraby and Eksteen, 2015), in cyanide starved conditions in the GlyCat™ process (Oraby and Eksteen, 2019) or in association with thiosulphate (Michel and Frenay, 1998; Feng and van Deventer, 2011). Because the major disadvantage to the thiosulphate leaching system is its reagent instability, the use of stabilising agents may improve the efficiency and profitability of the leach. Glycine, when used with thiosulphate, can act as a low-cost stabilising agent. The role of glycine within the thiosulphate is twofold, the glycine is capable of complexing directly with gold to form a stable compound as well as complexing with

the copper, to allow it to continue acting as a catalyst, but reducing its degradation effect on thiosulphate (Michel and Frenay, 1998; Feng and van Deventer, 2011).

1.4. Research Objectives

A review of the literature highlights the extensive study of various alternative lixiviants undertaken by many excellent researchers, however this research does show a lack of consistently incorporated mineralogical considerations. Additionally, while researchers have addressed certain mineralogical factors such as sulphides or carbon these studies rarely address the full suite of mineral present at an operational or production level. In response to these shortfalls the key objective of this study was to perform a comparative evaluation, of the different leaching methods in conjunction with the mineralogies commonly known to act as host to gold deposits. This was done with the aim to improve the understanding of gold and gangue mineral behaviour using alternative leaching methods. An additional objective was to determine if the understanding of leach behaviour could allude to the original geological environment that generated the ore, to assist in exploration target generation.

A diverse set of leaching methods were selected from the literature of existing research for testing. These were to be evaluated against an equally diverse set of mineralogies. Ores ranging the spectrum from free milling to double refractory were selected for testing to allow for a comparative study. This was done in order to understand the degree to which mineralogy could affect the leaching ability of lixiviants, on different ore types at a single set of conditions. The conditions selected and the motivations thereof are discussed to a greater degree in Chapter 9, once the ores have been appropriately characterised.

Chapter 2

Gold Deposits Selected for Study

2.1. Introduction

There are a number of important gold deposits within South Africa, which accounted for significant gold supplies in the past (Janisch, 1986). Although gold production has declined in recent decades, material from these deposits are still available for study. There are a number of deposits found throughout South Africa, ranging from extremely large, world class to small scale deposits. The deposits of study and their locations within South Africa are shown in Figure 2-1.

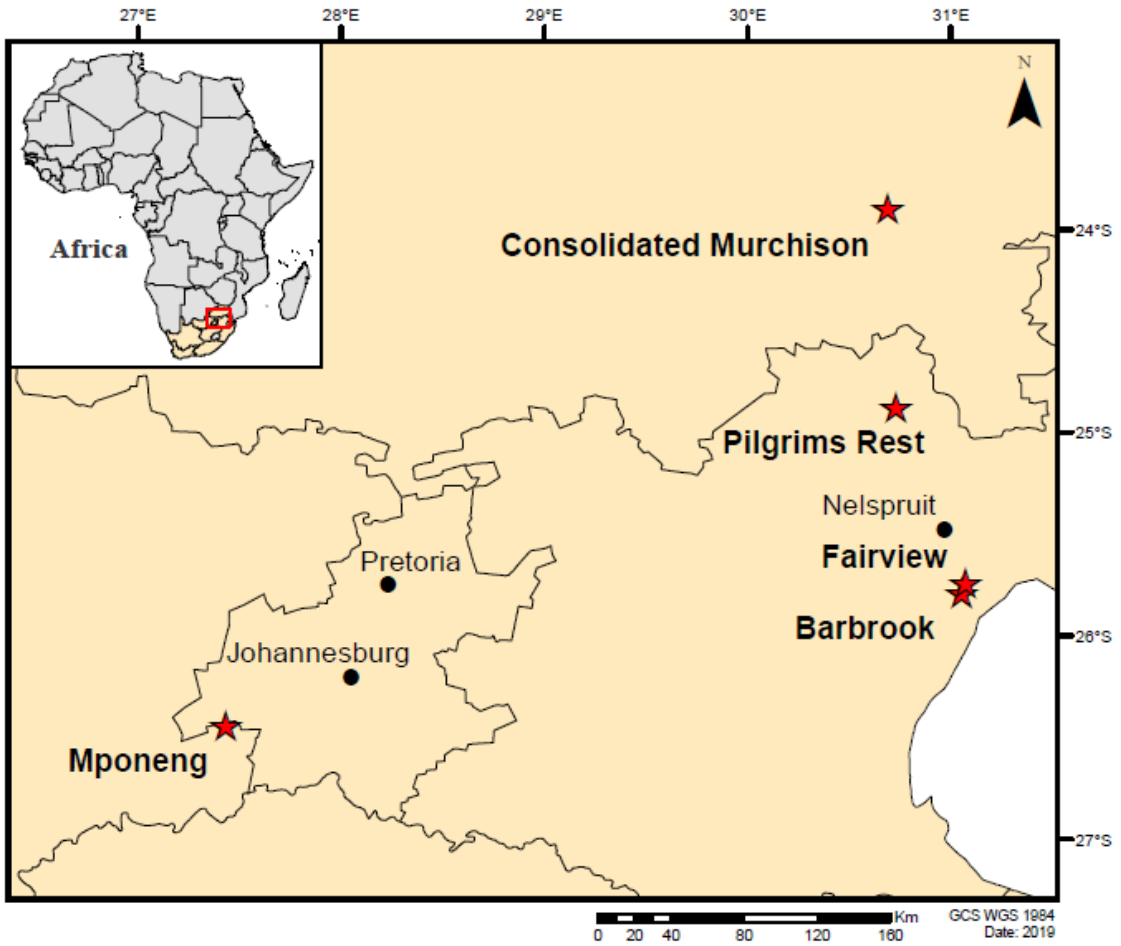


Figure 2-1 Localities of the gold deposits studied. The red stars are the exact locations of Mponeng (Ventersdorp Contact Reef), Fairview and Barbrook from the Barberton Greenstone Belt, Consolidated Murchison from the Murchison Greenstone Belt and Pilgrim's Rest from the Transvaal Basin.

The ores used in this thesis represent a spectrum of types of gold deposits and mineralogies from various localities in South Africa (Figure 2-2). These deposits include the Ventersdorp Contact Reef (VCR) at Mponeng Mine in the Witwatersrand Basin, Fairview and Barbrook, in the Barberton greenstone belt, Pilgrim's Rest- a Bushveld aged deposit on the eastern margin of the Bushveld Complex, and Consolidated Murchison within the Thabazimbi-Murchison lineament.

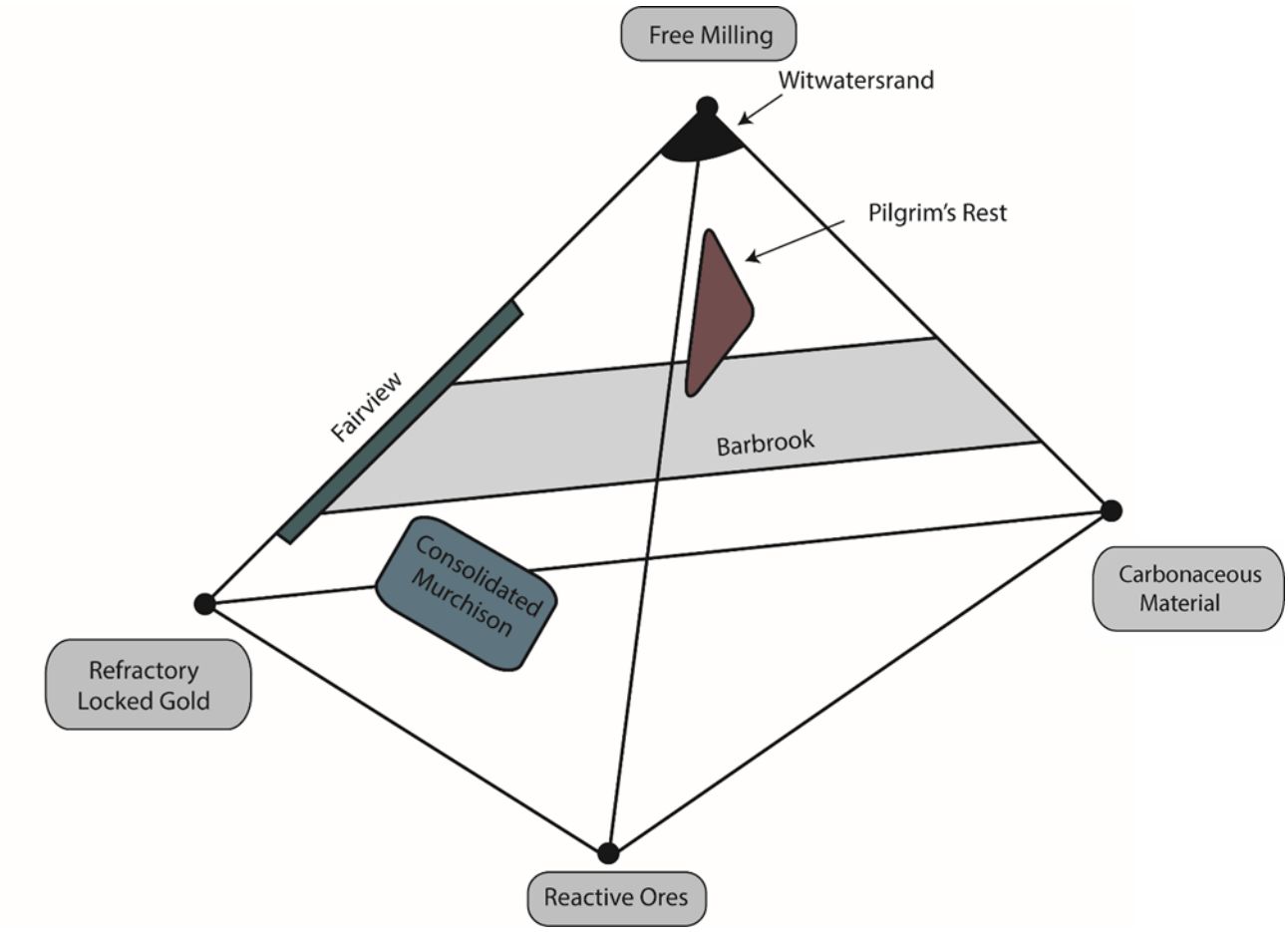


Figure 2-2 Gold deposits can span a vast range of responses to extraction, depending on their mineralogy which allows for the classification of deposits into free milling, refractory, reactive or carbonaceous (preg-robbing), however these classifications may overlap or vary within a single deposit. (Swash, 1988).

The VCR material was chosen as an example of high-grade free milling ore, where processing tends to be easy and compromising reagents (such as pyrrhotite) are only present in low quantities. Pilgrim's Rest represents a free milling ore with higher concentrations of reactive components. Fairview and Barbrook both represent refractory-hosted gold, however having both ores is important as Fairview represents singly refractory gold, while the presence of preg-

robbing carbon makes Barbrook a doubly refractory ore, that is significantly more difficult to process. Consolidated Murchison represents gold present in a gold mineral (aurostibite) as well as reactive and passivating associations. This range of material for study represents free milling, complex, and refractory ores; complex mineralogies defined by the presence of reagent consuming and preg-robbing materials, while the presence of gold-minerals, as well as sulphide-bound gold, results in refractory ore types. This diverse set of ores is an excellent setting against which to evaluate the respective leaching methods particularly in the context of cyanide performance.

2.2. Witwatersrand

The Witwatersrand Basin is host to the largest gold deposit in the world. This province is the largest gold bearing region, accounting for nearly a third of the world's total gold production (Frimmel, 2005). The gold of the Witwatersrand Basin was discovered in 1886, this sparked a significant gold rush in Johannesburg and subsequent tracing of the gold-bearing 'reefs' across the Witwatersrand Basin as extensive exploration took place (Borhcers, 1961; Handley, 2004). However, exploration at Carletonville, the location of the ore studied in this work, only took place during the 1930s, giving rise to the discovery of significant ore-bearing horizons (De Kock, 1941; McCarthy, 1994). The ore of study was obtained from Mponeng gold mine, which is owned and operated by Anglo Gold Ashanti. It is currently the deepest mine in the world. Mponeng Mine lies along the north-western rim of the Witwatersrand basin, near Carletonville, 80 km west southwest of Johannesburg (Figure 2-3). All production from Mponeng is sourced from the Ventersdorp Contact Reef (VCR), which occurs at the base of the Ventersdorp Lavas, and unconformably overlies the Witwatersrand Supergroup (Figure 2-4). Despite the fact that the VCR is not technically part of the Witwatersrand Supergroup, it does fall within the grouping of the 'Witwatersrand Triad' (Coward, 1995) and is traditionally considered with the Witwatersrand stratigraphy.

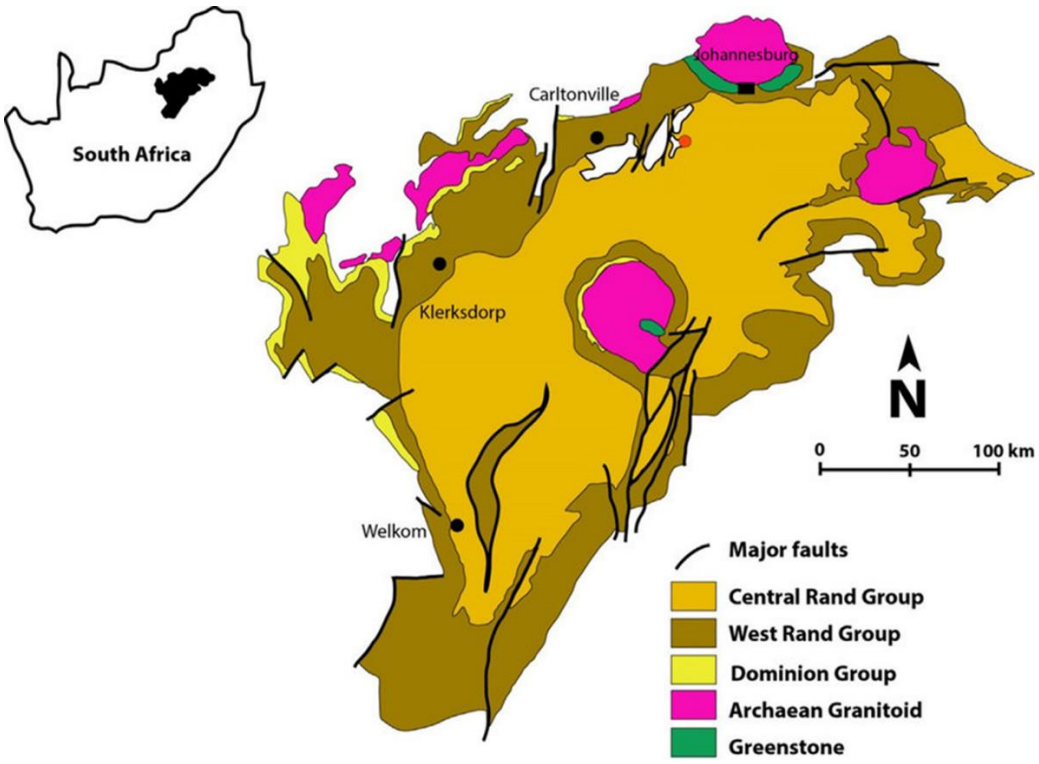


Figure 2-3 Schematic geological map of the Witwatersrand Basin (Mngadi et al., 2019). The Mponeng gold mine is the source of the Ventersdorp Contact Reef sample for study. The Mponeng mine is found within the Carletonville Goldfield, and the precise locality of the mine marked in red.

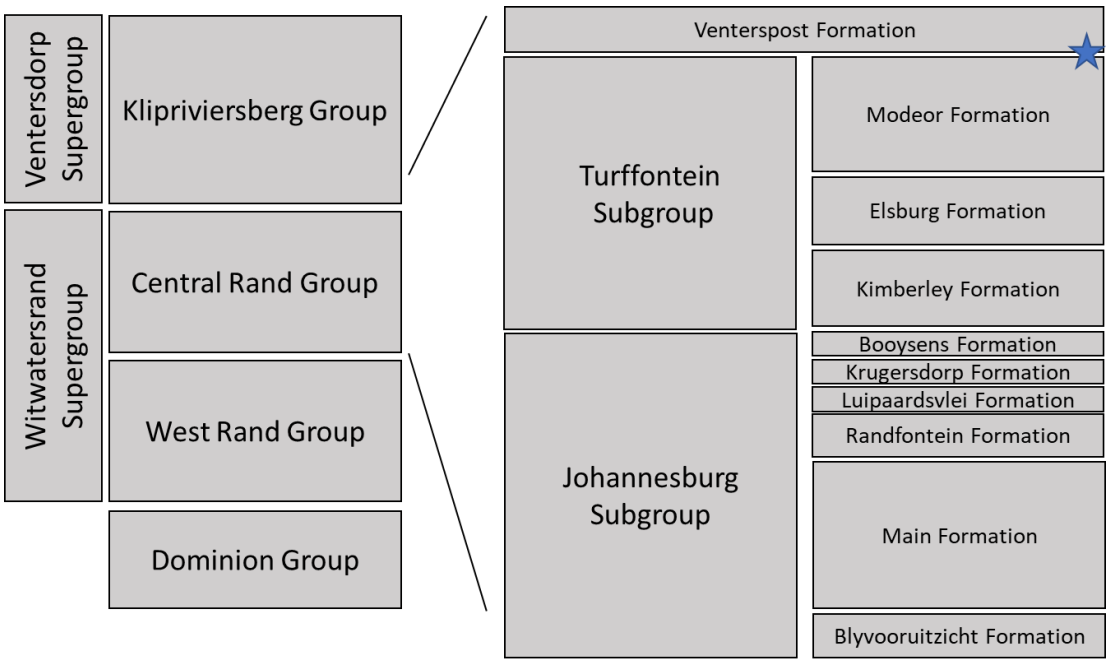


Figure 2-4 The stratigraphic column of the Witwatersrand Supergroup, with the position of the Ventersdorp Contact Reef (the ore of study) marked with a blue star. The Ventersdorp Contact Reef occurs at the contact between the Witwatersrand and the overlying Ventersdorp Supergroup, Venterspost Formation. The stratigraphic column was compiled from (Catuneanu and Biddulph, 2001).

2.2.1. Regional Geology and Formation

The VCR occurs at the base of the Ventersdorp Lavas (to which it owes its preservation), and unconformably overlies the Witwatersrand Supergroup (De Kock, 1941; Tucker et al., 2016). Economically viable VCR occurs along the western and northwestern margins of the Witwatersrand Basin, within the West Rand, West Wits and Carletonville goldfields (Tucker et al., 2016). The mineralisation occurs in metamorphosed quartz pebble conglomerates (McCarthy, 2006), and as such it is quartz-dominated ore, with significantly lesser amounts of chlorite, micas, pyrophyllite, chloritoid, pyrite, pyrrhotite, and lesser chalcopyrite, sphalerite and galena (Phillips and Law, 1994; Smits, 1994; Pretorius, 2012).

The VCR is thought to have formed directly from the erosion of the underlying Witwatersrand Basin (Krapez, 1985; Tucker et al., 2016). The gold, in addition to the major constituents such as quartz pebbles and the matrix material was likely to have been derived from the lithologies of the Witwatersrand Basin. Components such as quartz shards, volcanic ash and tuffaceous material that may be present in the VCR originated from the initial phases of the Ventersdorp lavas, this accounts for the darker matrix than typically seen in Witwatersrand ore (De Kock, 1941). There is evidence within the reef that hydrothermal activity has been responsible for the precipitation of gold (Gartz and Frimmel, 1999). The scale of gold remobilisation is contentious and beyond the scope of this work.

2.2.2. Local Geology and Mineralogy

While much of the Witwatersrand is extremely uniform across large areas, there is some variation. The VCR has been subdivided by McCarthy (1994) into two distinct types; the South Deep type and the Western Deep type. The Western Deep type is present in the Carletonville and Klerksdorp Goldfields where mineralisation is widespread throughout the conglomerates. The VCR can be up to 3 m thick locally. The angle of the unconformity is very low, approximately 4˚ or less (McCarthy, 1994). In contrast, the Southern Deep type has narrower mineralisation and larger degrees of the angle of unconformity. The footwall in Southern Deep Type mining areas is variable and includes; the Kimberley Formation, the Elsburg Formation and the Booysens Formation (Flitton, 2016). The local mineralogy is typical of the VCR, it is quartz-dominated, with chlorite alteration, pyrite, pyrrhotite, chalcopyrite, sphalerite and galena present in the matrix as well as gold. The gold is free milling and only associated with minor deleterious components (Flitton, 2016).

2.2.3. Extractive Metallurgy

The mineral assemblage at Mponeng (and in the Witwatersrand in general) has little effect on the cyanide extraction of gold, the free-milling gold is readily extracted (Pretorius, 2012). Therefore, standard milling and cyanide leaching is typically sufficient for the extraction process. The flowchart used to process the Mponeng ore has been outlined in Figure 2-5, constructed based on information available in the 2019 Annual Report for Anglo Gold Ashanti.

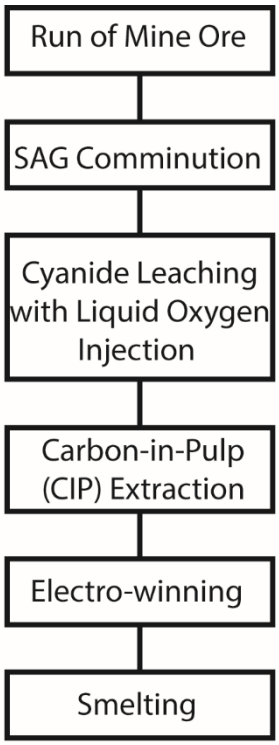


Figure 2-5 Processing flowchart for Mponeng Mine, important steps in this processing include cyanide leaching with Liquid Oxygen Injections and Carbon-in-Pulp extraction. .

The extraction process involves semi-autogenous (SAG) milling, cyanide leaching with liquid oxygen injection, extraction using carbon-in-pulp technology, followed by electro-winning and smelting in induction furnaces. The only part of the extraction process, that might be considered atypical is the addition of liquid oxygen, but this is included to counter the presence of any oxygen (and cyanide)-consuming pyrrhotite.

2.3. Barberton/ Fairview

The Barberton Greenstone Belt is near the town of Barberton in Mpumalanga Province, approximately 280 km east of Johannesburg (Figure 2-6). The Belt is one of the oldest and best preserved supracrustal sequences in the world. It is approximately 40 x 120 km and ranges in ages from 3.55 to 3.1 Ga (Lowe and Byerly, 2007).

Gold in Barberton was discovered in 1881, originally by Tom McLachlan, however that locality was malaria stricken and he was forced to withdraw. Later Augustus "French Bob" Roberts discovered a more feasible mining locality, and this was the trigger of the Barberton Gold rush (Anhaeusser, 2012). There have been a number of gold workings of varying sizes throughout the 137 years of mining taking place on the Barberton Greenstone Belt, although only four operating mines remain. Included in the four operating mines is Fairview, owned by Pan African Resources, from which the studied sample was collected.

2.3.1. Regional Geology and Formation

The Barberton Greenstone Belt comprises three supracrustal stratigraphic units (Figure 2-7); the lower Onverwacht, a predominantly volcanic unit, ranging from mafic to ultramafic, with minor felsic volcanics, tuffs and sediments. The overlying group, the Fig Tree, is argillaceous comprising greywackes, shales, mudstone and chemical sediments (Banded Iron Formation, BIF). The upper Moodies group contains conglomerates, quartz arenites and shales, with minor BIF and volcanics (Kamo and Davis, 1994; Hoffman et al., 1998; Lowe and Byerly, 2007; Anhaeusser, 2015). The greenstone belt has undergone greenschist to amphibolite metamorphism, with extensive siliceous, carbonate and potassic alteration (Dziggel et al., 2002).

The mechanism of mineralisation is a contentious issue amongst researchers on Barberton geology. In the past it was agreed that the mineralisation had an orogenic genesis, linked to the accretion of the greenstone belt (de Ronde et al., 1994; Groves et al., 1998; Goldfarb et al., 2001). However, a number of more recent papers (Dirks et al., 2009; Munyai et al., 2011; Dirks et al., 2013) suggest that the mineralisation occurred later than the accretion, and was likely to have been associated with an extensional event. Isotope studies suggest that the mineralising fluids originated from external sources that were probably magmatic, and these took

advantage of brittle-ductile shear zones and fractures (de Ronde et al., 1994; Ward, 1999; Munyai et al., 2011).

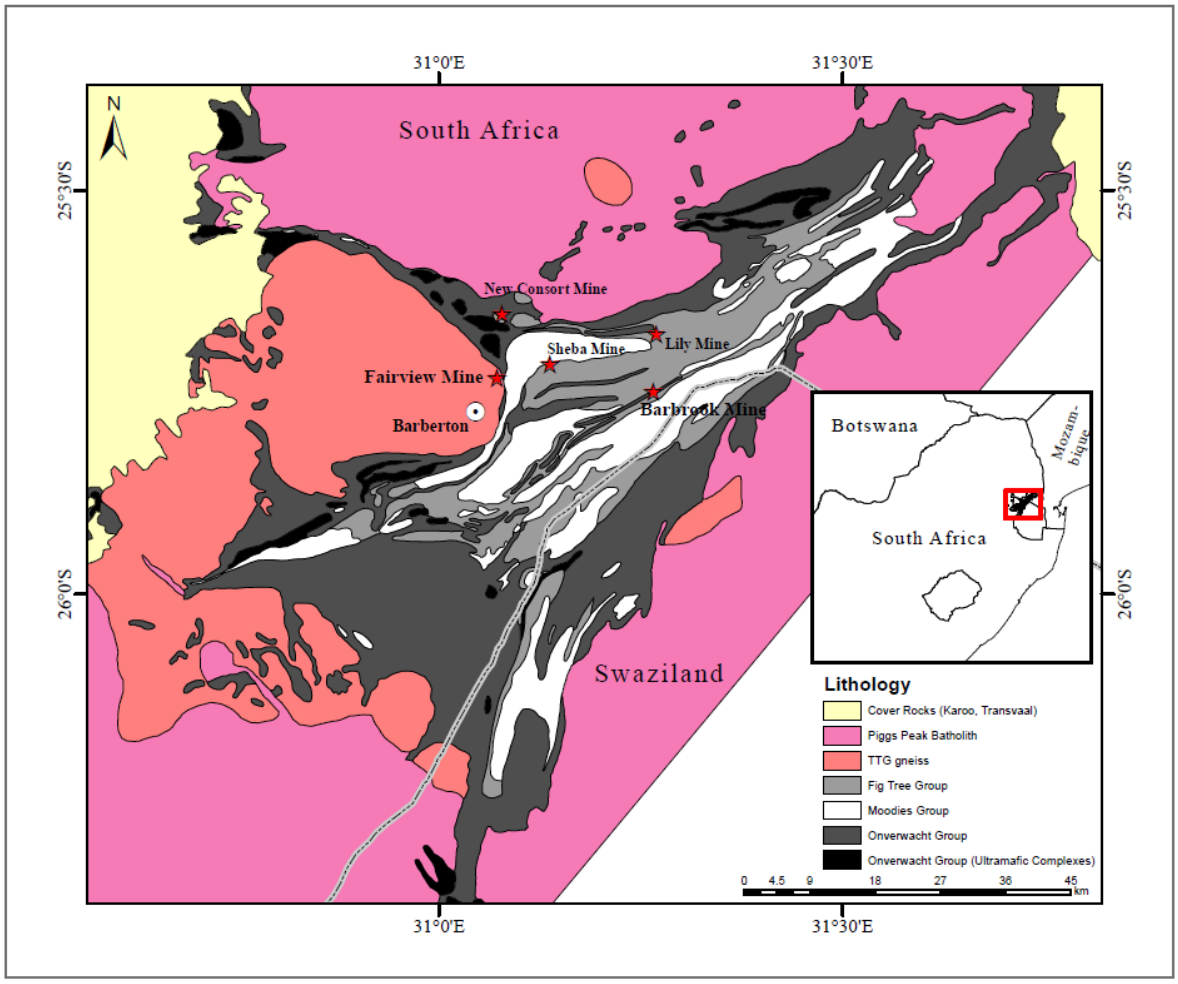


Figure 2-6 The geology of the Barberton Greenstone Belt with both Fairview and Barbrook mines marked in red (after Dirks et al., 2013).

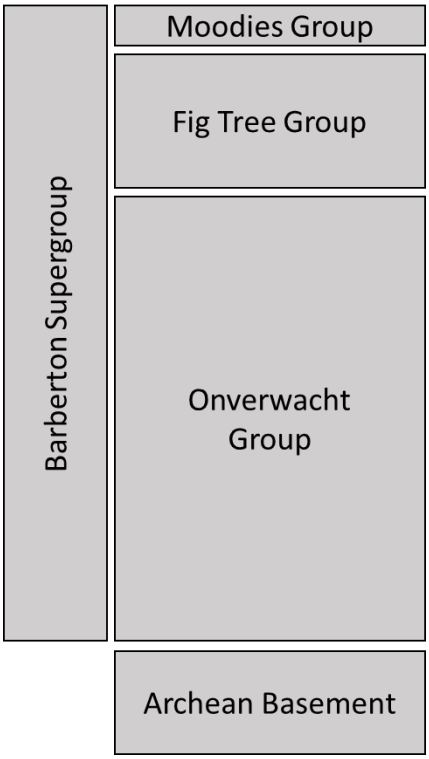


Figure 2-7 The stratigraphic sequence found in the Barberton Supergroup, with the Onverwacht, Fig Tree and Moodies Groups all of which are host to mineralisation in the Fairview locality. The Fig Tree Group is exclusively host to mineralisation in the Barbrook locality. The stratigraphic column was compiled from Lowe and Byerly, (2007).

2.3.2. Local Geology and Mineralogy

Fairview mine is located is 6 km north east of Barberton, within the Sheba Hills area. Mineralisation can be found in two forms 1) free milling, with commonly visible gold occurring in quartz veins, and 2) refractory sulphide ore, containing pyrite and arsenopyrite. Both of these forms of mineralisation are found in concordant and discordant fractures, ranging in size from 20 mm to 2 m in width. These fracture can also be up to 500 m in length (Anhaeusser, 1986; Anhaeusser, 2001). Ore minerals include pyrite and arsenopyrite with minor chalcopyrite, sphalerite and pyrrhotite. These ore minerals occur in calcite, quartz and sericite dominated gangue. Sheba-Fairview miners still produce 2.5-3 t of gold annually, with average grades of 12 g/ton (Pan African Resources 2019 Integrated Annual Report; Dirks et al., 2009). The mineralisation at Fairview is structurally hosted and occurs throughout all stratigraphic units, with no preference for lithology, although there is an increasing degree of mineralisation in the lowermost units (Ward, 1999).

2.3.3. Extractive Metallurgy

The extraction of gold from Fairview is complicated primarily due to the presence of refractory sulphide-hosted gold. In order to extract as much gold as possible, the sulphides hosting the gold must be destroyed. In the past this was done using roasters, however this process was abandoned due to a number of reasons (including maintenance and environmental concerns). The roasting process was replaced by biological oxidation (BIOX®). The BIOX® process was first designed and implemented in Barberton, after it was understood that certain bacteria (for example *Thiobaccilus ferrooxidans)* were capable of oxidising the sulphide minerals, making the gold accessible to cyanide leaching.

A simplified diagram based on a process description received from Fairview, (Figure 2-8) involves simple comminution, followed by flotation (in order to upgrade the sulphides), thickening, followed by BIOX. The conditions required for BIOX are far more acid than that required for an effective cyanide leach, and therefore neutralisation is needed prior to leaching. Cyanide leaching is completed, followed by gold recovery using CIP extraction and electro-winning.

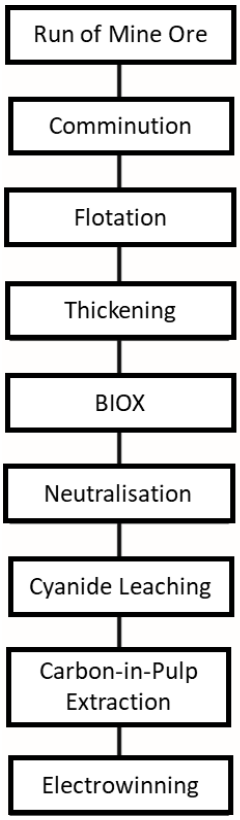


Figure 2-8 Processing flowchart for Fairview mine, in which it is evident that flotation, thickening and BIOX processes are implemented before cyanide leaching and carbon in pulp extraction.

2.4. Barberton/ Barbrook

Barbrook mine is part of the Barberton Greenstone Belt. It is located near the town of Barberton, Mpumalanga, approximately 280 km east of Johannesburg. Barbrook is approximately 20 km east northeast of Barberton. Due to the similarity of Barbrook and Fairview, the regional geology presented in Section 2.3.1 is relevant also for this deposit.

2.4.1. Local Geology and Mineralogy

Barbrook mine is comprised of a number of smaller deposits including Sofala, Barbrook, Cascades, Vesuvius, French Bobs, Crescent, Clifford Scott, Maid of the Mist, Crown, Bushbuck and Daylight (Pearton and Viljoen, 2017). Barbrook mine has been intermittently operational since 1915. These deposits occur exclusively within the Fig Tree Group, either in the BIF at the base of the greywacke units or along the margins of altered ultramafic schists (Lanyon, 1997). The ore deposits tend to occur along one of two major shear structures within the area, the Barbrook or Swartkoppies Line (Groenewald, 1975; Pearton and Viljoen, 2017). Mineralising hydrothermal fluids utilised these structures. The interaction between fluids and iron-rich lithologies caused sulphide mineralisation and gold precipitation (Groenewald, 1975).

Important minerals found in Barbrook ore include pyrite, arsenopyrite, pyrrhotite and chalcopyrite, with limited visible gold. The host rock gangue consists of ferruginous carbonaceous shale and carbonate-bearing chert, other minerals present may include vein quartz, ankerite, goethite and graphite (Groenewald, 1975; Grant, 2004). Higher gold grades are found in thicker BIF units, as well as quartz-filled shear zones and zones of arsenopyrite enrichment. These deposits tend to have limited economic viability due to the structurally complex and refractory nature of the ore (Pearton and Viljoen, 2017).

2.4.2. Extractive Metallurgy

Barbrook is currently owned by Vantage Goldfields, who have yet to make their processing procedures available to the public. However, Barbrook was previously owned by Caledonia Mining Resources, whose processing has been represented simplified in Figure 2-9.

The gold processing procedure is complicated by two factors; the occurrence of sulphide-hosted gold and preg-robbing carbonaceous material, making this ore double refractory. No other deposit of study contains a preg-robbing component and thus accounts for the need to

study two ores from the Barberton region. The Barbrook flowsheet is tailored to improve recovery despite these complications. Comminution is followed by flotation aimed at removing carbonaceous, preg-robbing material from the ore before further processing. A second flotation stage, under different conditions, is required to concentrate the sulphides acting as host to the gold. The sulphide concentrate undergoes an additional comminution phase followed by pressure oxidation. Cyanide leaching is then undertaken with the gold loading onto resin-in-leach (to reduce preg-robbing effects). The gold is eluted from the resin and can subsequently be smelted (Grant, 2004). Before Caledonia Resources sold Barbrook Mine in 2007, there were investigations into using BIOX® processes similar to those used at other mines in the region, such as Fairview (Grant, 2004). Despite promising investigations no further developments were made, probably due to financial constraints.

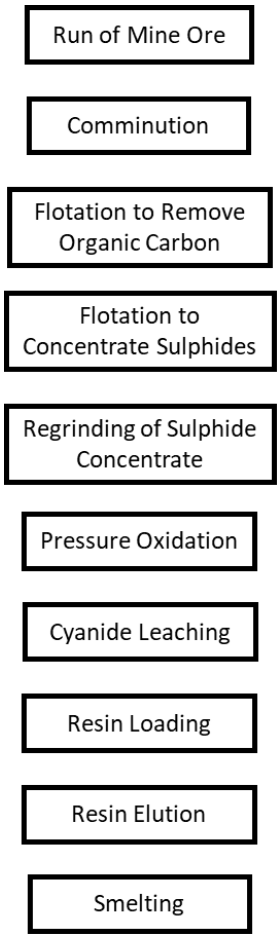


Figure 2-9 Processing flowchart for Barbrook, a simplified diagram generated from Caledonia Mining Resources Annual Reports. Organic carbon in this system is removed using flotation. The sulphides are also concentrated using flotation, after which they undergo further comminution, pressure oxidation and cyanide leaching.

2.5. Pilgrim's Rest

Sabie-Pilgrim's Rest is a gold district located in Mpumalanga Province, west of the Drakensburg, approximately 320 km east northeast of Johannesburg (Figure 2-10). It was discovered in 1872 by MacLauchlan, Valentine and Parsons, and it is one of the earliest exploited gold deposits in South Africa (Reinecke, 1929). Originally this gold locality was only an alluvial operation, but subsequent underground operations were initiated in the 1890s (Tyler and Tyler, 1996). Although there is currently no commercial mining in the Sabie-Pilgrim's Rest region, in the past it was an important source of gold, the third highest gold producing province in South Africa after the Witwatersrand Basin and Barberton (Boer et al., 1995).

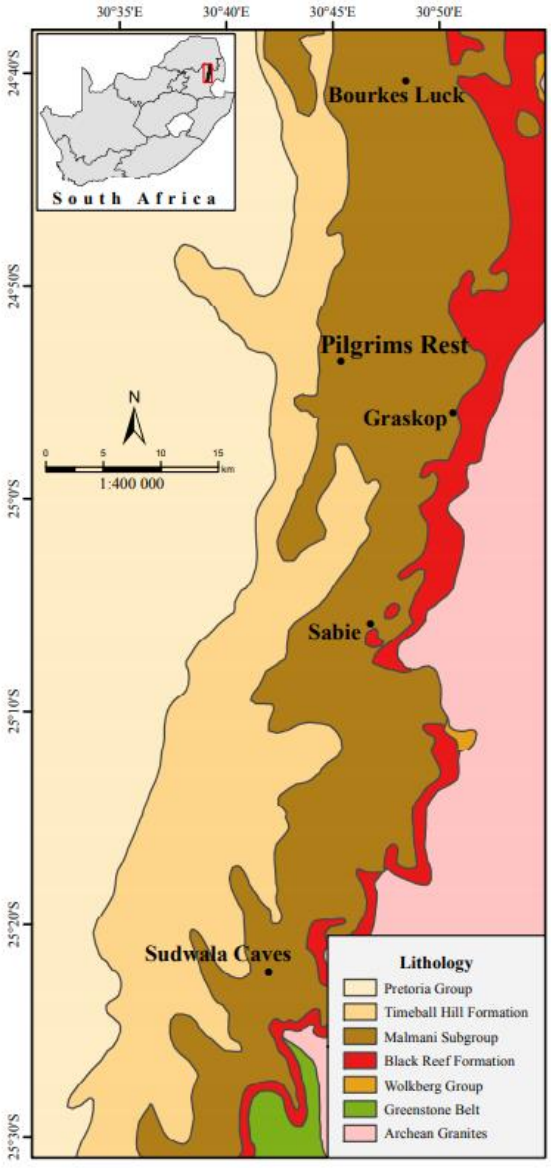


Figure 2-10 Schematic geological map of the Sabie-Pilgrim's Rest region, host to the Pilgrim's Rest gold deposit. Map produced from Jaguin et al., (2012).

2.5.1. Regional Geology and Formation

The Sabie-Pilgrim's Rest district is an epithermal deposit, hosted by the Proterozoic Transvaal Basin (Boer et al., 1995). The Transvaal Basin comprises the basal Black Reef Quartzites, overlain by the Chuniespoort Group (which contains the Malmani Dolomites, Penge Iron Formation and the Duitschland Subgroup), the Pretoria Group overlies the Chuniespoort (Figure 2-11). Of economic importance within the Transvaal basin is the Chuniespoort Group, which hosts most of the economic mineralisation of the Pilgrim's Rest region (Harley and Charlesworth, 1992; Boer et al., 1995).

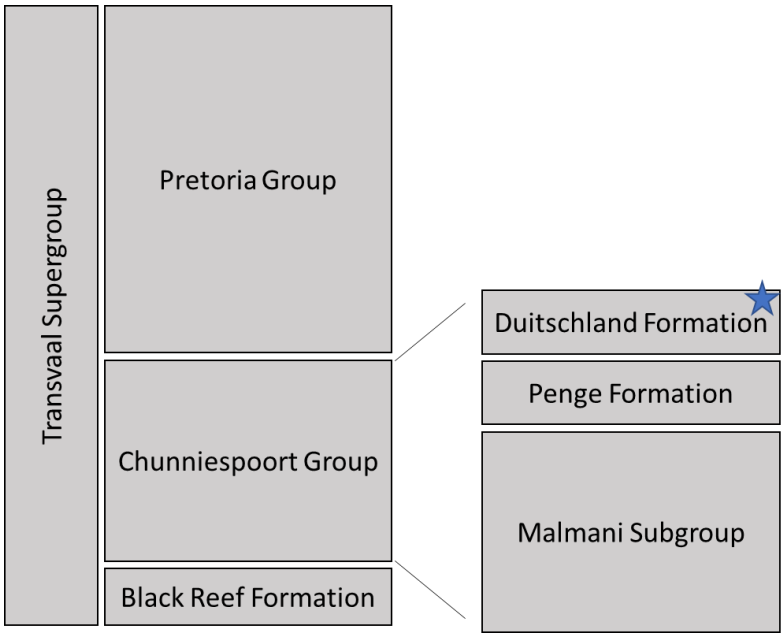


Figure 2-11 A portion of the stratigraphic sequence of the Transvaal Supergroup, is host to the Chuniespoort Group and subsequently the Duitschland Formation. Pilgrim's Rest mineralisation occurs within the Duitschland Formation (demarcated with a blue star). This stratigraphic sequence is based on Tyler and Tyler, (1996).

Mineralisation within the Pilgrim's Rest region is present as both concordant and discordant veins (relative to lithological units). The concordant quartz-carbonate veins, known as flat reefs, host the majority of the recovered gold. Both concordant and discordant veins range in thickness from 0.5 cm to 2 m, with an average of 20-30 cm (Swiegers, 1948). The ore mineralogy includes abundant pyrite, lesser chalcopyrite, arsenopyrite, bismuthinite, tetrahedrite, sphalerite and galena. Gold grade is locally extremely variable (ranging from 2-50 g/ton) (Harley and Charlesworth, 1996; Tyler and Tyler, 1996), however, gold grains typically post-date the

aforementioned metallic minerals. Gold occurs either as discrete, fine grains along micro fractures or as inclusions in sulphides. Mineralisation is thought to have been a multi-stage process, with much of the gold precipitating during later stages of hydrothermal activity (Meyer and Robb, 1996; Tyler and Tyler, 1996).

The gold mineralisation is thought to be contemporaneous with the formation of the Bushveld Complex. The Bushveld Complex is also responsible for the emplacement of the thrust system (exploited by the hydrothermal, mineralising fluids), but also the source of the fluids and heat required to form this epithermal gold deposit. Isotopic data, fluid inclusion studies and elemental fingerprints all support the Bushveld origin hypothesis (Tyler, 1986, Boer et al., 1995; Meyer and Robb, 1996; Tyler and Tyler, 1996).

2.5.2. Local Geology and Mineralogy

Little is known about the deposit from which the studied ore was claimed. The mine, Morgenzon, from which the samples were collected is not currently operational, nor is there any available research of this locality with any significant detail or importance. Therefore, all of the local geology is based on the research undertaken in this thesis.

2.5.3. Extractive Metallurgy

The mineralogy of the Pilgrim's Rest deposit is complex in terms of processing due to the presence of base metal sulphides and oxides (i.e. minerals containing copper and to a lesser degree iron). This deposit is also host to refractory gold however, the samples collected from Pilgrim's Rest for this study were not from the zones containing refractory gold (Tyler and Tyler, 1996). The complex and refractory ores were kept separate throughout processing using two discrete circuits that only merged for final cyanidation.

The operations represented in Figure 2-12 depict the processing procedure of the Pilgrim's Rest area in 1961, and while the flowchart may have changed in subsequent years there is no evidence for this in the literature. The non-refractory ore was simply processed via gravity concentration and subsequent cyanide leaching of the gravity tailings. The refractory ore required more complex processing in order to break down the refractory sulphide-hosted gold, and therefore additional steps of flotation and roasting were required before cyanidation could take place (Congress, 1961).

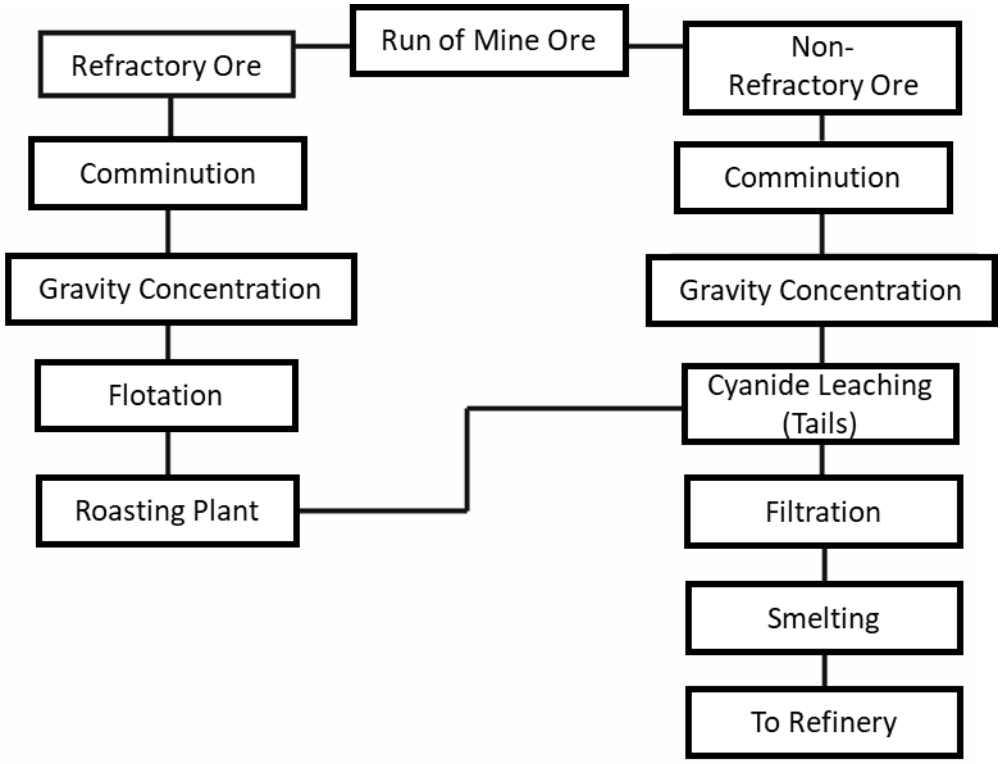


Figure 2-12 Processing flow chart for Pilgrim's Rest, a simplified diagram from 7th Commonwealth Mining and Metallurgical Congress, (1961). Discrete processes were applied for the two ore types, refractory and non-refractory.

2.6. Consolidated Murchison

Consolidated Murchison lies in the Northern Kaapvaal Craton, near Gravelotte, 50 km west of Phalaborwa and 370 km northeast of Johannesburg. It represents an important antimony-gold deposit within the Murchison Greenstone Belt (Figure 2-13), an Archean volcano-sedimentary belt (Davis, 1986; Anheusser, 2001). Gold was reportedly found as early as 1870, however it was only in 1886 that the area was declared a public digging. As the ores are refractory at depth, many of the original operations were unsuccessful after the surficial deposit was exploited (Davis, 1986).

While this deposit is not currently actively mined, it is owned by Stibium Mining and stockpiles are being processed. In the past it has been mined either for antimony, with additional gold credits or for gold, with additional antimony credits depending on the respective global demands and price (Davis, 1986).

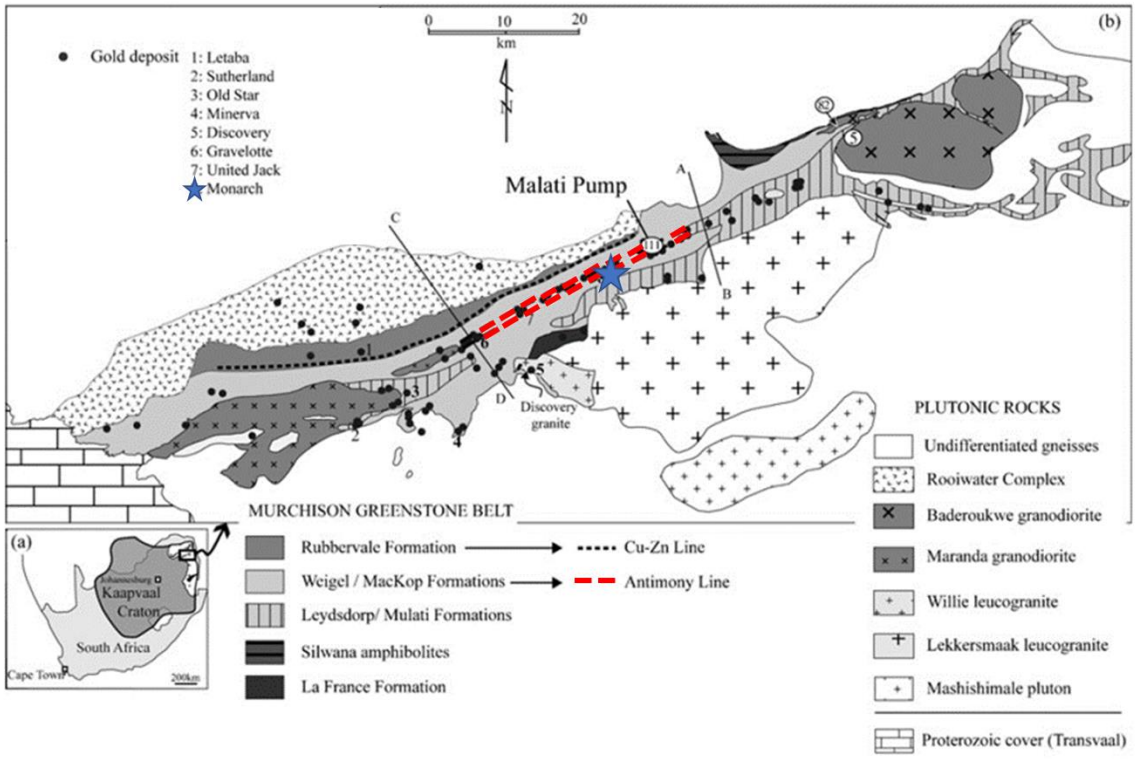


Figure 2-13 Schematic geological map of the Murchison Greenstone Belt, host to the Consolidated Murchison gold-antimony deposit (Jaguin et al., 2012). The ore was collected from the Monarch deposit, marked with a blue star, along the antimony line.

2.6.1. Regional Geology and Formation

While there is a large range of mineralisation present in the Murchison Greenstone Belt, only the antimony-gold deposits will be discussed. Gold and antimony are present in close association intermittently along a 35 km long epithermal vein system known as the Antimony Line, in the Murchison Greenstone Belt (Vearncombe et al., 1988). The Antimony Line is 250 m wide and occurs in the Weigel Formation (Figure 2-14). The Weigel Formation heterogeneously comprises quartzites, schists, iron formation and carbonate-bearing rocks (Vearncombe, 1988). The Antimony Line comprises a central core of carbonate material, surrounded by sheared talcose and chloritic schists. The mineralisation is significant in the central carbonates. The main mineralisation along the Antimony Line is stibnite (Sb_2S_3) and gold, these minerals tend to be associated with lesser amounts of arsenopyrite (FeAsS), berthierite ($FeSb_2S_4$), pyrite (FeS) and more atypically iron, nickel, lead and antimony sulphides (Davis, 1986). Gold is present in the Antimony Line in a number of forms ranging from coarse free grains, fine disseminations in sulphides to aurostibite (Davis, 1986; Vearncombe, 1988; Marsden and House, 2006).

There has been uncertainty regarding the metallogenesis of the Antimony Line. Models for both syngenetic and epigenetic origins have been proposed. However, the vast, complex, deformation undergone by this region makes accurate studies extremely difficult. The ores appear to be orogenic in nature as well as being associated with significant subsequent metamorphic remobilisation. The simultaneous occurrence of both gold and antimony minerals suggests that a single process was responsible for their enrichment (Pearton and Viljoen, 1986; Jaguin et al., 2012).

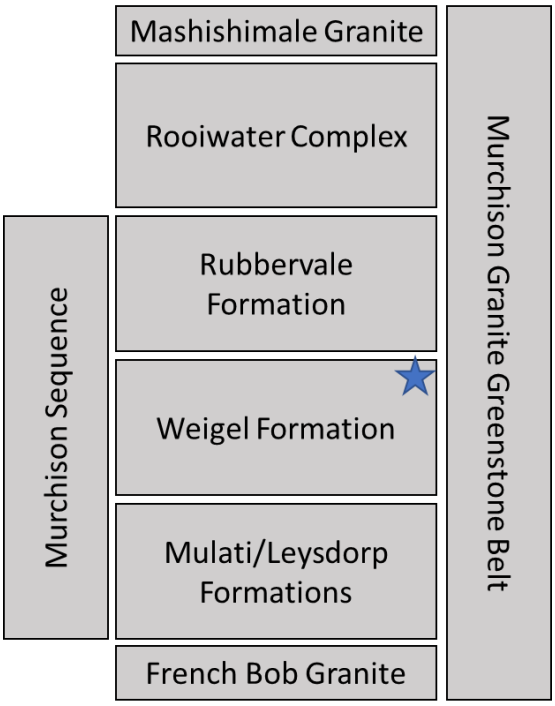


Figure 2-14 The stratigraphy of the Murchison Greenstone Belt, with the Weigel Formation, host to the Antimony Line and the Monarch deposit from which the studied samples were obtained. This column is based on the work in Poujol, (2001).

2.6.2. Local Geology and Mineralogy

There have been up to 19 antimony-gold mines at various points along the Antimony Line since its discovery, however the samples for this study have been obtained from the Monarch Shaft of Consolidated Murchison/Stibium Mopane. While the Monarch ore tends to be more abundant in gold than other localities (with average grades of 2.5 g/t) it also tends to be more metallurgically complex (Kirk Adam, Pers. Comms). Free gold occurs within both quartz veins and arsenopyrite/pyrite-rich chert zones. Minerals such as berthierite, gersdorffite,

tetrahedrite and ullmannite tend to be relatively abundant. Mineralisation is present within quartz vein stockworks in quartz carbonate rocks (Pearton, 1982; Pearton and Viljoen, 1986; Pearton and Viljoen, 2017).

2.6.3. Extractive Metallurgy

While antimony is readily extracted, its presence during cyanidation affects gold recovery in a number of ways. The antimony can act as an oxygen scavenger, while increasing both cyanide and lime consumption during leaching (Monhemius, 1987; Millard, 2005). There is also evidence that the surface of the gold becomes passivated during normal cyanidation processes in the presence of antimony (Millard, 2005; Rohner and Millard, 2016). In addition to the concerns surrounding the deleterious nature of antimony within the ore, the gold is also present in a range of forms from coarse free gold to sulphide-hosted. The amount of gold occurring in inaccessible forms (such as sulphide-hosted) varies with location and will significantly affect the efficiency of gold extraction. Therefore, in order to completely predict the degree of refractoriness of the ore a good gold deportment study is required, as attempted here.

The refractory nature of this ore has implications for gold dissolution, which cannot be effectively or economically done using simple cyanidation. The processing procedures used to extract this ore are more complex than the previous deposits described. The flowchart of its processing has been simplified and outlined in Figure 2-15.

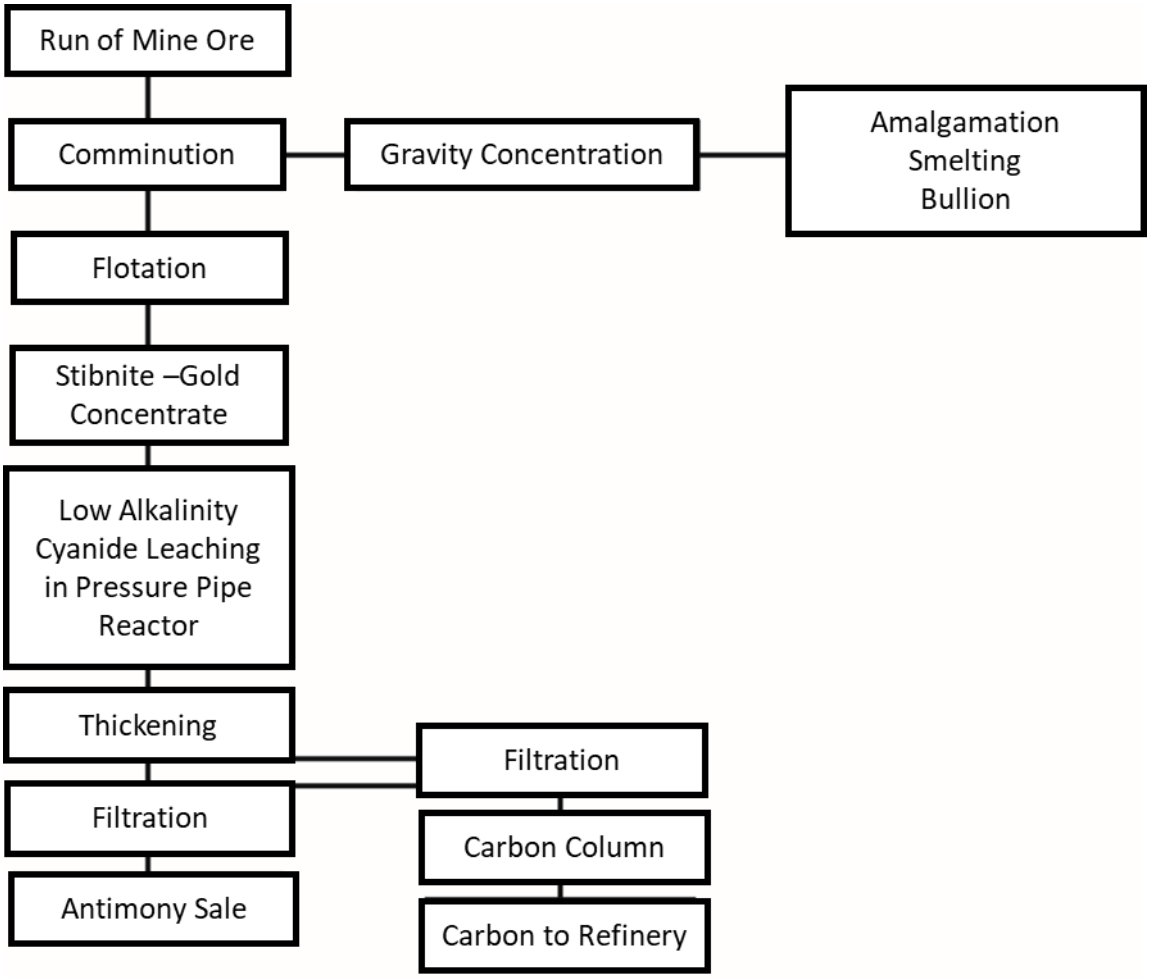


Figure 2-15 Processing flow chart for Consolidated Murchison, a simplified diagram (Davis, 1986; Marsden and House, 2006). Low alkalinity pressure leaching of this ore has been used to deal with the highly refractory nature of this ore.

The processes shown in the flowsheet represented in Figure 2-15, have been described by a number of workers in varying detail (Davies and Paterson, 1986; Marsden and House, 2006). In 1986 (the time of publication of Davis and Paterson), the gravity concentrate step was recovering up to 35% of the total gold (more recent data are not available). The remaining ore was subjected to flotation, concentrating both gold and stibnite. The gold is subsequently recovered from the flotation concentrate using a low alkalinity, high pressure (8.8 MPa) cyanide leach, in a 1.5 km pressure pipe reactor (Davies and Paterson, 1986; Millard, 2005). This process also requires pre-screening and conditioning. Oxygen consumption is on average 57 kg/ton (Marsden and House, 2006).

2.7. Discussion

The samples collected from the Witwatersrand Basin, Barberton, Consolidated Murchison and Pilgrim's Rest span a wide variety of mineralogies. The individual ores are each processed using very different flow sheets in order to optimise recovery relative to the variable nature of gold and gangue.

The formation of these deposits is very different, with some mineralisation related to alluvial processes with ore hosted within reefs, while others owe their metallogenesis to orogenic and hydrothermal processes, with ore that is not stratigraphically bound.

Grades range from extremely high for Pilgrim's Rest and the VCR, moderate for Fairview and low for Consolidated Murchison and Barbrook. These same ores have varying degrees of refractory hosted gold, particularly in Fairview and Barbrook, while Consolidated Murchison has gold present in aurostibite. VCR and Pilgrim's Rest contain native, liberated gold. Barbrook contains preg-robbing carbon, and a consuming component can be found in Pilgrim's Rest and Consolidated Murchison.

The nature of these ores has been briefly summarised, however, these characterisations are generally based on predicted responses to cyanide and factors such as cyanide consuming components may have limited relevance with regards to leaching using alternative lixiviants. This sample diversity should allow for a good evaluation of the leaching techniques described in Chapter 1.

Chapter 3

Sample Preparation and Comminution

3.1. Introduction

Sample preparation was undertaken to enable the study to better achieve the objectives. The first objective was to characterise the mineralogy to predict leaching behaviour (with considerations to mineralisation targeting in addition to providing information for leaching with alternative lixiviants). The subsequent aim is to understand the gold and gangue mineral behaviour in the context of alternative lixiviants.

To ensure that the characterisation was performed optimally, appropriate sample preparation was performed for respective analyses,

- the preparation of a statistically significant number of thin sections for study
- milling, splitting and pulverising samples for XRD and assay
- gravity concentration on feed to ensure statistically significant gold grains

In order to understand the gold and gangue mineral behaviour in the different leaching systems the material was also prepared for bottle roll leaching. This will be discussed to a greater extent in section 3.2.

Sample preparation of gold is notoriously difficult (Hoffman et al., 1998), but critical to producing quality results. Ensuring the samples used for the required analyses are representative is paramount, and as such, steps to prevent biased sub-sampling or cross contamination were taken. The complete flowsheet for sample preparation is found in Figure 3-1.

Five samples from discrete deposits within South Africa were selected for study. The deposits were selected to represent a wide range of leach behaviours (from free milling to double refractory), and therefore the mineralogy of each deposit needed to be well understood.

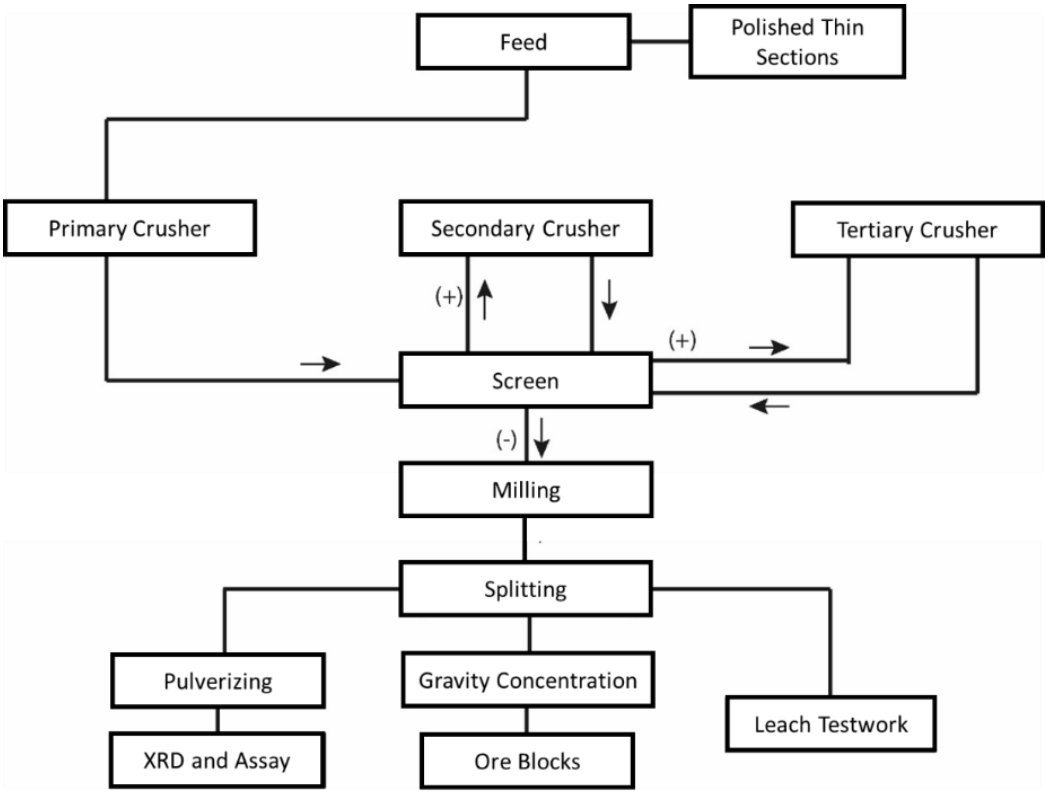


Figure 3-1 Flowsheet for sample preparation for XRD, assay, ore blocks for gold characterisation and leaching. Crushing, milling and splitting were required for the leaching test work, while subsequent pulverisation was needed for XRD and assay. Gravity concentration was used to prepare ore blocks for gold characterisation.

The samples used for this study were run of mine, obtained from direct feed to the processing plants. The sampling took place after coarse crushing but before any other processing was implemented. Attempts were made to ensure that the mineralogy was representative of the typical feed of the deposit, as determined by the on-site geologists.

3.2. Methodology

Polished Thin Sections

As the samples are composites of run of mine ore, it was necessary to prepare several thin sections, an average of 10 from each deposit were selected, to ensure that the petrological studies were completed on the full range of lithologies to obtain an accurate representation of the respective deposits. The polished thin sections were made directly from the ore, prior to the crushing required for the remaining test work. The sample surfaces were selected in order to be as representative as possible of the run of mine material, while still including important

mineralisation features. Pictures were taken of the selected areas for polished thin sections for later correlation.

Comminution

Much of the additional mineralogical test work, as well as subsequent leaching required comminution of varying degrees.

Comminution is a vital first step in the processing of most commodities and particularly in gold extraction. Quality comminution is vital to ensure that the mineral/metal of interest is sufficiently exposed or liberated to be susceptible to the extraction processes. Comminution can refer to several processes such as blasting, crushing and milling (Fuerstenau and Han, 2003; Napier-Munn and Wills, 2006). The typically recommended grind size for gold leaching is 80% passing 75 µm (Marsden and House, 2006), and therefore all of the aforementioned processes are required in order to achieve this grain size. In the literature a coarser grind size has been suggested for gold quantification, for example 50% passing 75 µm recommended by Coetzee et al. (2011). However, a typical grain size for leaching gold is 80% passing 75 µm, and in order to ensure that the material of study is representative of the leaching material, the same milling was performed on all material used in this research. As gold leaching is typically performed on material 80% passing 75 µm, studying material of this nature was most practical. The gravity concentrates were used for the polished blocks, so a larger number of gold grains were available for study.

Crushing reduces the material from run of mine size to a standard diameter of between 2 and 0.5 cm, commonly in several stages. Crushing requires compression of material against fixed surfaces. Types of crushers include jaw, cone and impact crushers, and crushing tends to be done dry (Fuerstenau and Han, 2003; Napier-Munn and Wills, 2006).

Milling reduces particle size from within the crushing range to the point where the particles are sufficiently small to differentiate the economic component from gangue, this exact size will vary for respective ores. Milling differs from crushing, as impact and abrasion are the mechanisms responsible for particle size reduction in milling, resulting from interaction with media capable of free motion (Fuerstenau and Han, 2003; Napier-Munn and Wills, 2006). Milling can be completed either wet or dry, and common types include, ball, rod and semi/autogenous mills.

For this research project, the total run of mine ore received from the respective mines varied in mass from 25 to 40 kg, while individual rocks ranged in size from approximately 10 to 35 cm. The variation in mass for the respective deposits is due to the sample size deemed necessary by the respective mine geologists, thus this mass was not reduced until milling and splitting had taken place to ensure this representativity remained throughout the study.

Due to the size of the material, jaw crushing was required for the initial phase of comminution. Secondary phase crushing was completed using two cone crushers, the first cone crusher (or secondary crusher) reduced the particles to 3.35 mm or less, while the second cone crusher (or tertiary crusher) was operated in closed circuit and ensured a crush size of 100% passing 1.7 mm. Following crushing, the samples underwent a series of splitting processes using a rotary splitter to ensure the samples were homogenous, as well as to separate the total mass into several equal aliquots of less than 10 kg each, to allow for milling.

The aim of the milling was to achieve 80% passing 75 µm. Aliquots of approximately 8 kg were milled using a ball mill. The ball mill shell had an internal diameter of 40 cm, an internal length of 40 cm, the mill was rotated at 49 rpm while the grinding media has a mass of 55.02 kg. The time required to mill samples to 80% passing 75 µm varied depending on the mass being milled and the inherent strength of the material. Therefore, to obtain the desired grind for the respective samples, the first aliquot was milled for a series of short time periods. After each short milling period a subsample of this comminuted aliquot was screened to determine the percentage of material 75 µm or less. This process was repeated several times until a milling curve could be generated, capable of predicting the milling time required to achieve the correct grind for the respective samples. Using the milling curves subsequent aliquots to be milled quickly and efficiently to the correct grind size. The milling curves for each deposit (found in Figure 3-2) were generated using exponential trends. Each deposit shows a concordant trend with an initially rapid decrease in grain size (evident in a steep gradient) before a moderate tapering out (a shallower gradient) of grain size reduction. It is likely that this initially rapid size reduction occurred because of breakage along numerous pre-existing zones of weakness within coarse grained particle originally present (Napier-Munn and Wills, 2006). These zones of pre-existing weaknesses are predominantly due to residual effects of the crusher, as well as poorly cemented clastic material. As these zones of weakness became less abundant, the rate of decreasing size as a product of milling decreased, and the curves became flatter.

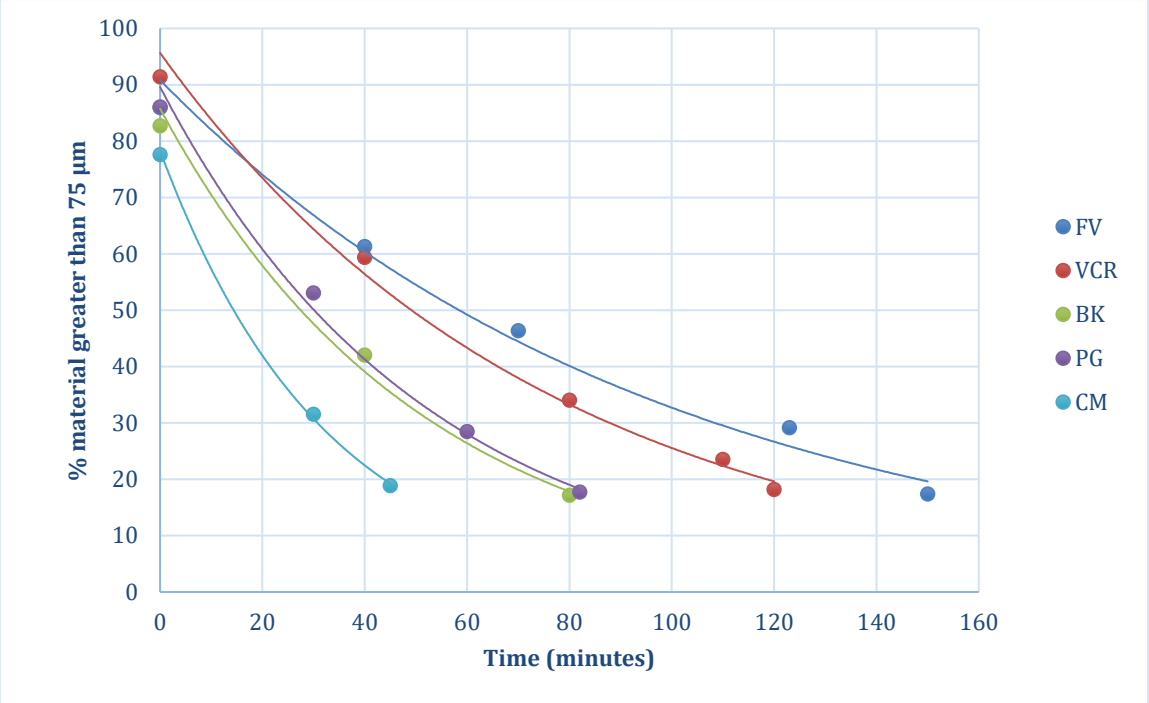


Figure 3-2 Milling curves produced for the milling masses of Ventersdorp Contact Reef (VCR), Fairview (FV), Barbrook (BK), Pilgrim's Rest (PG), and Consolidated Murchison (CM). Each curve shows an initially rapid decrease in grain size (evident in a steep gradient) before a moderate tapering out (a more shallow gradient) of grain size reduction.

Using the exponential equations of the trendlines, the exact milling time required for the ore was calculated. It is noteworthy that the milling times for different deposits is vastly different, despite the original partical sizes (after crushing) being very similar. This variation is likely to be due to two reasons 1) mineralogical variations (constituent minerals and textural features) influencing intrinsic strength, 2) total mass being milled (Table 3-1). The variation in milling times cannot be accounted for solely due to mass, as is evident in Figure 3-3, where there is no direct relationship between mass and time. An additional factor (intrinsic strength) is required to account for the difference between milling time for the respective deposits.

Table 3-1 Average masses used for milling for all deposits, and their ideal milling times in order to achieve a grind size of 80% passing 75 µm. Fairview and Ventersdorp Contact Reef (VCR) have significantly longer milling times than Pilgrim's Rest and Barbrook and to an even greater extent Consolidated Murchison.

Deposit	Mass (kg)	Milling Time (min)
Fairview	9.62	151.5
VCR	8.6	120.5
Pilgrim's Rest	8.6	79
Barbrook	8.2	73
Consolidated Murchison	8.2	44

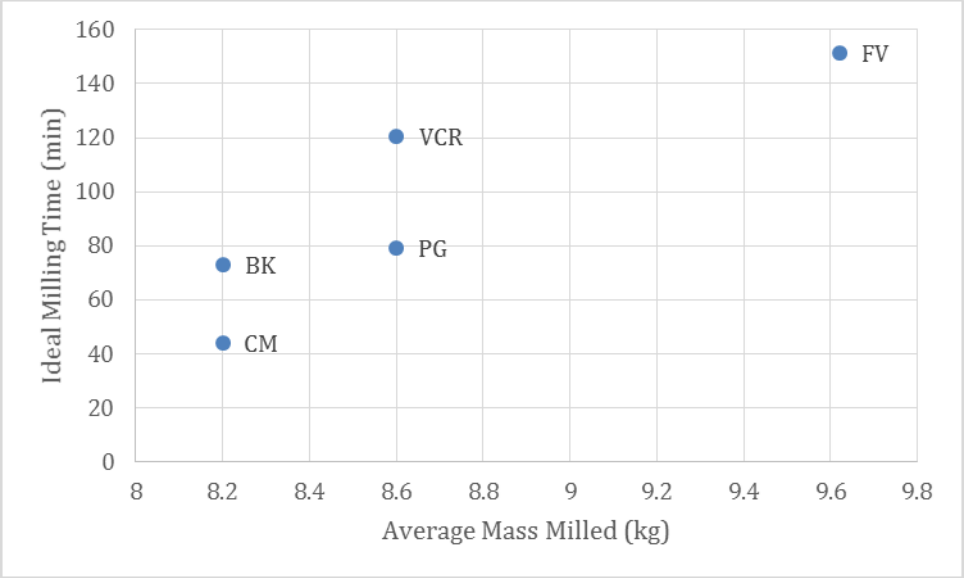


Figure 3-3 Graph representing the relationship between the mass of the material being milled and the time required to obtain a grind of 80% passing 75 µm. This graph highlights that the milling time is not only related to the mass of the milling material.

XRD and Assay

Good sample preparation is vital to obtain the best XRD results. The grain size of the powder to be analysed needs to be extremely fine to reduce noise levels, avoid preferred orientation, and to ensure that a representative number of particles are involved in the diffraction analyses (Klug and Alexander, 1974). The samples give the best results when the particles are between 0.1 µm and 40 µm, to allow for significant diffraction to take place without affecting the accuracy of the results, highlighting the importance of the sample preparation for XRD (Speakman, 2016).

The material used for assay also requires pulverisation to homogenise the samples, allowing for representative subsampling.

Leaching Testwork

The leaching test work was undertaken on ore at 80% passing 75 µm, thus no additional sample preparation was required. This grind size was selected with the aim of achieving sufficient gold liberation/exposure to allow for the leaching behaviour of the ore to be controlled by the chemistry and mineralogy, rather than the grind. This was guided by typical commercial plant practice but was supported by mineralogical analyses of the samples conducted prior to the leach test work (Coetzee et al., 2011).

Gravity Concentration

The gravity concentration was done using a Knelson Concentrator, one of the most effective techniques to concentrate gold using gravity. Originally developed in in the 1970's, it utilises two main principles, increased gravitational force and material fluidisation (Knelson, 1992). The Knelson Concentrator consists of a central concentrating cone, which has a series of grooves capable of trapping heavy minerals. Surrounding this, is a water jacket, which contains pressurised water that moves into the central concentrating cone (Huang, 1996). The material (a slurry) is fed into the central cone, which rotates rapidly, enhancing the gravitational force. This rapid rotation forces the material to the extremities of the cone, where depending on the specific gravity of the material, it will either become trapped in the grooves of the cone, or move up and out of the cone (Huang, 1996; Burt, 1999). The concentration, specifically the process of increasingly heavy minerals replacing lighter minerals in the grooves is reinforced by the pressurised water injected into the central cone, which allows the material to remain fluidised. The light material overflows and continuously discharges from the system, while the heavy material is concentrated within the cone and must be periodically recovered (Knelson, 1992; Huang, 1996).

Gravity concentration was done on aliquots of approximately 5 kg, for each sample, with a water pressure of 0.5 psi and a centrifugal acceleration of 120 G. From the Knelson Concentrator approximately 100 g of concentrate sample was generated, which was subsequently hand panned to further concentrate the samples and upgrade the gold. The hand panning was done to generate approximately equal amounts of panned concentrate (conc) and

remaining gravity concentrate (mids). The concentrates and middlings were used to make three polished blocks respectively, to allow for a study of the gold present in quantities that would not otherwise have been possible. Higher grade in the material of study allow for better characterised gold.

The degree to which the gravity concentration upgraded the gold from the feed to the concentrate samples is roughly calculated using the concentration of the initial material and the remaining gravity tailings (Table 3-2). It is interesting to note the varying degree to which the concentrates have increased in grade; these variations in grade (from feed to concentrate) range from 4 to 44%, which reflects the textural nature of the gold in the respective samples, specifically the coarseness and liberation, and therefore the degree to which it can be separated from gangue.

Table 3-2 The masses involved in the gravity concentration process, which were then used to generate a theoretical/calculated grade within the concentrate and relative percentage increase in grade from sample to conc.

Deposit	Sample Grade (g/ton)	Initial Sample (g)	Tails (g)	Gravity Conc (g)	Panned Conc (g)	Panned Midds (g)	Conc Grade (g/ton)	Grade Increase (%)
VCR	23.6	4907	4734	119.01	65.65	53.36	766	32.4
Fairview	12.5	5274	5054	115.53	69.52	46.01	128	10.3
Barbrook	2.08	4860	4724	123.9	67.48	56.42	24	11.7
Consolidated Murchison	3.04	4929	4823	95.11	57.92	39.19	13	4.4
Pilgrim's Rest	29.4	5284	5149	118.8	48.46	70.34	1300	44.2

3.3. Discussion

All of the described sample preparation was performed with consideration to the quality requirements/difficulties applicable to gold ore. Samples were taken from the run of mine material to prepare polished sections in order to facilitate an understanding of the minerals, textures and associations of the samples prior to comminution. Subsequently all the remaining material was crushed and milled as is required for leaching. The grind size was selected in order to achieve the objective of understanding the relationship between gangue composition and leaching. Thus the grind 80% passing 75 µm was selected to achieve sufficient gold liberation/exposure to allow for the leaching behaviour of the ore to be controlled by the chemistry and mineralogy, rather than the liberation/exposure of the gold. Some additional

sample preparation was required for other characterisations and methods. An aliquot of the milled material was pulverised to allow for XRD and assay. A larger aliquot was separated and underwent gravity concentration to provide a sample of enriched gold concentrations to properly characterise the gold from each deposit.

Chapter 4

Petrography

4.1. Introduction

Petrography was a vital component of this study as it allowed for the characterisation of gangue minerals, particularly those with the potential to interfere with gold extraction. For example, the identification of deleterious components such as carbonates or carbon are particularly relevant to the leaching response. As the samples were composites it was necessary to prepare a number of thin sections from the samples of each deposit in order to ensure that the descriptions were an accurate representation of the received material. The thin sections were examined using a petrographic microscope under transmitted and reflected light microscopy. Transmitted light microscopy is useful for characterising gangue mineralogy, particularly for minerals that are semi-transparent in thin section. Examples of such semi-transparent minerals include silicates and carbonates which may have a prohibitive effect on processes such as crushing, milling and even leaching. Minerals in transmitted light microscopy can be studied using plane polarised light (PPL) and crossed polarised light (XPL). The difference between plane and cross polarised light microscopy relates to the set up and presence of polarisers within the microscope. The use of both of these setups provides more diagnostic characteristics for mineral identification. Reflected light microscopy, in contrast to transmitted light, is relevant to opaque minerals. Opaque minerals tend to be metal-bearing, commonly occurring as sulphides. Reflected light microscopy was particularly advantageous for locating and identifying economic minerals as well as potentially deleterious sulphides. Both reflected and transmitted light microscopy provide insight with regards to texture and mineral association. Full thin section scans from each deposit have been incorporated to highlight the overall mineralogy and textures.

The run of mine material was studied in order to provide a holistic understanding of both the gangue and economic minerals. While gold was present in the material studied for the petrography in certain samples, the observations made regarding gold were used in conjunction with the polished blocks and MLA results to confirm gold mineralogy, all of which is found in Chapter 6. Therefore, the work in this chapter generally focuses on the gangue mineralogy of the samples with a focus on deleterious components.

4.2. VCR

The VCR is, in many ways, a very typical Witwatersrand ore. The VCR sample was a conglomeratic ore-bearing reef. The sampled conglomeratic host tended to be matrix supported meaning that the individual clasts did not form the framework of the conglomerate (Figure 4-1). However there was some variability in this throughout the composite sample. The conglomerate was polymict, with clasts of both quartzites and vein quartz, and significantly lesser chert pebbles. The pebbles were rounded to well-rounded (Figure 4-1), and commonly extended along a single axis in an ovular shape. The pebbles ranged in size from 2 mm to 5 cm. The quartz pebbles were commonly milky white, dark grey/smoky, and in certain instances the pebbles displayed variation from dark margins to white centres. The matrix was dark green to black, and was composed of fine, rounded to sub-angular quartz grains, with interstitial chloritic and sericitic material (Figure 4-2A). Metallic minerals present included pyrite, pyrrhotite and minor chalcopyrite as well as gold (Figure 4-2B).

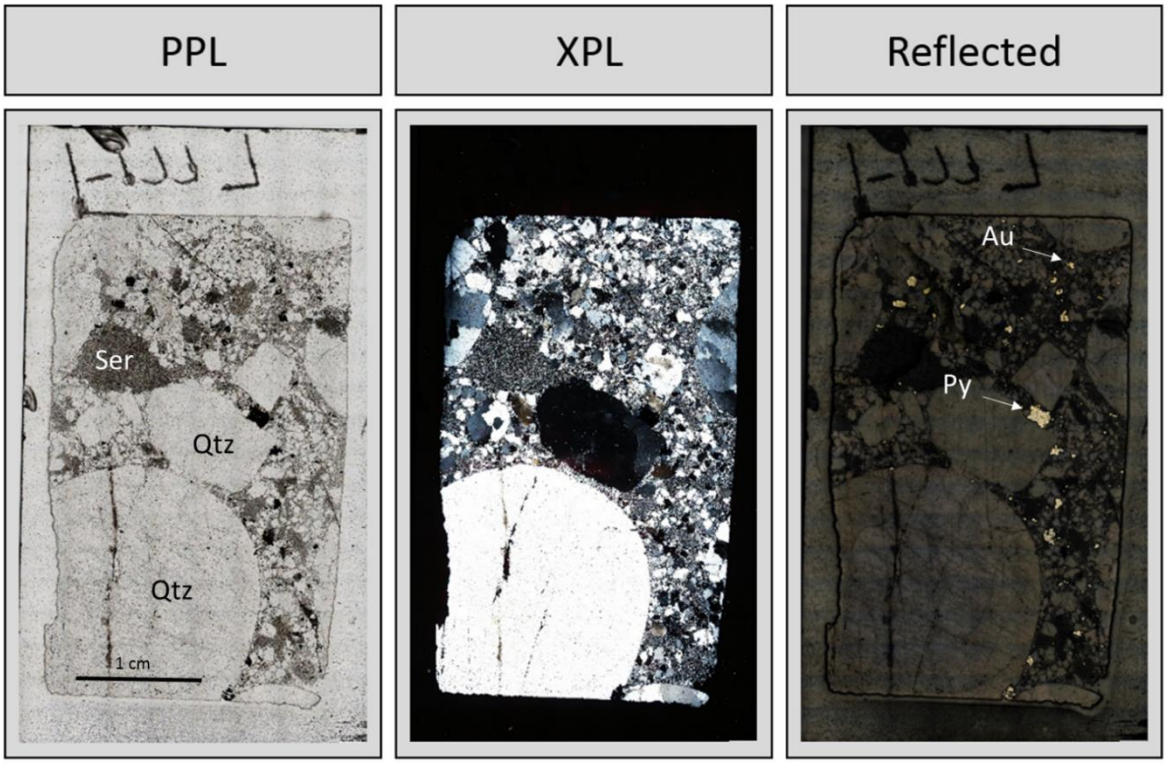


Figure 4-1 Overview photomicrographs of transmitted (PPL and XPL) and reflected light images, these highlight the overall mineralogy and texture of the sample. As is evident in the image, the Ventersdorp Contact Reef sample was a matrix-supported conglomerate. The PPL and XPL images highlight the presence of predominantly quartz (Qtz) clasts. The abundant matrix contained fine-grained quartz, chlorite (Chl) and sericite (Ser). The thin section in reflected light highlights the abundant pyrite (Py) as well as minor gold (Au).

Quartz

The dominant mineral (quartz) was present in at least 4 forms;

1) Quartzite pebbles

2) Quartz vein pebbles;

Both pebble types generally occurred in the same form; typically ranging from 2 mm to 3 cm in size, rounded, barrel-shaped grains. The pebbles contributed a variable amount to the total mineralogy, due to the variation associated with its large size, however it formed approximately 35-40% of the total rock.

3) Fine interstitial quartz grains represented the majority of the matrix. These grains were rounded to sub-angular, ranging in size from very fine to 2 mm. Approximately 50-55% of the total rock was comprised of fine-grained quartz matrix.

4) Quartz was also present in amalgamations of extremely fine grained, microcrystalline quartz, however, this contributed to less than 2% of the total.

Chlorite

The chlorite occurred in large agglomerations or interconnected masses, comprised of smaller chlorite grains. No preference for habit was evident in the fine chlorite grains, as well as little to no foliation. The chlorite regularly occurred with inclusions of muscovite. Chlorite constituted approximately 1% of the total ore.

Muscovite/ Sericite

The muscovite ranged in size from very fine grained to 0.5 mm, with an average of around 0.1 mm. The muscovite was commonly associated with chlorite as well as microcrystalline quartz and generally occurred interstitially within the matrix. The extremely fined-grained, well distributed micas contributed less than 2% to the total ore.

Pyrite

The nature of the pyrite was rather variable with a range in size from 10 µm to 5 mm, with a generally anhedral to subhedral habit. The pyrite formed in a single event as no evidence of overgrowth is present. The pyrite was generally more porous than compact and occurred within the matrix material, more commonly associated with the chlorite and muscovite than the fine quartz.

	The abundance of pyrite was highly heterogeneous but with an average of the whole rock comprising 0.5% pyrite.
Pyrrhotite	Individual pyrrhotite grains ranged in size from 10 µm– 0.5 mm, although it should be noted that the pyrrhotite commonly occurred as an interconnected network and could therefore forms chains of significant length. Only trace amounts of pyrrhotite were present.
Chalcopyrite	Trace amounts of fine (<0.2 mm) chalcopyrite were present. The chalcopyrite generally occurred as interstitial grains with a blebby habit. This mineralisation, when present was typically associated with pyrite.
Gold	Trace amounts of gold were evident, typically as irregular grains. The grain sizes ranged from less than 5 µm to more than 350 µm.
Sphalerite	Trace amounts of sphalerite were present in the sample, which was generally associated with galena.
Galena	Trace amounts of galena were present in the sample. The galena was commonly associated with sphalerite.
Alteration minerals	Muscovite/sericite and chlorite were present in the sample, particularly as fine grains within the matrix.
Texture and structure	The sample was dominated by clasts typically of large rounded pebbles, with finer interstitial material. In some instances, there were angular clasts present, but this was rarely the case.
Veining	Minor veining was present, these fine veins were generally comprised of quartz and sericite.

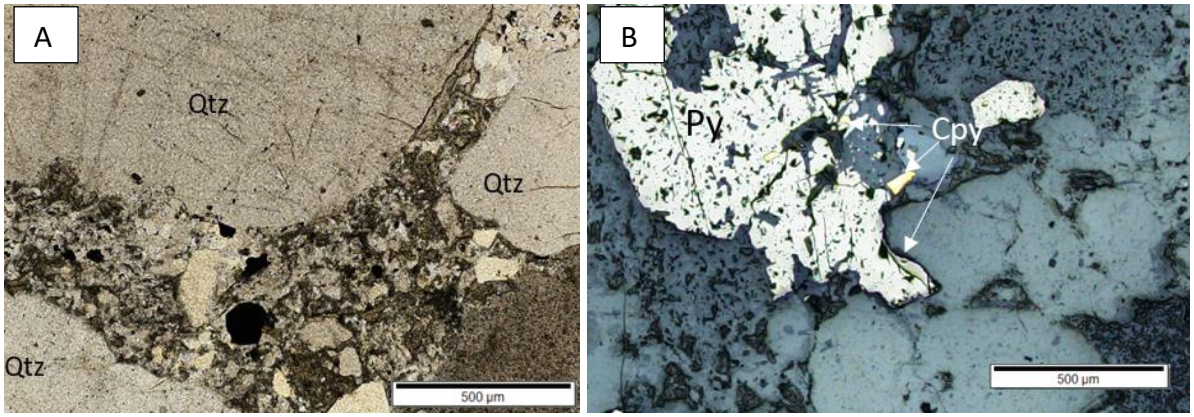


Figure 4-2 Microphotographs from Ventersdorp Contact Reef thin sections. A) A PPL transmitted light image showing the abundant quartz (Qtz) pebbles, interacting with a fine-grained matrix of quartz, chlorite and sericite, pyrite was also present within the matrix (present as black grains in transmitted light). B) A reflected light image of a pyrite (Py) grain associated with chalcopyrite (Cpy) at the margins as is typical for the VCR material.

4.3. Fairview

The mineralisation found in the Fairview sample was predominantly hosted by an altered micaceous schist and, in more quartz rich regions, quartzite (Figure 4-3). The sedimentary units contained lithic quartz fragments with a fine-grained matrix of quartz, chlorite, dolomite, magnesite and to a lesser extent muscovite (Figure 4-4A). Ore and metallic minerals were commonly present in veins/fractures (typically large and euhedral) (Figure 4-4B), or as disseminations within the main lithology (commonly fine and sub- to anhedral). Metallic minerals present included pyrite, arsenopyrite and minor visible gold. Significant veining was present, commonly closely associated with mineralisation. Veins were either quartz or calcite.

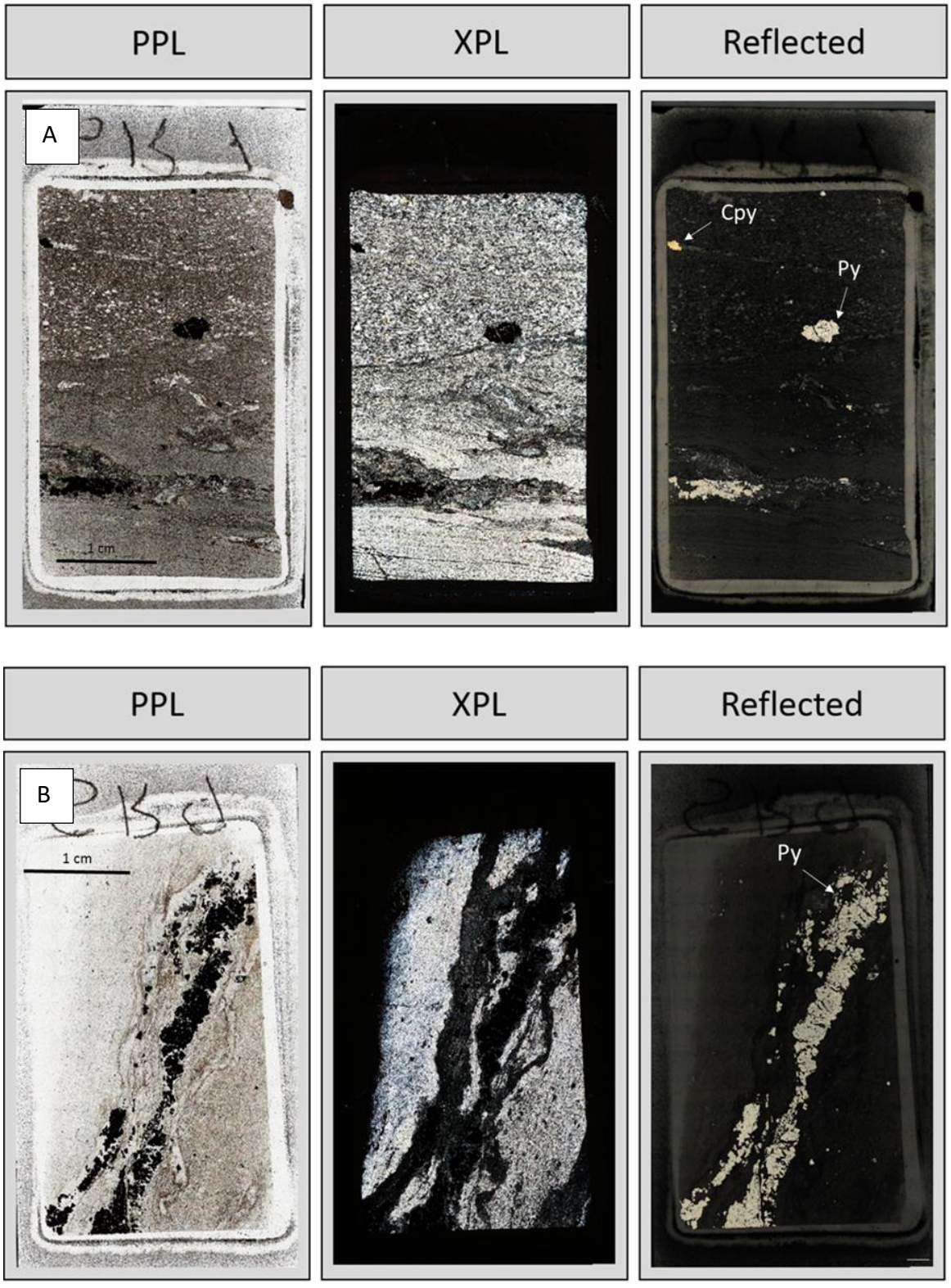


Figure 4-3 Transmitted (PPL and XPL) and reflected light images highlight the overall mineralogy and texture of the Fairview sample. A) The PPL and XPL images depict a matrix of quartz, chlorite, and dolomite. Individual minerals grains were typically not differentiable at low magnification. B) Veins are common in Fairview and are predominantly quartz with minor calcite. The reflected light image highlights the pyrite (py) found in veins and fractures.

Quartz	Quartz was present as lithic fragments, commonly with a moderate alignment, found within the finer grained matrix. Fragments ranged in size from 0.1 mm to 0.7 mm, with an average grain size of approximately 0.3 mm. The grains were generally angular to sub-angular, commonly with fractured margins. These lithic fragments constituted approximately 20% of the total rock. In addition to the clasts, there were also finer grained subhedral quartz within the matrix, approximately 0.1 mm or less in size. This accounted for approximately 20% of the total material.
Calcite	Calcite was a dominant phase within the fine-grained groundmass and was generally highly granulated (not present as distinct grains). However, the calcite masses did tend to align with the foliation of the overall lithology. It contributes approximately 10% to the total rock.
Muscovite/Sericite	Sericite was present as extremely fine grains (less than 100 µm) with indiscriminate distribution throughout the groundmass. However, it still contributed a significant proportion to the total ore, approximately 20-25%.
Chlorite	Chlorite was present as fine subhedral grains (<200 µm) disseminated throughout the groundmass. Chlorite was a major mineral, forming approximately 15% of the total ore.
Pyrite	Pyrite was present in minor concentrations, as disseminations as well as with vein-hosted occurrences. The disseminations tended to be anhedral and finer-grained, ranging in size from approximately 100 to 400 µm, while the pyrite occurring within veins tended to be coarse and euhedral, with the largest individual grain approximately 2 mm in size. The larger, vein-hosted pyrite was sometimes found amalgamated, into larger, semi-amorphous masses.
Arsenopyrite	The sample contained minor to trace amounts of very fine-grained arsenopyrite which was typically associated with pyrite.

Chalcopyrite	The chalcopyrite was not abundant and when present it was extremely fine-grained.
Gold	Only minor amounts of gold are present, typically very fine, less than 50 µm in size. Gold was generally associated with sulphides, particularly pyrite and arsenopyrite.
Alteration minerals	The alteration minerals present in the Fairview sample included sericite and chlorite
Texture and structure	The sample tended to be fine-grained, with a distinct alignment of minerals most commonly visible in the lithic fragments of sedimentary quartz.
Veining	Quartz and calcite veins were abundant. These veins were commonly associated with mineralisation.

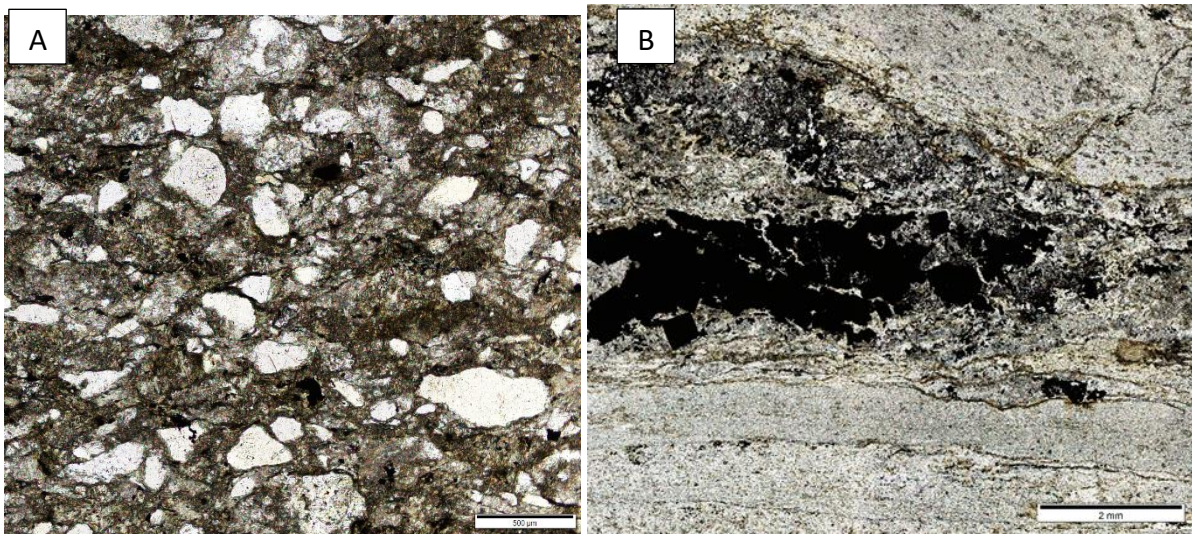


Figure 4-4 Transmitted light (PPL) images of Fairview material. A) Lithic quartz fragments with a fine-grained matrix of quartz, chlorite, dolomite, magnesite and to a lesser extent muscovite. B) Quartz and calcite vein crosscutting the fine grained micaceous schist. Subhedral to euhedral pyrite occurs within the vein (evident in transmitted light as the black mineral).

4.4. Barbrook

The Barbrook sample host lithology was predominantly an altered silicic carbonate, with extensive veins (Figure 4-5). The groundmass was comprised of broadly anhedral dolomite, fine-grained quartz fragments and finer chlorite, muscovite and carbon (Figure 4-6A). The extensive network of veins gave the appearance of a stockwork texture. The veins were predominantly carbonate with minor quartz (Figure 4-6B). The amount of iron-rich material present in the sedimentary units was variable and could range from highly abundant to entirely absent. Secondary iron oxides such as limonite and goethite were present. Sulphides evident in the Barbrook samples included pyrite and arsenopyrite, with minor chalcocite, galena and pyrrhotite.

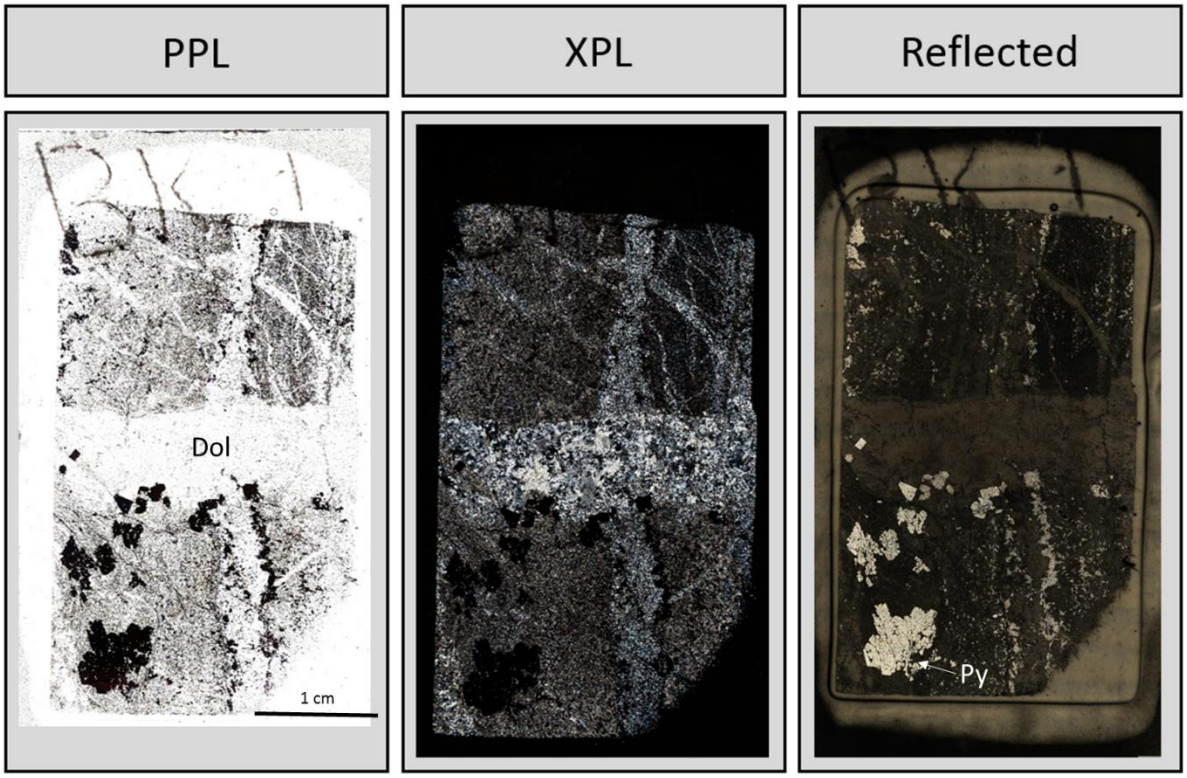


Figure 4-5 Transmitted (PPL and XPL) and reflected light images highlighting the overall mineralogy and texture of the Barbrook sample. The fine-grained groundmass common in this sample is evident, comprised of quartz, chlorite and muscovite. Dolomite (Dol) was abundant in the cross-cutting veins. Pyrite (py) is also present.

Calcite	Calcite contributed to a significant proportion of the mineralogy of the Barbrook ore. The calcite occurred either as fine-grained material within the groundmass or as larger anhedral grains present within, but notably different from, the finer groundmass. Calcite was also the main mineral associated with the extensive veining throughout the sample. In total the different types of calcite contributed approximately 60% to the total volume of the rock.
Quartz	Quartz occurred in a very similar format to that of the calcite, as fine grains in the matrix, and as larger crystals and veins. However, the quartz was less prevalent than calcite, accounting for approximately 30% of the total rock.
Chlorite	Chlorite was present as fine-grained disseminations throughout the groundmass (<200 µm). Chlorite distribution was variable but on average of formed approximately 5% of the ore.
Muscovite/sericite	Muscovite/sericite was present in only minor amounts, associated with chlorite and more altered units. It comprised less than 2% of the mineralised lithology.
Kaolinite	The kaolinite was very fine-grained and present in minor amounts. When present the kaolinite was typically associated with chlorite and muscovite.
Carbon	Carbon was present in trace amounts, generally as fine grains, associated with veins and/or highly altered zones (Figure 4-6).
Pyrite	Pyrite was present throughout the sample, in minor quantities. The pyrite occurred (in a very similar form to that previously described for Fairview) as both disseminations and within veins. The disseminations tended to be anhedral and fine-grained, ranging in size from 50 to 500 µm. However, the pyrite that was associated with veins tend to be euhedral and coarse with some of the grains reaching 5 mm in size.

Pyrrhotite	Pyrrhotite generally had a variable distribution. When present it was typically fine-grained, occurring in veins and associated with pyrite. Pyrrhotite was present in trace quantities.
Arsenopyrite	Trace arsenopyrite was present as fine masses, occurring within or associated with veins as well as with pyrite.
Chalcocite	Trace chalcocite was present as fine masses, associated with veins, as well as pyrite and arsenopyrite.
Galena	Trace amounts of galena were present throughout the sample.
Alteration minerals	The alteration minerals sericite, chlorite and kaolinite were present in minor quantities and were homogenously distributed.
Texture and structure	The sample was generally fine-grained, with a distinct stockwork texture due to the abundance of veining.
Veining	The stockwork texture that was evident in the sample material was generated by significant quartz and calcite veins, and commonly associated with mineralisation. There were several stages of calcite vein formation, both early and late, while the quartz formed at an intermediate stage.

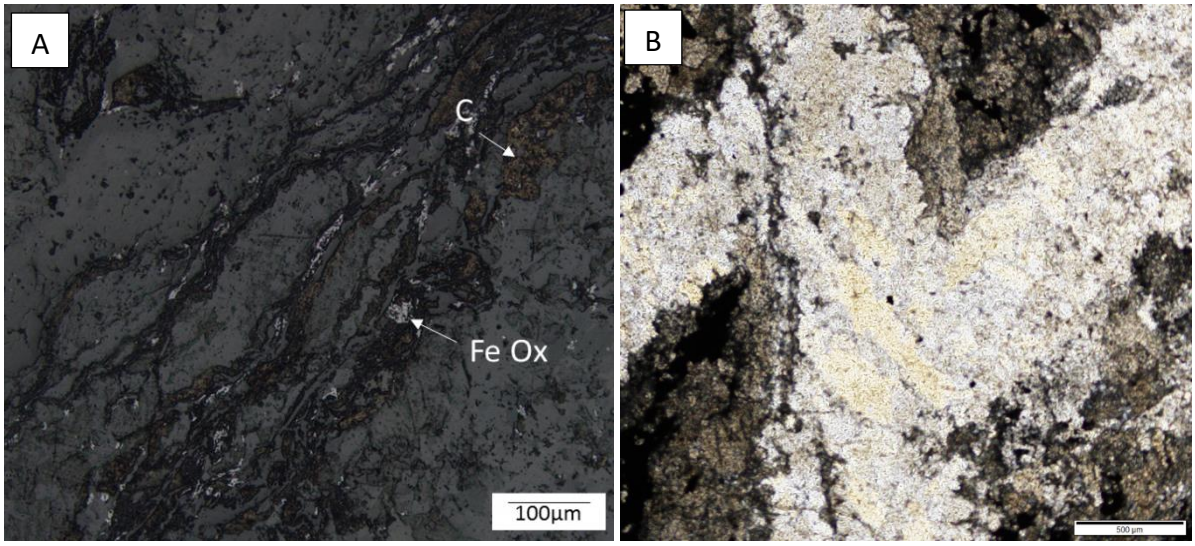


Figure 4-6 Microphotographs from the Barbrook sample. A) Example of carbon (C) found in Barbrook, associated with iron oxides (Fe Ox). B) One of many calcite veins within the Barbrook sample. These calcite veins are present in both early and late stages.

4.5. Consolidated Murchison

The mineralisation within the Consolidated Murchison sample was hosted by dominantly carbonate and quartz-bearing schist, with chlorite, biotite, magnesite and locally abundant tourmaline (Figure 4-7, Figure 4-8A). Ore minerals present included stibnite (Sb_2S_3), pyrite, berthierite ($FeSb_2S_4$), and lesser arsenopyrite (Figure 4-8B). The mineralisation was generally present as homogenous disseminations throughout the schist.

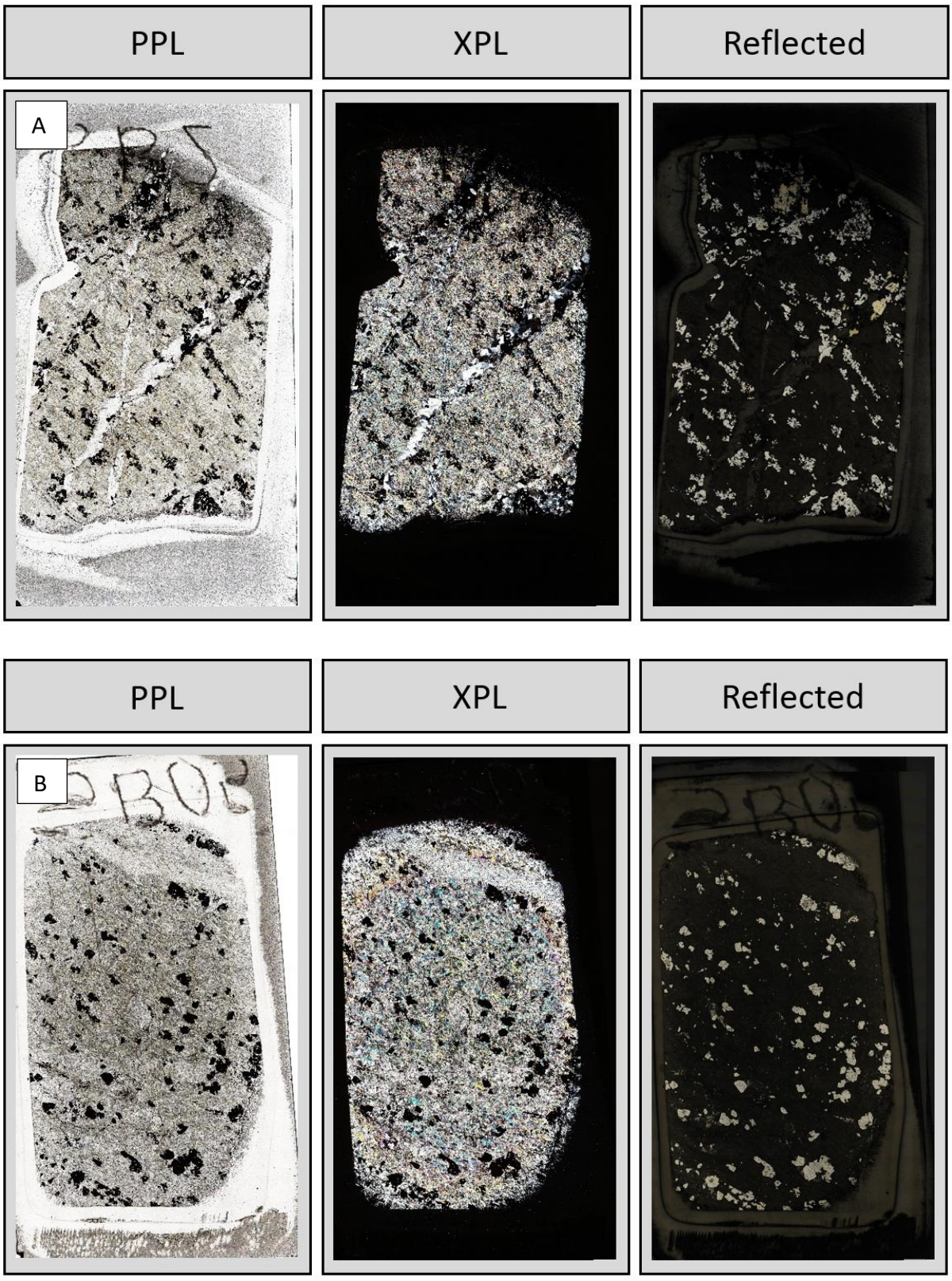


Figure 4-7 Transmitted (PPL and XPL) and reflected light images highlight the overall mineralogy and texture of Consolidated Murchison. A) Photomicrograph of the schist that is commonly host to the mineralisation B) Less prominent alignment of the micas found in this sample. The majority of the minerals comprising these samples cannot be determined at on overview scale due to its fine-grained nature. Pyrite is visible in reflected light, distributed homogenously across much of the sample.

Magnesite — Magnesite was the dominant carbonate mineral present in the Consolidated Murchison sample. It tended to be cryptocrystalline and contributed significantly (approximately 40%) to the total gangue. The magnesite was associated with fine-grained quartz.

Quartz — The quartz occurred as grains throughout the lithology, as well as in lesser amounts of vein quartz. However, the quartz was a less prevalent component than magnesite, forming approximately 20% of the total ore.

Tourmaline — Tourmaline was locally abundant, with concentrations up to 5% of the overall sample. The tourmaline occurred as elongate laths, with lengths reaching 2.5 mm. The tourmaline grains occurred with the elongate axis aligned along the trend of schistosity.

Biotite — Biotite was present in minor amounts (5%). It was generally fine- grained (<100 µm), subhedral and aligned to form schistosity.

Chlorite — Chlorite was present as fine grains (<150 µm), disseminated homogeneously throughout the ore. Chlorite was associated with biotite and contributes approximately 5% of the total sample material.

Stibnite — Stibnite was present throughout the ore in moderate quantities (5%). The mineralisation of stibnite (as well as subsequent ore minerals described) tended to occur as homogeneous disseminations. In contrast to Fairview and Barbrook there was no increased mineralisation in veins. The stibnite grain size range was bimodal, with a coarser class ranging 0.5 mm to 2 mm that had a sub-to-euhedral habit, and a finer size fraction that ranged from 50 µm to 200 µm with a sub-to-anhedral habit.

Pyrite — The pyrite was associated with stibnite, with similar abundances. Pyrite tended to be subhedral, ranging in size from 50 µm to 1.5 mm.

Arsenopyrite — The arsenopyrite was typically anhedral and less than 1 mm in size, with a strong association with pyrite.

Berthierite	Only limited amounts of berthierite were present. Grains typically had a subhedral habit, with a range in size from 0.5 mm to 1 mm. The berthierite was commonly associated with stibnite.
Alteration minerals	The alteration minerals in the Consolidated Murchison sample included biotite and chlorite.
Texture and structure	The predominant texture of the Consolidated Murchison sample was fine-grained schist with crosscutting networks of quartz veins.
Veining	A stockwork texture was generated by significant quartz and calcite veins, associated with mineralisation.

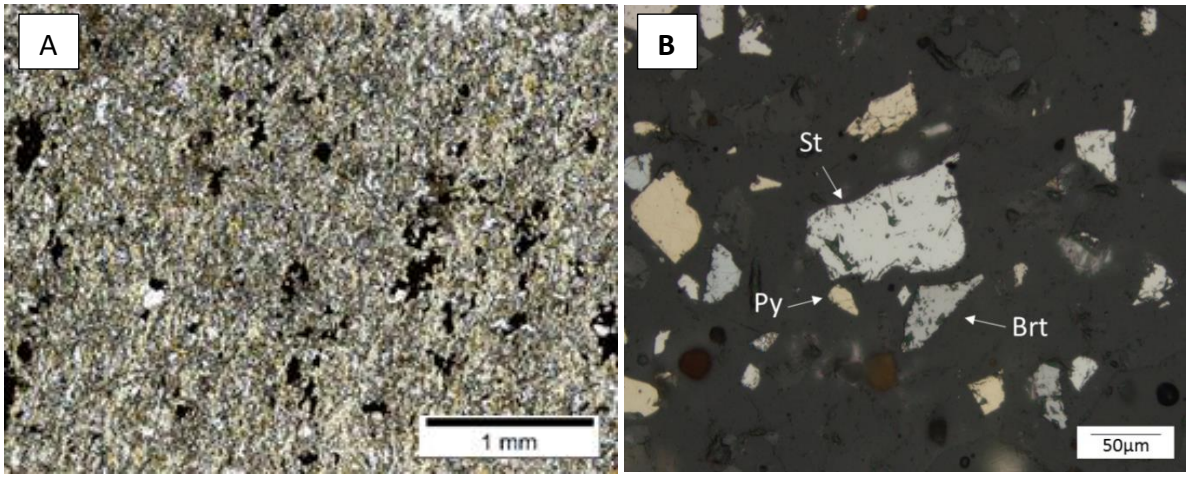


Figure 4-8 Microphotographs of typical components of the Consolidated Murchison sample. A) Fine grained quartz carbonate schist, with chlorite, biotite and magnesite. B) An example of mineralisation at Consolidated Murchison including stibnite (St), pyrite (Py) and berthierite (Brt).

4.6. Pilgrim's Rest

The Pilgrim's Rest sample host lithology was variable with fine-grained siltstones and shales, which were commonly associated with fine calcite and muscovite. All original lithologies were crosscut by quartz/carbonate veins (Figure 4-9). In the region the ore was sampled, the amount of quartz veins and associated pyrite mineralisation was significantly more abundant than the original host lithologies (Figure 4-10A). The sulphide mineralogy included abundant pyrite with lesser pyrrhotite, chalcocite and chalcopyrite (Figure 4-10B). Gold, when present, was

associated with these sulphides. Pyrite represented approximately one third, by mass, of the feed. The mineralisation occurred extensively within veins, and to a lesser extent as disseminations and even as massive mineralisation.

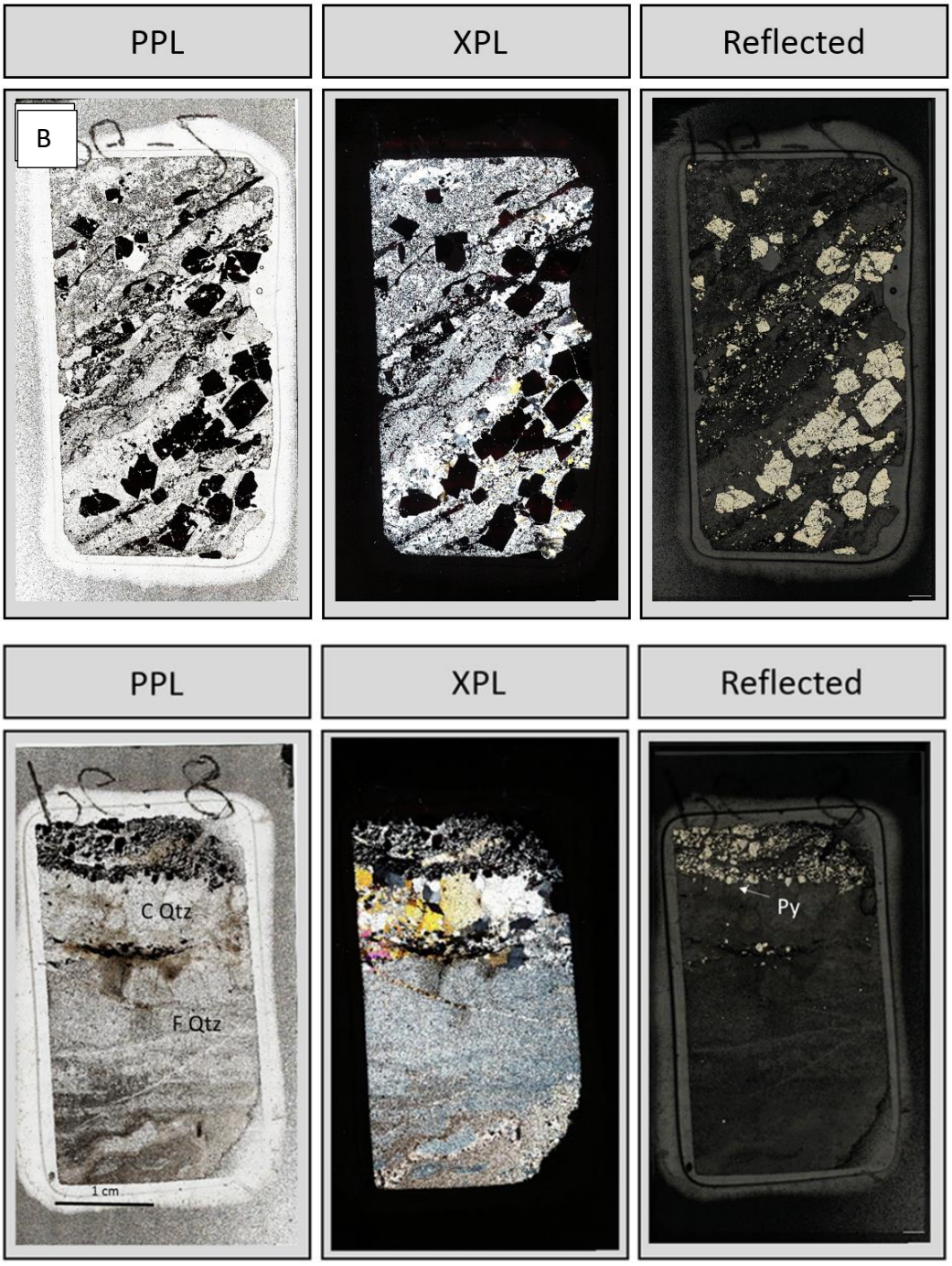


Figure 4-9 Transmitted (PPL and XPL) and reflected light images highlight the overall mineralogy and texture of the Pilgrim's Rest sample. A) The Pilgrim's Rest sample has abundant quartz-rich, pyrite-bearing vein that spans the entire sections. B) The sedimentological units found in Pilgrim's Rest include both siltstones comprising of coarse quartz (C qtz) and shales with fine quartz (F qtz) as well as other minerals such as calcite and muscovite.

Quartz	The quartz was abundant and occurred in two forms; 1) as crosscutting veins, associated with mineralisation. This quartz tended to be coarse, angular, interlocking grains, ranging in size from 0.1 mm to 0.5 cm. 2) Fine-grained quartz which occurred in the sedimentary units. Fine quartz was also commonly associated with alteration minerals such as chlorite or sericite.
Pyrite	Pyrite was present in unusually large quantities, associated with veins in addition to occurring as disseminations. The pyrite crystals ranged in size from 5 mm to 50 µm and tend to be euhedral to subhedral. The pyrite was commonly closely related to quartz veins, however when abundant (massive) pyrite is present this relationship with quartz is not evident.
Dolomite	The dolomite/calcite was present in both the metasedimentary facies units and also as cross cutting veins. The calcite typically occurred as either fine-grained masses or larger crystals reaching approximately 2 mm.
Muscovite	Very fine muscovite was present, this was generally associated with carbonates. The muscovite typically occurred as interstitial material.
Chalcopyrite	Minor chalcopyrite was present, commonly associated with pyrite. The chalcopyrite tended to be anhedral and fine-grained (50 µm to 300 µm).
Pyrrhotite	Minor pyrrhotite was present, typically anhedral, with a diameter of 100-300 µm.
Gold	Visible gold occurred as discrete, fine grains, ranging in size from 1.5 µm to 75 µm.
Alteration minerals	Muscovite/sericite alteration was present in addition to the silicification and carbonatisation phases. Extensive veins were associated with pyrite development.

Texture and structure	Sedimentary layering was a predominant feature in the Pilgrim's Rest sample, as was the extensive veination.
Veining	Extensive quartz and calcite veination was evident. Associated with the veins was the occurrence of pyrite

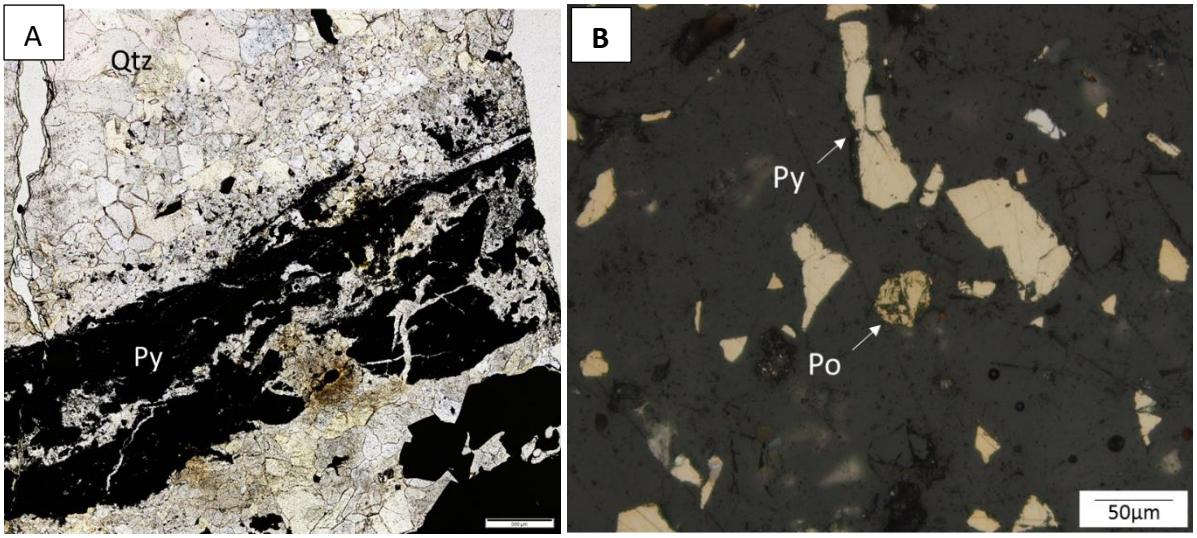


Figure 4-10 Microphotograph of certain features within the Pilgrim's Rest samples. A) A transmitted PPL image of a vein with ubiquitous quartz (Qtz) and lesser pyrite (Py- black) that emphasises the abundance of quartz veins, which are commonly present to a greater degree than the host lithologies, within the Pilgrim's Rest sample. B) Reflected light mineralisation at Pilgrim's Rest, extensive pyrite (Py), and lesser pyrrhotite (Po).

4.7. Summary

Petrography was an important feature in this study as it allowed for the characterisation of gangue minerals, particularly those with the potential to interfere with gold extraction. Additionally, it highlighted the mineralogical differences between samples. The thin sections were examined using a petrographic microscope under transmitted and reflected light.

The VCR sample was a matrix-supported conglomerate. The conglomerate was polymict, with clasts of both quartzites and vein quartz, and significantly lesser chert pebbles. The matrix was dark green to black, and was composed of fine, rounded to sub-angular quartz grains, with interstitial chloritic and sericitic material. Metallic minerals that were present included pyrite, pyrrhotite and minor chalcopyrite, as well as gold.

The mineralisation found in the Fairview sample was hosted within an altered micaceous schist as well as quartzites. The schist contained a fine-grained matrix of quartz, chlorite, dolomite, magnesite and to a lesser extent muscovite. Ore minerals were present in either veins/fractures (typically large and euhedral), or as disseminations within the main lithology (commonly fine and sub- to anhedral). Metallic based minerals that were present included pyrite, arsenopyrite and minor visible gold. Significant veining was present, commonly closely associated with mineralisation. Veins were generally either quartz or calcite.

The Barbrook sample host lithology was predominantly an altered silicic carbonate, with extensive veins. The groundmass was comprised of broadly anhedral dolomite, fine-grained quartz fragments and finer chlorite, muscovite and carbon. The veins present were predominantly carbonate with minor amounts of quartz. The sulphides that were evident in the Barbrook samples include pyrite, arsenopyrite, with minor chalcocite, galena and pyrrhotite.

The mineralisation within the Consolidated Murchison sample was hosted by dominantly carbonate and quartz-bearing schist, with chlorite, biotite, magnesite and locally abundant tourmaline. Ore minerals that were present in the Consolidated Murchison sample include stibnite (Sb_2S_3), pyrite, berthierite ($FeSb_2S_4$), and lesser arsenopyrite.

The Pilgrim's Rest sample host lithology was variable with fine-grained siltstones and shales, commonly associated with fine calcite and muscovite. All original lithologies were crosscut by quartz/carbonate veins. The sulphide mineralogy included abundant pyrite with lesser pyrrhotite, chalcocite, chalcopyrite and these sulphides tended to be associated with native gold.

Chapter 5

XRD and Assay

5.1. Introduction

Understanding the chemical nature of the ore and particularly the gangue component is important for predicting the feasibility of leaching. XRD is used to help provide insight into the major mineralogical composition of the ore. This method is important, in conjunction with other mineralogical determinations, as it allows for the identification of all minerals present at concentrations greater than 1%. XRD is particularly relevant in the identification of minerals that may be too fine-grained, uncommon or indistinguishable from other minerals to be identified during optical microscopy. XRD analyses fine, homogenised material which allows for the determination of the bulk mineralogical composition of the sample. This method provides a more representative mineralogical understanding than many other techniques.

The presence of certain minerals will have varying effects on mechanisms throughout the extraction process. In the context of leaching, the presence of carbonates, sulphides and clays are all potentially deleterious, thus precise identification is important. Additionally, XRD can quantify the mineralogy of the residues after leaching, which makes this method extremely valuable for determining the effects of leaching on the ore, one of the main objectives of this study.

The assays utilised in this work includes the analyses for gold using fire assay collection and AAS (atomic absorption spectroscopy), an accepted industry standard. Additional assays, such as X-Ray Fluorescence (XRF), Inductively Coupled Plasma Mass Spectrometry (ICP-MS), Inductively Coupled Plasma Atomic Emission Spectroscopy (ICP-OAES) were utilised to supplement mineralogical data and to give a further indication of the elements likely to be found in later leaches. XRF provides information regarding the major elements (SiO_2, Al_2O_3), while ICP-MS/AES is relevant to respective trace elements (Cu, Ni, Pb, Zn, Sb, etc.)

5.2. XRD

The XRD analyses were run on the respective gold-bearing samples to determine the mineral compositions of each deposit. This acts as an inexpensive and convenient way to identify major

minerals present. This is particularly relevant for minerals present that might negatively affect the leaching process.

XRD uses scattered X-Rays to determine the spacing between crystal lattices, employing wave interference patterns generated by crystals responsible for scattering the X-Rays. XRD is based on the principles of Bragg's Law/Equation (Equation 5.1) where *n* represents an integer, d the spacing of the crystal lattice, λ wavelength of the X-Ray, θ the angle between the incident X-Ray and the plan of atoms (Kraus and Hunt, 1920).

$$n\lambda = 2\mathrm{d}\sin\theta \qquad \text{(Equation 5.1)}$$

The wavelength of X-Rays tends to be constant, while individual minerals in a sample will have an almost unique d-spacing from which they can be identified.

An X-Ray diffractometer comprises a number of components, the basics of which include an X-Ray source, collimators (to convert X-Rays to parallel beams), a sample holder and a detector (Speakman, 2016). The X-Rays produced by the source are scattered by the sample material, generating wave interference patterns (a plot of X-Ray intensity vs θ)(Speakman, 2016). From these interference patterns, minerals are identified by comparison with a standard database of mineral powder diffraction files (PDFs). This database is maintained by the International Centre for Diffraction (previously known as the JCPDS) (Stanjek and Häusler, 2004).

Analyses were done using a Co-Kα radiation source, as commonly the iron-rich nature of the ores causes X-ray fluorescence inducing high background levels when Cu-radiation is used (Coetzee et al., 2011). XRD is important for analysis of clays and other phyllosilicates that may hinder the leaching processes (Schnitzer et al., 1988). All XRD analyses were performed using a Panalytical diffractometer. XRD analyses were done on both the composite sample feed, gravity concentrate and the leached residue. However only the composite sample and gravity concentrate will be discussed in this chapter, whilst the results for the leach residue are found in Chapter 10.

While the XRD is an excellent tool for phase identification it can also be used to extract further information using additional features such as peak intensity and width. A method known as Rietveld refinement is used for approximate quantitative analysis using the XRD patterns. Rietveld refinement is a curve-fitting procedure, which uses a series of parameters and the least

squares method to generate a theoretical curve close to that of the observed curve (Speakman, 2016). Rietveld refinement requires all minerals present to be identified, after which all structural information (e.g. unit cell dimensions) for respective minerals can be gathered (Young, 1993). By adjusting the concentration of individual minerals, in conjunction with the known structural data for individual minerals, the observed diffraction pattern can be replicated. The concentrations that produce a theoretical pattern that is the most similar to the real diffraction patterns are output as the determined concentration (Dreele, 2003). In addition to structural data, factors such as background interference and peak broadening due to equipment must also be accounted for (Young, 1993; Speakman, 2016).

Both the feed and the gravity concentrate underwent XRD analyses. The composite feed was pulverised to ensure the best possible results. However, the gravity concentrate was not pulverised as there was a limited amount of sample material and the material used for XRD was subsequently intended for petrographic study. The pulverisation of the concentrates would prevent the gold used for characterisation of being representative of the material used for leaching.

5.2.1. Results

Only major mineral phases present in composite samples of respective deposits have been determined from XRD. A lower detection limit of 1% is typically accepted for XRD analyses. The assay data have been used to confirm the results of the XRD data as far as possible using calculated mineral abundances.

From the assorted peaks at respective d-spacings, several minerals have been identified from the VCR sample diffractogram (Figure 5-1). VCR feed and concentrate material had all the same mineral phases; quartz, chlorite, calcite, pyrite and pyrrhotite. There was a major increase in the sulphide abundances after undergoing the concentration process, as evident in the minerals such as pyrite and pyrrhotite.

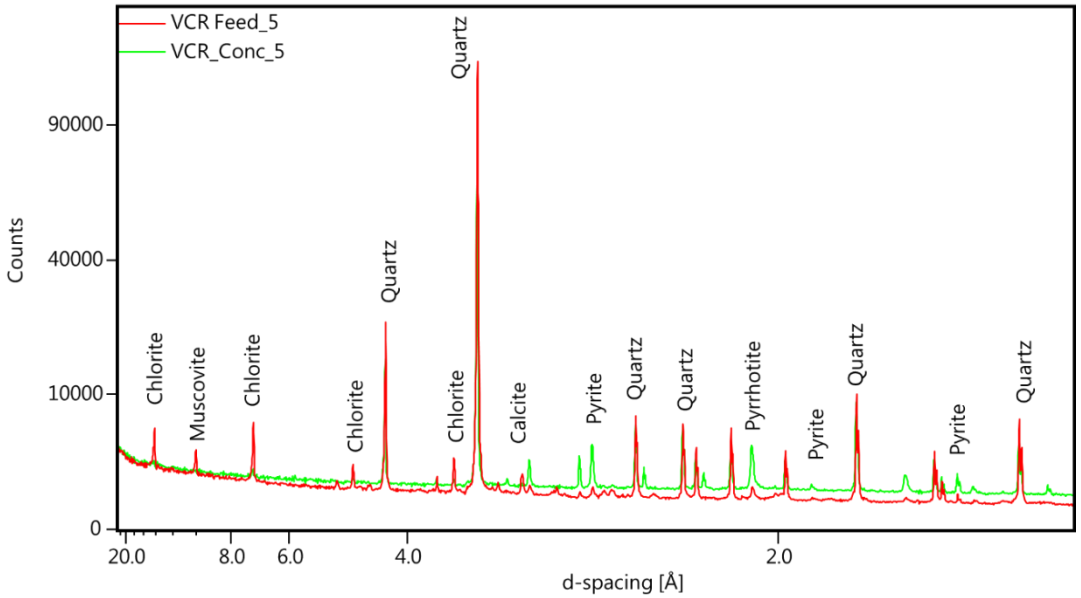


Figure 5-1 X-Ray diffractograms for the Ventersdorp Contact Reef feed (red) and gravity concentrate (green). The gravity concentration process increased the amount of pyrite and pyrrhotite from the feed to the concentrate.

The XRD pattern generated from the Fairview sample (Figure 5-2), shows the presence of quartz, dolomite, chlorite, muscovite, magnesite, pyrite and arsenopyrite. The minerals that increased in abundance in the concentrate in the Fairview sample included pyrite and arsenopyrite.

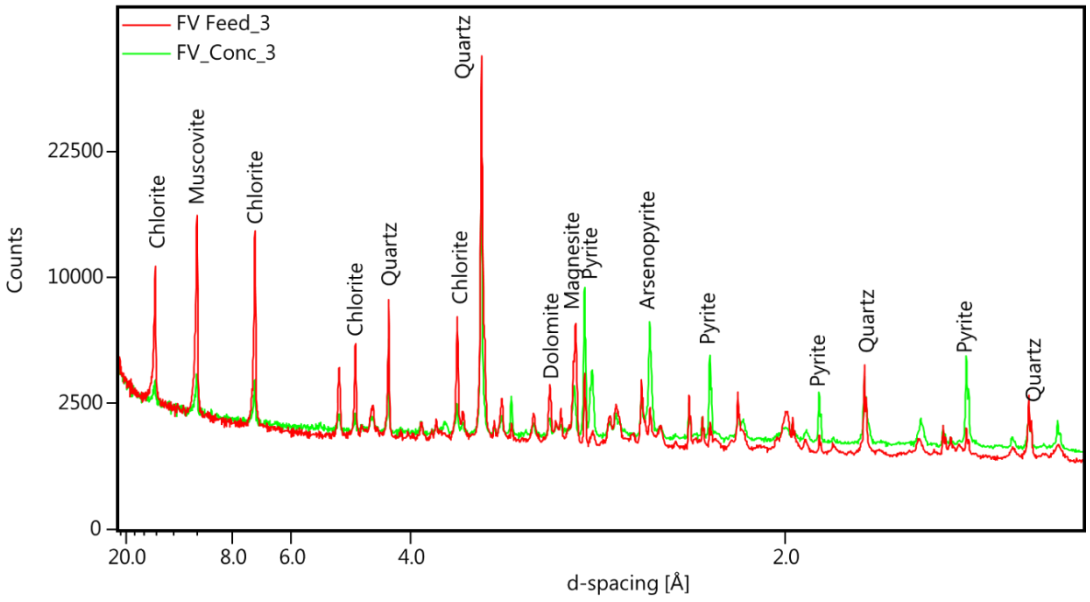


Figure 5-2 X-Ray diffractograms for Fairview feed (red) and gravity concentrate (green). The minerals that were upgraded in the gravity concentrate include pyrite and arsenopyrite.

The Barbrook sample feed (Figure 5-3) contained dolomite, quartz, chlorite, pyrite, muscovite, calcite and kaolinite. There was sufficient upgrading of heavy minerals in the concentrate that siderite and pyrrhotite were present in the XRD pattern of the concentrate but not in the feed.

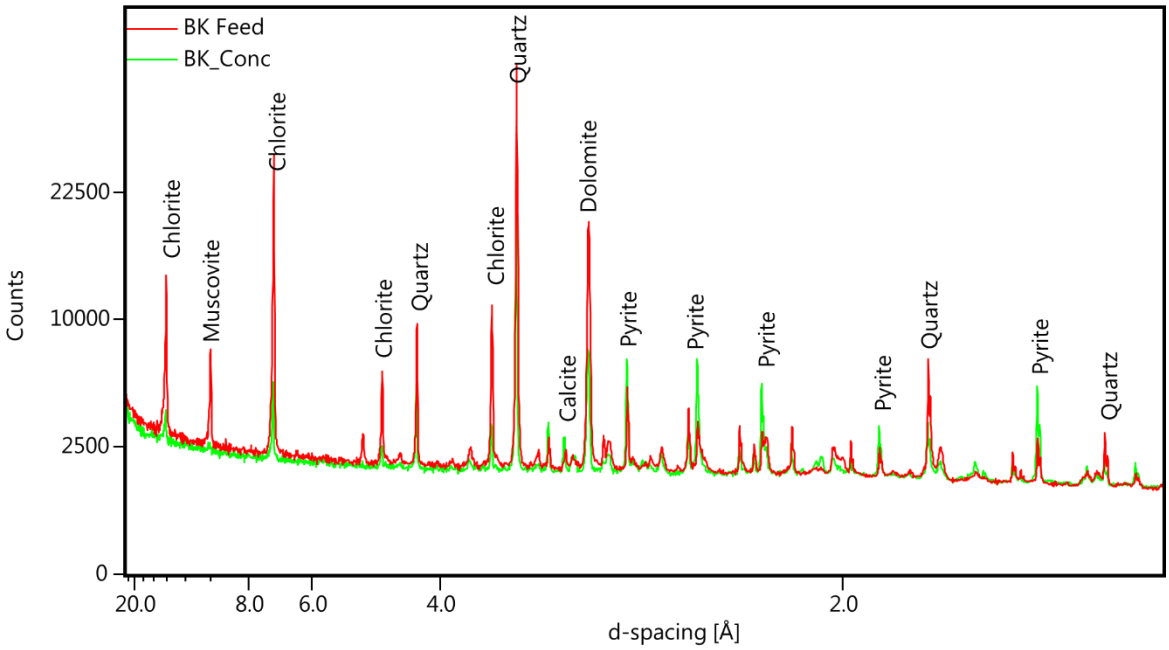


Figure 5-3 X-Ray diffractograms for Barbrook feed (red) and gravity concentrate (green). The amount of pyrite present has increased from the feed to the concentrate.

The Consolidated Murchison sample had a large number of mineral phases present, including chlorite, biotite, talc, quartz, berthierite, stibnite, dolomite, magnesite, tourmaline and arsenopyrite (Figure 5-4). The XRD pattern for the concentrate showed an increase in arsenopyrite and pyrite.

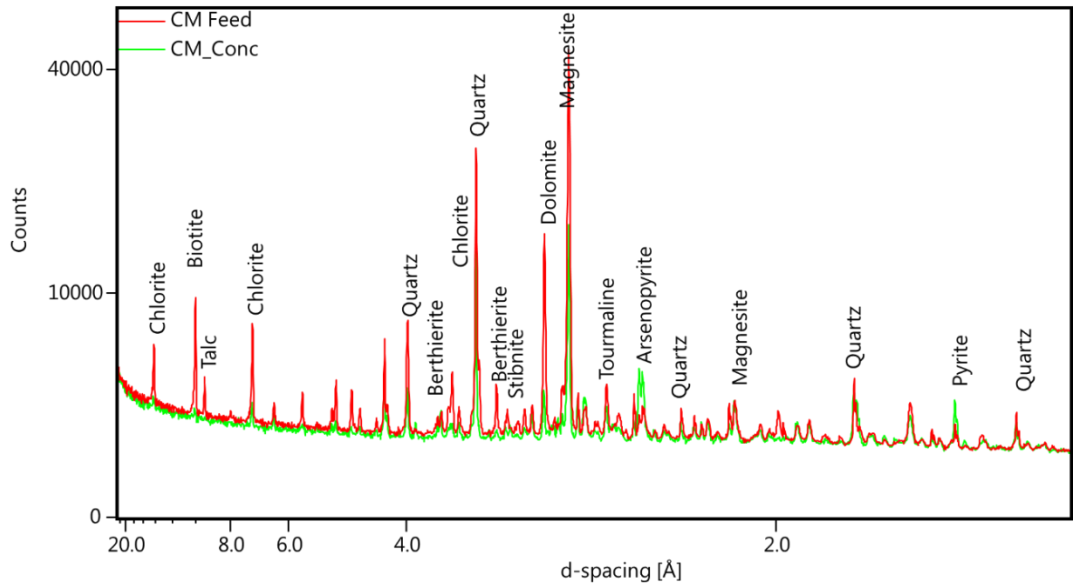


Figure 5-4 X-Ray diffractograms for Consolidated Murchison feed (red) and gravity concentrate (green). The diffractogram shows the increase in arsenopyrite and pyrite in the gravity concentrate.

From the diffractogram for the Pilgrim's Rest sample (Figure 5-5) major minerals present included quartz, pyrite, dolomite, chalcopyrite and muscovite. The pyrite was upgraded in the gravity concentrating process.

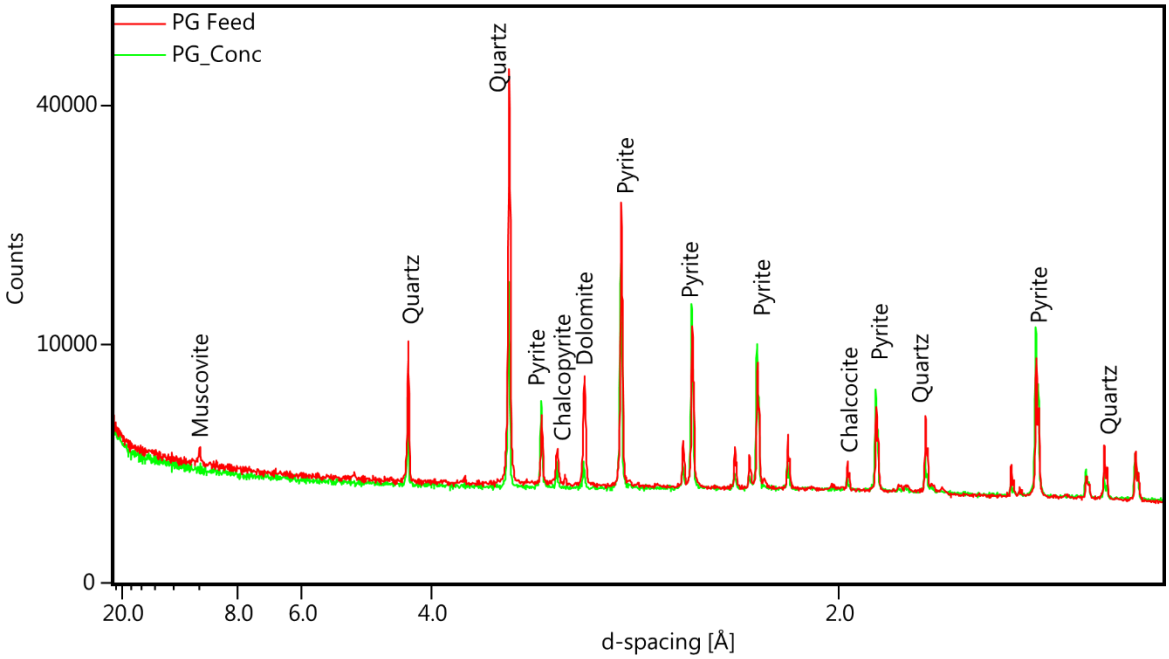


Figure 5-5 – X-Ray diffractograms for Consolidated Murchison feed (red) and gravity concentrate (green). There is a significant increase in the amount of pyrite present in the concentrate relative to the feed.

The information generated using Rietveld analyses, found in Table 5-1, gives mineral mass percentage determined by XRD within the composite sample. Unfortunately the number of minerals present in the Consolidated Murchison sample prevented the software from accurately obtaining quantitative analyses and therefore these results have not been included in the main table, however approximate values are found in Table 5-2. The results found in Table 5-1 and Table 5-2 have been confirmed, as far as possible, by calculating the mineral abundances, using a combination of assay results and minerals phases present (known from XRD). While this method can act as confirmation for the Rietveld analyses, it has serious limitations when multiple minerals that contain the same elements are present.

From the mineral mass of both the feed and concentrate it is interesting to note the minerals that were upgraded, and the degree to which this was done. This is also noticeable in the diffractograms, where denser mineral (e.g. pyrite) peaks are higher in the concentrate than the feed, and vice versa for less dense minerals (e.g. muscovite).

The relative increase in quartz and muscovite in the VCR sample XRD data was due to the decrease in both pyrite and arsenopyrite and therefore an overall decrease in sample mass. The gravity concentration process is equally evident in the Fairview sample where pyrite, arsenopyrite and magnesite were increased in the sample. The Barbrook sample gravity concentrate was upgraded in pyrite, siderite and pyrrhotite, with pyrrhotite and siderite increased from below the limit of detection in the feed. Pilgrim's Rest material showed a significant increase in pyrite from 30% in the feed to 60% in the concentrate. The mineral changes from feed to concentrate for the Consolidated Murchison sample could not be quantified successfully.

Table 5-1 Mineral quantities for respective deposits, both feed and gravity concentrate, based on Rietveld analyses, XRD. Sulphides such as pyrite, pyrrhotite and arsenopyrite generally have increased abundance in the concentrate relative to the feed.

Rietveld Analyses		
	VCR Feed	VCR Conc
Mineral	Mass %	Mass %
Quartz	96.9	88
Muscovite	1.1	-
Chlorite	1	1.4
Calcite	0.4	-
Pyrrhotite	0.4	7.1
Pyrite	0.3	3.5
	Fairview Feed	Fairview Conc
Mineral	Mass %	Mass %
Quartz	41.1	37.7
Muscovite	24.8	10.6
Chlorite	13.4	10
Dolomite	10.8	3.2
Magnesite	7.1	13.7
Pyrite	2.7	16.3
Arsenopyrite	0.2	8.6
	Barbrook Feed	Barbrook Conc
Mineral	Mass %	Mass %
Dolomite	58	41.4
Quartz	29.8	28
Chlorite	6.4	10.6
Pyrite	3.4	15.3
Muscovite	1.7	1.8
Calcite	0.4	1.6
Kaolinite	0.3	1.2
	Pilgrim's Rest Feed	Pilgrim's Rest Conc
Mineral	Mass %	Mass %
Quartz	60.5	32.6
Pyrite	32.6	60
Chalcopyrite	1.2	1.1
Dolomite	4.9	6.3
Muscovite	0.8	

Table 5-2 Mineral quantities for Consolidated Murchison feed on XRD analyses. Antimony minerals that are present in sufficient concentrations to be identified in XRD include stibnite and berthierite.

Cons Murch Feed	
Mineral	Mass %
Quartz	38
Magnesite	24
Tourmaline	8
Stibnite	6
Arsenopyrite	6
Pyrite	4
Dolomite	4
Chlorite	4
Biotite	4
Berthierite	1

5.3. Assay

Several methods were used to determine the chemical nature of the samples, but the main techniques used included XRF, ICP-MS/OES and AAS.

Results from the assays have been confirmed using standards, blanks and repeats, which were always found to be within an acceptable range of the given/expected values, with a single minor exception. Examples of standards used include SARM5, SARM33, GGC08, AMIS0377, AMIS0441, AMIS0381, GS903-4, GGC-09. These standards vary in concentrations, lithologies, company and country of origin and therefore represent a wide variety from which to confirm the quality of data.

The assays were undertaken to determine the presence of oxides using XRF (Table 5-3) and metallurgically significant elements, using ICP MS/OES, examples of which include As, Cu, Co, Ni, Sb, Zn and Pb (Table 5-4). The major element compositions have been determined to help confirm major mineralogy, particularly in conjunction with XRF. The metallurgically significant concentrations cannot, on their own, identify definite leach issues, they act as indicators. Elevated concentrations include Cu and Zn concentrations of the Barbrook sample, the Ni and Sb concentration of the Fairview and Consolidated Murchison samples respectively, as well as the Cu concentrations of the Pilgrim's Rest sample. The assays have also provided gold

concentrations of the samples which range from 30 g/ton (Pilgrim's Rest) to 1.6 g/ton (Barbrook) which will be discussed more intensively in Chapter 6.

Table 5-3 XRF results for the respective deposits, providing major oxide composition for the Ventersdorp Contact Reef (VCR), Fairview (FV), Barbrook (BK), Consolidated Murchison (CM) and Pilgrim's Rest (PG). LDetection is the lower detection limit of the analyses while Udetection is the upper detection limit.

WITS - 001	Al_2O_3	SiO_2	Fe_2O_3	CaO	MgO	MnO	K_2O	Na_2O	TiO_2	P_2O_5	V_2O_5	Cr_2O_3	LOI
METHOD	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V
LDETECTION	0.05	0.05	0.01	0.01	0.05	0.01	0.01	0.05	0.01	0.01	0.01	0.01	-50
UDETECTION	100	100	100	100	100	100	100	100	100	100	100	100	100
UNITS	%	%	%	%	%	%	%	%	%	%	%	%	%
VCR	3.52	87.9	4.6	0.82	1.50	0.04	0.73	0.10	0.20	0.033	<0.01	0.11	1.75
FV	14.0	53.0	10.5	1.01	6.59	0.22	3.72	0.30	0.70	0.096	0.03	0.18	7.91
BK	5.94	40.0	21.8	6.87	3.62	0.44	0.77	0.20	0.21	0.071	0.01	0.11	12.25
CM	8.58	32.7	13.7	2.12	9.44	0.19	0.35	0.63	0.32	0.053	0.02	0.21	17.54
PG	0.56	35.7	36.5	1.30	0.82	0.05	0.18	<0.05	0.03	<0.01	<0.01	0.09	18.7

Table 5-4 Assay results composition for the Ventersdorp Contact Reef (VCR), Fairview (FV), Barbrook (BK), Consolidated Murchison (CM) and Pilgrim's Rest (PG). Elements assayed for include those that have the potential to affect the efficiency of gold extraction through cyanidation, such as Cu, Ni, Co, Ni and Pb. Additional data associated with assays such as blanks, standards and repeats are in Appendix A. LDetection is the lower detection limit of the analyses while Udetection is the upper detection limit.

HEAD ASSAY RESULTS													
WITS - 001	Au	S	S^{2-}_S	C	ORG C	As	Cu	Fe	Co	Ni	Sb	Zn	Pb
METHOD	FAA303	CSA06V	CSA08V	CSA01V	CSA03V	AAS11C	ICP90A	ICP90A	ICP90A	ICP90A	ICP90A	ICP90A	ICP90A
LDETECTION	0.02	0.01	0.01	0.01	0.05	0.01	10	0.01	10	10	50	10	20
UDETECTION	100	100	30	73.8	40	5	100000	30	100000	100000	100000	100000	100000
UNITS	G/T	%	%	%	%	%	PPM	%	PPM	PPM	PPM	PPM	PPM
VCRA	22.2	1.19	1.09	0.31		0.09	139	-	43	228	-	75	193
VCRB	25.6												
VCRC	23.0												
FVA	12.4	2.86	2.76	1.27		0.81	84	-	50	491	-	124	<20
FVB	12.5												
FVC	12.6												
BKA	1.7	8.06	-	3.94	0.23	0.23	597	15.3	-	472	65	2330	579
BKB	2.4												
BKC	2.2												
CMA	3.1	6.29	-	3.55	0.13	2.96	164	10.2	-	700	70100	98	87
CMB	3.0												
CMC	3.1												
PGA	30.3	29.6	-	0.70	0.14	0.35	7230	25.5	-	84	931	47	108
PGB	28.5												
PGC	29.4												

5.4. Discussion

The chemical and mineralogical quantification of the samples using XRF and ICP analyses, as well as XRD and petrography was important in order to understand the nature of the gangue and mineralisation.

A comparison of the data obtained from optical petrography, XRD and assay allowed for the identification and confirmation of all phases present.

Abundant minerals identified in VCR material included quartz, muscovite, chlorite, calcite and base metal sulphides such as pyrite and pyrrhotite, additionally minor amounts of chalcopyrite, sphalerite and galena were located. Phases such as quartz, muscovite, chlorite, dolomite, magnesite, pyrite and arsenopyrite (in addition to trace chalcopyrite) have been identified in Fairview while Barbrook is host to dolomite, quartz, chlorite, pyrite, muscovite, calcite, kaolinite, minor base metal sulphides and preg-robbing carbon. Consolidated Murchison is host to several base metal sulphides such as arsenopyrite, pyrite, berthierite, and stibnite in addition to magnesite, tourmaline, quartz, dolomite, chlorite and biotite. Pilgrim's Rest is host to quartz, pyrite, chalcopyrite, dolomite and muscovite.

From the characterisation of the gangue mineralogy, several phases were identified as potentially deleterious to leaching based on literature (Table 5-5). Within the samples, phases of concern relating to cyanide leaching identified include pyrrhotite, arsenopyrite, antimony sulphides and carbon. Phases of concern for chloride-hypochlorite and thiourea leaching include pyrrhotite, arsenopyrite, antimony sulphides, pyrite and chalcopyrite. The thiosulphate leaching system is predicted by literature to be sensitive to pyrrhotite, arsenopyrite and antimony sulphides.

Table 5-5 Summary of the phases present in the samples that have been identified in literature as potentially deleterious.

Lixiviant	Predicted Deleterious Mineral Groupings	Potentially Deleterious Minerals Identified in Samples
Cyanide	**Base metal sulphides (particularly Cu, As, Sb bearing minerals)**	Pyrrhotite
		Arsenopyrite
		Antimony sulphides
	Carbon	Carbon
Chloride-Hypochlorite	**Sulphides**	Pyrrhotite
		Arsenopyrite
		Antimony sulphides
		Pyrite
		Chalcopyrite
Thiourea	**Sulphides**	Pyrrhotite
		Arsenopyrite
		Antimony sulphides
		Pyrite
		Chalcopyrite
Thiosulphate	**Base metal sulphides**	Pyrrhotite
		Arsenopyrite
		Antimony sulphides

Chapter 6

Gold Mineralisation

6.1. Introduction

The chemical and physical properties of gold within a specific ore are determined and characterised using standardised features such as gold mineralogy, grain size, association, liberation and exposure. The modal mineralogy can be determined along with these gold characterisations. Gold mineralogy relates to the mineral in which the gold occurs. Commonly gold is found in its native metallic form, but in more complex instances gold can occur as gold minerals such as petzite (Ag3AuTe2), sylvanite ((AgAu)Te2), or aurostibite (AuSb2). Gold can also be locked within other minerals as solid solution or inclusions within arsenopyrite. The mineralogy of the gold will strongly control if leaching is feasible.

Gold grain size distribution within an ore is also vital information to ensure effective processing. In particular, milling and leaching durations will be affected by the grain size distribution to allow for maximum recovery. Minerals that are associated with gold will affect the response to certain processing methods, for example with leaching, the association with an oxygen consumer such as pyrrhotite would be deleterious. Liberation refers to the area percent of the mineral of interest, relative to the area of the total host particle. Liberation is relevant to physical separation processes such as gravity concentration. Exposure refers to the percentage of the grain that has a free, exposed surface and quantifies the amount of gold exposed to lixiviants that would be available for leaching. Most of these characteristics were acquired using the mineral liberation analyser (MLA), while the gold mineralogy was determined using the electron probe micro analyser (EPMA).

6.2. Mineral Liberation Analyser (MLA)

Gold from the samples of study were characterised using a Mineral Liberation Analyser (MLA), in order to obtain important data regarding grain size distribution, mineral association, liberation and exposure. The gold deportment was performed on polished blocks which had undergone gravity separation as recommended (Coetzee et al., 2011). The material will therefore be biased towards dense ore minerals due to the separation process, and the mineral ratios present may not be representative. However, understanding the

characteristics of the maximum number of gold grains was the priority. The number of gold grains located and characterised varies with each sample (Table 6-1) and therefore the reliability of the characterisations made may vary. For example, the Barbrook and Consolidated Murchison samples, had fewer than the ideal number of gold grains and therefore the conclusions made are less reliable than that of the VCR, Pilgrim's Rest and Fairview samples. The nature of the gold as discussed below has been confirmed using optical petrography where possible.

Table 6-1 The number of gold-containing particles and gold grains identified for each deposit and subsequently used for characterisation. With an increasing number of identified gold grains within a sample the more reliable the data becomes.

Sample	Native Gold			Aurostibite			
	Particles	Grains	Au (%)*	Particles	Grains	Au (%)	Total Gold Bearing Grains
VCR	266	380	100	0	0	0	380
FV	224	321	100	0	0	0	321
BK	47	50	100	0	0	0	50
CM	22	23	37.22	44	44	62.78	67
PG	156	169	100	0	0	0	169

*Au (%) here refers to the amount that the gold contributes to the particle in which it is hosted.

6.2.1. Grain Size Distribution

The grain size distribution is an important factor for consideration in processing as it will affect the milling size and leach times required for extraction (Coetzee et al., 2011). The grain sizes referred to in this section are represented in equivalent circular diameter (ECD)

The VCR sample had a range in grain sizes from 0.5 µm to 425 µm (Figure 6-1, Figure 6-2), with the majority of gold weight percent occurring in the coarser-grained fraction (200 -425 µm), despite a significant number of individual gold grains in the finer fraction. The presence of coarse gold had implications for the leach lengths required to ensure maximum recovery. Coarse gold generally requires extended leaches as individual gold grains take longer to undergo complete dissolution.

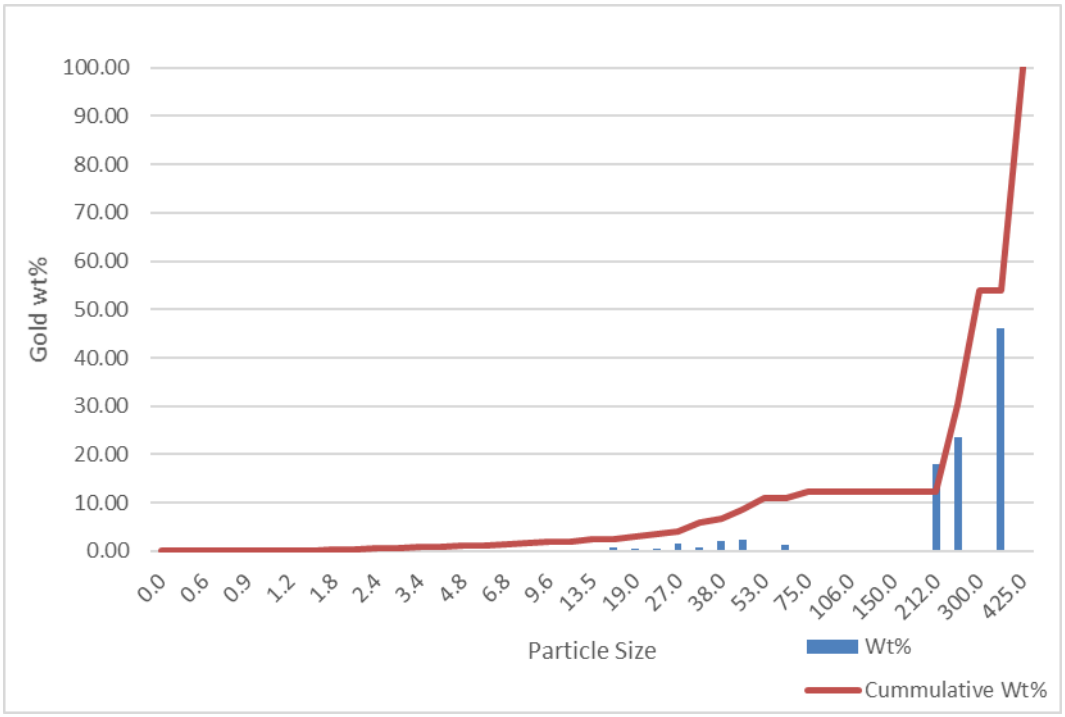


Figure 6-1 The gold grain size distribution for the Ventersdorp Contact Reef sample, with a bar graph representing weight percent contribution for certain size fractions, and line graph representing cumulative weight percent. The gold in the Ventersdorp Contact Reef sample ranged in grain size from 0.5 µm to 425 µm with the majority of gold weight percent occurring in the coarser grained fraction (200 -425 µm)

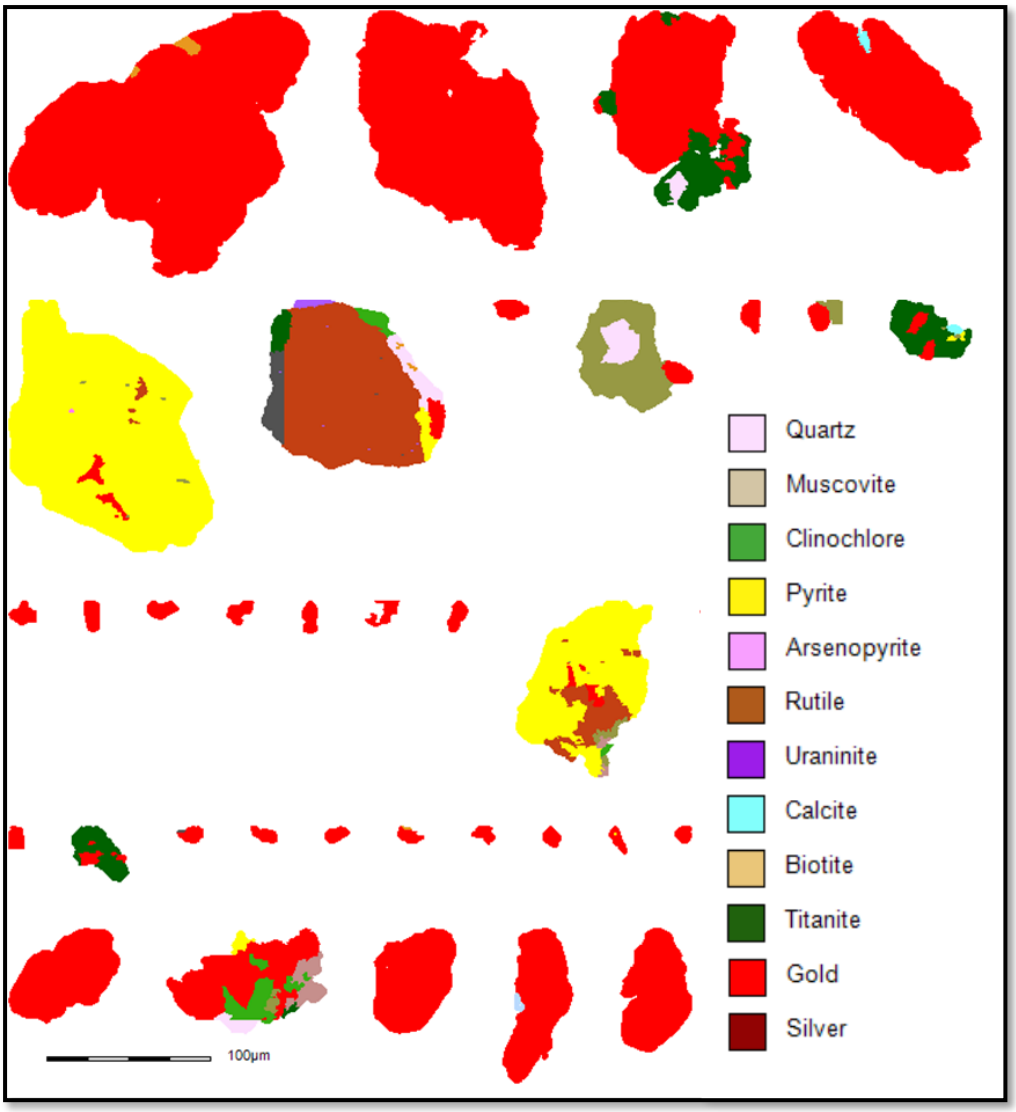


Figure 6-2 False colour images from the MLA of gold found in the Ventersdorp Contact Reef sample. This image highlights the range in grain size of gold from 0.5 µm to 425 µm, with the majority of gold weight percent occurring in the coarser grained fraction (200 -425 µm).

The Fairview samples had an unexpectedly large range in grain sizes, from 0.5 µm to 425 µm (Figure 6-3, Figure 6-4A). The large range, and coarse gold contribution, was similar to that of the VCR sample, although with noted inconsistencies in distribution due to the absence of gold grains between sizes of 16 µm and 45 µm, between 45 µm and 180 µm, and 180 µm and 355 µm. Fairview had significant nugget gold, and this will require consideration for extraction.

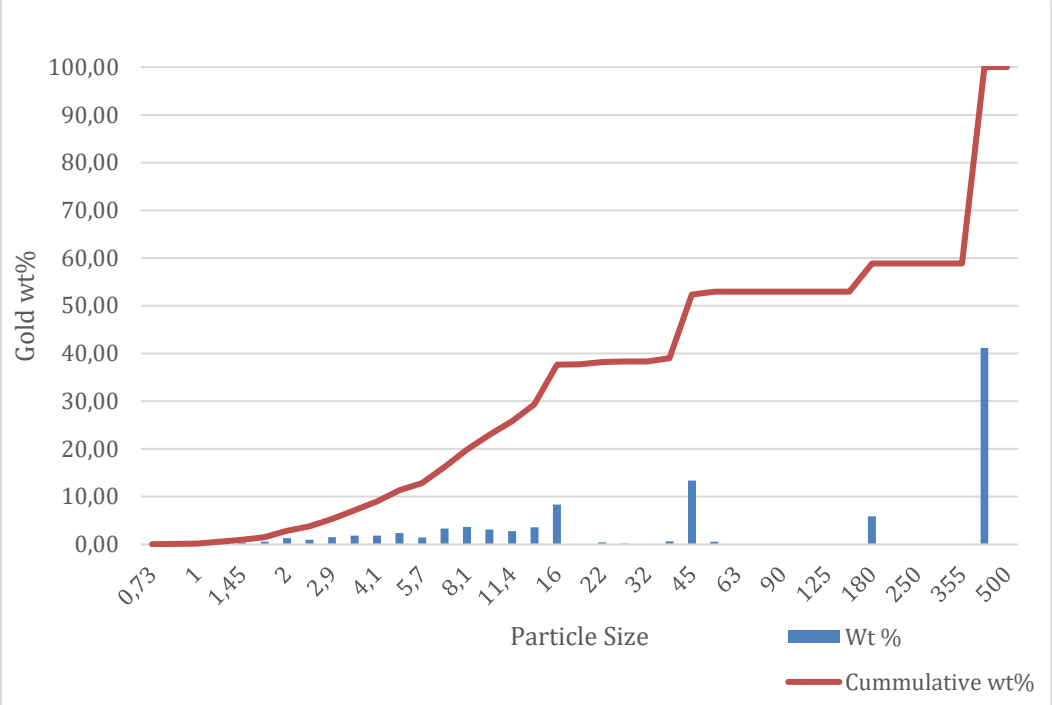


Figure 6-3 Graph representing the gold grain size distribution for the Fairview sample, with a bar graph representing weight percent contribution from certain size fractions, and a line graph representing cumulative weight percent. The gold in the Fairview sample had a large range in grain sizes, from 0.5 µm to 425 µm. There were inconsistencies in distribution due to the absence of gold grains between sizes of 16 µm and 45 µm, between 45 µm and 180 µm, and 180 µm and 355 µm.

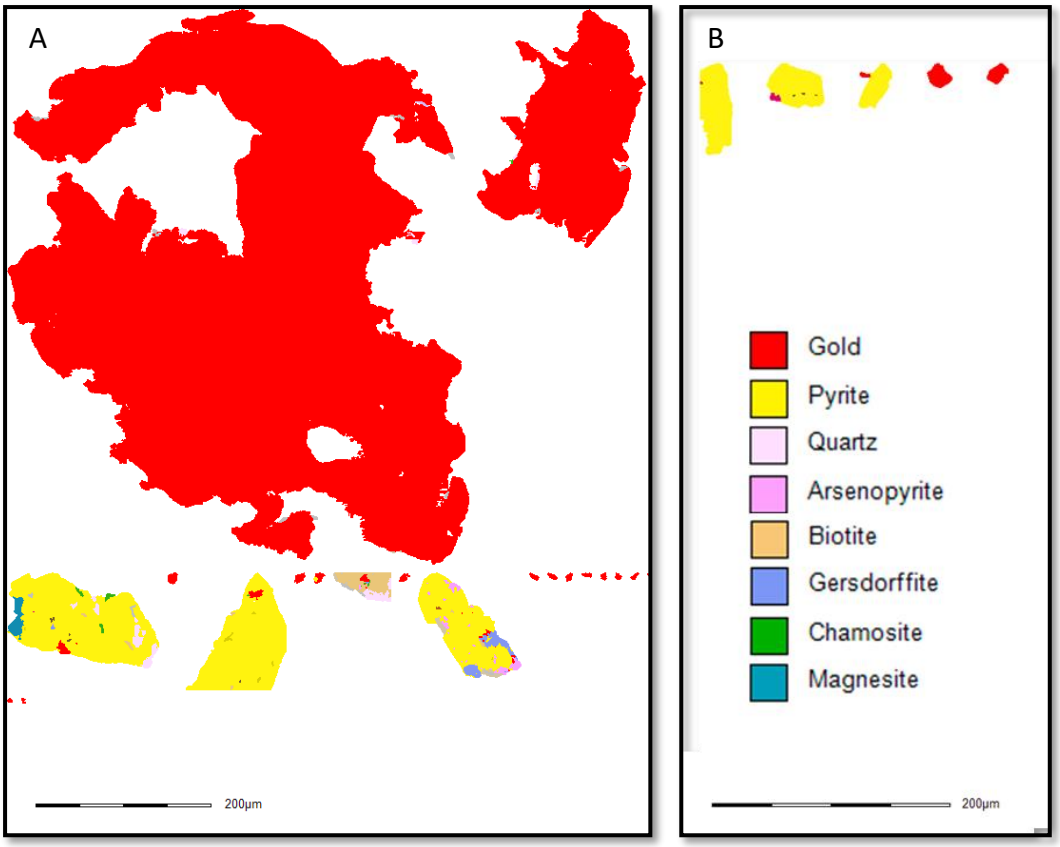


Figure 6-4 False colour MLA images from the Barberton samples (Fairview and Barbrook) A) An example of the vast range in sizes of gold grains in the Fairview sample, from 0.5 µm to 425 µm. B) The Barbrook sample gold grains, with pyrite association represented. The Barbrook sample gold grains ranged in size from 1 µm to 53 µm.

The Barbrook sample contained gold grains with a limited range in size, from 1 µm to 22 µm (Figure 6-5, Figure 6-4B), with a single exception of one larger grain at 53 µm. While this range may have been a product of the limited number of gold grains analysed, it still indicates that much of the gold present was fine. Ultra-fine milling may improve extraction; however, it seems likely (based on literature and calculated grades) that much of the gold present is not native gold.

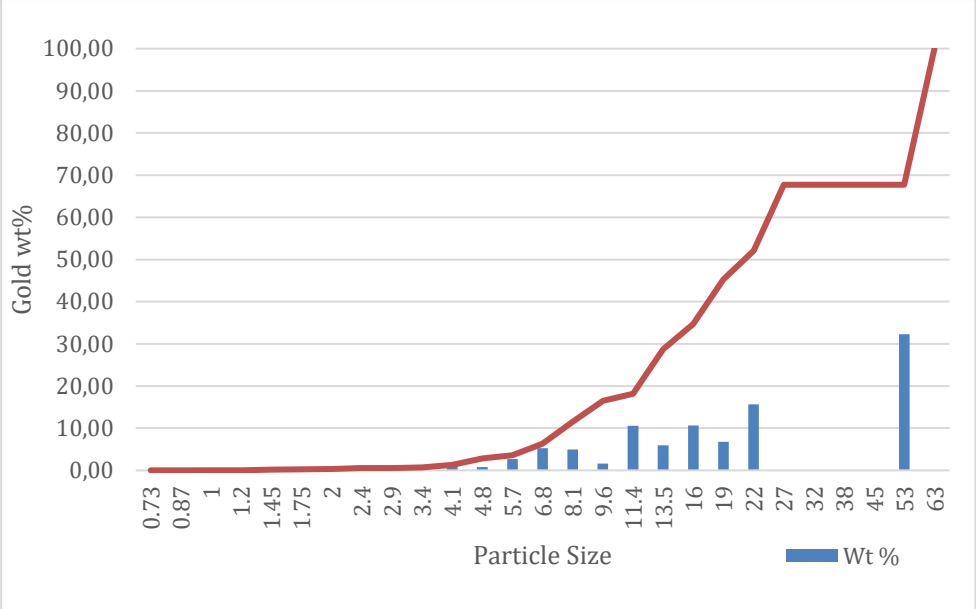


Figure 6-5 Graph representing the gold grain size distribution for the Barbrook sample, with a bar graph representing weight percent contribution from certain size fractions, and a line graph representing cumulative weight percent. The gold grains ranged in size from 1 µm to 22 µm, with a single exception of one larger grain at 53 µm.

The Consolidated Murchison sample, unlike the other deposits, had two identified gold bearing minerals; native gold and aurostibite. The native gold had a limited grain size distribution from 2 µm to 27 µm (Figure 6-6) with much of the gold occurring predominantly in the range from 3-10 µm. The aurostibite grains tended to be significantly larger than the native gold grains, with a range in size from 0.73 µm to 53 µm (Figure 6-8), with the vast majority of the total aurostibite occurring in the coarser fractions. The total gold-bearing mineral grains found in Consolidated Murchison ranged in size from 0.73 µm to 53 µm.

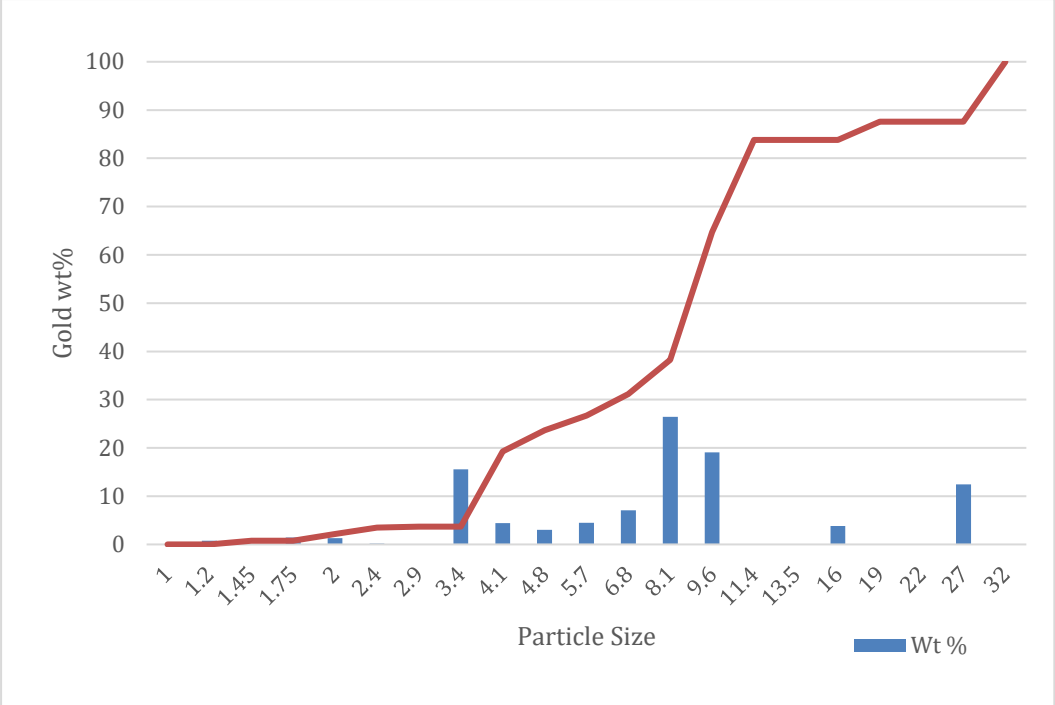


Figure 6-6 Graph representing the native gold grain size distribution for the Consolidated Murchison sample, with a bar graph representing weight percent contribution from certain size fractions, and a line graph representing cumulative weight percent. The native gold had a limited grain size distribution from 2 µm to 27 µm with much of the gold occurring predominantly in the range from 3-10 µm.

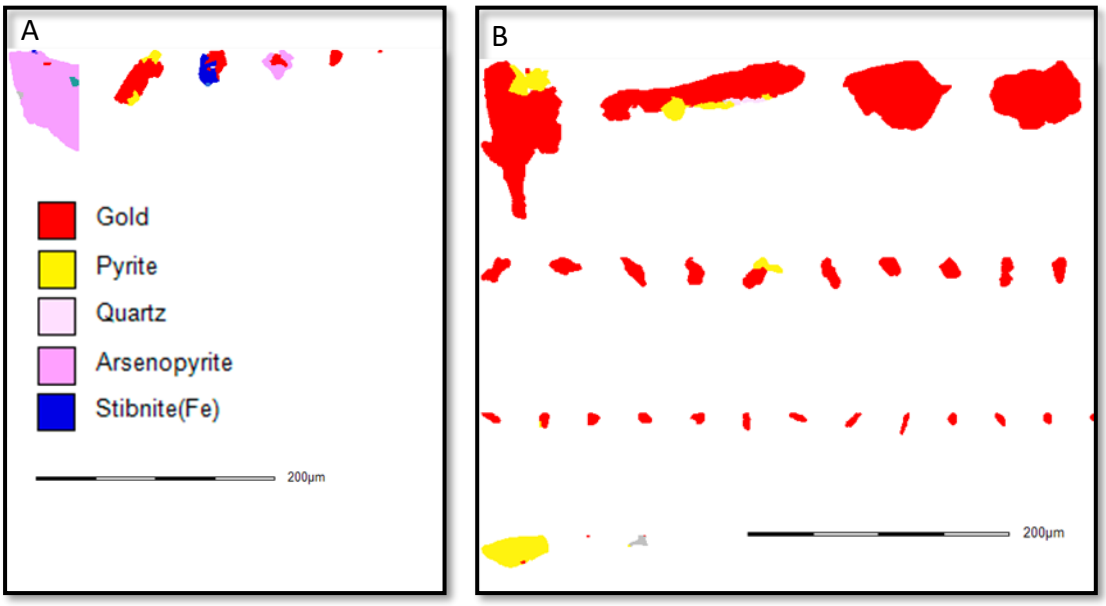


Figure 6-7 False colour MLA images A) Consolidated Murchison sample gold grains, highlighting the grain size distribution, limited number of grains and the lower degree of liberation. The native gold had a limited grain size distribution from 2 µm to 27 µm. B) Pilgrim's Rest sample gold grains, highlighting the range in size, excellent liberation and exposure.

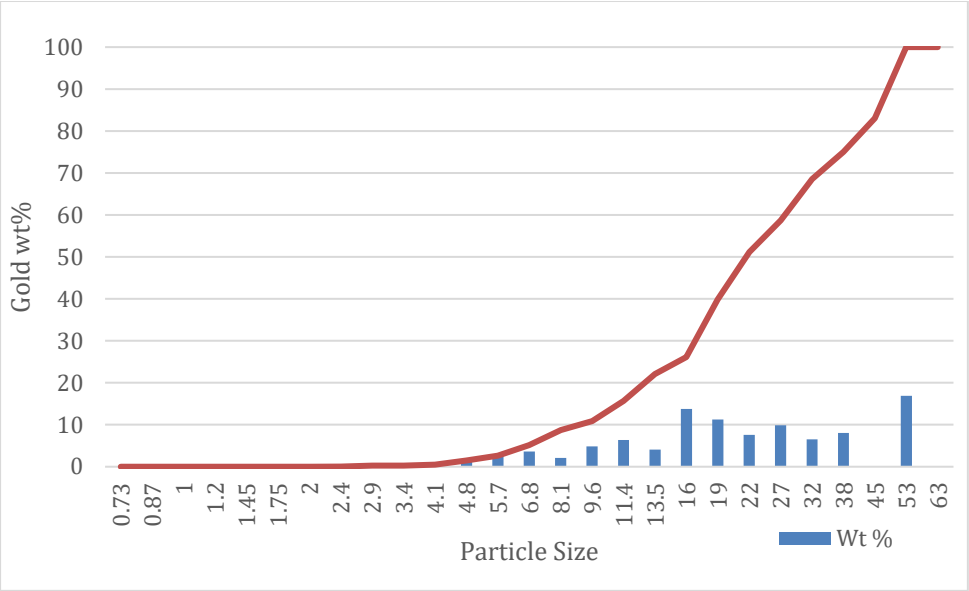


Figure 6-8 Graph representing the aurostibite grain size distribution for Consolidated Murchison, with a bar graph representing weight percent contribution from certain size fractions, and a line graph representing cumulative weight percent. Aurostibite grains tended to be significantly larger than the native gold grains, with a range in size from 0.73 µm to 53 µm

The Pilgrim's Rest sample had gold grains ranging in size from 0.75 µm to 63 µm (Figure 6-7B, Figure 6-9B). There was no coarse gold present, this is stated with some confidence as a large number of grains have been located in this studied material. The distribution of grain sizes was quite evenly spread, with a number of size fractions contributing ± 5%. Approximately equal proportions of gold come from size fractions 6.8 µm to 45 µm.

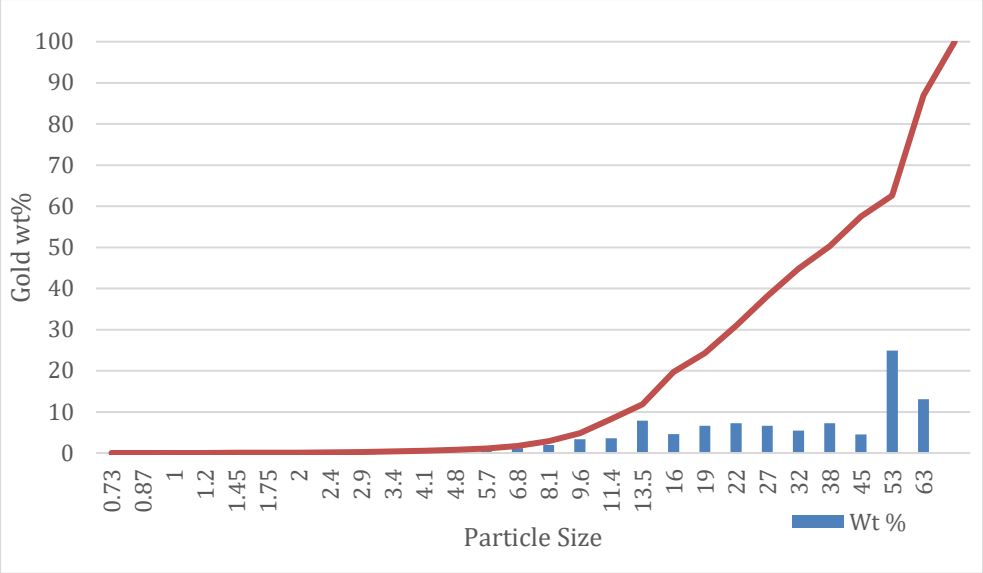


Figure 6-9 Graph representing the gold grain size distribution for the Pilgrim's Rest sample, with a bar graph representing weight percent contribution from certain size fractions, and a line graph representing cumulative weight percent. The Pilgrim's Rest sample has gold grains ranging in size from 0.75 µm to 63 µm

6.2.2. Mineral Associations

Mineral associations will significantly affect the response to certain processing methods, for example the response to flotation, gravity separation and leaching.

The minerals associated with gold vary significantly in the studied deposits. Variations include the mineral compositions, the total number of minerals and the degree to which they are associated. The complete results including trace associations is in Appendix B.

The VCR sample had several minerals associated with gold, however the degree to which the associations occur was typically limited. Mineral associations included quartz (6.44%), pyrrhotite (4.32%), coffinite (1.99%), chamosite/chlorite (1.60%), pyrite (1.39%), mica (0.39%), brannerite (0.17%), while other minor minerals present included other silicates (4.25%) such as orthoclase, plagioclase and amphiboles, other sulphides (2.29%) such as chalcopyrite, arsenopyrite, and galena, and carbonates (1.99%) which included siderite and ankerite. Lesser amounts of iron manganese oxides and phosphates were also present. Free surface gold constituted 75% of the total gold.

Minerals associated with the gold in the Fairview sample included pyrite (54.59%), arsenopyrite (16.68%), pyrrhotite (1.44%), as well as other sulphides (0.41%). Micas (2.49%), quartz (1.07%), chlorite (1.54%), and iron manganese oxide minerals (0.23%) were also associated with the gold. The free surface accounted for only 20.71% of the mineral associations.

The gold found within the Barbrook sample was predominantly associated with sulphides; pyrite (4.11%), arsenopyrite (3.06%) as well as other sulphides (0.40%), there was also an association with ankerite (1.97%), chlorite (1.53%), siderite (0.76%) and micas (0.26%). The Barbrook sample had a significant free surface (87.91%) available.

The mineral associations for the Consolidated Murchison sample were subdivided into two groups; those associated with native gold, and those associated with aurostibite. Minerals associated with native gold included arsenopyrite (23.32%), magnesite (5.86%), pyrite (2.67%), stibnite (2.39%) as well as other sulphides (1.32%). The native gold had a free surface of 64.44%. The aurostibite had limited mineral association (Figure 6-10) with only minor arsenopyrite (0.81%), and an otherwise free surface (99.18%).

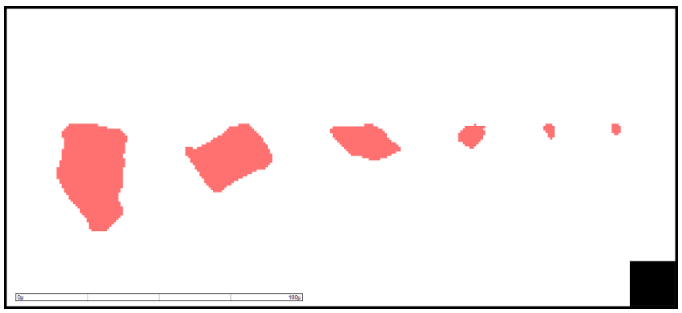


Figure 6-10 False colour image of aurostibite grains, highlighting the high degree of free surface and lack of mineral association.

The Pilgrim's Rest sample gold grains are associated with pyrite (7.71%), other sulphides (1.00%) such as chalcopyrite, pyrrhotite and arsenopyrite, quartz (0.35%), biotite (0.05%) and arsenopyrite (0.02%), with a free surface of 90.20%.

6.2.3. Liberation

Mineral liberation is an important characteristic to consider when ensuring effective processing procedures. Mineral liberation will control the ability of the ore to undergo

successful flotation and gravity separation. Liberation is defined in a number of ways (Berube and Marchand, 1984; Barbery, 1991) but the most common and simple definition involves the area % of the mineral of interest relative to the area of the total host particle (Figure 6-11). This value is determined from a number of representative particles (King and Schneider, 1998; Crop, 2013; Mariano, 2016). The definition above applies if working with a 2D section, however volume may be substituted for area if a 3D environment is considered.

The degree of liberation is typically grouped into incremental grades. The grades can be in small increments of 10%, or as used in this work, larger increments; <30% liberated material is described as locked, 30-60% is middlings, whereas > 60 % is defined as liberated (Mariano, 2016).

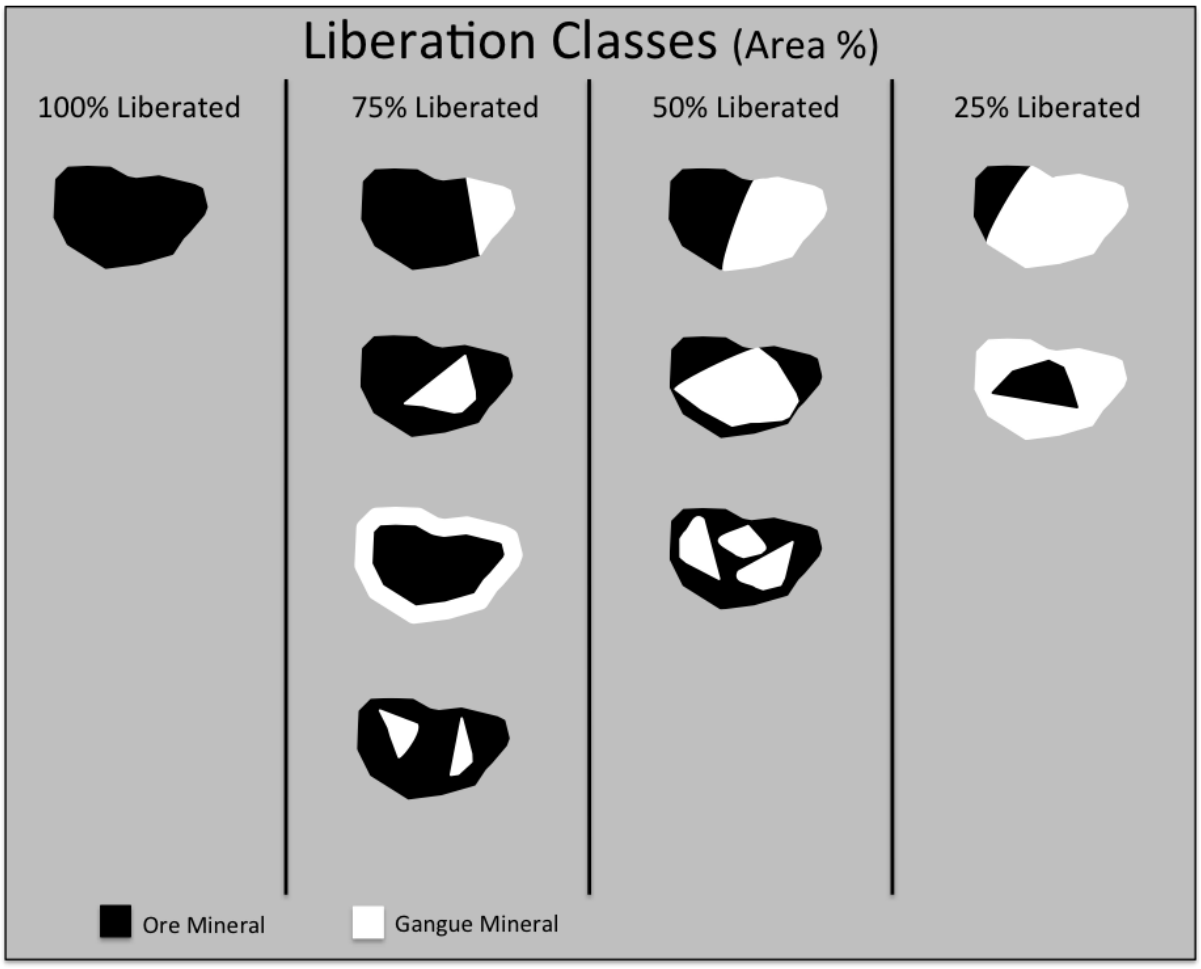


Figure 6-11 The range of liberation classes, e.g. 100% liberated, where the ore mineral is 100% of the area of the particle containing the ore mineral to 25% liberation, where the gold only comprises 25% of the area of the host particle (Crop, 2013).

The full liberation data are extensive and are given in Appendix B. A short summary of the relevant features related to the liberation are shown in Figure 6-12. The liberation data are presented for samples from each deposit and subdivided into different classes, based on the gold composition of the particle in weight percent. It is important to note that the textural information provided for both liberation and exposure only refer to particulate gold, which is not the only gold present in these samples (this is particularly relevant to gold from the Fairview and Barbrook samples).

The VCR sample has a vast range of mineral associations which spanned all the gold liberation classes, however, the majority of the gold (94.53 wt%) occurred in particles between 95 and 100% liberated. This highlights the high degree to which gold particles were liberated. The minor amount of gold particles that aren't well liberated were commonly associated with pyrite, pyrrhotite, quartz, coffinite, uraninite, plagioclase, Fe-oxide, biotite, muscovite, chlorite and galena. The total weight percent of gold grains that were liberated are 98.49%, 1.17% middlings, while the remaining 0.33% is locked.

The Fairview sample material contained gold with a range of liberations, with 25% of the gold locked (<30% of the particle is gold), 3% as middlings and 72% liberated. The gold particles that were not well liberated are commonly associated with pyrite, arsenopyrite, quartz, biotite, muscovite, magnesite and ankerite.

The Barbrook material had a limited number of gold grains, however, a sufficient number were identified to gain an understanding of the distribution, which highlights excellent liberation found in the Barbrook sample. Approximately 97.12% of the total gold was hosted within particles with a liberation of 90% or more. While 85.75% of the total gold occurred as 100% liberated gold.

The Consolidated Murchison sample had gold both in the native form and in aurostibite. The native gold had approximately 12% locked in particles of less than 30% liberation. While more gold, 28%, occurred in the 75- 80% liberation class, with a remaining 20% and 34.48%, in the range of 90-95% and 100% respectively. The aurostibite was extremely well liberated with more than 96% of the total aurostibite falling within 70% or greater liberation class. Only a limited number of minerals including arsenopyrite, stibnite, pyrite and siderite were associated with gold.

Of the gold found within the Pilgrim's Rest sample, more than 97% of the total gold was more than 95% liberated. Gold liberation was good, while the middlings (2.34%) and locked gold (0.5%) were associated primarily with pyrite, as well as chalcopyrite and quartz.

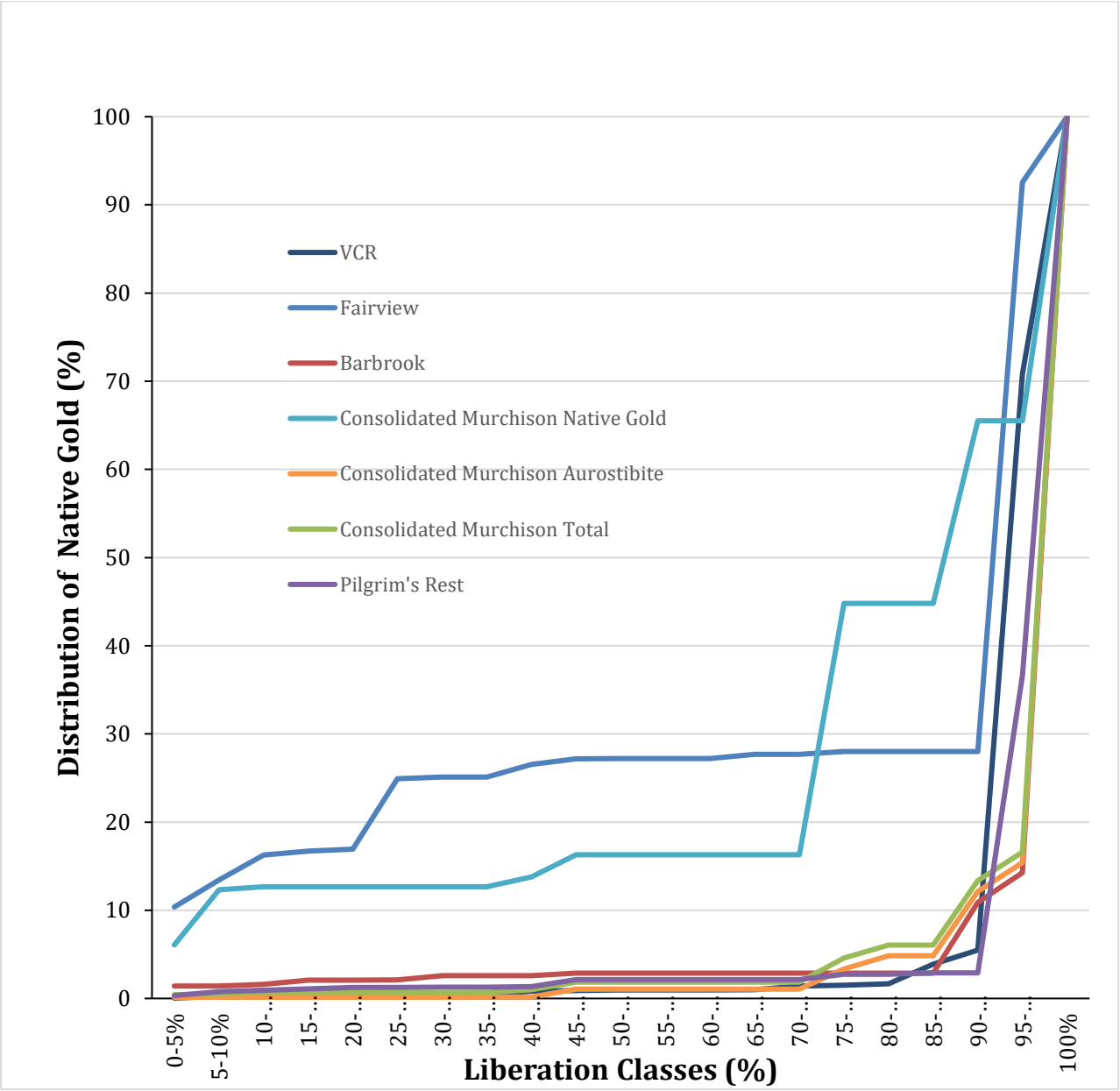


Figure 6-12 Graph representing the cumulative gold liberation by particle composition for the deposits of interest. The lower degree of liberation associated with the Fairview and Consolidated Murchison Native gold is clearly evident. The Ventersdorp Contact Reef, Barbrook and Pilgrim's Rest samples all have good liberation.

6.2.4. Exposure

Mineral exposure can be described as the percentage of the grain that has a free, exposed, surface (Figure 6-13) (Crop, 2013). The portion of the grain with an exposed surface is available for interaction with reagents in leaches or flotation (Napier-Munn and Wills, 2006). The degree of exposure is defined either using percentage increments, or as presented in this work as either unexposed/poorly exposed (<10% free surface area), or exposed (>10% free surface area).

It is interesting to note that while liberation and exposure are both important characteristics in gold extraction, and have similar definitions, they are not directly related. An example of this is evident in Figure 6-13, where in the first row/class all minerals have 100% exposure, but significant variation in liberation.

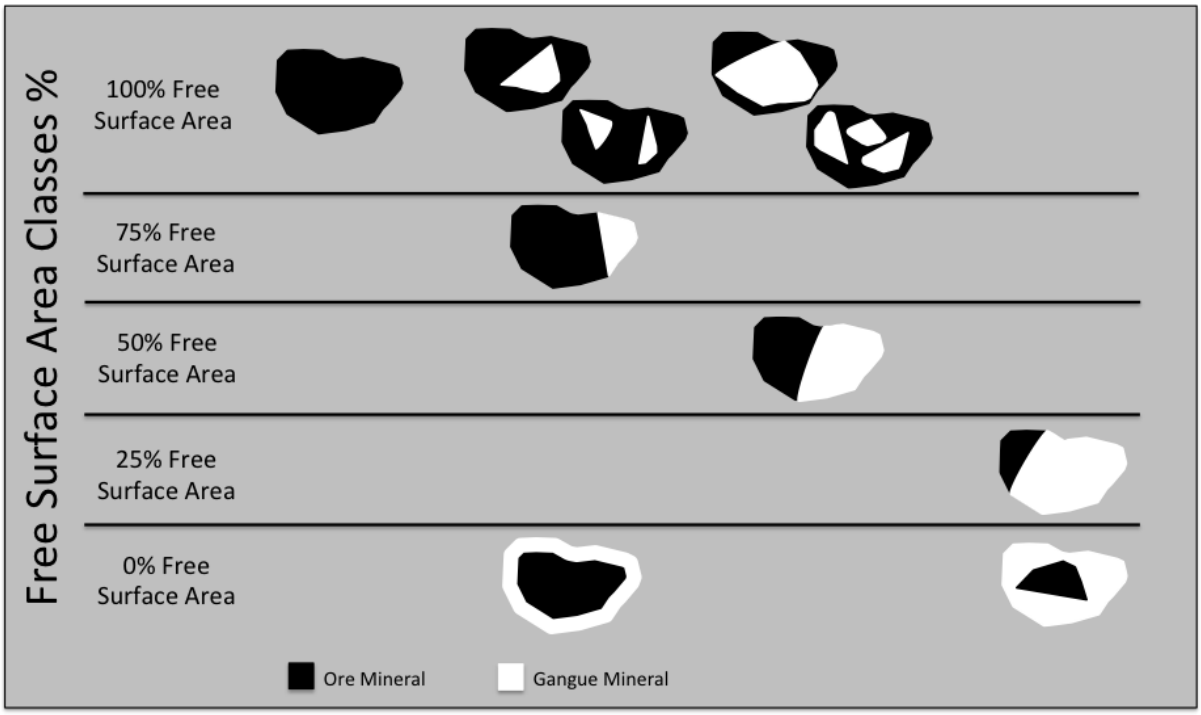


Figure 6-13 The range of classes of free surface area/exposure. Ranging from 100% free surface exposure to 0% free surface exposure (Crop, 2013). The 100% free surface area highlights the difference between liberation and exposure. All of the grains in the top row have 100% exposure, while liberation varies from 100% of the left to less than 25% on the right.

The full exposure data are extensive and are given in Appendix B. A short summary of the relevant features related to the liberation are shown in Table 6-2 and Figure 6-14. The

liberation data are presented for each deposit and subdivided into different classes, based on the gold free surface of the particle percentage.

The VCR sample had excellent gold exposure, with only 0.15% of gold locked and 0.18% partially exposed. The majority of the exposed gold occurred in 2 classes, one with 95-100% exposure, and the other with 100% exposure.

The Fairview material had a moderate exposure of 50% with 22.80% of the remaining gold is locked.

The Barbrook material had an excellent gold exposure of 93.58%.

The Consolidated Murchison material had native gold grains with a moderate exposure of 96.98%, with the spread of total gold (in weight percent) ranging from classes at both low (0-10%), and high (100%) degrees of exposure. The aurostibite grains were 100% exposed.

The Pilgrim's Rest sample had a good exposure of 98.37%, which comes from the classes ranging from 75-100% free surface particle gold.

Table 6-2 Gold Mineral Exposures for each deposit, subdivided into 3 classes, not exposed, partially exposed, and exposed. Fairview is the only sample present with limited exposure, the other samples all have exposures greater than 90%.

	VCR	Fairview	Barbrook	CM Native Au	CM Aurostibite	PG
Not Exposed	0.15	22.80	2.05	0.00	0.00	0.55
Partially Exposed	0.18	27.22	4.40	3.02	0.00	1.08
Exposed	99.67	49.98	93.55	96.98	100.00	98.37
Total	100.00	100.00	100.00	100.00	100.00	100.00

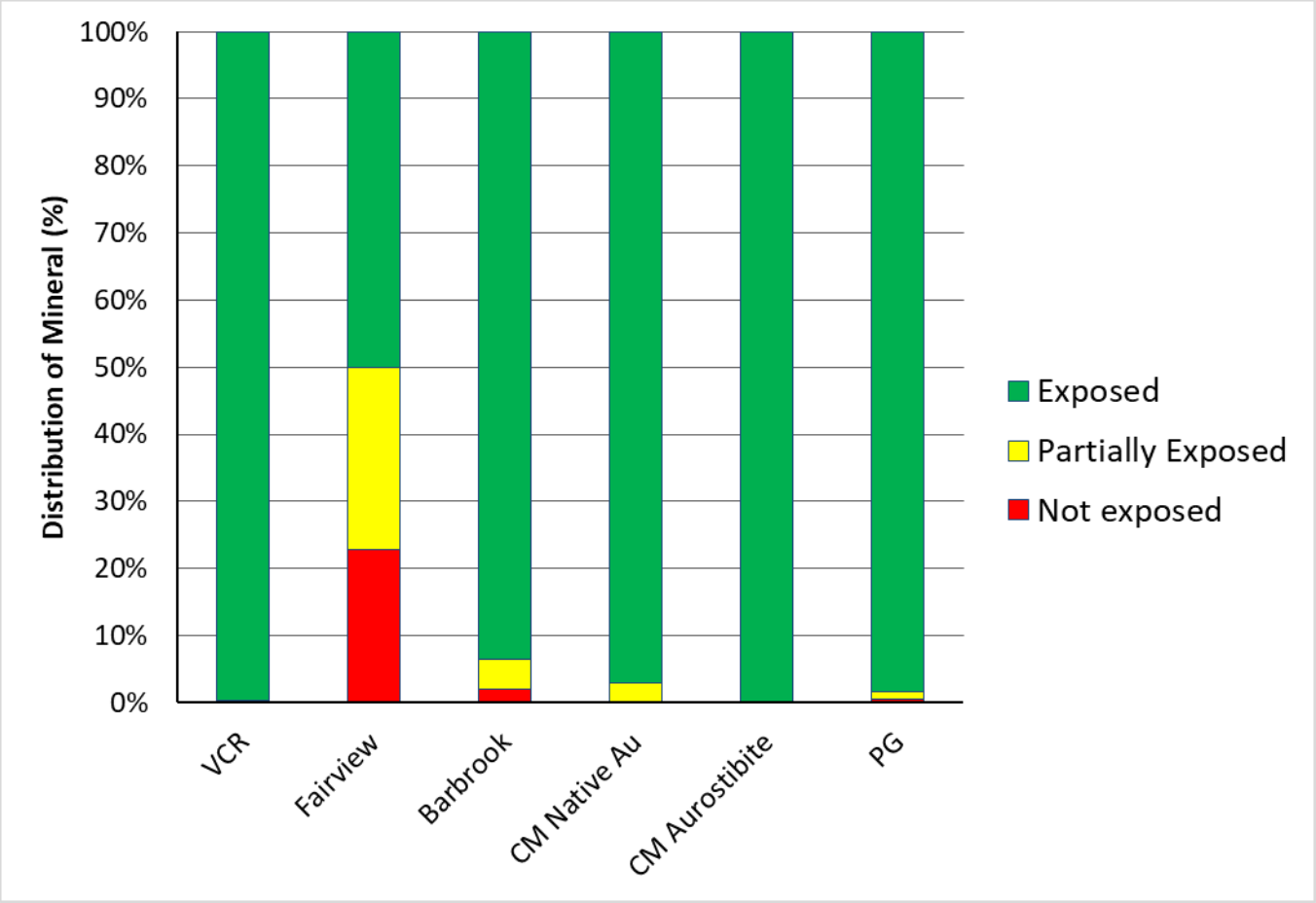


Figure 6-14 Gold mineral exposures for each sample, subdivided into not exposed, partially exposed, and exposed. The Fairview sample has notably moderate exposure while the remaining samples have excellent exposure. CM – Consolidated Murchison, PG – Pilgrim's Rest.

6.2.5. Gravity Concentrate Modal Mineralogy

The modal mineralogy is determined using rapid identification and quantification of all constituent minerals. Unfortunately, this method had slightly skewed results for two reasons; the surface of the block was analysed (rather than the traditional 90˚ cut sections) and the gravity concentrates were used as material for the blocks, however this information should still provide some valuable insight despite these limitations. The modal mineralogy has been summarised in Figure 6-15.

The determined modal mineralogy for the VCR sample was dominated by quartz (42.17%), pyrite (22.27%), pyrrhotite (20.76%), Fe (Mn)-oxides (4.83%) and phosphates (0.26%).

The modal mineralogy of the Fairview sample had a number of contributors; pyrite (39.52%), arsenopyrite (14.09%), quartz (13.15%), micas (12.77%), chlorite (5.10%) as well as a number of minerals that contributed less than 5% (orthoclase, plagioclase, pyrrhotite, gersdorffite, coffinite, brannerite, magnesite and siderite).

The modal mineralogy of the Barbrook sample included pyrite (27.90%), quartz (23.51), ankerite (14.55%), chlorite (8.66%) and micas (4.18%). Other notable minerals include siderite, pyrrhotite, arsenopyrite, amphiboles and feldspars.

The gold from the Consolidated Murchison sample was associated with a number of minerals including arsenopyrite (19.30%), quartz (13.15%), pyrite (9.13%), berthierite (7.80%), tourmaline (5.60%), magnesite (5.40%), siderite (5.41%), ankerite (3.86%) and stibnite (3.67%).

The minerals association for Pilgrim's Rest was dominated by pyrite (78.68%), with quartz (12.69%) and pyrrhotite (1.11%) the only minerals present in significant quantities.

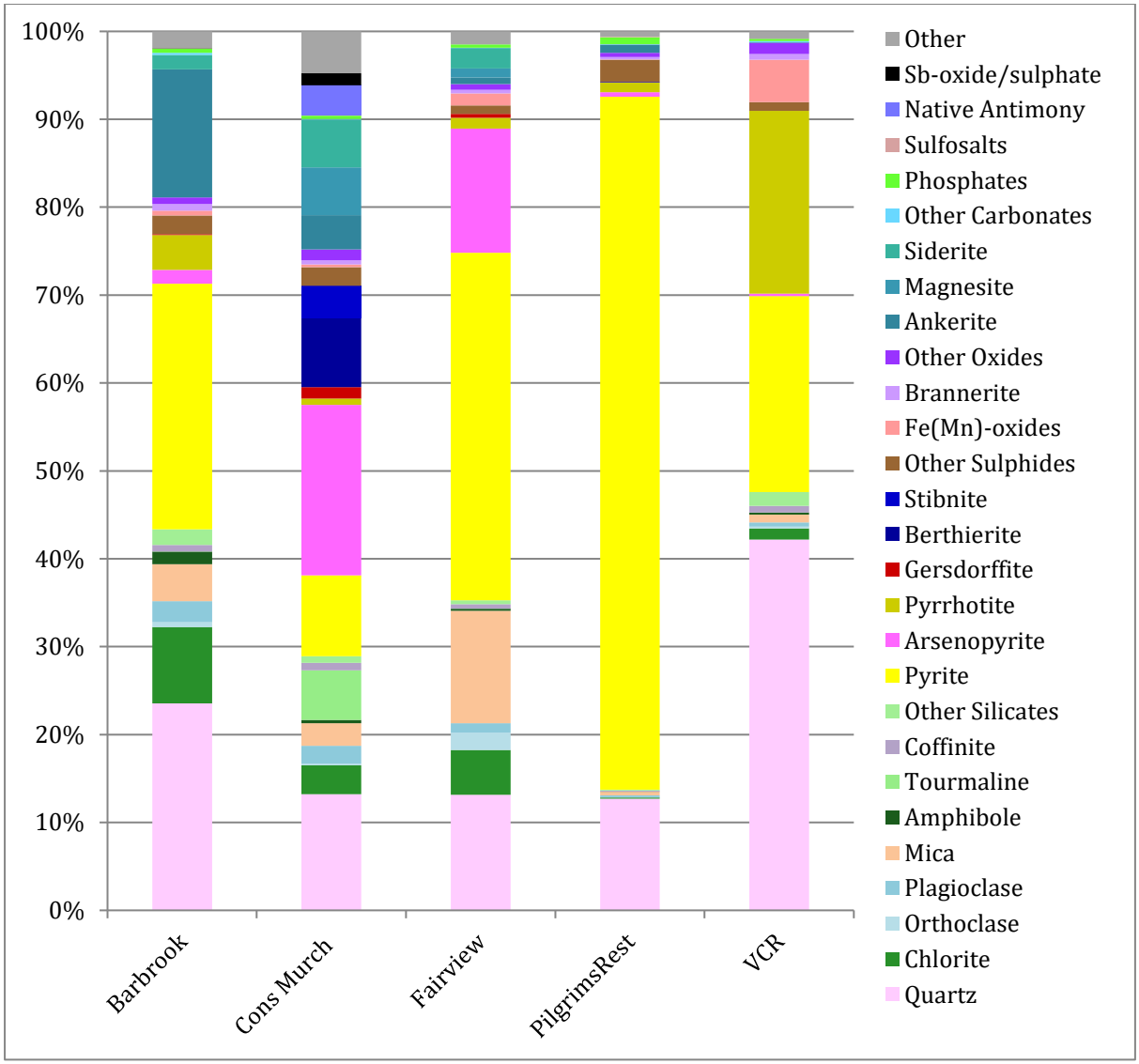


Figure 6-15 Normalised modal mineralogy summarised for the ore deposits of interest. All the samples have abundant quartz and to a lesser extent pyrite. With the exception of Pilgrim's Rest, all samples also have feldspars, chlorite and mica.

6.3. Electron Probe Micro-Analyser (EPMA)

Electron Probe Micro Analysis (EPMA) was undertaken with two main goals; 1) to locate and better understand the nature of the "invisible gold" thought to be present in the refractory gold ores of the Fairview, Barbrook and Consolidated Murchison samples, 2) to determine the nature of the native gold and the possible silver content of the gold grains.

EMPA work was undertaken using a CAMECA SX100. In order to locate gold in sulphides, the detectors were set to target sulphur, iron, copper, arsenic, antimony and gold. These

elements were analysed as they are the most common gold-bearing sulphides (confirmed by the mineralogy of the deposits). Gold is typically found within arsenopyrite, arsenian pyrite, or at Consolidated Murchison, associated with the antimony. The targeted peaks and conditions for the respective elements have been summarised in Table 6-3. In order to get both high concentration elements (e.g. S and Fe) and low concentration elements (Au), two conditions were required (Table 6-3). The first set of conditions was used to analyses the first 5 elements (S, Fe, Cu, As and Sb) where the beam was set to 30 keV and 10 nA, while the second set of conditions was used to determine gold concentration which required far more intense beam conditions of 30 keV and 200 nA. The whole analyses could not be done under the second set of conditions as these would be too destructive to use over extended periods and would affect the results obtained.

Table 6-3 The targeted peaks and conditions set up on the EPMA for the respective elements of interest, when analysing potentially gold-bearing sulphides. While most of the elements were analysed for at 10 nA, the low gold concentrations required the beam conditions for analysis to be 30keV, 200 nA.

Elements analysed	Peaks targeted	Beam Conditions
Sulphur	Kα	30 keV, 10 nA
Iron	Kα	30 keV, 10 nA
Copper	Kb	30 keV, 10 nA
Arsenic	Lα	30 keV, 10 nA
Antimony	Lα	30 keV, 10 nA
Gold	Lα	30 keV, 200 nA

From these analyses, some gold-bearing sulphides were located. While concentrations may not be accurate; there was certainly a notable increase in gold content in specific sulphides relative to the average. The sulphides with an elevated gold concentration tended to be arsenic or antimony- rich, which are the grains that would be expected to contain gold, based on the literature.

Sulphide-hosted gold was only found in the refractory deposits, of the Fairview and Consolidated Murchison samples. No refractory gold was located in the Barbrook material; however, this may be a product of the lower gold grade. The EPMA work was done to confirm the presence of these refractory, gold-bearing sulphides rather than to quantify their gold content.

The gold located in the Fairview sample (Figure 6-16, Table 6-4) was found exclusively in arsenic-enriched minerals, with the elemental ratios expected for arsenopyrite. Of the arsenopyrite grains analysed, 50% were gold-bearing. Of the points analysed, point 11 was the only point with an elemental total significantly less than 100%. This poor result is likely to be due to the grain not being a sulphide and therefore the elements analysed for were not abundant. The abundance of elements such as sulphur, iron and copper are minor in most silicates and thus only a low total was reported for the silicate that was unintentionally analysed.

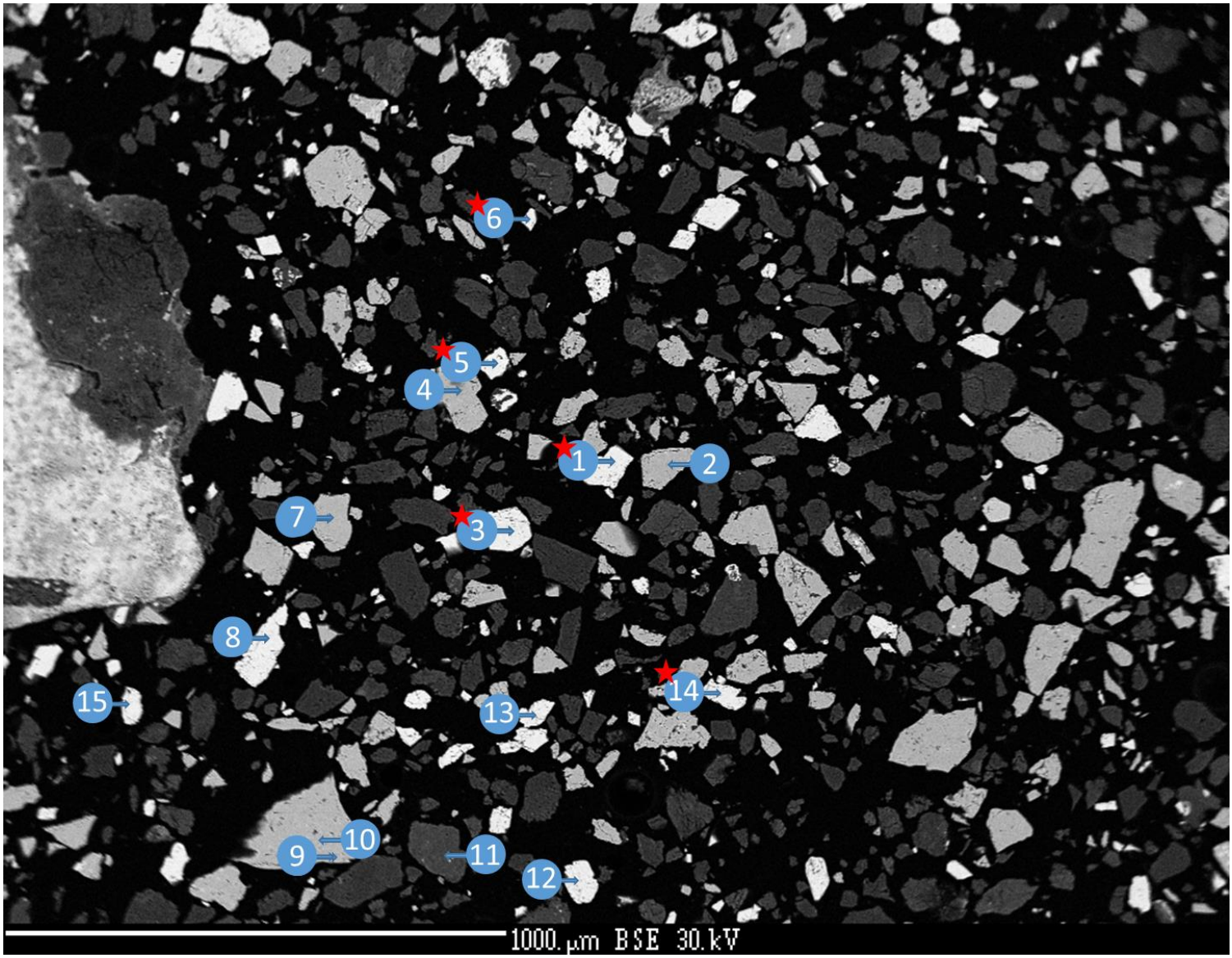


Figure 6-16 The points of analyses for the Fairview material. Minerals with gold detected have been marked in red. Each point correlates with the data found in Table 6-4. These gold-bearing points (1,3,5 and 14) are all arsenopyrite grains.

Table 6-4 The data determined from EPMA analysis for the Fairview sample. The data set corresponds to the labelled grains in Figure 6-16. Gold-bearing grains have been highlighted. Grains analysed were generally sulphides, with the exception of point 11 which was probably a silicate.

Fairview EPMA points							
Data Set	S	Fe	Cu	As	Sb	Au	Total
1	22.84	35.93	0.00	40.24	0.03	0.02	99.07
2	52.54	46.37	0.00	0.23	0.03	0.01	99.19
3	23.42	35.55	0.00	39.33	0.02	0.01	98.34
4	52.26	45.72	0.01	0.33	0.00	0.00	98.33
5	22.61	35.14	0.00	41.17	0.02	0.02	98.97
6	22.31	34.81	0.08	41.58	0.01	0.02	98.80
7	52.34	45.87	0.00	0.92	0.00	0.00	99.13
8	22.45	35.50	0.00	41.67	0.02	0.00	99.65
9	52.78	46.24	0.00	0.54	0.02	0.00	99.60
10	52.96	46.40	0.02	0.50	0.00	0.00	99.89
11	0.00	7.19	0.00	0.09	0.07	0.00	7.35
12	24.01	35.29	0.00	39.83	0.06	0.01	99.20
13	23.59	35.20	0.00	39.93	0.03	0.00	98.76
14	22.91	34.91	0.00	41.10	0.05	0.03	99.00
15	22.75	35.15	0.00	40.62	0.04	0.01	98.57

The Barbrook material (Figure 6-17, Table 6-5) exhibited no gold-bearing sulphides, however, this may be due to the low gold grade in this deposit. The generally low gold grade could mean that the gold-bearing sulphides had concentrations too low to be detectable or the gold-bearing grains are not abundant and therefore difficult to locate.

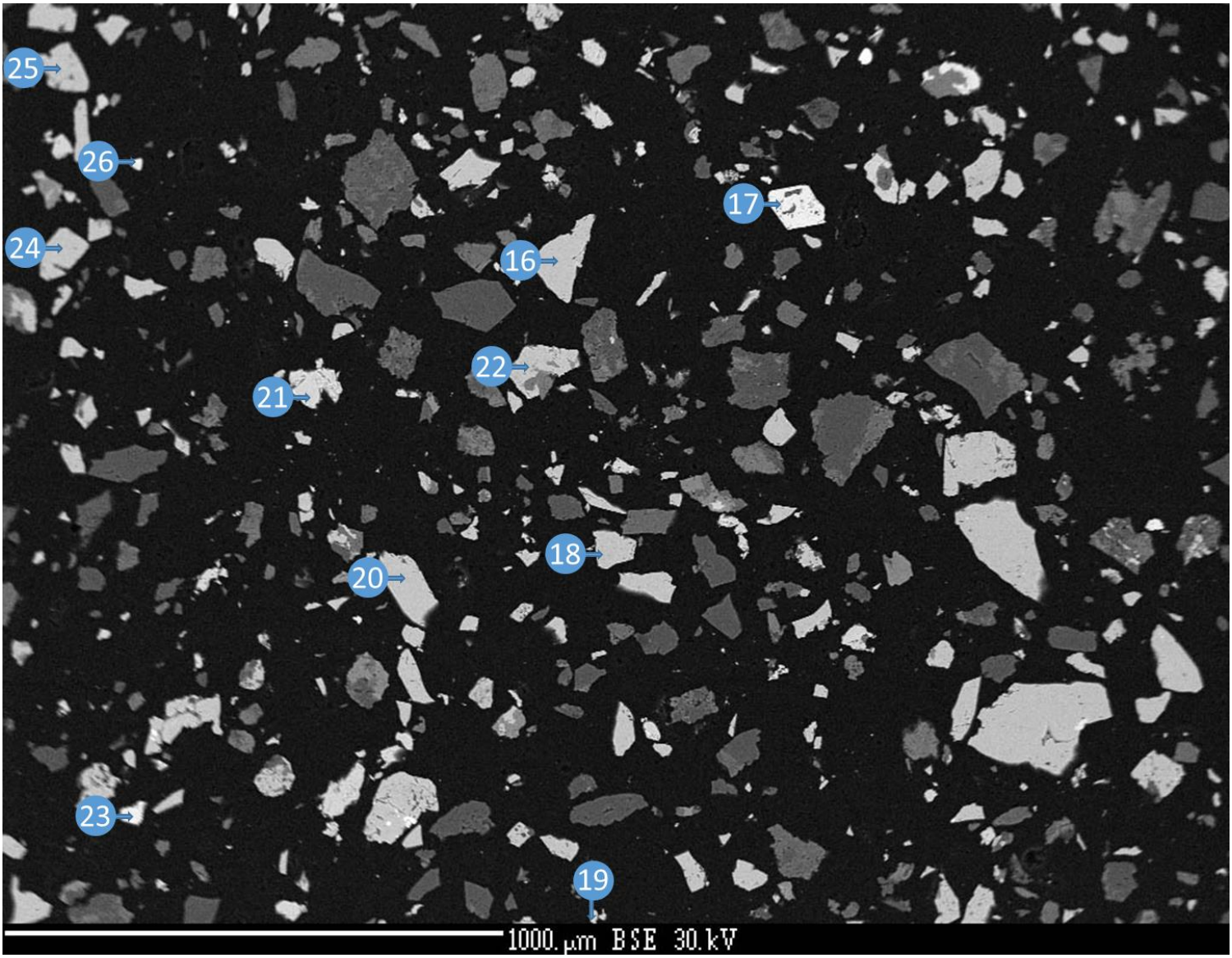


Figure 6-17 The points of analyses for the Barbrook material. Unfortunately, no gold was detected within the sulphides in this sample. Each point correlates with the data found in Table 6-5.

Table 6-5 The data determined by EPMA analysis for the Barbrook sample, the data set value corresponds to the labelled grains in Figure 6-17. No gold bearing grains were located in the Barbrook sample.

Barbrook EPMA Points							
Data Set	S	Fe	Cu	As	Sb	Au	Total
16	53.26	45.90	0.00	0.03	0.02	0.00	99.21
17	22.11	34.77	0.00	41.63	0.04	0.01	98.57
18	53.01	45.78	0.03	0.10	0.03	0.00	98.94
19	53.38	46.05	0.00	0.10	0.00	0.00	99.53
20	53.19	46.17	0.00	0.25	0.00	0.00	99.61
21	39.39	58.21	0.04	0.00	0.00	0.00	97.65
22	52.57	45.33	0.00	0.00	0.01	0.00	97.91
23	23.10	34.69	0.00	40.55	0.00	0.01	98.35
24	53.01	45.94	0.00	0.00	0.00	0.01	98.96
25	52.56	45.99	0.00	0.19	0.00	0.00	98.74
26	21.74	33.27	0.00	41.49	0.03	0.00	96.52

While only two gold-bearing grains were located in the Consolidated Murchison ore (Figure 6-18, Table 6-6), these grains both have the elemental composition of stibnite.

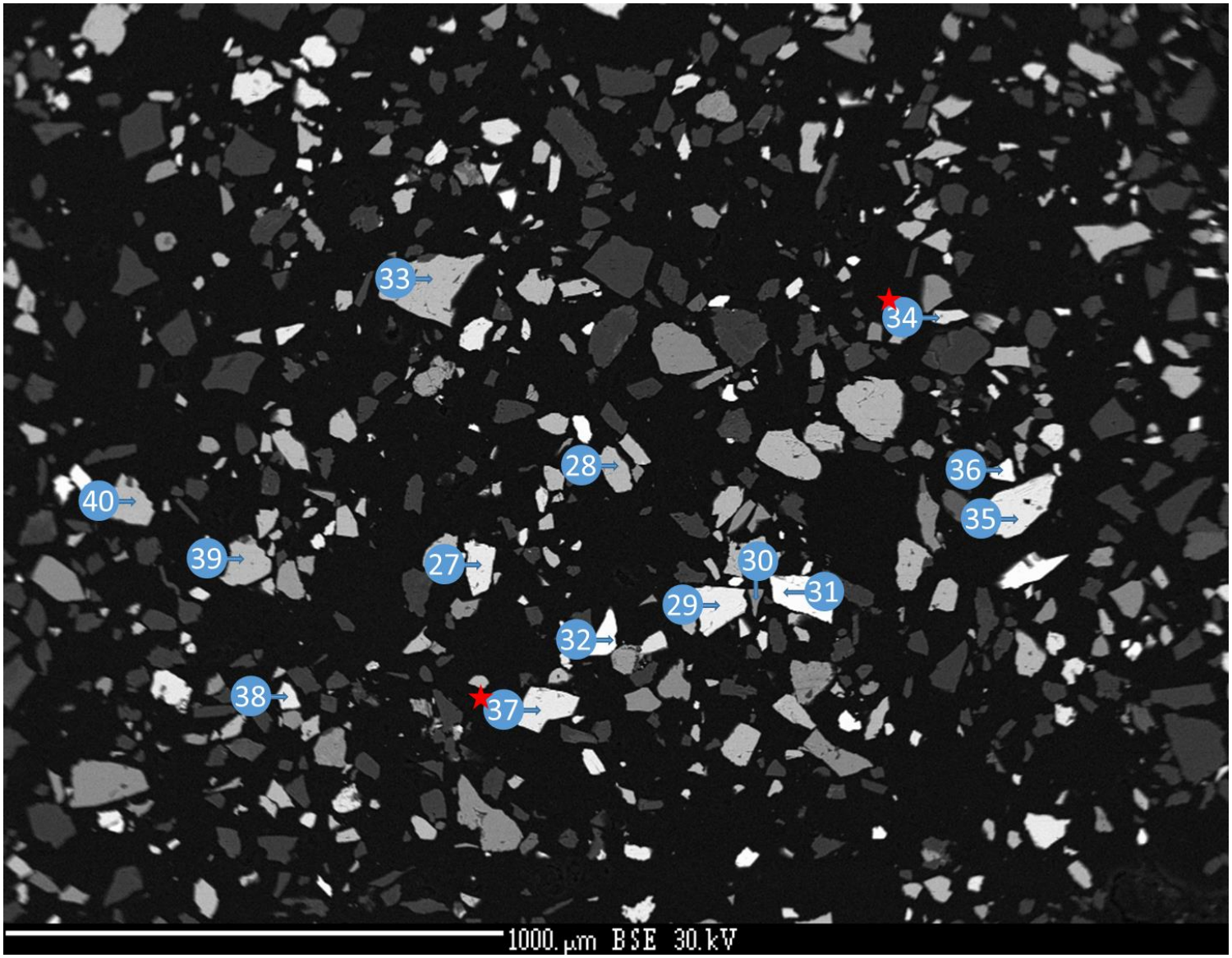
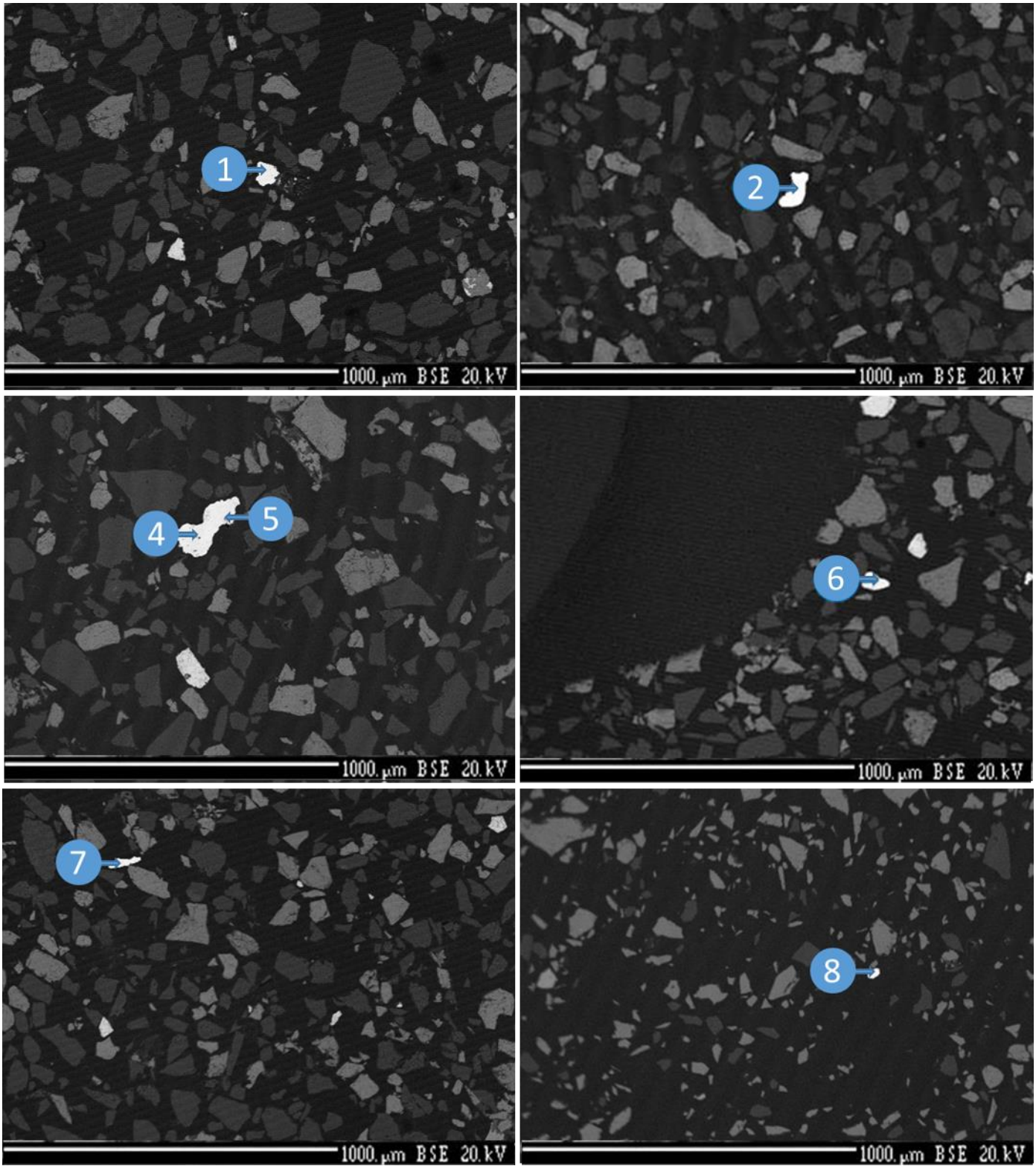


Figure 6-18 The points of analyses for the Consolidated Murchison material. Points with gold detected have been marked in red. Each point correlates with the data found in Table 6-6 and have the chemical composition of stibnite.

Table 6-6 The data determined by EPMA analysis for the Consolidated Murchison sample, which corresponds to the labelled grains in Figure 6-18. The two gold-bearing grains have been highlighted.

Cons Murch EPMA points							
Data sets	S	Fe	Cu	As	Sb	Au	Total
27	15.69	26.49	0.00	0.46	57.21	0.00	99.85
28	21.40	33.97	0.05	42.69	0.15	0.00	98.27
29	27.78	0.12	0.00	0.40	71.33	0.01	99.63
30	52.82	45.56	0.00	0.13	0.05	0.00	98.55
31	0.00	0.16	0.00	5.02	93.48	0.00	98.67
32	0.03	0.09	0.05	4.68	92.67	0.00	97.53
33	21.75	34.56	0.02	42.23	0.17	0.01	98.73
34	27.90	0.50	0.00	0.39	71.47	0.02	100.28
35	27.85	0.06	0.04	0.35	71.17	0.00	99.47
36	0.01	0.14	0.00	4.81	94.01	0.01	98.98
37	28.36	0.24	0.00	0.34	71.36	0.02	100.33
38	15.41	26.78	0.00	0.44	58.70	0.01	101.34
39	20.60	34.26	0.00	43.80	0.18	0.00	98.83
40	20.94	34.33	0.00	43.39	0.18	0.01	98.83

Analyses of the gold grains in the native gold samples (i.e. VCR and Pilgrim's Rest) were briefly done to ensure that these grains were typical and with no major components likely to affect the leaching results (Figure 6-19). Of primary interest was the ratio of gold to silver in "native gold" grains. The VCR sample had an average of 9.7% silver within the gold grains (Table 6-7), while the Pilgrim's Rest sample had predominantly lower silver grades (approximately 5.5%). However, a single grain within the Pilgrim's Rest sample had a distinctly elevated silver concentration of 21.3% which would theoretically fall within the range of electrum (Allan and Woodcock, 2001).


1
1000.μm BSE 20.kV
2
1000.μm BSE 20.kV
4
5
1000.μm BSE 20.kV
6
1000.μm BSE 20.kV
7
1000.μm BSE 20.kV
8
1000.μm BSE 20.kV

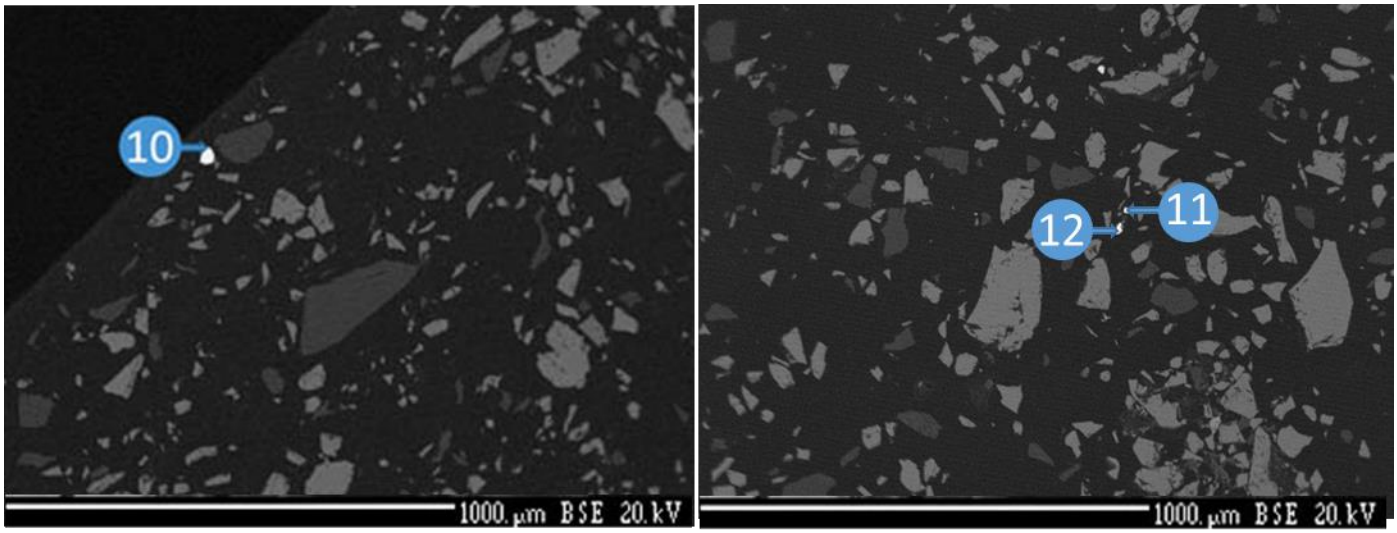


Figure 6-19 Gold grains analysed for Au/Ag content from the Ventersdorp Contact Reef sample (points labelled 1-7) and the Pilgrim's Rest sample (labelled 8-12). Most of the grains analysed were native gold with between 5% and 10% silver.

Table 6-7 EPMA data for gold grains from A) the Ventersdorp Contact Reef (VCR) sample and B) the Pilgrim's Rest sample. While most points located had varying degrees of gold and silver, point 3 was not gold-bearing.

A) VCR EPMA points			
Data Set	Ag	Au	Total
1	10.214	87.435	97.65
2	10.218	86.651	96.868
3	0.377	0	0.377
4	8.641	89.126	97.767
5	8.137	89.666	97.804
6	10.838	86.864	97.702
7	10.342	84.897	95.239

B) Pilgrim's Rest EPMA points			
Data sets	Ag	Au	Total
8	5.624	93.419	99.043
9	5.939	91.744	97.683
10	0	0	0
11	0	0	0
12	5.456	90.851	96.307
13	21.267	78.623	99.89

6.4. Discussion

The VCR sample had a range of gold grain sizes, from 0.5 to 425 µm. A significant proportion of total gold came from the coarse gold fraction (212 to 425 µm), which will affect the leach times required to extract all available gold. The extended leach times required for coarse gold relate to the amount of time needed for the lixiviant to access the entire particle and achieve complete dissolution. The VCR sample had excellent exposure (99.8%), and liberation (98.49%), while its limited mineral associations are unlikely to significantly interfere with any

extraction procedures. Pyrrhotite was present within the VCR sample however, only in generally very minor quantities. The native gold grains that were found in the VCR sample were quite typical with approximately 10% silver present.

The Fairview material, at a grind of 80% -75 µm, has visible native gold that was 72% liberated, with a surface exposure of 75%. Gold grains ranged in size from 0.5 µm to 425 µm, with the most significant weight percent contribution to total gold occurring in the range of 180 µm and 425 µm. The large range in grain size, as well as presence of coarse gold will affect the required leach time. Pyrite, arsenopyrite, pyrrhotite and micas were commonly associated with gold. The extent of the visible gold, relative to the assayed grade (12.5 g/ton Au) suggested that there was gold invisible to both optical petrology and MLA. EPMA work confirmed the presence of gold hosted within arsenopyrite, as predicted by several sources of literature (Anhaeusser, 2001, Anhaeusser, 2015).

In the Barbrook ore studied, the limited number of gold grains identified using MLA reduced the scope of the conclusions that could be made. However, based on all data obtained, the visible gold had a liberation of 97%, while 99% of gold was exposed for leaching. Increased grinding would have a negligible positive impact on the gold recovery. Of the gold grains found, the grain size distribution ranged from 1 µm to 53 µm. The gold tended to be associated with pyrite, arsenopyrite, ankerite and chlorite which is unlikely to significantly affect leaching conditions (Anheusser, 2001; Pearton and Viljoen, 2017). While it is plausible that there is refractory gold present in this sample, none was located which is likely to be due to low grades/concentrations.

The MLA results for Consolidated Murchison material, while restricted due to the number of gold-bearing minerals, suggested that the native gold had good liberation (88%) and exposure (85%), while the aurostibite had excellent liberation (96%) and exposure (100%). These grains ranged in size from 0.73 µm to 52 µm. Mineral associations included arsenopyrite (15.74%), pyrite (13.65%) and stibnite (11.57%). The presence of antimony-rich minerals, such as stibnite, will affect the gold leaching, due to both reagent consumption and passivation. While there was limited native gold, there was evidence for aurostibite from MLA results, as well as gold associated with stibnite from EPMA analyses. This confirms comments in the literature that state that gold is present in the Antimony Line in a number of forms (Davis, 1986; Vearncombe et al., 1988; Marsden and House, 2006).

In the Pilgrim's Rest sample gold occurred as discrete, fine grains, ranging in size from 0.75 µm to 53 µm. The gold present had small amounts of silver (5%), with a minor presence of electrum (with silver contributing more than 20% to a grain). The presence of electrum may have a deleterious effect on leaching rates as it tends to leach more slowly than gold in cyanide leaching and reduces overall leaching in the chloride-hypochlorite due to the formation of passivating AgCl. While the literature suggests that this deposit typically contains significant proportions of gold less than 5 µm in size (Swiegers, 1948), the weight percent contribution of gold from the sample that was less than 5 µm was insignificant. More than 85 wt% of the gold was from grains between 10 µm to 53 µm in size. Very fine grinding would not increase gold availability significantly. This assessment was supported by the MLA liberation and exposure data which show that the gold was well liberated (97%) and exposed (98%) with an 80% passing75 µm grind size. Major minerals present include pyrite and quartz which were unlikely to affect the gold extraction. However minor amounts of pyrrhotite and other sulphides have the potential to be deleterious.

In the context of VCR and Pilgrim's Rest, the native gold grains are well exposed and no additional gold is thought to be locked within other mineral types, as such any incomplete dissolution of the gold would need to be ascribed to factors relating to leach conditions such as reagent depletion or slow dissolution kinetics, rather than the nature of the ore. In contrast, the nature of the Fairview, Barbrook and Consolidated Murchison ores present dissolution issues due to their mineralogical nature, and thus potential insufficient dissolution could be due to the aforementioned factors in addition to non-ideal exposure/gold occurring locked within refractory sulphides.

Chapter 7

Formational Observations

7.1. Introduction

In recent years there has been a major deficit in the discovery of new gold deposits. Global gold exploration investment from 2007 to 2016 amounted to $65 billion, while the value of gold deposit discoveries in this same time period accounts for less than half of this (McKeith et al., 2010; Schodde, 2016). Additionally, in the past 15 years, of the discoveries that have been made, only an extremely limited number of these could be considered world-class (greater than 15 million ounces) (Holmes, 2017). This is a stark contrast from previous decades where discoveries containing more than 30 million ounces were not unusual (Holmes, 2017; Ashutosh, 2019). High grade deposits are also increasingly rare. In the last 50 years of mining the average grade of gold mined has decreased from 10 g/ton to 1.4 g/ton (Newman, 2019). This declining state has forced the gold industry to rely on alternative methods for ensuring sufficient resources and reserves are owned by individual companies in order to maintain competitive and relevant within the mining industry. Examples of these alternative methods include exploiting increasingly geologically/metallurgically complex resources and depending on business-based additions, such as mergers and acquisitions, to the reserves. Complex and refractory deposits are increasingly developed beyond the exploration stage due to a lack of feasible mining alternatives. Additionally, pre-existing deposits that have been depleted in free-milling ore are now being exploited for more refractory gold at depth. Important examples of these business-based additions is evident in many large companies such as Barrick and Newmont as an alternative, low risk approach to maintaining their reserves (Ashutosh, 2019). These alternatives methods currently appear to be cheaper and more effective than replacing gold lost to production by exploration, but they are unfortunately not sustainable long-term solutions.

The lack of new discoveries has been claimed by some to be due to a dearth of world class deposits remaining (Ashutosh, 2019), however it seems much more plausible that most of the readily discoverable deposits have been found and what remains are more enigmatic, sub-surficial deposits (Groves et al., 2018).

If these deposits exist at depth, new more targeted exploration methods will be required. The integration of theoretical and empirical geosciences as well as new exploration techniques will become vital for future successful discoveries (McCuaig and Hronsky, 2000; Groves et al., 2018). Different exploration techniques will be needed, as conventional methods result in high risk with limited outcome. It is imperative that these new methods be able to see through cover, some examples of which include geophysics, biogeochemistry and hydro-geochemistry (Desjardins, 2018). Of similar importance to improved techniques will be early stage targeting in order to avoid further high-risk exploration with limited outcomes. Targeting can include identifying regions associated with gold mineralisation both temporally and spatially. If during the targeting stage, areas are selected that would be less likely to contain refractory gold, this would be a major economic advantage. The knowledge of the formation of refractory deposits could be an important consideration. For this reason, understanding the geological controls of the formation of the different types of refractory ores may be important to the future of gold exploration, mining and processing.

7.2. Complex/Refractory Deposit Formation

There are several groups of gold deposit classified based on their genesis. Of these groups some are more likely to be refractory or complex than others. Free milling ores are more likely to have formed in alluvial deposits, reduced intrusions and low sulphidation epithermal deposits. In contrast, high sulphidation epithermal, Carlin, and of relevance to this study, orogenic greenstone deposits of disseminated stockwork type, are far more likely to be refractory (Rowe and Zhou, 2007).

There are a number of factors capable of affecting the relative recovery of gold from the respective deposits, which is evident in the processing procedures selected to optimise the recovery for each respective deposit. The use of biological oxidation at Fairview is used to ensure the destruction of gold-bearing sulphides. Whereas at Barbrook the many flotation steps, ultra-fine milling and pressure oxidation is used to counter the refractory gold and preg-robbing nature of the ore. Separation processes are utilised at Consolidated Murchison to reduce the passivating and reactive effect of antimony minerals, while high pressure, low alkalinity cyanidation is utilised to ensure the antimony-locked gold is effectively leached (Monhemius, 1987; Millard, 2005; Rohner and Millard, 2016). The heavy reliance of separation processes and discrete processing circuits for Pilgrim's Rest emphasises the effect

of the reactive nature of the abundant base metal sulphides and oxides associated with this ore.

The presence of abundant quantities of gold within the lattice of arsenopyrite is a common feature of refractory gold deposits and is evident in Fairview and Barbrook. The incorporation of gold into crystal lattices is thought to occur during mineralisation at lower greenschist facies temperatures and pressures. Increasing metamorphic conditions allows for the gold to be progressively expelled from the lattice, into zones of weakness within the mineral forming occlusions and crack-hosted gold. Increased metamorphism (above greenschist facies) will ultimately allow for the formation of gold outside the sulphides and within the gangue (Vaughan and Kyin, 2004).

Considering the notable absence of South African orogenic belts that have undergone high level metamorphism, the refractory nature of these deposits within South Africa is characteristic of their metamorphism and the geological history (Foster and Piper, 1993).

Arsenic enrichment, which is required to facilitate the formation of arsenopyrite occurs predominantly within sedimentary lithologies. The enrichment of arsenic in sedimentary rocks seems to occur during the sedimentation process, commonly to a great degree in finer grained lithologies (Tanaka, 1988).

Antimony is an important reactive element, particularly in the context of Consolidated Murchison. Antimony enrichment processes are not well constrained; however, a large-scale sampling project was conducted to determine antimony concentrations. Enrichments are generally found in sediments such as shales and argillites, in addition to andesites and rhyolites (Boyle and Jonasson, 1984).

Preg-robbing carbon is also a particularly common source of gold processing complications. However, the origin of carbon is wide-ranging and may form due to both biotic and abiotic sources. Additionally biotic factors are subject to numerous influences including oxygen conditions, burial rates, iron concentration, preservation and metamorphism (Welte, 1969; Pedersen and Calvert, 1990; Meyers, 2007). The variability of these controls is evident in the differences between Fairview and Barbrook, two gold deposits that are only a few kilometres apart, formed by the same mineralisation processes acting on the same stratigraphic lithologies, but forming distinctly different carbonaceous natures.

The presence of pyrrhotite is a common but deleterious processing factor in gold deposit and an example of such an ore in this study is Pilgrim's Rest. The formation of pyrrhotite occurs in intrusives, particularly those associated with a mafic to ultramafic system. In addition to intrusive systems pyrrhotite can also be associated with hydrothermal fluids (Vaughn, 1982; Gordon and McDonald, 2015). Mafic/ultramafic associated hydrothermal fluids are likely to form pyrrhotite, unfortunately this is not the only hydrothermal environment that can precipitate pyrrhotite. The ability for pyrrhotite to form in a number of different hydrothermal settings is due to the fact that the components required for formation (iron and sulphur) may have come from different sources during hydrothermal fluid movement (Reed and Palandri, 2006). Despite this, having an association with mafic magmatic material is still a good indication of a potential complex (high reagent consuming) nature, as demonstrated by Pilgrim's Rest (Kissin and Scott, 1982).

In addition to avoiding regions of disadvantageous geological history, one could also include temporal relationships by incorporating age data with the periods of mineralisation. Work by Groves et al. (2018) highlights the relationship between orogenic gold deposits and tectonic conditions, which can subsequently be related to certain time periods. Three major periods of orogenic gold mineralisation occur within the range of 2.75 and 2.55 Ga, 2.1 and 1.75 Ga and 600 to 50 Ma. If the ages of certain orogenic regions are known, in addition to their geological history, this data can be readily incorporated into targeted regions of exploration.

7.3. Discussion

The understanding of regional geological formations in conjunction with desktops studies could be an important factor in the early stages of deposit targeting which will be a critical phase of exploration in the future. Should it be possible to target regions with higher chances of gold mineralisation, particularly free-milling gold, it would certainly be a consideration worth factoring. Additionally, understanding the mineralogy and therefore the metallurgical nature of the deposit early in a project will prevent later complications. Utilising the geological history of a region will allow for more informed exploration decisions. This is currently highly relevant as the ongoing elevated gold prices may cause a surge in gold exploration.

Chapter 8

Hydrometallurgy Theory

8.1. Introduction

Once the ores have been successfully removed from the deposit, the extraction of the valuable metal or material of interest from the mined resource becomes of paramount importance. This process, known as extractive metallurgy, is conducted using three main methods, pyrometallurgy, hydrometallurgy and electrometallurgy. Should hydrometallurgy be a feasible extraction method, it tends to be the preferential operation processing in the gold industry, due to typically lower operating costs, moderate extraction temperatures as well as improved environmental considerations (Fleming, 1992; Smit, 2001).

Hydrometallurgy refers to metal extraction involving aqueous-based mechanisms. Examples of hydrometallurgical processes include leaching, solvent extraction, ion-exchange and precipitation. Of relevance to this work is leaching which involves the preferential dissolution of the metal/s of interest from the solid feed material to a liquid phase (Free, 2013). A number of variables must be controlled throughout the leach in order for it to be successful, these include pH, redox, reagent concentration, temperature, pressure etc. (Smit, 2001; Gupta, 2017).

8.2. Leaching Conditions

Selective leaching allows for the separation of the metal of interest (transferred to the liquid phase/pregnant solution) from the unwanted components of the ore. The unwanted components (residue) can be disposed of while the metal hosted within the solution can undergo subsequent recovery processes.

It is important that optimal leaching conditions are maintained to ensure that 1) sufficient quantities of the desired material are being extracted and minimal amounts are lost to the residue and 2) that only minor amounts of unwanted material should report to the pregnant solution, to provide an acceptable separation of desired and waste material for subsequent processing steps.

8.3. Leaching Systems

The degree to which a reaction takes place is controlled by a number of factors; the diffusion of the reagents to the surface of the mineral of interest, the ability of the reagents to absorb on to the surface of the relevant mineral, the desired reaction taking place, the removal of the desired product from the mineral surface and transferal into the bulk solution, and the rate at which these steps can occur. The slowest of all these steps will be known as the rate-determining step (Smit, 2001). While different leaching processes can be evaluated on a theoretical basis using thermodynamics and kinetics described above, a more relevant evaluation can also be done on the basis of

1) the degree to which the dissolution can take place (which would be a direct product of the previous factors)

2) the selectivity of leaching and

3) cost of leaching system (Smit, 2001; Free, 2013).

In order to ensure the highest degree of dissolution is maintained, optimal conditions must be maintained. Evaluating leaching based on selectivity is crucial as an ore will contain a variety of different minerals that have the potential to compete for leaching reagents, which will increase reagent consumption, reduce reactivity and increase unwanted components in the leach liquor. The cost of leach will significantly impact on the price of overall extraction. The nature of the leach will also directly impact on the cost of equipment (Gos and Rubo, 2001).

8.4. Bottle Roll Leaching

Bottle roll leaching is a common analytical tool which involves exposing the ore to leach solution in bottles that are rolled, to induce agitation. This technique is an important first step in understanding the ores response to the selected leaching system on a small scale. This evaluation will consider important factors such as the total amount of leachable metal, reagent consumption and degree of monitoring required. Bottle roll test work acts as the basis for which leaches are assessed and considered for upscaling (Cetin et al., 2017).

8.5. Leaching Factors and Variables

To ensure effective leaching takes place, several parameters can be varied and controlled. Some of these variables will be dependent on the nature of the leaching system, for example reagent concentration, pH, temperature and catalysts. While other components such as particle size and leaching time may be optimised according to the individual ores rather than the leaching system.

8.5.1. Intrinsic Variables

Variables intrinsic to the leach, must be controlled to ensure that the desired reaction occurs to the maximum extent. While some of these factors could vary slightly from ore to ore, there will be a leaching standard. The leach agents and their respective concentrations are the basis of any leaching system, without the correct components such as oxidising and complexing agents, no leaching could effectively take place. While increasing concentrations would tend to increase the reaction rate, overly high concentrations may increase the reagent consumption while only having a moderate effects on the kinetics of the desired interaction (Hilson and Monhemius, 2006b). Therefore, reagent concentration is a balance of kinetic and economic factors. Increasing concentration of catalysts (should any be required) would, within a certain range, increase the reaction rate, however this would stop when sufficient catalyst is present, or another factor becomes limiting.

The pH and reduction potential represent two factors that significantly affect the performance of a leaching system. Under certain conditions these factors can be subject to rapid and unpredictable change and therefore must be a priority for optimising leaching. pH refers to the negative common logarithm of the activity of hydrogen ions, and more simply describes the acidity/alkalinity of the solution (Brown, 2009). Reduction potential is a measure of the tendency of a chemical species to gain an electron and thereby be reduced (Brown, 2009; Atkins and De Paula, 2013). These factors are important as they control the nature and stability of the complexes/species, this in turn controls the ability of metals of interest to dissolve in solution. During the extraction processes, certain minerals can be more precisely targeted by controlling these conditions, which is important to both increase the extraction of the desired metal and reduce the extraction of undesirable metals. Most reagents have a window of operating conditions where the extraction is efficient. This

window is notably small for the alternative lixiviants under consideration (as seen in Figure 1-2) (Aylmore, 2016a). Controlling these factors is important, not only to ensure effective leaching, but to reduce costs associated with excessive reagent consumption, subsequent extraction processes, and in certain instances even safety. A notable example of controlling pH for safety is found in cyanide leaching, when the cyanide-bearing solution is maintained at a pH of between 10 and 11. At more acidic conditions cyanide becomes increasingly stable as lethal HCN (Johnson, 2015).

Factors such as pressure and temperature significantly affect the kinetics of the reaction. While some reactions will require higher temperatures to initiate (Brown, 2009), almost all reactions operate faster at higher temperatures (with the exception of certain precipitation reactions). In this work all the reactions required for leaching are capable of operating at ambient conditions. Additionally, the financial considerations of large-scale leaching at elevated temperatures tend not to be feasible, despite the increased kinetics of this leach. The same rationale is applied to pressure which tends only to be economic when treating concentrates, rather than whole ores.

8.5.2. Extrinsic Variables

Extrinsic variables are defined as variables unrelated to the specific leaching system used, and can either be optimised to individual ores, or standardised for consistency. While the leaching system may be moderately affected by these variables (and vice versa), it is not a dominant control. Particle size is an important example of an extrinsic variable. The particle size required for a successful leach will be strongly dependant on the nature of the mineralogy of the deposit (Egan et al., 2016), rather than the extraction technique. The ore must undergo comminution to sufficiently expose the desired material to the leaching solution. For leaching processes, the degree of liberation (i.e. the degree to which the economic mineral comprises the host particle) is of less importance than the surface exposure which allows the solution to interact and dissolve the metal of interest (Egan et al., 2016).

Pulp density of a leach solution and its optimal conditions are not commonly described in literature; however, it is typically maintained between 35 to 50% (Marsden and House, 2006). The optimal percentage is based on several factors including specific gravity of ore as well as

particle size. The selected pulp density should be based on a balance between the increased mass transport associated with low densities, and the increased retention time associated with high densities (Marsden and House, 2006). Reagent consumption can also be reduced with higher densities, as lower dosages are required to achieve desired concentrations.

The total volume of the sample is important to ensure material is representative of the ore of study. This is especially important when working with coarse-grained gold, where increased masses may reduce the appearance of the nugget effect (Dominy et al., 2003).

The rotation of bottles is a factor that is typically consistent, although this can be adjusted depending the nature of the bottle roller. This component is important as the agitation caused by bottle rolling will improve the diffusion of reagents, increasing the rate of leaching.

Time could be considered both an intrinsic and extrinsic variable, as certain leaches will operate under specific kinetic conditions, controlling leaching rates, and therefore the time required for efficient extraction is controlled by the leaching system (Aylmore, 2016a). However, time could also be considered an extrinsic factor as the nature of the ore and its mineralogy will also have a distinct control on the time required for efficient leaching. Time can be standardised for different leaches, relative to their kinetics, to allow for a consistent comparison of different leaching systems, as will be done in subsequent works.

8.6. Conditions for Leaching Test

In preparation for leaching, samples were precisely weighed out and mixed with appropriate quantities of water, and subsequently reagent. The pH and Eh conditions were maintained at target throughout the leach by the addition of adjusting reagents, as required. A summary of the selected leaching conditions used in this work (based on literature) for both cyanide as well as alternative gold extraction, including parameters such as percent solids, pH, Eh, duration, reagent addition have been summarised in Table 8-1. Extrinsic variables such as grind size, rotation speed, temperature and time are held constant to evaluate the leaching systems fairly.

The leach periods were selected based on the kinetic information relating to the individual leaches, in addition to the mineralogical nature of the ore. The rate of reactions reported by Senanayake (2004) and Hasab et al. (2014) in addition to the nature of the gold as

characterised in Chapter 6 were considered to ensure that sufficient leach times were given for complete dissolution to prevent the results from being hindered by time.

Once leaching was completed, the solution was separated through filtration, into pregnant leach liquor and residue. The residue was thoroughly washed, and both the liquor and the residue were submitted for the appropriate assays. The wash solution was made to simulate the conditions during leaching, to prevent the precipitation of gold in solution due to pH or Eh changes. The wash solution was prepared by adding the appropriate pH and Eh adjusting reagents, and approximately 30% of the active reagent.

8.6.1. Cyanide

Standardised leaching conditions for cyanidation are well defined and are described below, based on conditions found in the literature (Schaeffer, 1882; Eisele et al., 1988; Dadgar, 1989; Sparrow and Woodcock, 1995; Jeffrey et al., 2001; Swanepoel, 2001). These are the conditions at which the leaches were operated and maintained.

Duration of cyanide leaching can typically range from 10 – 72 hours. However, after a study of the nature of the gold from the respective deposits, it is evident that there is no extremely coarse-grained gold that requires extended leaching periods, and therefore a 24-hour time span was selected as an appropriate for the leach.

Reagent Addition for cyanide is variable (ranging from 0.2 – 2.5 g/L) depending on the ore. Should the ores be highly reactive the required concentrations will be high. However, the lowest effective concentrations are desirable, not only due to the costs associated with the purchase of cyanide but also the subsequent disposal procedures required to reduce any environmental and safety hazards (Laitos, 2012). A high sodium cyanide concentration of 1 g/L was selected to ensure sufficient reagent.

pH is typically maintained between 10 and 11, 10.5 was selected for this work.

Temperature and Pressure were maintained at ambient conditions, under which the leach can successfully operate. Ambient conditions tend to be the most economic setup. In addition to the cyanide leaching systems, the alternatives can also operate in ambient conditions and therefore these variables will be consistent across tests.

Percent solids of 50 is standard for most gold processing and were used consistently across all bottle roll leaching.

In addition to the conditions outlined above, CIL (carbon-in-leach) was also implemented for all cyanide leaches to account for the potential presence of preg-robbing carbon. As the use of CIL is commonly utilised to address preg-robbing carbon, CIL was included to ensure that cyanidation was not disadvantaged by problems that have already been addressed in the industry.

8.6.2. Chloride-Hypochlorite

The conditions for the chloride-hypochlorite leaching system have been describe extensively in the literature (Puvvada and Murthy, 2000; Jeffrey et al., 2001; Baghalha, 2007; Nam et al., 2008; Hasab et al., 2013; Hasab et al., 2014; Adams, 2016). The conditions described are those that were selected for leaching.

Duration for this leaching system was 8 hours. This is due to both rapid leaching kinetics, and the tendency of the gold chloride complex to break down with time causing the precipitation of native gold.

Reagent Addition described in the literature tends to be quite consistent, with chloride concentrations of 80 g/L, and 10 g/L sodium hypochlorite recommended for faster leaching

pH required to ensure the most stable conditions for gold chloride complexes must be kept at a pH of 4.5.

8.6.3. Thiourea

The conditions for the thiourea and ferric iron leaching system have been selected and used, based on several articles in the literature (Groenewald, 1976; Chen et al., 1980; Schulze, 1984; Eisele et al., 1988; Ubaldini et al., 1998; Li and Miller, 2006).

Duration of thiourea-based leaches tends to be shorter than the cyanide leaching systems due to rapid kinetics of the gold dissolution. In order to avoid the breakdown of thiourea as well as allowing this leach the opportunity to display its kinetic advantage, the leaching duration was set at 8 hours.

Reagent Addition found in the literature for thiourea ranges from 1 g/L to 10 g/L, with 3-5 g/L addition of ferric iron. The higher end of the concentrations (10 g/L thiourea and 5 g/L iron) were selected for the leaches conducted. Redox potential should be maintained around 400 mV.

pH for optimal leaching conditions, should be maintained between 1 and 2.

8.6.4. Thiosulphate

The conditions for thiosulphate, copper, ammonia leaching system described in the literature are quite variable (Chen et al., 1980; Schulze, 1984; Langhans et al., 1992; Abbruzzese et al., 1995; Ubaldini et al., 1998; Aylmore and Muir, 2001; Jeffrey et al., 2001; Aylmore, 2016b).

Duration of this leach was selected to be 24 hours, as the kinetics of this system are thought to be similar to that of cyanide. Similar leach lengths would allow for an equal comparison of these leaching techniques.

Reagent Addition described in the literature seems to define two end members, high and low reagent concentration (with little mention of intermediate conditions). The high concentration end members have concentrations of around 40 g/L thiosulphate, 34 g/L ammonia and 6 g/L copper. The lower concentration conditions, which are used to minimise the decomposition of reagents (especially thiosulphate) have typical concentrations of 22.5 g/L thiosulphate, varying concentrations of ammonia, which can be both low (3.4 g/L) or more commonly high (65 g/L), 2 g/L copper and in some instances glycine as a stabiliser (3 g/L). Oxygen fugacity is commonly atmospheric. In order to ensure that sufficient reactivity occurs, high concentration conditions, 40g/L thiosulphate and 6 g/L copper were used, as this is consistent with the selected conditions of cyanide leaching

pH required for successful leaching can range from 7-11, however optimal results seem to be achieved at a pH of 9.5.

*Temperature*s used in this leaching system range from ambient to up to 80˚C. Ambient conditions will be used in these leaches (as well as all other systems to ensure consistent conditions for evaluation).

Percent solids used in literature tends to be lower percent solids than typical gold leaches, however there is no quoted reason for this and therefore a consistent percent solid (50%) was chosen.

Table 8-1 Summary of the typical leaching conditions for the cyanide, thiosulphate, thiourea and chloride-hypochlorite respective leaching systems.

Leaching system	Time (hrs)	Component addition (g/L)	Reagent addition (g/L)	pH (controlling reagent)	Eh (controlling reagent)	Temperature (°C)	Percent Solids
Cyanide	24	.85 g/L CN^-	1.6 - 2.4 g/L NaCN	10.5	Ambient	Ambient	50%
			0.26 - 3.52 g/L $Ca(OH)_2$	$Ca(OH)_2$	Air		
Thiosulphate	24	40 g/L S_2O_3	56.40 g/L $Na_2S_2O_3$	10.5	150-250 mV	Ambient	50%
		34 g/L NH_3	34 g/L NH_3	NH_3	Cu (II)		
		6 g/L Cu	15.07g/L $CuSO_4$				
Thiourea	8	10 g/L CH_4N_2S	10 g/L CH_4N_2S	1.5	400 mV	Ambient	50%
		5 g/L Fe (III)	$Fe_2(SO_4)_3$ (III)	H_2SO_4	$Fe_2(SO_4)_3$		
Chlorine	8	80 g/L Cl^-	100g/L NaCl	2.5	1000 mV	Ambient	50%
		10 g/L ClO^-	14.47 g/L NaClO	HCl	NaClO		

8.7. Titrations

Understanding the consumptions of leaching reagents is an important part of evaluating the feasibility of a leaching system. High reagent consumption incurs additional expenses in order to ensure sufficient gold dissolution.

Each leaching system will require certain titrations to determine respective consumptions, a summary of this is in Table 8-2. The reagent preparation for these titrations is in Appendix C.

8.7.1. Cyanide

The concentration of cyanide in solution can be determined using titration. While many methods exist and were utilised in the past (Pohlandt, 1983) the industry standard is now the complexation of free cyanide with silver nitrate. The end point is determined with the addition of a rhodamine indicator (Breuer et al., 2011). When a known quantity of silver nitrate is added to a solution containing unknown quantities of cyanide, they complex to form silver cyanide (Equation 8.1).

$$Ag^+ + 2CN^- \rightarrow Ag(CN)_2^- \qquad \text{(Equation 8.1)}$$

This method is used to determine the free cyanide content in solutions, however there are known interferences e.g. the presence of copper. Copper complexes with cyanide and prevents gold leaching, however this complex will disassociate and report as free cyanide during the titration (Barnes et al., 2000).

8.7.2. Chloride-Hypochlorite

While it is possible to determine the chloride concentration using argentometric titration which is described below, this is of limited relevance to the chloride-hypochlorite leach. The limitations of this titration are relevant exclusively to this leaching system as excess sodium chloride is added, where the chloride would detach from sodium in order to bond with silver, reporting significantly higher chloride concentrations, and making the titration results somewhat irrelevant. While the titration results for respective leaches are not reported in the subsequent chapters, the method has been described below.

Argentometry is a method used to determine the amount of chlorides present, with a solution of silver nitrate as a titrant. There are three common methods that are associated with argentometric titrations: the Mohr method, the Volhard method and the Fajans method, each of which has different conditions and indicators (Harris, 2010; Skoog et al., 2013).

The Volhard method is the only argentometric titration effective under acidic conditions, and therefore would be used to determine the reagent consumption for the chloride-hypochlorite based leach. The Volhard method is a more complex titration than the previous and subsequent systems, as it is what is known as a back titration. This back titration involves adding a known, excess amount of silver nitrate. Only a certain amount of the silver nitrate will complex with chlorides, forming silver chlorides (Equation 8.2). The precise amount of silver nitrate remaining is determined using an additional titration with potassium thiocyanate (Equation 8.3) and a ferric iron indicator (Harris, 2010), which is used to determine the original concentration of chloride ions in a solution.

$$Cl^- + Ag^+ \rightarrow AgCl + Ag \qquad \text{(Equation 8.2)}$$

$$Ag+ SCN^- + Fe^{3+} \rightarrow AgSCN + FeSCN \qquad \text{(Equation 8.3)}$$

Unfortunately, silver thiocyanate can replace silver chlorides formed during the excess addition of silver nitrate. To prevent this affecting the titration an organic liquid is added, which allows the silver chloride to be separated from water, leaving it inaccessible to thiocyanate interaction (Caldwell and Moyer, 1935).

8.7.3. Thiourea

Determining thiourea consumptions using titration methods have been briefly mentioned in literature (Schmidt et al., 1993; Örgül and Atalay, 2002; Gönen et al., 2007), however the most commonly utilised method is a complex formation titration using potassium iodate (KIO_3), in conjunction with a starch indicator. However, experimental work has confirmed that the titration used for thiosulphate is also effective in this instance, with a more clearly defined endpoint and more consistent result than the methods described in the literature. Thus, thiourea consumptions were determined using iodine-based titrations. The reaction of interest operates by Equation 8.4 (Calokerinos and Hadjiioannou, 1983; Deutsch, 2012) where the thiourea is oxidised by iodine.

$$2C_4N_2S + I \rightarrow C_8N_4S_2 + I^- + 3H_2O \qquad \text{(Equation 8.4)}$$

Once the iodine has reacted with all the thiourea present, free iodine becomes available, the soluble starch stains the solution a deep blue-black colour. The exact thiourea concentrations may be difficult to determine due to interference from dissolved iron, sulphates and other salts (Örgül and Atalay, 2002).

8.7.4. Thiosulphate

The concentration (and therefore consumption) of thiosulphate in solution can be determined using an iodimetric titration (Zipperian et al., 1988; Deutsch, 2012). In this titration, a known concentration of iodine solution is titrated against an unknown concentration of thiosulphate-bearing leach solution (Deutsch, 2012). The thiosulphate iodine titration operates by the Equation 8.5, where thiosulphate is oxidised by iodine.

$$2S_2O_3^{2-} + I \rightarrow S_4O_6^{2-} + 2I^- \qquad \text{(Equation 8.5)}$$

When the above reaction has consumed all thiosulphate, free iodine becomes available and the soluble starch stains the solution a deep blue-black colour (Ha et al., 2010). This titration however has a number of potential interfering agents, including the copper complexes which

may reduce obscure the colour change associated with the end point of the titration (Deutsch, 2012).

Table 8-2 Summary of the conditions required for titration to determine the reagent concentration (and subsequently the consumptions) for the respective leaching systems.

Leaching system	Reagent	Titration System	Titrant	Concentration	End point change
Thiosulphate	S_2O_3	Iodimetric	Iodine solution	0.01 M	Deep blue
			Starch indicator	2 ml of 1%	
Thiourea	CH_4N_2S	Iodimetric	Iodate solution	0.1 M	Deep blue
			Starch indicator	2 ml of 1%	
Chlorine	Cl^-	Volhard	Silver nitrate solution	180 mL of 0.1 M*	Dark red
			Potassium thiocyanate	0.1 M	
			Ferric sulphate	1 mL saturated	
			Choroform	3 mL	

8.8. Safety Precautions

Safety was made a paramount priority throughout all laboratory work. While working with the reagents and conditions as described in section 6 (leaching conditions) of this chapter some inherent dangers were present, however an extensive effort was made to ensure this work was conducted in the safest environment possible. Dangers relating to the work were identified and ranked based on likelihood and severity. The highest ranked concern related to the interaction with chemicals, particularly the potentially carcinogenic thiourea, which emphasised the importance of the use of personal protective equipment (PPE), the fume hood and spill management strategies. Materials safety data sheets (MSDS) were available for all reagents in use. Incompatible reagents were kept in separate, well demarcated storage regions, while reagent waste was disposed of into separate and clearly marked acid and general waste containers. The lab was equipped with a safety shower, eye wash station and first aid kit. Dust inhalation when handling milled samples is very likely and thus dust masks and fume cupboards were always utilised. Additional safety precautions taken included ensuring loose clothes are not present in the lab, particularly around the bottle roller.

8.9. Discussion

The metallurgical extraction of the commodity of interest is a vital step in the value chain of the minerals industry. Hydrometallurgical extraction/leaching is the generally accepted method for gold recovery, which allows the gold to be selectively transferred away from the gangue components of the ore. Cyanide, while being the typical lixiviant for gold recovery, is not the only option. In order to evaluate alternatives to cyanide, the use of bottle roll leaching was implemented, which is a common small scale technique. Leaching conditions must be maintained at their respective optimal conditions to ensure high levels of recovery of the component of interest, and low levels of unwanted materials. Factors that should be controlled include pH, Eh, leach duration, reagent concentration, temperature and pressure. The conditions used for the leaching experiments were selected based on conditions described in the literature. Generally, more extreme conditions were selected for each leach system to ensure none were disadvantaged due to unexpected consumptions.

All leaches were operated at ambient temperature and pressure conditions and a solid percentage of 50. Cyanide was leached for 24 hours, with a sodium cyanide concentration of 1 g/L and a pH of 10.5. Chloride-hypochlorite leaching was conducted over 8 hours, while the pH was kept below 4.5. Chloride was added to ensure an initial concentration of 80 g/L, while 10 g/L sodium hypochlorite was added to maintain a high Eh. Thiourea was also leached over an 8 hour period, at a pH of 1.5. The initial thiourea concentration was 10 g/L while 5 g/L ferric iron was also required. Thiosulphate was leached over 24 hours, with 40 g/L thiourea and 6 g/L copper. The optimal pH of 9.5 was maintained for this leach.

Determining the concentrations and consumptions at the end of the leaching period was vital to evaluate leaching. Cyanide values were determined using free acid titrations with silver nitrate and a rhodamine indicator. Thiourea and thiosulphate concentrations were determined using an iodimetric titration in the presence of a starch indicator.

Safe laboratory practises were implemented throughout the leaching and titration procedures.

Chapter 9

Experimental Leaching

9.1. Introduction

The lixiviants of study; cyanide, thiourea, thiosulphate and chloride-hypochlorite were evaluated using bottle roll leaching, where each lixiviant was to be tested in conjunction with each ore in order to appropriately evaluate the lixiviant in a variety of mineralogical contexts. The target conditions for respective leaches were outlined in Chapter 8 and throughout the leaching period these conditions were monitored and maintained as far as possible. Not all leaches were successfully completed, due to the limitations of certain techniques when used in conjunction with reactive ores. However, the leaches that were successfully completed were evaluated by assaying the gold in the leaching components (residue and leach liquor).

9.2. Cyanide Leaching

The precise conditions maintained during cyanide leaching are given in Appendix C. A one-hour precondition step was completed, which involved adjusting the pH to the target range, to ensure the pH is stable when the cyanide is added. This preconditioning step is important for safety reasons to ensure that cyanide is not added to the leach at dangerous pH conditions (where hydrogen cyanide formation is favoured).

Once leaching was completed, the leached pulp was separated through filtration, into pregnant leach liquor and residue. The residue was thoroughly re-pulped, washed and filtered several times, and both the liquor and the residue were submitted for the required assays. Gold grades of the leach liquor and residue are vital for understanding the success of leaching. The success of a leach can also be evaluated based on reagent consumption which will be discussed in Chapter 11. The gold dissolution is calculated using the head grade relative to the assayed residue or solution (Table 9-1). Where the gold is stable in solution, a quality control step was implemented to determine the accountability of gold. Accountability is calculated by comparing the gold in the leaching products, the leach solution and the residue, to the gold in the feed as a percentage. While an accountability nearing 100% is most desirable, in the context of the sample grades and precision of the chemical analyses, accountabilities within 10% of 100% were considered acceptable. Where the gold is unstable

in solution over extended periods, this control step is not applicable. While a certain degree of gold precipitation will occur during leaching, causing gold to report to the residue and maintaining accountability, a significant amount of gold will precipitate from solution between leaching and assay which would produce low accountabilities.

The gold dissolution calculation incorporates sample mass differences associated with the dissolution of gangue mineralogy (Equation 9.1). Similarly, the accountability data, when present, was also calculated with consideration of mass balancing (Equation 9.2).

$$\frac{(\text{Feed Au Grade x Feed Mass}) - (\text{Residue Au Grade x Residue Mass}) \text{ x } 100}{\text{Feed Au Grade x Feed Mass}} \quad \text{(Equation 9.1)}$$

$$\frac{(\text{Residue Au Grade x Residue Mass}) + (\text{Solution Au Grade x Solution Mass}) \text{ x } 100}{\text{Feed Au Grade x Feed Mass}} \quad \text{(Equation 9.2)}$$

9.2.1. Cyanide Leaching Results

The range in gold dissolution for different samples emphasises the variability in ore types, with dissolution ranging from just less than 100% to 14%. The VCR sample had almost complete dissolution, further emphasising its free milling nature. The Pilgrim's Rest sample, despite having had the highest consumptions for both cyanide and lime, had a dissolution of almost 90%. The Fairview sample had the next best gold dissolution; however, this degree of dissolution is still low with an average of 66.6%. Both the Barbrook and Consolidated Murchison samples attained very poor dissolutions, with 26.6% and 14% respectively.

Table 9-1 The gold assay results based on cyanide leaching, calculated gold dissolution % and accountability. The calculated dissolution was determined by the differences in mass and grade of material from feed to residue. Additional information regarding masses etc. used to calculate the dissolution percentage is given in Appendix C.

Ore	Feed		Residue		Dissolution	Accountability
	Mass	Au Grade	Mass	Au Grade	Feed/Res	-
	g	g/t	g	g/t	%	%
VCR	433.7	23.60	432.5	0.93	96.07	104.7
Fairview	478.1	12.50	477.9	3.98	68.17	96.5
Barbrook	457.3	2.08	453.9	1.59	24.13	104.3
Consolidated Murchison	509.8	3.05	505.9	2.42	21.26	92.3
Pilgrim's Rest	453.8	29.40	450.5	3.27	88.96	103.0

9.3. Chloride-Hypochlorite Leaching

Samples ranging between 400 and 500 g were mixed with equivalent masses of water, with the appropriate reagent additions to ensure that a minimum concentration of 100 g/L of sodium chloride and 21.5 g/L of sodium hypochlorite was achieved. The chloride-hypochlorite leaches were run over an 8-hour period at ambient temperature and pressure conditions. The pH was maintained using hydrochloric acid, and the redox using the sodium hypochlorite. The precise conditions maintained during leaching are found in Appendix C, however the conditions were maintained at a pH of 4.5, while the Eh was generally 1100 mV. In order to maintain target conditions, the pH and redox were determined and adjusted on an hourly basis.

9.3.1. Chloride-Hypochlorite Leaching Results

The leach was found to be extremely unstable as well as highly acid consuming. The consumption, reactivity and instability of this leaching technique was so extensive that it was only possible to successfully complete leaching on the VCR material. The reagent consumption will be more extensively covered in the Chapter 11. The dissolution of the gold within the VCR sample in the chloride-hypochlorite leach achieved was 39% (Table 9-2). The limited degree of dissolution found in the ore cannot be explained by instability of the gold-chloride complex in solution, as this gold was present during assay, and the accountability is good. Therefore, it is necessary to gain a better understanding of the characteristic of the remaining gold, which is considered in Chapter 10.

Table 9-2 Gold dissolution data for the chloride-hypochlorite leach of the Ventersdorp Contact Reef sample. Dissolution was calculated from the difference in mass and grade of material from feed to residue. While accountability of gold was high, dissolution was limited.

Feed		Residue		Dissolution	Accountability
Mass	Au Grade	Mass	Au Grade	Feed/Res	-
g	g/t	g	g/t	%	%
436.10	23.6	417.14	15.0	39.20	98.7

9.4. Thiourea Leaching

The thiourea leaching responses were evaluated using samples ranging between 400 and 500 g, this material was mixed with equivalent masses of water, while the appropriate reagent masses were added to ensure that there was a minimum concentration of 10 g/L thiourea,

and 5 g/L ferric sulphate. This leach was run over a period of 8 hours at ambient temperature and pressure conditions. The pH was maintained using sulphuric acid as required, while the redox conditions were achieved using the copper sulphate. The precise conditions maintained during leaching are presented in Appendix C, however the overall pH of 1.5 was maintained, while the Eh was generally 420 mV.

9.4.1. Thiourea Leaching Results

The appropriate leaching conditions were achieved only for the VCR and Pilgrim's Rest samples. This was largely due to the reagent consumptions associated with this leach which were unfeasibly high for the Consolidated Murchison, Fairview and Barbrook samples. The difference in dissolutions achieved by the VCR and Pilgrim's Rest samples were significant (Table 9-3), however neither achieved sufficient degrees of dissolution relative to that achieved using cyanide leaching. The free milling gold found within the VCR sample reached a dissolution of 30%, while the Pilgrim's Rest sample only achieved dissolutions of 14%.

Table 9-3 The thiourea leaching results for Ventersdorp Contact Reef (VCR) and Pilgrim's Rest. The gold dissolution has been calculated from the difference in mass and grade of material from feed to residue. VCR dissolution is significantly higher than that of Pilgrim's Rest

Ore	Feed		Residue		Dissolution
	Mass g	Au Grade g/t	Mass g	Au Grade g/t	Feed/Res %
VCR	440.6	23.6	426.7	17.0	30.2
Pilgrim's Rest	452.1	29.6	425.4	26.9	14.5

9.5. Thiosulphate leaching

A range of thiosulphate leach conditions were tested, including glycine present and absent leaching on both calcined and uncalcined ores. The ore material was subdivided into aliquots of between 400 and 500 g, where they were leached with the equivalent mass of water (to ensure a 50% solid percentage), while the appropriate amount of reagent was added to ensure approximately 60 g/L hydrated sodium thiosulphate, 7.5 g/L copper sulphate and 3 g/L glycine. The pH was maintained using ammonia and sulphuric acid as needed. The leaches were run over a 24-hour period, with the conditions adjusted approximately once an hour until stable conditions were achieved. The ores, when calcined, were done so at 540˚C, over a 12-hour period, with large surface exposure to allow for sufficient oxygen interaction. The leach results have been subdivided into different thiosulphate leaches; thiosulphate leaching

on uncalcined ores, glycine and thiosulphate leaches on uncalcined ores, thiosulphate leaches on calcined ore and finally glycine and thiosulphate leaches on calcined ores.

9.5.1. Thiosulphate Leaching Results

The original thiosulphate leaches had varying but limited results (Table 9-4), the free milling ores experienced the highest gold dissolution with the VCR sample achieving 42% and the Pilgrim's Rest material with 46%. Of the remaining refractory samples, the Barbrook sample achieved 5% dissolution, however at these concentrations it seems likely that the represented 5% dissolution is as likely a sampling or assay error as much as it is actual dissolution. The remaining ores from Fairview and Consolidated Murchison samples both achieved 0% dissolution. To gain a better understand as to the repeatability of these leaches, two additional leaches were run using identical conditions on the Pilgrim's Rest material, with dissolutions achieving identical results within the error of the assays.

Table 9-4 The thiosulphate leaching results for the Ventersdorp Contact Reef (VCR), Fairview, Barbrook, Consolidated Murchison and Pilgrim's Rest samples. The gold dissolutions are determined from the different mass and grade of the feed and residue. The free milling VCR and Pilgrim's Rest samples achieved much better dissolution than the refractory Fairview, Barbrook and Consolidated Murchison samples.

Ore	**Feed**		**Residue**		**Dissolution**
	Mass g	Au Grade g/t	Mass g	Au Grade g/t	**Feed/Res** %
VCR	446.3	23.6	438.8	13.9	42.1
Fairview	474.3	12.5	450.0	15.0	0.0
Barbrook	473.9	2.1	471.9	2.0	5.2
Consolidated Murchison	523.5	3.1	504.0	3.2	0.3
Pilgrim's Rest	443.3	29.6	426.2	16.5	46.4
Pilgrim's Rest Repeat 1	396.0	29.6	379.6	16.8	45.6
Pilgrim's Rest Repeat 2	371.4	29.6	350.3	16.2	48.4

In order to understand the limitations of the thiosulphate system, leaches were subsequently run with the addition of the stabiliser, glycine. The dissolutions achieved in a thiosulphate glycine system (Table 9-5) were a significant improvement on the previous leaches, the increased dissolution ranged from only 7% with the Consolidated Murchison sample and up to 55% for Barbrook sample. With the exception of the Consolidated Murchison sample, all of the dissolutions achieved between 40 and 65%, with the highest dissolutions from the VCR ore (65%), followed closely by the Barbrook sample which achieved an unexpectedly high

dissolution of 60%. The Pilgrim's Rest sample achieved a gold dissolution of 54% while the Fairview sample achieved 43%. The Consolidated Murchison sample was the only refractory ore not to experience notable improvements in gold dissolution. The Barbrook ore seems to be the most amenable to this process and achieved significantly higher dissolutions than were obtained by cyanidation.

Table 9-5 The thiosulphate glycine leaching results for the Ventersdorp Contact Reef (VCR), Fairview, Barbrook, Consolidated Murchison and Pilgrim's Rest samples. The samples all achieved moderate gold dissolution (above 40%) with the exception of Consolidated Murchison.

Ore	Feed		Residue		Dissolution
	Mass g	Au Grade g/t	Mass g	Au Grade g/t	Feed/Res %
VCR	434.9	23.6	399.3	8.8	65.8
Fairview	473.9	12.5	459.3	7.3	43.7
Barbrook	462.3	2.1	432.7	0.9	60.0
Consolidated Murchison	514.2	3.1	501.6	2.9	7.2
Pilgrim's Rest	474.2	29.6	456.3	13.9	54.8

Many of the samples studied had quantities of gold locked within refractory minerals which limited the degree to which they could leached. In order to increase the gold exposure, the ores were calcined and then leached using thiosulphate only (Table 9-6). It is interesting to note that the VCR achieved lower dissolutions than non-calcined material under identical conditions. This is evident in both glycine free and glycine assisted leaches (as will be discussed later). The lower degree of dissolution is due to the formation of iron oxides capable of encapsulating gold during roasting. The encapsulation of gold by iron oxides is a process documented by many researchers (Li et al., 2017; Zhang et al., 2017) and is further reinforced by mineralogical evidence of hematite formation in the calcined material (XRD results in Chapter 10). However, the refractory ores achieved significant improvements in leaching results when the ores were calcined prior to leaching. For Fairview the dissolutions increased from 0% to 45%, while Barbrook increased from 5% to 40%. This increased dissolution may be due to an increased amount of gold available or to a reduced amount of sulphides which impacted on the stability of thiosulphate.

Table 9-6 The results for calcined ores, the Ventersdorp Contact Reef (VCR), Fairview and Barbrook samples, that have undergone thiosulphate leaching. The Fairview and Barbrook samples both achieved moderate gold dissolution of 40% or greater. VCR only achieved less than 30%.

Ore	Feed		Residue		Dissolution
	Mass g	Au Grade g/t	Mass g	Au Grade g/t	Feed/Res %
VCR	446.7	23.6	426.0	18.1	26.9
Fairview	444.7	12.5	433.0	7.0	45.7
Barbrook	433.4	2.1	424.4	1.3	40.2

The final stage of leach testing was performed on calcined ores, with thiosulphate and glycine to best optimise the dissolutions. Under these leaching conditions the highest gold dissolutions were seen for thiosulphate (Table 9-7), while the refractory ores achieved higher dissolutions than achieved using the cyanide. The dissolution for the VCR sample is likely to have been limited (60%) due to the previously discussed formation of oxidised sulphides, reducing the availability of the gold. The Fairview and Barbrook samples have however both achieved the highest dissolutions attained in both thiosulphate and cyanide leaching.

Table 9-7 The results for calcined ores, Ventersdorp Contact Reef (VCR), Fairview and Barbrook, with subsequent thiosulphate, glycine leaching. All samples achieved dissolutions greater than 60%.

Ore	Feed		Residue		Dissolution
	Mass g	Au Grade g/t	Mass g	Au Grade g/t	Feed/Res %
VCR	414.86	23.6	397.19	9.82	60.16
Fairview	445.56	12.5	436.27	3.34	73.84
Barbrook	439.23	2.08	381.29	0.58	75.79

9.6. Discussion

Sufficient quantities of exposed gold were present in all the ore types to allow for at least some lixiviant interaction and subsequent leaching. The VCR ore underwent almost complete gold dissolution during cyanide leaching, despite the coarse-grained nature of the gold. Thus, the selected leaching time of 24 hours seems sufficient not only for the coarse-grained gold from the VCR sample but also the remaining samples. The leach duration selected for the alternative techniques were based on the leach time required for cyanide dissolution in conjunction with the kinetics of cyanidation, relative to the kinetics of the alternatives. Should the kinetics remain consistent with what is described in the literature, the limited results attained from the alternative methods should be unrelated to insufficient time.

The chloride-hypochlorite leach is exceptionally unstable, with highly variable pH and redox conditions. High reagent additions are required to achieve target conditions, this is due to high gangue reagent consumptions. More information is provided in this regard in Chapter 11. These consumptions were prohibitively high for all but VCR material, which is a predominantly quartz-bearing, unreactive ore. Maintaining the pH and redox conditions for an 8-hour period required constant monitoring and reagent additions. The VCR sample, the only ore where leaching was technically feasible, achieved a gold dissolution of 40%. This limited dissolution is not due to the instability of the gold-chloride complexes, as the gold was found in solution more than a week after leaching. However, the limited dissolution may be due to the instability of the system, extreme reactivity with gangue minerals (discussed in Chapter 10) and subsequently target conditions only being achieved temporarily and inconsistently, preventing continuous leaching. While the leach was only run for 8 hours the kinetics associated with this system should have allowed for sufficient dissolution of even the most coarse-grained gold within the VCR sample.

The thiourea leaching system also only achieved limited results, with only the VCR and Pilgrim's Rest samples proving practical to leach. While excessive reagents were required to attain target conditions (discussed in the Chapter 12), once these conditions were achieved the leach remained relatively stable with only moderate adjustments required. Despite this overall stability, the leaches still only achieved gold dissolutions of 14% and 40% for the Pilgrim's Rest and VCR samples respectively. As thiourea is inherently unstable and the thiourea-gold complex is capable of reprecipitating gold, it was unsurprising that gold was not found in solution assays. The instability of this lixiviant is responsible for the lack of effective dissolution. The differences in leaching results for the Pilgrim's Rest and VCR sample are due to mineralogical differences between the ore types, this is addressed in Chapter 10.

The thiosulphate leaches consumed moderate amounts of reagent to attain ideal leaching conditions and generally these conditions were simple to maintain, with only minor adjustments required. The leaching achieved, using conditions suggested in the literature, resulted in poor gold dissolutions even with free milling ores, while practically no gold dissolution was evident from the refractory ores. The limited leaching associated with the refractory ores could not be due to the presence of sulphides within the leach, as the sulphide-bearing Pilgrim's Rest showed a greater degree of dissolution despite the sulphide

concentration being an order of magnitude higher than most of the refractory ores. There may be a single element responsible for these underperforming ores, for example this system could be more sensitive to nickel or magnesium, which is present in significantly higher concentrations in the refractory ores than in the free milling. The failure to leach could also be due to individual deleterious components such as the antimony concentration for Consolidated Murchison. The leaching could also be stunted due to the specific presence of arsenopyrite rather with sulphides in general. Arsenopyrite is associated with gold in all of the refractory deposits used.

In order to gain a better understanding of the limitations associated with thiosulphate leaching, adjustments were made to optimise the conditions. Glycine was added to see if a stabilising agent, capable of reducing thiosulphate degradation would improve and maintain gold dissolution. The ores were calcined in order to understand what changes the exposure of refractory gold and the reduction of available sulphides had on the dissolution.

The degree of gold dissolution achieved using the VCR samples was reduced by the calcining process. This is due to the formation of iron oxide (generated from the oxidation of iron sulphides), known for encapsulating gold during calcining. However, the addition of glycine, particularly to the uncalcined material, allowed the sample to achieve higher gold dissolution.

The dissolution of the gold within the Fairview sample was significantly improved by calcining and glycine addition both separately and in conjunction. The use of a stabiliser in thiosulphate leaching on uncalcined ores achieved dissolutions of 44% (approximately 65% of the total gold exposed) which is a major improvement from the original thiosulphate leach. The calcined ores also achieved moderate dissolutions which is likely to have been due to an increased availability of gold. However, the addition of the stabiliser was still required to allow for significant (approximately 70%) recoveries to be achieved. From this, it can be concluded that limited leaching results associated with thiosulphate are due almost entirely to the instability of the lixiviant, rather than any interactions with sulphides. This is also evident in the Chapter 11 where reagent consumptions are discussed.

The dissolution of the gold from Barbrook material using thiosulphate, glycine-bearing leaches was significant (59.9%) even outperforming direct cyanidation (24.13%). This

increased dissolution highlights the amount of Barbrook gold that is lost to preg-robbing and the advantage that thiosulphate has over cyanide even in excess carbon-in-leach (CIL) conditions. The calcined, glycine-bearing thiosulphate-leached ores reached 75% dissolution, suggesting that approximately 75% of the total available gold was leached.

Results from the glycine-bearing leaches are encouraging as improved dissolutions for thiosulphate leaches were achieved for all ore types in the presence of a low-cost stabilising agent. The Barbrook ore, when leached with thiosulphate and glycine far surpassed the results attained with CIL cyanidation. Unfortunately, with the exception of the Barbrook sample, the thiosulphate leaches consistently achieved only ~ 70% of the dissolution achieved by cyanide, even with the addition of a stabilising agent.

Thiosulphate leaches with calcined material for Consolidated Murchison and Pilgrim's Rest samples were evaluated, however their responses were limited and due to the limited applicability of these ores they were not further processed.

As the calcined refractory ores achieved dissolutions similar to that of VCR sample, it suggests that the increase in dissolution was due to an increase in exposed gold rather than a decrease in sulphides.

Chapter 10

Residue and Solution Data

In order to appropriately understand the interaction between the lixiviants of study and the ore types utilised, the solid residue and the pregnant leach solution were analysed to determine the mineralogical change during leaching and the components within solution.

10.1. Residue

The comparison of the residue mineralogy, relative to that of the feed, is an important method used to evaluate the leach, as well to understand the processes and mineral interactions occurring throughout. The mineralogy of the residue was determined using XRD on all material and MLA on selected samples.

10.1.1. Chloride-Hypochlorite

The XRD data from the chloride-hypochlorite leach residue is overlain with the original feed (Table 10-1, Figure 10-1). From this comparison it is evident that major mineralogical changes occurred during the leach. Using mass balance calculations, the amount of mineral consumed was quantified and the minerals were subsequently categorised as non-reactive (<5% dissolution), slightly reactive (5-15% dissolution), moderately reactive (15-30%) or highly reactive (>30%). Quartz, which is unreactive in all the leaching conditions under consideration, was used as a control in assessing the significance of change in concentration and the determination of whether a phase is reactive or not. The uncertainty (standard deviation) on the mineral dissolution results is 4.6% and thus the range of 0-5% was selected as non-reactive. This uncertainty is related to the limitations associated with the semi-quantitative nature of XRD. Pyrite, pyrrhotite and calcite have been classified as highly reactive, while chlorite is moderately reactive. Quartz and muscovite are non-reactive. The presence of halite in the residue was due to the reagents used in the wash solution. As the wash solution was made to simulate the leach solution (in order to prevent the precipitation of any gold grains not yet separated from the residue) excess sodium chloride/halite was added to the solution which is evident in the residue.

Table 10-1 Mineralogical difference between the Ventersdorp Contact Reef (VCR) feed and residue of the chloride-hypochlorite leach, as determined by XRD. The dissolution % of individual minerals has been calculated and their respective reactivity categorised.

Mineral	VCR Feed	VCR-CL Residue	Dissolution	Classification
Units	**%**	**%**	%	-
Quartz	87.4	88.9	2.7	Non reactive
Chlorite	8.1	7.0	17.3	Moderately reactive
Muscovite	2.7	2.7	4.3	Non reactive
Pyrrhotite	0.8	0	100.0	Highly reactive
Pyrite	0.4	0	100.0	Highly reactive
Calcite	0.6	0	100.0	Highly reactive
Mineral	**VCR Feed**	**VCR-CL Residue**	**Formation Mass Balance**	**Classification**
Halite	0	1.4	100.0	New mineral forming

Feed Mass (g)	436.1
Residue mass (g)	417.14

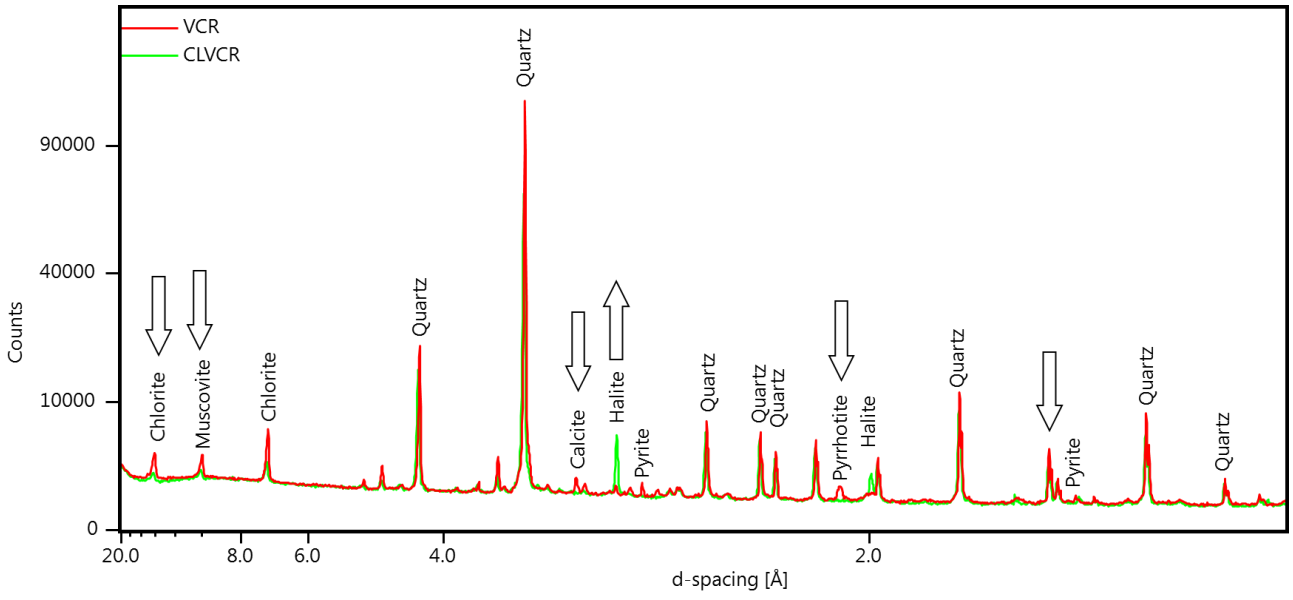


Figure 10-1 XRD pattern for the Ventersdorp Contact Reef (VCR) feed (red) and residue (green) overlain to highlight mineralogical differences after chloride-hypochlorite leaching. The arrows represent the increasing or decreasing abundance from feed to residue. Consumption of chlorite, calcite, pyrrhotite and pyrite is evident.

A number of blocks were prepared from the remaining residue in order to study the remaining gold surface and to understand the cause of the limited leaching results. Only a limited number of grains were located, of these many are entirely locked and would not have been leached with cyanide either. There is a notable lack of liberated, coarse gold grains located in the residue using the MLA. This absence of coarse gold grains may show that these grains underwent dissolution, however it seems more likely that the coarse grains of interest were present in insufficient quantities to be located in the few blocks produced from the residue. Any coarse gold present, but not located, would account for significantly more gold than represented in the extremely fine grains in Figure 10-2. Due to the absence of statistically significant gold the nature of the gold located served to provide little information. However, some of these gold grains tend to be associated with certain minerals, particularly quartz, pyrite, chlorite and pyrrhotite. Several of these are reactive minerals which may have been responsible for conditions being too unstable to maintain leaching.

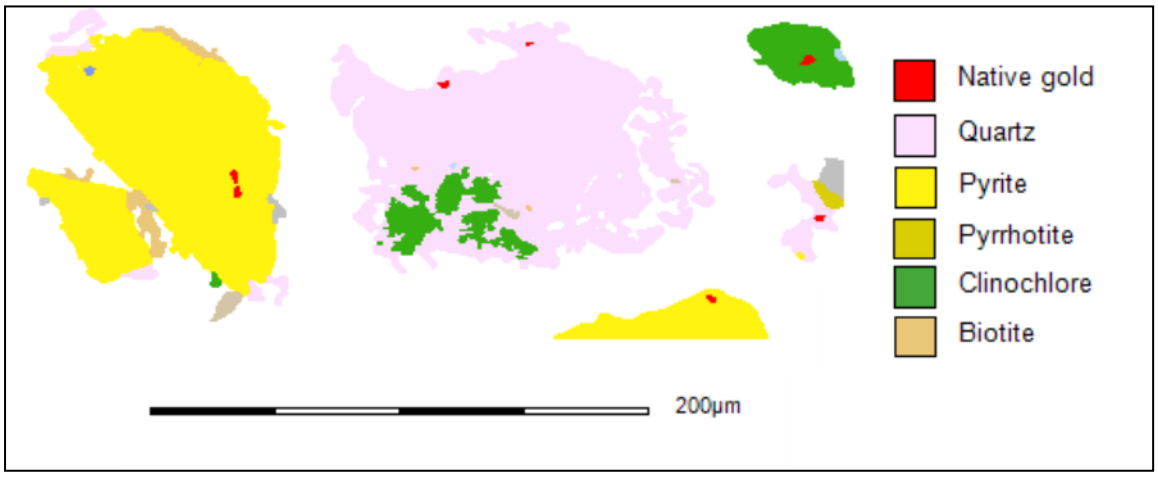


Figure 10-2 False colour images of the remaining gold grains in the Ventersdorp Contact Reef residue from the chlorite-hypochlorite leach. These gains have little to no exposure and are consistently less than 10 µm.

10.1.2. Thiourea

The thiourea system was also highly reactive as is evident in the XRD results. The XRD data for the feed and residue have been determined for the VCR sample (Table 10-2A, Figure 10-8) and Pilgrim's Rest sample (Table 10-2B, Figure 10-4). The dissolution percentage of the mineralogies, calculated using mass balance of feed and residue, shows a number of minerals that are reactive in thiourea leaching. Pyrite, pyrrhotite, calcite and dolomite are all highly

reactive, while chlorite, muscovite and chalcopyrite are significantly less reactive. The semi-quantitative nature of the XRD analyses and the previously defined uncertainty is evident in the quartz results, as the dissolution percentage is slightly negative, however within this range quartz can still be considered non-reactive. In addition to the loss of certain minerals, it is interesting to note that two new minerals have formed during the leach; pyracmonite and nitrammite. Pyracmonite [(NH4)3Fe(SO4)3] is a colourless to white mineral, with a vitreous lustre and a trigonal crystal system. The formation of pyracmonite in this leach is understandable based on the components of the leach – thiourea (CH4N2S) and ferric iron, in addition to the oxygen incorporated into the leach during agitation. This logic can similarly be applied to nitrammite (NH4,K)(NO3), a white mineral with an orthorhombic crystal system. The formation of these new minerals, pyracmonite and nitrammite highlights the degree of thiourea decomposition as thiourea is the only source of nitrogen within this leach. Therefore, notable quantities of thiourea would have degraded in order for the formation of pyracmonite and nitrammite to have formed in detectable quantities.

Table 10-2 The mineral abundances in the feed and residue as determined by XRD for A) the Ventersdorp Contact Reef sample and B) the Pilgrim's Rest sample. TU represents thiourea leaching residue. The dissolution % of individual minerals has been calculated and their respective reactivity categorised.

A)

Mineral	VCR Feed	VC B Residue	Dissolution Mass Balance	Classification
Units	**%**	**%**	%	
Quartz	87.4	91.1	-0.9	Non reactive
Chlorite	8.1	3.8	54.6	Highly reactive
Muscovite	2.7	2.5	10.3	Slightly reactive
Pyrrhotite	0.8	0.4	51.6	Highly reactive
Pyrite	0.4	0.3	27.4	Moderately reactive
Calcite	0.6	0	100.0	Highly reactive
Mineral	**VCR Feed**	**VCR-TS Residue**	**Formation Mass Balance**	**Classifcation**
Pyracmonite	0	1.2	100.00	New mineral forming
Nitrammite	0	0.7	100.00	New mineral forming

Feed Mass (g)	440.6
Residue mass (g)	426.7

B)

Mineral	PG Feed	PG TU Residue	Dissolution Mass Balance	Classification
Units	**%**	**%**	%	
Quartz	47.9	53.2	-4.5	Non reactive
Dolomite	16.6	0.8	95.5	Highly reactive
Pyrite	33.4	37.3	-5.1	Non reactive
Chalcopyrite	1.2	0.9	29.4	Moderately reactive
Muscovite	0.9	0.9	5.9	Slightly reactive
Mineral	**PG Feed**	**PG TU Residue**	**Formation Mass Balance**	**Classifcation**
Pyracmonite	0	3.7	100.00	New mineral forming
Nitrammite	0	3.2	100.00	New mineral forming

Feed Mass (g)	452.11
Residue mass (g)	425.37

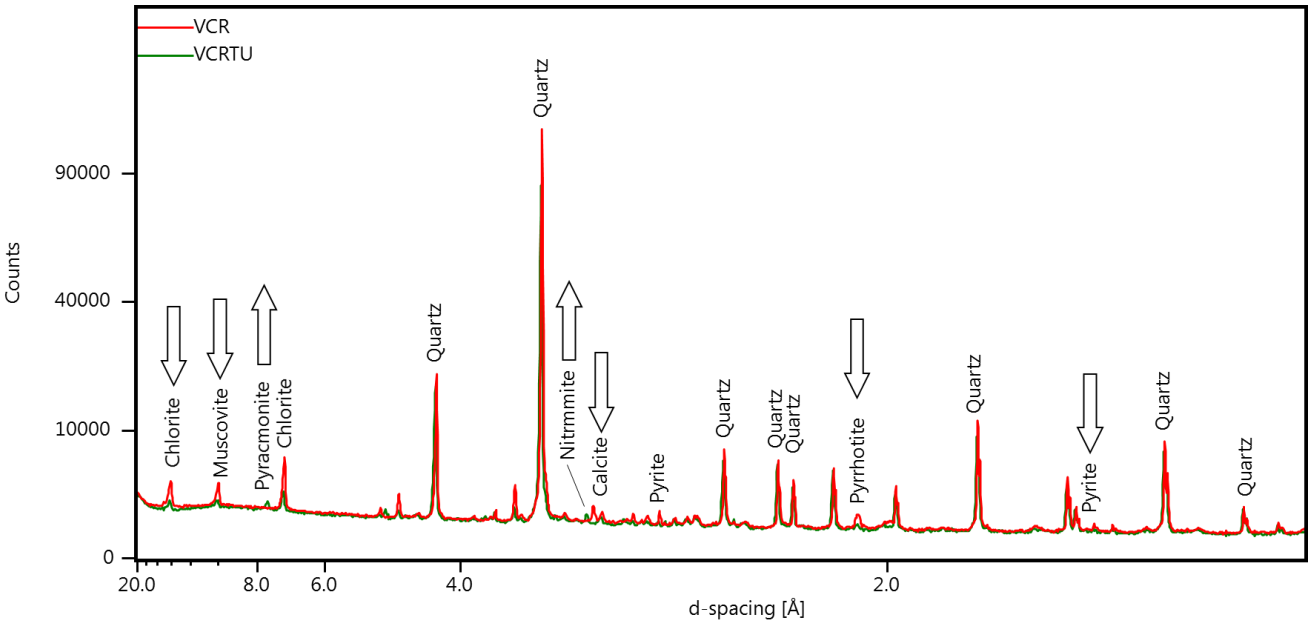


Figure 10-3 XRD pattern for Ventersdorp Contact Reef (VCR) feed (red) and residue (green) overlain to highlight the mineralogical similarities before and after thiourea leaching. As is evident no mineralogical changes took place during leaching.

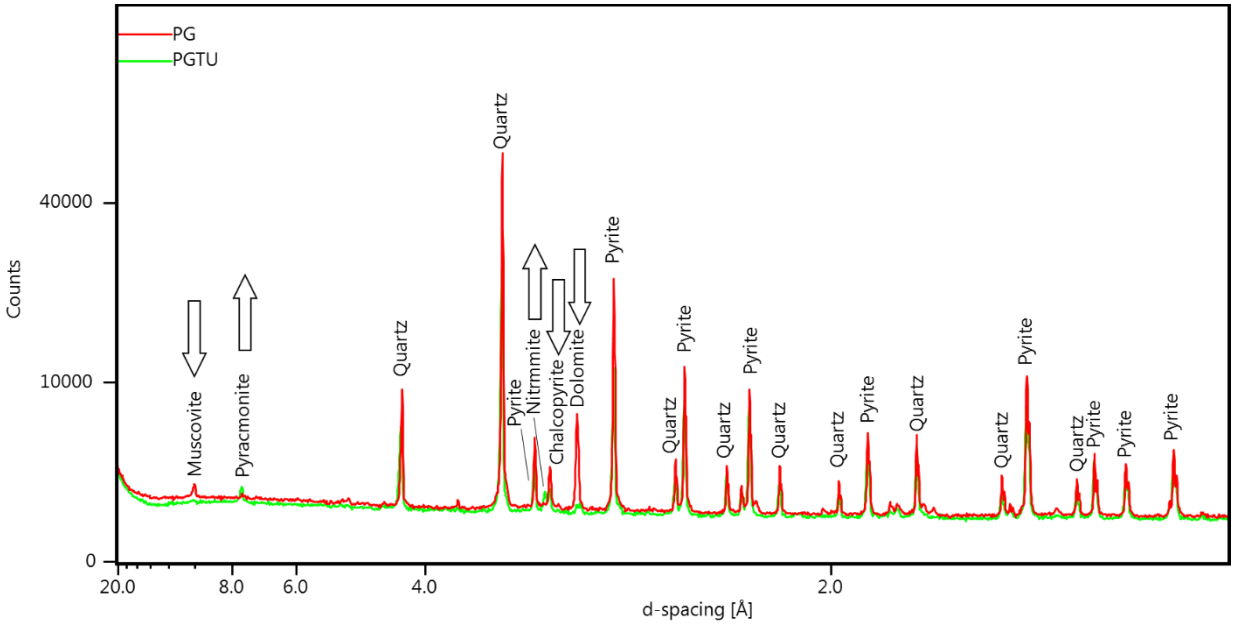


Figure 10-4 XRD pattern for Pilgrim's Rest feed (red) and residue (green) overlain to highlight the mineralogical nature before and after thiourea leaching. The mineralogy was unaffected during this leach

10.1.3. Thiosulphate

The XRD patterns for all thiosulphate leaches tend to show little variation between the feed and residue. This applies to ores that have experienced thiosulphate leaching both with and

without glycine additions. When the ores were calcined, the sulphides were oxidised to form iron oxides etc. while the carbonates were also oxidised to varying degrees. Despite the major mineralogical variation from feed to calcine, the calcined ores also showed very little change during leaching, with the exception of anhydrous minerals which were hydrated. Some of the XRD diffractograms that highlight the absence of mineralogical change during leaching have been included, the remaining ones can be found in Appendix D.

The feed mineralogy of the VCR sample (Table 10-3) was predominantly quartz, with minor amounts of reactive components such as chlorite, pyrrhotite and pyrite. Despite the presence of reactive components, the XRD patterns showed no variation in the mineralogies between the feed and residue, for both thiosulphate leaches and thiosulphate glycine leaches. The similarity of the mineralogy is evident in the XRD patterns (Figure 10-5) where the three patterns (feed, thiosulphate residue and thiosulphate + glycine residue) are barely differentiable. It is relevant to note that even the more reactive minerals that would have been consumed in cyanide leaching such as pyrrhotite remained during the thiosulphate leaches. For this reason, the mass balance calculations and subsequent reactivity classifications were not included for thiosulphate leaches of any kind, as all minerals were classified as non-reactive. The calcined VCR sample has lost chlorite and muscovite, while the pyrrhotite and pyrite were oxidised to form hematite. The increase in the abundance percentage for quartz (from 87 to 91%) represents an overall decrease in the mass from calcining while the quartz quantity remained unaffected. The calcined material also experienced no mineralogical changes during leaching.

Table 10-3 The mineralogy of the Ventersdorp Contact Reef (VCR) feed, both before and after calcining, in comparison to the thiosulphate (glycine-bearing and glycine-absent) leached residue. TS refers to thiosulphate leaching, while the G represents glycine addition. No mineralogical change due to leaching is evident in both standard and calcined feed.

Mineral	VCR Feed	VCR TS Residue	VCR TSG Residue	CVCR Feed	CVCR TS Residue	CVCR TSG Residue
Units	**%**	**%**	**%**	**%**	**%**	**%**
Quartz	87.4	87.4	87.4	91.0	91.0	91.0
Chlorite	8.1	8.1	8.1	5.4	5.4	5.4
Muscovite	2.7	2.7	2.7	2.0	2.0	2.0
Pyrrhotite	0.8	0.8	0.8			
Pyrite	0.4	0.4	0.4			
Calcite	0.6	0.6	0.6	0.5	0.5	0.5
Hematite	-	-		1.1	1.1	1.1
Total	100	100	100	100.0	100.0	100.0

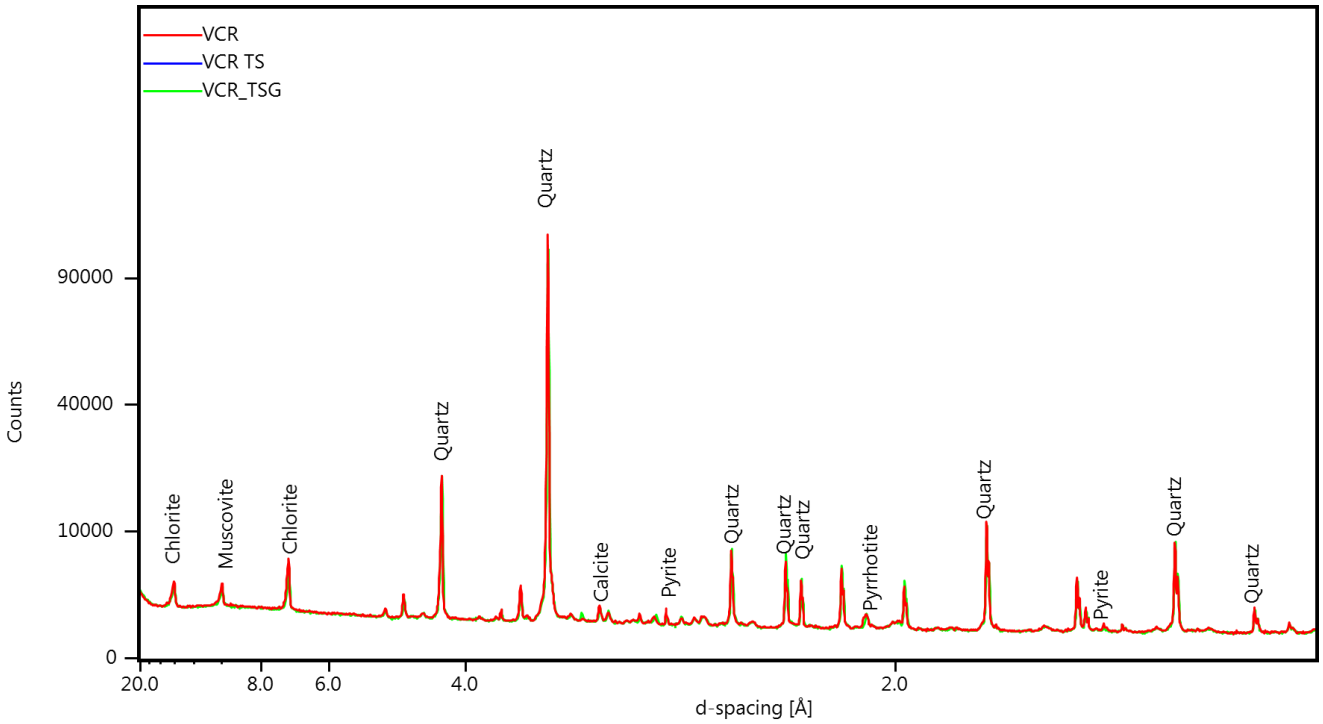


Figure 10-5 The XRD patterns for the Ventersdorp Contact Reef feed material (VCR) and the residue for thiosulphate (VCR TS) and thiosulphate + glycine (VCR TSG) leaching. These patterns are barely differentiable, highlighting the absence of mineralogical change throughout thiosulphate leaching.

The Fairview sample had abundant quartz, but also a high content of muscovite, dolomite, chlorite, magnesite, pyrite and arsenopyrite. These more reactive minerals were not

consumed throughout the leach (Table 10-4). The calcined feed contained significantly less dolomite and magnesite, in addition to the destruction of pyrite and arsenopyrite to form hematite. In the calcined material the mineralogy also remained the same throughout the 24-hour thiosulphate glycine leach.

Table 10-4 The mineralogy of the Fairview feed, both before and after calcining, in comparison to the thiosulphate (glycine-bearing and glycine-absent) leached residue showing the absence of mineralogical change during leaching. During calcining the samples lost significant dolomite, magnesite, pyrite and arsenopyrite. FV- Fairview feed, CFV- Calcined Fairview feed, TS- Thiosulphate leach residue, TSG – Thiosulphate + glycine.

	FV Feed	FV TS Residue	FV TSG Residue	CFV Feed	CFV TS Residue	CFV TSG Residue
Mineral	**FV**	**FV-TS**	**FV-TSG**	**CFV**	**FV-TU**	**FV-TSG**
Units	**%**	**%**	**%**	**%**	**%**	**%**
Quartz	40.5	40.5	40.5	52.8	52.8	52.8
Muscovite	25.5	25.5	25.5	33.0	33.0	33.0
Dolomite	14.2	14.2	14.2	3.0	3.0	3.0
Chlorite	9.2	9.2	9.2	9.7	9.7	9.7
Magnesite	6.7	6.7	6.7	-	-	-
Pyrite	2.9	2.9	2.9	-	-	-
Arsenopyrite	1.0	1.0	1.0	-	-	-
Hematite	-	-	-	1.5	1.5	1.5
Total	100	100	100	100.0	100	100

The Barbrook sample (Table 10-5) contained abundant dolomite, quartz, chlorite and pyrite, which was unaffected by the leach. The calcined ore had significant amounts of quartz and chlorite, while much of the dolomite was consumed in addition to pyrite, kaolinite, calcite, and muscovite. Anhydrite and hematite were formed during the roasting process. The calcined material showed some variation between feed and residue (Figure 10-6), however this is only due to the change from anhydrite to gypsum caused by hydration during leaching.

Table 10-5 The mineralogy of the Barbrook feed, both before and after calcining, in comparison to the thiosulphate (glycine bearing and absent) leached residue. No mineralogical change is evident during thiosulphate leaching; however, the calcining process has consumed pyrite, kaolinite and calcite. BK- Barbrook feed, CBK- Calcined Barbrook feed, TS- Thiosulphate leach residue, TSG – Thiosulphate + glycine leach residue.

Mineral	BK Feed	BK TS Residue	BK TSG Residue	CBK Feed	CBK TS Residue	CBK TSG Residue
Units	**%**	**%**	**%**	**%**	**%**	**%**
Dolomite	52.3	52.3	52.3	11.7	11.7	11.7
Quartz	25.2	25.2	25.2	41.6	41.6	41.6
Chlorite	14.7	14.7	14.7	20.7	20.7	20.7
Pyrite	4.9	4.9	4.9	-	-	-
Kaolinite	1.0	1.0	1.0	-	-	-
Calcite	0.4	0.4	0.4	-	-	-
Muscovite	1.5	1.5	1.5	5.6	5.6	5.6
Anhydrite	-	-	-	7.7	-	-
Gypsum				-	7.7	7.7
Hematite	-	-	-	12.7	12.7	12.7
Total	100	100	100	100.0	100	100

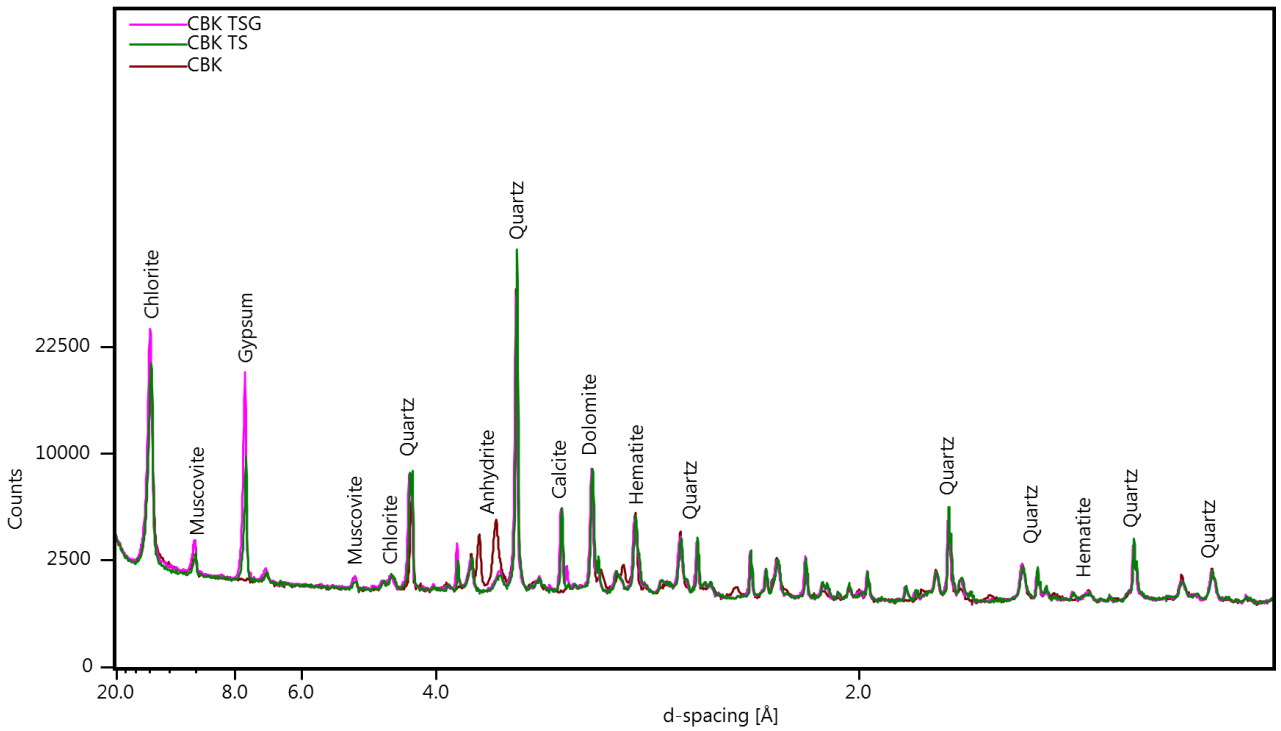


Figure 10-6 XRD pattern for calcined Barbrook material showing very little mineralogical variation from the feed to the residue with the exception of the change from anhydrite to gypsum. CBK- Calcined Barbrook feed, TS- Thiosulphate leach residue, TSG – Thiosulphate + glycine leach residue.

The Consolidated Murchison sample was host to many minerals including quartz, tourmaline, magnesite, dolomite, chlorite, biotite, pyrite, arsenopyrite, berthierite, stibnite and talc. Even the most unstable of these minerals were still present after the leach (Table 10-6). The roasting process consumed magnesite, pyrite, arsenopyrite, berthierite, stibnite and talc and formed magnetite, hematite, periclase, senarmonite and tripuhyite, all of which are still present after leaching. Senarmonite (Sb_2O_3) and tripuhyite ($FeSbO_4$) are antimony oxides formed from the oxidation of the antimony sulphides (stibnite and berthierite).

Table 10-6 The mineralogy of the Consolidated Murchison feed, both before and after calcining, in comparison to the thiosulphate (glycine bearing and absent) leached residue. No mineralogical variation as a result of leaching is present. Calcining however, consumed magnesite, pyrite, arsenopyrite, berthierite, stibnite and talc while forming new minerals such as magnetite, hematite, periclase etc. CM – Consolidated Murchison feed, CCM – Calcined Consolidated Murchison feed, TS – Thiosulphate residue, TSG – Thiosulphate + glycine residue.

Mineral	CM Feed	CM TS Residue	CM TSG Residue	CCM Feed	CCM TS Residue	CCM TSG Residue
Units	**%**	**%**	**%**	**%**	**%**	**%**
Quartz	28.1	28.1	28.1	38.1	38.1	38.1
Tourmaline	20.4	20.4	20.4	34.0	34.0	34.0
Magnesite	19.7	19.7	19.7	-	-	-
Dolomite	15.1	15.1	15.1	11.6	11.6	11.6
Chlorite	6.7	6.7	6.7	9.5	9.5	9.5
Biotite	2.5	2.5	2.5	2.3	2.3	2.3
Pyrite	2.1	2.1	2.1	-	-	-
Arsenopyrite	2.1	2.1	2.1	-	-	-
Berthierite	1.9	1.9	1.9	-	-	-
Stibnite	1.8	1.8	1.8	-	-	-
Talc	0.4	0.4	0.4	-	-	-
Magnetite				1.1	1.1	1.1
Hematite	-	-	-	0.9	0.9	0.9
Periclase	-	-	-	0.7	0.7	0.7
Senarmontite				0.2	0.2	0.2
Tripuhyite	-	-	-	1.6	1.6	1.6
Total	100.8	100.8	100.8	100.0	100	100

The Pilgrim's Rest sample (Table 10-7) is predominantly composed of quartz, pyrite and dolomite, with lesser chalcopyrite and muscovite. These minerals remained present throughout the leach. When the ore underwent roasting, all of the pyrite was oxidised into hematite and bassanite ($CaSO_4$) was also formed. The mineralogy of the Pilgrim's Rest feed, both before and after calcining, was consistent with the leaching residues. Thus, no

mineralogical change occurred during the 24 hour thiosulphate leach, with the exception of the change from anhydrite to gypsum caused by hydration during leaching.

Table 10-7 The mineralogy of the Pilgrim's Rest feed, both before and after calcining, in comparison to the thiosulphate (glycine bearing and absent) leached residue, again, highlighting the lack of mineralogical change from leaching. The effect of roasting in the Pilgrim's Rest sample is most evident in the oxidation of pyrite to form hematite. PG – Pilgrim's Rest feed, CPG – Calcined Pilgrim's Rest feed, TS – Thiosulphate residue, TSG – Thiosulphate + glycine residue.

Mineral	PG Feed	PG TS Residue	PG TSG Residue	CPG Feed	CPG TS Residue	CPG TSG Residue
Units	**%**	**%**	**%**	**%**	**%**	**%**
Quartz	47.9	47.9	47.9	50.5	50.5	50.5
Pyrite	33.4	33.4	33.4	-	-	-
Dolomite	16.6	16.6	16.6	2.0	2.0	2.0
Chalcopyrite	1.2	1.2	1.2	-	-	-
Muscovite	0.9	0.9	0.9	1.3	1.3	1.3
Anhydrite	-	-	-	0.8	-	-
Gypsum				-	0.8	0.8
Hematite	-	-	-	45.1	45.1	45.1
Bassanite	-	-	-	0.3	0.3	0.3
Total	100	100	100	100	100	100

The VCR residue from the thiosulphate-glycine leach was selected for gold residue study, as it had the highest gold dissolution. In order to study the remaining gold, the residue was upgraded using heavy liquid separation (HLS). The HLS was prepared using tetrabromoethane (TBE) at a specific gravity of 2.96 g/cm^3 in a centrifuge to ensure fine gold was able to settle out (Coetzee et al., 2011). The floats were assayed to ensure the HLS was successful, while the sinks were used to prepare polished sections for MLA study.

From the residues, 509 gold grains were located within 241 particles (Figure 10-7). The gold particle sizes were limited (Figure 10-8), ranging between 1.2 to 27 µm, with the majority of the weight percent distributed approximately equally from the larger weight classes. Vast amounts of data for the liberation and exposure is given in Appendix D, however briefly summarised the gold grains in the VCR sample were 45% liberated, 45% locked, and 10% of the gold fell within the middlings classification.

The exposure data for gold within the VCR residue sample could be similarly grouped, with approximately 50% of the gold exposed by more than 10%, 29% of the gold occurred as poorly

exposed (between 0 and 10%) and the remaining 21% as entirely enclosed. The gold was associated with a number of minerals; allanite (19%), pyrite (16%), arsenopyrite (7%), cobaltite (6.5%), quartz (4%) and coffinite (3.05%). When comparing the gold in the residue to that in the feed, one of the notable differences was the amount of free surface in the residue, which was more than 50% less than in the feed. This means that while significant amounts of gold would readily interact with the lixiviant in the feed, significantly less gold would be available in the residue to do so.

The previously unreported allanite associated with gold is also worth noting and was likely upgraded extensively during the HLS due to its high density. The other components associated with gold both in feed and residue were present in insufficient quantities to determine if these differences are due to leaching effects or more likely mineralogical variations within the ore. The most obvious difference between the feed and residue gold grains was the grain size distribution which shows a strong contrast between the feed (with well liberated gold grains, 220 µm or greater), and the residue (with a maximum gold grain size of 27 µm).

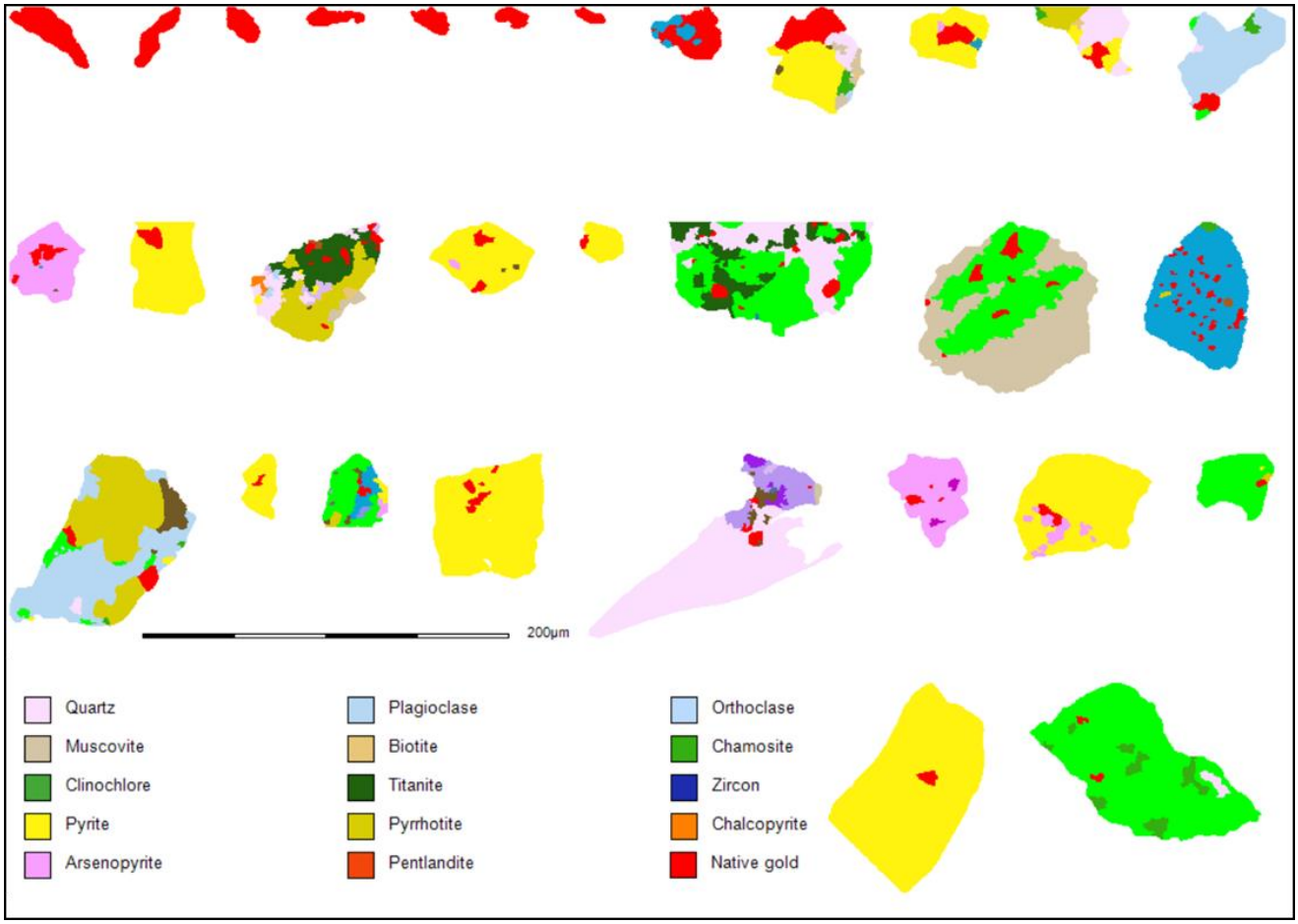


Figure 10-7 False colour images from MLA of gold grains (red) and their associated minerals found in the Ventersdorp Contact Reef thiosulphate glycine residue. This image emphasises the number of fine grained (less than of 27 µm equivalent circular diameter) gold particles in the residue.

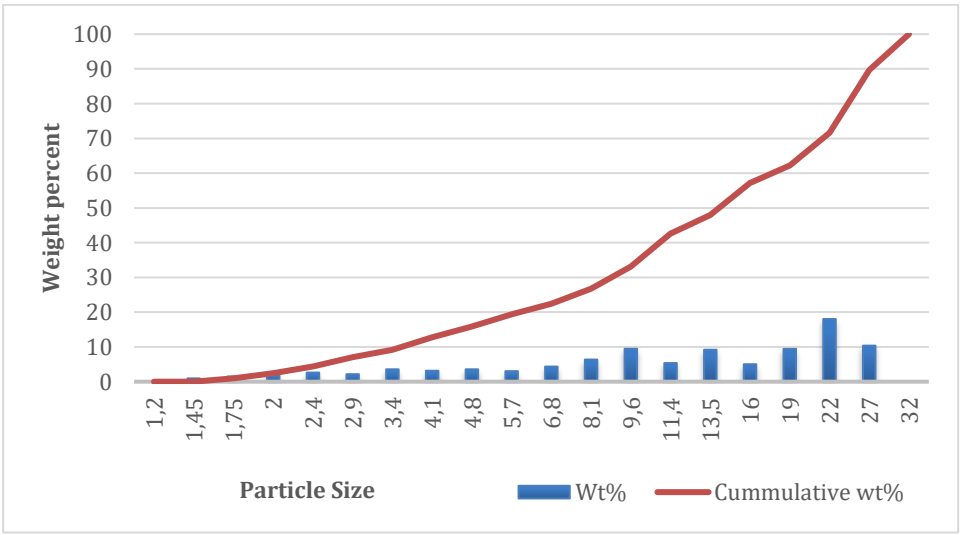


Figure 10-8 Gold grain size distribution from thiosulphate glycine leached Ventersdorp Contact Reef residue.

The abundance of fine-grained, liberated gold grains evident in the feed but not the residue would suggest that the gold grains originally present have either undergone incomplete dissolution or subsequent re-precipitation. A study of the BSE images of the same false colour gold grains shown in Figure 10-7, give a better indication of whether the gold grains represent incomplete dissolution or re-precipitation. The BSE images of the remaining gold (Figure 10-9) show compact gold with embayments that suggest incomplete dissolution, rather than precipitation.

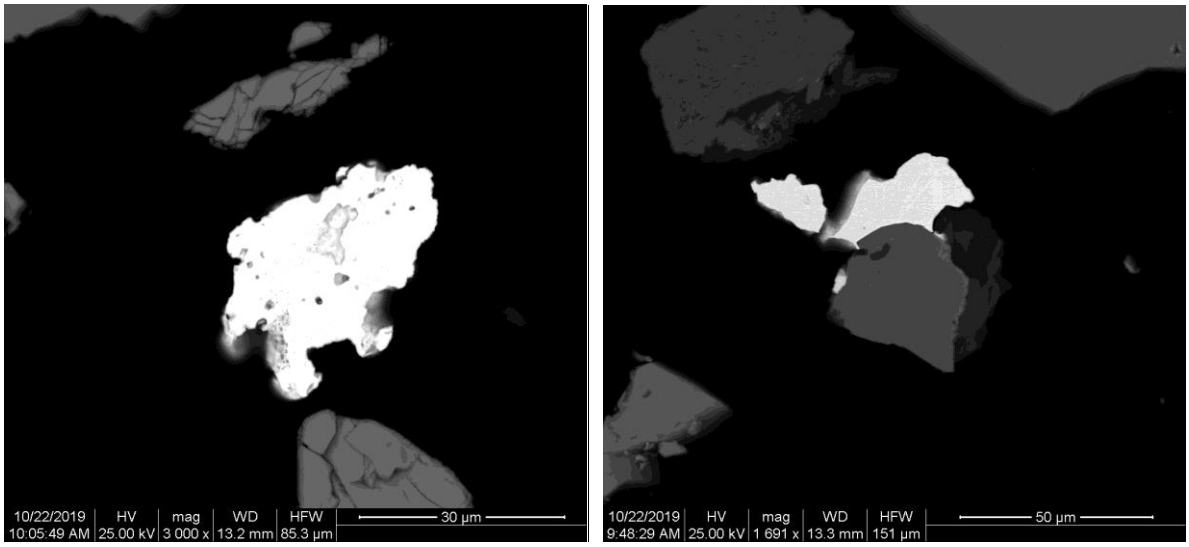


Figure 10-9 BSE images of gold (white) found in calcined, thiosulphate glycine leach of the Ventersdorp Contact Reef residue. The compact nature of the gold, along with the embayments along the margins would suggest that these grains underwent incomplete dissolution.

10.2. Solution Assay

In order to gain a better understanding of the consumptions in the leaches, additional assays were done to determine the presence of potentially deleterious metals within the leach solution. The elements assayed for tend to be reactive and would increase the reagent consumption and reducing leaching efficiency.

10.2.1. Cyanide

The metal concentrations found in the cyanide leach are shown in Table 10-8. While the concentrations present in this leach solution don't generally constitute very high values, relative to gold concentrations they are significant. Notable proportions of cyanide capable of leaching gold, are being utilised to complex with other, far more abundant metals. This

complicates not only leaching, but also subsequent recovery due to dilution of the gold in solution. Substantial concentrations of deleterious components are found in most of the deposits of interest but are most noteworthy in the Pilgrim's Rest ore. The Pilgrim's Rest leach solution contained significant copper concentrations, as well as elevated sulphur and to a lesser extent, iron. The Barbrook leach solution had raised concentrations of copper, iron, zinc, manganese and even limited amounts of antimony. Additional concentrations of concern included the arsenic, sulphur and antimony in the Consolidated Murchison leach as well as elevated arsenic, iron and sulphur in the Fairview leach. The VCR ore produced a leach solution with no notable concentrations of reactive components, with the exception of slightly elevated copper concentrations. The copper, zinc, manganese and antimony found in solution and their concentrations strongly relate to the cyanide consumptions previously described. High concentrations of deleterious elements within solution correspond with high cyanide consumption as these elements complex with cyanide. However, these same elements do not appear to have a specific effect on the consumption of thiosulphate.

Table 10-8 Assay results for deleterious metals in the cyanide leach solution. Additional information regarding repeats, standards and blanks are in Appendix D. LDetection refers to the lower detection limit in the elemental assay. VCR – Ventersdorp Contact Reef, FV – Fairview, BK – Barbrook, CM - Consolidated Murchison and PG – Pilgrim's Rest. Copper concentrations for Pilgrim's Rest may be deleterious, to a lesser extent Barbrook and Consolidated Murchison also have potentially deleterious elements in solution.

METHOD	Cu ICP84B	Fe ICP84B	Ni ICP84B	S ICP84B	Zn ICP84B	Co ICP84B	Mn ICP84B	Sb ICP84B	As ICP84B
LDETECTION	0.02	0.02	0.05	0.6	0.17	0.12	0.5	0.31	0.34
UNITS	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L
VCR	11.8	4.72	0.17	126	3.2	0.1	0.6	<0.31	0.7
FV	5.33	17.1	1.27	194	2.9	0.3	2.7	<0.31	14.1
BK	47.4	37.7	0.88	819	28.6	0.3	6.9	1.5	1.1
CM	3.45	6.78	0.21	552	1.3	<0.12	1.2	63.4	16.7
PG	266	21.6	0.09	1650	<0.17	<0.12	<0.5	<0.31	<0.34

10.2.2. Chloride-Hypochlorite

The solution assay for the chloride-hypochlorite leach (Table 10-9) shows high concentrations of copper, iron, sulphur, manganese and arsenic. These elevated concentrations are indicative of the dissolution of the sulphides present in the VCR ore.

Table 10-9 Assay results for deleterious metals in the chloride-hypochlorite leach solution. Additional information regarding repeats, standards and blanks are in Appendix D. LDetection refers to the lower detection limit in the elemental assay. VCR – Ventersdorp Contact Reef.

	Cu	Fe	Ni	S	Zn	Co	Mn	Sb	As
METHOD	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B
LDETECTION	0.02	0.02	0.05	0.6	0.17	0.12	0.5	0.31	0.34
UNITS	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L
VCR	38.9	4930	16.5	3530	31.6	4.8	60.2	0.8	131

10.2.3. Thiourea

The solution assay for thiourea (Table 10-10) shows varying degrees of consistently elevated element concentrations relative to the values found in cyanide solution. The high iron concentration is primarily due to the addition of ferric iron sulphate as a reagent that is required for leaching. High concentrations of sulphur are likely to be due to the presence and breakdown of thiourea as well as ferric sulphate. The leach using the Pilgrim's Rest sample showed a significantly higher copper concentrations relative to the VCR leach. The difference in copper concentration is likely to be due to the occurrence of copper sulphides (chalcopyrite), as well as higher arsenic and manganese from sulphides. Nickel concentrations in the VCR leach are somewhat elevated, as is cobalt, however the limited concentrations of these metals in solution highlights the general nature of the VCR material which tends to be a non-reactive ore. As the consumptions of these leaches are similar despite significantly different mineralogy and solution chemistries, it seems likely that the loss of thiourea over the 8-hour leach period reflects the reagent instability rather than mineral interactions.

Table 10-10 Assay results for deleterious metals in the thiourea leach solution. Additional information regarding repeats, standards and blanks are shown in Appendix D. LDetection refers to the lower detection limit in the elemental assay. VCR – Ventersdorp Contact Reef, FV – Fairview, BK – Barbrook, CM - Consolidated Murchison and PG – Pilgrim's Rest. The high iron concentration is primarily due to the addition of ferric iron sulphate.

	Cu	Fe	Ni	S	Zn	Co	Mn	Sb	As
METHOD	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B
LDETECTION	0.02	0.02	0.05	0.6	0.17	0.12	0.5	0.31	0.34
UNITS	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L
VCR	17.9	38200	10.6	53600	4.4	2.6	120	1.1	22.7
PG	1050	32300	3.41	51300	6.6	<0.12	255	1.7	72.8

10.2.4. Thiosulphate

The metals present in the leach solution in the thiosulphate system show consistently high values for copper (Table 10-11). It is interesting to note that the lowest copper concentrations are present in the Pilgrim's Rest leach solution, which had the highest copper grades in the feed. However more than 9 000 mg/L of copper was added to the solution to facilitate leaching. Copper solubility is not likely a relevant factor in the solution concentrations, as concentrations of up to 32 000 mg/L can be attained at 20˚C (Dalton et al., 2016). Iron, cobalt, manganese and arsenic are present in all the ores, but in insignificant quantities. The low concentrations of metals in leach liquor highlight the limited reactivity of the thiosulphate leaching system relative to previous lixiviants. Iron content is particularly noteworthy as concentrations less than the limit of detection are consistently present, even in ores where reactive iron sulphides such as pyrrhotite are present. While there are minor amounts of arsenic and antimony present in the Consolidated Murchison leach liquor this represents less than 0.2 and 0.02% of the total arsenic and antimony present in the feed.

Table 10-11 Assay results for deleterious metals in the thiosulphate leach solution. LDetection refers to the lower detection limit in the elemental assay. VCR – Ventersdorp Contact Reef, FV – Fairview, BK – Barbrook, CM - Consolidated Murchison and PG – Pilgrim's Rest. Copper concentrations reflect the addition of copper sulphate rather than a deleterious component in the ore.

	Au	Cu	Fe	Ni	S	Zn	Co	Mn	Sb	As
METHOD	SOL81X	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B
LDETECTION	0.001	0.02	0.02	0.05	0.6	0.17	0.12	0.5	0.31	0.34
UNITS	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L
VCR	0.005	5460	<0.02	<0.05	25900	42.8	<0.12	<0.5	0.8	<0.34
FV	0.003	4760	<0.02	<0.05	24300	38.3	<0.12	<0.5	<0.31	4.3
BK	0.004	5090	67.4	0.18	34100	0.5	<0.12	0.6	<0.31	<0.34
CM	0.002	4520	<0.02	<0.05	27700	35.6	<0.12	<0.5	12	63.2
PG	0.143	1120	0.8	0.08	21300	9.1	<0.12	<0.5	0.5	<0.34

As all leaches were completed on VCR feed, a comparison of reactivity of the alternative lixiviants is possible based on the elements found in the leach solution (Table 10-12), this has been represented as a factor relative to cyanide. This data further emphasises the greater degree of reactivity for chloride-hypochlorite and thiourea, while thiosulphate consistently shows lower reactivity than cyanide.

Table 10-12 A comparison of the elements found in solution, as a factor relative to cyanide. The elements in red are significant as they have been added as reagent required for leaching, rather than a product of leach reactivity.

	Cu	Fe	Ni	S	Zn	Co	As
Cyanide	1.00	1.00	1.00	1.00	1.00	1.00	1.00
Chloride- Hypochlorite	3.30	1044.49	97.06	28.02	9.88	48.00	187.14
Thiourea	1.52	8093.22	62.35	425.40	1.38	26.00	32.43
Thiosulphate	462.71	0.00	0.00	205.56	13.38	0.00	0.00

10.3. Discussion

Using both XRD analyses on residue as well as assay results on leach solution, it is possible to confirm the mineralogical changes due to lixiviant interaction. Table 10-13 indicates the reactivity of the lixiviants of study with the respective mineralogies which have been classified using both mineralogical changes from feed to residue as well as solution chemistry.

While XRD data was not available for the cyanide leach, an examination of the leach solution assays highlights the presence of moderate amounts of copper, iron, sulphur and antimony in solution. The solution chemistry is indicative of moderate reactivity with pyrrhotite, arsenopyrite and stibnite/berthierite as predicted in the literature (Deschênes and Prud'homme, 1997; Komnitsas and Pooley, 1989; Chryssoulis and McMullen, 2005; Li and Miller, 2006; Karimi et al., 2010).

The leaching ability of chloride–hypochlorite was extremely limited, it displayed severe gangue consumption even with the most stable ore (VCR), this is evident both in the change in mineralogy from feed to residue as well as the number and concentration of metals in the leach liquor. In addition to this, the gold dissolutions were extremely low despite a high reagent concentration. A study of the gold in the residue indicates that these gold grains

were not successfully leached as the ore contained minerals that were reactive with the leach solution, preventing optimal leach conditions and extensive leaching from occurring.

The thiourea was also highly reactive, which is evident in the consumption of muscovite, pyrrhotite, pyrite and calcite during leaching in addition to the elevated concentrations of copper, iron, manganese and nickel in solution. These minerals are clearly vulnerable to the conditions of this leach. The formation of new minerals, pyracmonite and nitrammite, highlights the degree to which thiourea degrades within the leach, as thiourea is the only source of nitrogen allowing for the formation of these new minerals.

The thiosulphate leaches showed a consistent lack of reactivity to the mineralogy of all the ores studied, despite high reagent concentrations. Additionally, the leach solution shows consistently lower metal concentrations than that of the cyanide leaches, which further reinforces that thiosulphate leaches do not react with any of the minerals present across all the ore types. This lack of reactivity is also evident in the thiosulphate-glycine leaches. The lack of reactivity of all thiosulphate leaches has positive implications for reagent consumption and solution purity. The mineralogy from the glycine-bearing thiosulphate leach residue shows a number of well liberated gold grains. The average grain size in the residue is significantly smaller than was present in the feed. The texture of the residual gold grains suggest that these particles have undergone incomplete dissolution. Therefore, improved dissolutions could be expected with longer periods of effective leaching (further discussed in Chapter 12).

Table 10-13 A classification of the identified mineral sensitivities and their relative degrees of reactivity based on solution chemistry and residue mineralogy.

Lixiviant	Mineral Sensitivities	Classification
Cyanide	**Pyrrhotite**	Moderately reactive
	Arsenopyrite	
	Antimony sulphides	
Chloride-Hypochlorite	**Pyrrhotite**	Highly reactive
	Pyrite	
	Calcite	
	Chlorite	Moderately reactive
Thiourea	**Pyrrhotite**	Highly reactive
	Chlorite	
	Carbonates	
	Pyrite	Moderately reactive
	Chalcopyrite	
	Muscovite	Slightly reactive
Thiosulphate	-	No observed mineral sensitivities

The degree of reactivity observed for both chloride-hypochlorite and thiourea are significantly greater than anything suggested in the literature (Deschenes and Ghali, 1988; Van Deventer et al., 1990; Sparrow and Woodcock, 1995; Adams, 2016). While the sulphide consumption was to be expected, the reactivity with generally stable minerals such as chlorite and muscovite is not described within literature but it entirely unique to this study. The lack of reactivity during thiosulphate leaching observed in this study is also very different from that of the literature (Aylmore, 2016a; Aylmore, 2016b) which states a sensitivity to sulphides, however this system was consistently less reactive than all other leaching systems, including cyanide.

Chapter 11

Consumption, Reagent Cost and Detoxification

11.1. Introduction

In order to evaluate alternative leaching systems, additional factors that will directly affect the economics of extraction must be considered. Consumption, reagent cost and detoxification will be briefly considered as they are a relevant aspect of any processing technique. While gold dissolution is of paramount importance, the expenses associated with the technique will strongly affect if it is integrated into industry or not.

11.2. pH Adjusting Reagent Consumption

The acid consumption for the low pH leaches (thiourea and chloride-hypochlorite) has been predicted based on carbonate content in the respective ores, in conjunction with their reactivity. The lime consumption for cyanide leaching is also included.

Chloride-Hypochlorite Leaching System

The chloride–hypochlorite system operates in an acidic stability field and thus concerns for consumption associated with the mineralogy relate to reaction between the acid and carbonates or other basic ore component. In the chloride leaching system the pH controlling reagent is hydrochloric acid (HCl). The reaction of HCl with common carbonates is a well understood process. Reactions with calcite, magnesite and dolomite (predominant carbonates identified in the ores) have been outlined in Equations 11.1 to 11.3 respectively;

$$CaCO_3 + 2HCl \rightarrow CaCl_2 + CO_2 + H_2O \qquad \text{(Equation 11.1)}$$

$$MgCO_3 + 2HCl \rightarrow MgCl_2 + CO_2 + H_2O \qquad \text{(Equation 11.2)}$$

$$MgCa(CO_3)_2 + 4HCl \rightarrow MgCl_2 + CaCl_2 + 2CO_2 + H_2O \qquad \text{(Equation 11.3)}$$

In general carbonates have high dissolution rates relative to that of other minerals, which is further emphasised by the XRD results for feed and residue that show complete dissolution of these carbonates during chloride-hypochlorite leaching. The equations used to determine the rate of reaction of carbonates with hydrochloric acid is in Equations 11.4 to 11.6, where M represents the metal ion;

$$MCO_3 + H^+ \rightarrow M^{2+} + HCO_3^- \quad k1 \quad \text{(Equation 11.4)}$$

$$MCO_3 + H_2CO_3 \rightarrow M^{2+} + 2HCO_3^- \quad k2 \quad \text{(Equation 11.5)}$$

$$MC0_3 \rightarrow M^{2+} + CO_3^{2-} \quad k3 \quad \text{(Equation 11.6)}$$

As per the stoichiometry of the Equations 11.4 to 11.6 as well as the reaction order of the equation the rate of the reaction between carbonates and HCl is expressed in Equation 11.7;

$$R = k1\ \alpha_{H^+} + k2\ \alpha_{HCO_3}^- + k3 \quad \text{(Equation 11.7)}$$

The rate constants (k1, k2 and k3), are values that define the relationship between the concentration of compound in a reaction (e.g. calcite) and the rate of that reaction (e.g. calcite reacting with HCl). The values of these constants were acquired experimentally by Chou et al., (1989) and is given in Table 11-1.

Table 11-1 The rate constants (defined based on their relationship with the components in Equation 11.7) for calcite, magnesite and dolomite reacting with hydrochloric acid. These constants correlate with the thermodynamic and calculated values (Chou et al., 1989).

	Calcite Rate Constant $mol.cm^{-2}.s^{-1}$	Magnesite Rate Constant $mol.cm^{-2}.s^{-1}$	Dolomite Rate Constant $mol.cm^{-2}.s^{-1}$
k1*	8.91^{-05}	2.51^{-09}	2.60^{-07}
k2*	5.01^{-08}	6.02^{-10}	1.00^{-08}
k3†	6.46^{-11}	4.47^{-14}	2.20^{-12}

Additional factors that are required to determine the rate of the neutralisation of carbonates is the activity of H^+ (α_{H^+}) and HCO_3 ($\alpha_{HCO_3}^-$).

The α_{H^+} is a factor that accounts for the deviations of substances from an ideal mixture. Deviations from ideal behaviours of chemicals within solution are accommodated by modifying the activity coefficient. Thus, the activity defines the actual amount of acid available to maintain the pH.

The activity of H^+ is correlated from the concentration of H^+ multiplied by the activity coefficient of H^+. The activity of H^+ is defined using several equations including the calculation to determine pH (Equation 11.8), as well as use of the Debye-Huckle (Equation 11.9) and Davis equations (Equation 11.11).

The extended Debye-Huckle formula (Equation 11.9) determines the activity coefficients using the ionic strength (I), the charge of the ion (Z), and the hydrated radius of the ion in pm (α). To determine the ionic strength (Equation 11.10), the concentration of the ion (c) and charge (z) are required. The Equation 11.10 works well for I ≥ 0.1M. To find activity coefficients for ionic strengths above 0.1 M, more complicated equations are usually used.

$$pH = -\log(\alpha_{H^+}) \qquad \text{(Equation 11.8)}$$

$$\log \gamma = \frac{-0.51\ zi^2\ (I)^{0.5}}{1 + [\alpha\ (I^{0.5})/305]} \qquad \text{(Equation 11.9)}$$

$$I = 0.5\ \Sigma\ cz2 \qquad \text{(Equation 11.10)}$$

There are several shortfalls of the extended Debye- Huckle equation, particularly with regards to factoring the ionic strength calculations. Thus, an alternative is necessary, the best of which is the Davis equation (Equation 11.11);

$$\log \gamma = -0.51\ z^2 \left(\frac{(I)^{0.5}}{1 + I^{0.5}} - 0.3I\right) \qquad \text{(Equation 11.11)}$$

The values for the activity constant of H^+ have been determined using the above equations (Equation 11.10 and 11.11), where Y = 0.2 and αH+ = 1.43. The activity of H_2CO_3 has been assumed to be the saturation of carbonic acid (5.71×10^{-5})(Greenwood and Earnshaw, 2012). While this value may theoretically be higher, the rapid agitation associated with bottle rolling will reduce the extent to which this value could surpass saturation. The limitations to this assumption are mitigated by the fact that this value is only a second order component to the rate calculation.

Based on the above defined values and using Equation 11.7 the reaction rates for calcite, dolomite and magnesite reacting with HCl is determined (Table 11-2).

Table 11-2 The reaction rates determined for carbonates found within the ores.

Reaction Rates ($mol.cm^{-2}.s^{-1}$)	
Calcite	1.28E-04
Magnesite	3.59E-09
Dolomite	3.72E-07

Using the reaction rates, the geometric surface area and the length of leach, the amount of carbonate minerals that could be leached in a specific leaching period is determined (Table

11-3). The surface area of the carbonates was calculated using a geometric surface assuming spherical particles guided by the sizes seen in the mineralogy. Consumption was then determined using Equation 11.12.

Consumption = Reaction rate x leach length x carbonate surface area (Equation 11.12)

[moles] [$mol.cm^{-2}.s^{-1}$] x [s] x [cm^2]

Table 11-3 The variables required in order to calculate the total mass of respective carbonates which could be consumed in a given leach (given as mass consumed). The calculated masses consumed in the leaching period are all greater than the total amount of sample used in a leach.

Minerals	Total Surface Area for a 500 g Sample (cm^2)	Time (sec)	Consumption (mol)	Molar Mass (g/mol)	Mass Consumed (g)
Calcite	208870	28800	362102.20	100.09	36242809.18
Magnesite	199309	28800	9.74	84.31	821.07
Dolomite	189946	28800	960.65	184	176759.08

The total masses of carbonates that could be consumed in an 8-hour acid leach are 36 245 kg for calcite, 0.8 kg for magnesite and 176 kg for dolomite. All these masses are greater than the total amount of sample used in a single leach (0.5 kg). Therefore, it can be assumed that all carbonates present in the sample would be consumed entirely and thus the total carbonate mass has been used to define acid consumptions.

Calculated acid consumption ranges from 2.9 kg/ton to 461.5 kg/ton (Table 11-4). These consumptions have been calculated based on pure HCl (however the maximum concentration of hydrochloric acid is 37% due to solubility limits). Furthermore, these values have been calculated based on the consumptions of carbonates but there is evidence that other components (chlorite etc.) have also been consumed. While these minerals have been consumed to a lesser extent, this will still contribute to the acid consumption.

Table 11-4 Calculated acid consumption for the ores of study in the chloride-hypochlorite leach. Acid consumption ranges from 2.9 kg/ton for Ventersdorp Contact Reef (VCR) to 461.5 kg/ton for Barbrook.

			HCl Consumption / Mineral	HCl Consumption Total
Ore	Mineral	Mass %	kg/ton	kg/ton
VCR	Calcite	0.4	2.9	2.9
Fairview	Dolomite	10.8	85.4	146.8
	Magnesite	7.1	61.4	
Barbrook	Dolomite	58.0	458.6	461.5
	Calcite	0.4	2.9	
Pilgrim's Rest	Dolomite	4.9	38.7	38.7
Consolidated Murchison	Magnesite	38.0	328.6	360.2
	Dolomite	4.0	31.6	

Thiourea Leaching System

Sulphuric acid is used to control the pH in this system and the reactions that occur with calcite, magnesite and dolomite respectively are described in Equation 11.13 to 11.15;

$$CaCO_3 + H_2SO_4 \rightarrow CaSO_4 + CO_2 + H_2O \quad \text{(Equation 11.13)}$$

$$MgCO_3 + H_2SO_4 \rightarrow MgSO_4 + CO_2 + H_2O \quad \text{(Equation 11.14)}$$

$$MgCa(CO_3)_2 + 2H_2SO_4 \rightarrow MgSO_4 + CaSO_4 + 2CO_2 + 2H_2O \quad \text{(Equation 11.15)}$$

The degree of reactivity of these carbonates is related to several factors including acid concentration, carbonate particle size, and other metals in solution. Of these components the acid (H^+) concentration/activity is the most significant factor (Chou et al., 1989; Maree and Du Plessis, 1994).

Using the same free acid calculations described in Equations 11.4 to 11.6, in addition to the stoichiometry (Equation 11.12 to 11.14), the rates for dissolution of the respective carbonate minerals are determined (Table 11-5). Rate reactions are then used to determine the total carbonate mass consumed in an 8 hour leach – which is 43 446 kg for calcite, 0.098 kg for magnesite and 21 kg for dolomite (Table 11-6). The masses for calcite and dolomite are greater than the total masses of the sample and therefore it can assumed that there was complete consumption of the respective carbonates present in the material used for leaching (Table 11-7). However, any sample that contains more than 98.46 g (i.e. 20 % of the sample)

would be limited in the consumption of magnesite. Thus, leaches involving Consolidated Murchison ore would be unable to consume all carbonates present during the leach duration, as the carbonate content of 38% magnesite, is greater than the calculated maximum 20% consumption possible in an 8-hour leach. This has limited the acid consumption to 100 g/ton due to magnesite kinetic reactivity.

The total acid consumption for this leaching system relative to the ores of study ranges from 3.92 to 620.90 kg/ton, most of these consumptions are far too high to be considered economic. It should be noted that calcite interacting with sulphuric acid may form a passivating layer of gypsum, which would reduce the degree of consumption (Booth et al., 1997). Calcite, however, is present in limited quantities (less than 0.4%) and therefore this is not a major consideration in this work.

Table 11-5 The reaction rate determined for the carbonates found in the ores, when reacting with sulphuric acid.

Reaction Rates ($mol.cm^{-2}.s^{-1}$)	
Calcite	7.22E-06
Magnesite	2.03E-10
Dolomite	2.11E-08

Table 11-6 The variables required to determine the total mass of carbonates that could be consumed in the duration of the leach. The masses for calcite and dolomite are greater than the total masses of a leaching sample. However, for magnesite, any material that contains more than 98.46 g (i.e. 20 % of the sample) would be limited in the consumption of magnesite.

Minerals	Total Surface Area for a 500 g Sample (cm^2)	Time (sec)	Consumption (mol)	Molar Mass (g/mol)	Mass Consumed (g)
Calcite	208870	28800	43407.91	100.09	4344698.14
Magnesite	199309	28800	1.17	84.31	98.46
Dolomite	189946	28800	115.17	184	21191.72

Table 11-7 Calculated acid consumption for the ores of study in the thiourea leach. The total acid consumption for this leaching system, relative to the ores of study range from 3.92 kg/ton for Ventersdorp Contact Reef (VCR) to 620.90 kg/ton for Barbrook.

			H_2SO_4 Consumption /Mineral	H_2SO_4 Consumption Total
Ore	Mineral	Mass %	kg/ton	kg/ton
VCR	Calcite	0.4	3.92	3.92
Fairview	Dolomite	10.8	114.89	197.48
	Magnesite	7.1	82.59	
Barbrook	Dolomite	58	616.98	620.9
	Calcite	0.4	3.92	
Pilgrim's Rest	Dolomite	4.9	52.12	52.12
Consolidated Murchison	Magnesite	38	84.73	100.54
	Dolomite	4	15.81	

Cyanide Leaching System

Throughout cyanide leaching, the pH of the system was maintained at 10.5 using the addition of lime. The conditions for the leach were consistently monitored and adjusted in order to maintain the target values. Due to variations within the ore, pH adjusting reagent additions vary for respective samples.

Increased lime consumption was incurred for Fairview ore which only required a minor amount of lime addition (1.2 mL), as well as for the VCR (1.3 mL), Barbrook (1.9 mL) and Consolidated Murchison (2.2 mL) while Pilgrim's Rest required an order of magnitude more reagent (17.6 mL). The acidic nature of the Pilgrim's Rest sample is likely to be due to the significant concentration of iron sulphide minerals (predominantly pyrite), which tend to be oxidised during the cyanide leach, thus resulting in the formation of acid (Heinen et al., 1978). The amount of sulphur in solution (Table 10-8) was used to determine the amount of pyrite oxidised during leaching and the subsequent formation of acid in order to predict lime (CaO) consumption The potential formation of gypsum was considered, however calculations determining the potential presence of gypsum using the solubility product constant along with solution activity confirmed that gypsum saturation was not obtained and therefore the formation of gypsum (and the sulphur component therein) was not taken into consideration. The predicted and actual consumptions of CaO are generally similar (Table 11-8), however for Barbrook (BK) and Consolidated Murchison (CM) the predicted consumption is

significantly more than the actual consumption of CaO. This is explained by the mineralogy of the ores, as Barbrook and Consolidated Murchison have abundant carbonates. Barbrook contains 58% dolomite while Consolidated Murchison has 28% carbonates, thus the leach solution for these respective ores are already alkaline and less CaO is required for neutralisation.

Table 11-8 Tabulated data comparing the predicted and actual CaO consumption during cyanidation.

	Predicted CaO Consumption	Actual CaO Consumption
Deposits	g	g
VCR	0.11	0.13
Fairview	0.17	0.12
Barbrook	0.72	0.22
Consolidated Murchison	0.48	0.19
Pilgrim's Rest	1.44	1.76

An attempt was also made to predict CaO consumptions using the feed mineralogy, however this was found to be impractical as there are many factors controlling the generation of acid. While pyrite is an acid-forming mineral in solution, it has limited degrees of reactivity, where pyrrhotite is significantly more reactive and in trace quantities may have a greater impact than pyrite in major quantities. Furthermore, the extent of oxidation in pyrite is dependent on a number of factors including morphology, surface texture and prior oxidation in situ (Weisener and Weber, 2010; Cole, 2018). For this reason, the mineral abundances are insufficient for predicting the lime consumption of cyanide leaches.

11.3. Main Reagent Consumption

The consumption of the main reagent is relevant to evaluating the leach as they determine the amount of reagent required in addition to the effectiveness of the reagent used.

The reagent consumption for cyanide, thiourea and thiosulphate was determined using the titration methods outlined in Chapter 8. These consumptions have been represented in Table 11-9 and Figure 11-1.

The reagent consumptions for cyanide, in terms of kg/ton, are significantly lower than the other leaching systems. Cyanide consumption increases from 1.2 kg/ton for the VCR sample, with further increasing values for Fairview, Consolidated Murchison, Barbrook and up to 2.41 kg/ton for the Pilgrim's Rest sample. The cyanide remaining at the end of the leach was between 10% and 30% of the original value, with the exception of the Pilgrim's Rest leach. Therefore, sufficient cyanide was present throughout most of the leaches. The higher cyanide consumptions for the Pilgrim's Rest and Consolidated Murchison samples was likely due to the reactive components of the ore. The consuming agents within the Pilgrim's Rest sample are likely to be the abundant pyrite with a minor contribution from pyrrhotite, chalcocite and chalcopyrite. The consuming agents within the Consolidated Murchison sample are likely to be stibnite (Sb_2S_3), pyrite, berthierite ($FeSb_2S_4$), and lesser arsenopyrite. These assumptions are supported by the solution assays that are discussed later.

The reagent consumptions for the thiourea leach are significantly higher than consumptions incurred for the cyanide leach. There is little difference between the consumption of the VCR and Pilgrim's Rest samples, which despite having very different mineralogies both incurred approximately 10 kg/ton thiourea consumption. Due to these high consumptions, less than 1% of the original thiourea was present at the end of the 8-hour leach. It is therefore evident that this high reagent consumption is due to the instability of the thiourea, rather than the reactivity of the ore mineralogy.

The thiosulphate consumption is extremely high for all ores, however the VCR, Fairview, Barbrook and Consolidated Murchison samples all tended to have consumptions of approximately 60 kg/ton, with approximately 30% of the original reagent remaining at the end of the leach, despite vastly different mineralogies. The Pilgrim's Rest sample however consumed significantly more reagent at 85 kg/ton, leaving none of the original thiosulphate available for leaching after 24 hours. It is interesting to note that the Pilgrim's Rest ore is the highest consuming ore, as pyrite, the most dominant sulphide within this sample is generally considered unreactive, particularly with cyanide leaching, while some of the other ores have extremely reactive sulphides which showed little additional contribution to consumption.

The addition of glycine to thiosulphate leaching made a notable contribution to the reduction of thiosulphate consumption, bringing the average consumption down to approximately 50 kg/ton of thiosulphate, with notable variations in individual ores. The Pilgrim's Rest ore

consumed the most thiosulphate, followed by VCR, Barbrook, Fairview and Consolidated Murchison samples. It is interesting to note that the VCR sample had the second highest consumptions despite having very limited reactive components. Considering the high presence of reactive sulphides within the Consolidated Murchison sample, it is also surprising that the Consolidated Murchison leach consumed the least amount of reagent.

The calcined thiosulphate leaches showed significant variations in the total reagent consumed, with an average consumption of 52 kg/ton showing slightly higher values than the consumptions for thiosulphate and glycine leach solution. However, there is a significant variation between individual ores of more than 30 kg/ton. The Barbrook ore consumed the most thiosulphate, followed by the Fairview and Pilgrim's Rest samples. With notably less consumption in the Consolidated Murchison and VCR material. These relative consumptions are unrelated to addition of any reagent required for pH adjustment.

The thiosulphate and glycine leaches on calcined ores had an average consumption of 54 kg/ton, which is a higher reagent requirement than in the previous leaches (both calcined and glycine bearing), but still lower than the original unaltered thiosulphate leach. The consumption for each ore tended to be quite similar, with the highest consumption for the Pilgrim's Rest sample at 60.38 kg/ton and the lowest for the Consolidated Murchison sample at 48.20 kg/ton.

Table 11-9 Reagent consumption for the leach systems (cyanide, thiourea and thiosulphate) and deposits (Ventersdorp Contact Reef, Fairview, Barbrook, Consolidated Murchison and Pilgrim's Rest). The cyanide consumption is extremely low, approximately 3 kg/ton, and is a stark contrast relative to that of thiourea with 10 kg/ton and thiosulphate, with up to 80 kg/ton consumption.

Leach	Ore	Consumption (kg/ton)
Cyanide	VCR	1.20
	Fairview	1.33
	Barbrook	1.47
	Consolidated Murchison	2.18
	Pilgrim's Rest	2.41
Thiourea	VCR	9.91
	Pilgrim's Rest	10.01
Thiosulphate	VCR	62.01
	Fairview	62.12
	Barbrook	62.18
	Consolidated Murchison	61.55
	Pilgrim's Rest	85.31
Thiosulphate + Glycine	VCR	56.83
	Fairview	45.51
	Barbrook	51.42
	Consolidated Murchison	37.22
	Pilgrim's Rest	59.13
Thiosulphate + Calcine	VCR	32.83
	Fairview	63.91
	Barbrook	73.03
	Consolidated Murchison	38.91
	Pilgrim's Rest	62.39
Thiosulphate + Calcine + Glycine	VCR	56.69
	Fairview	54.46
	Barbrook	53.12
	Consolidated Murchison	48.20
	Pilgrim's Rest	60.38

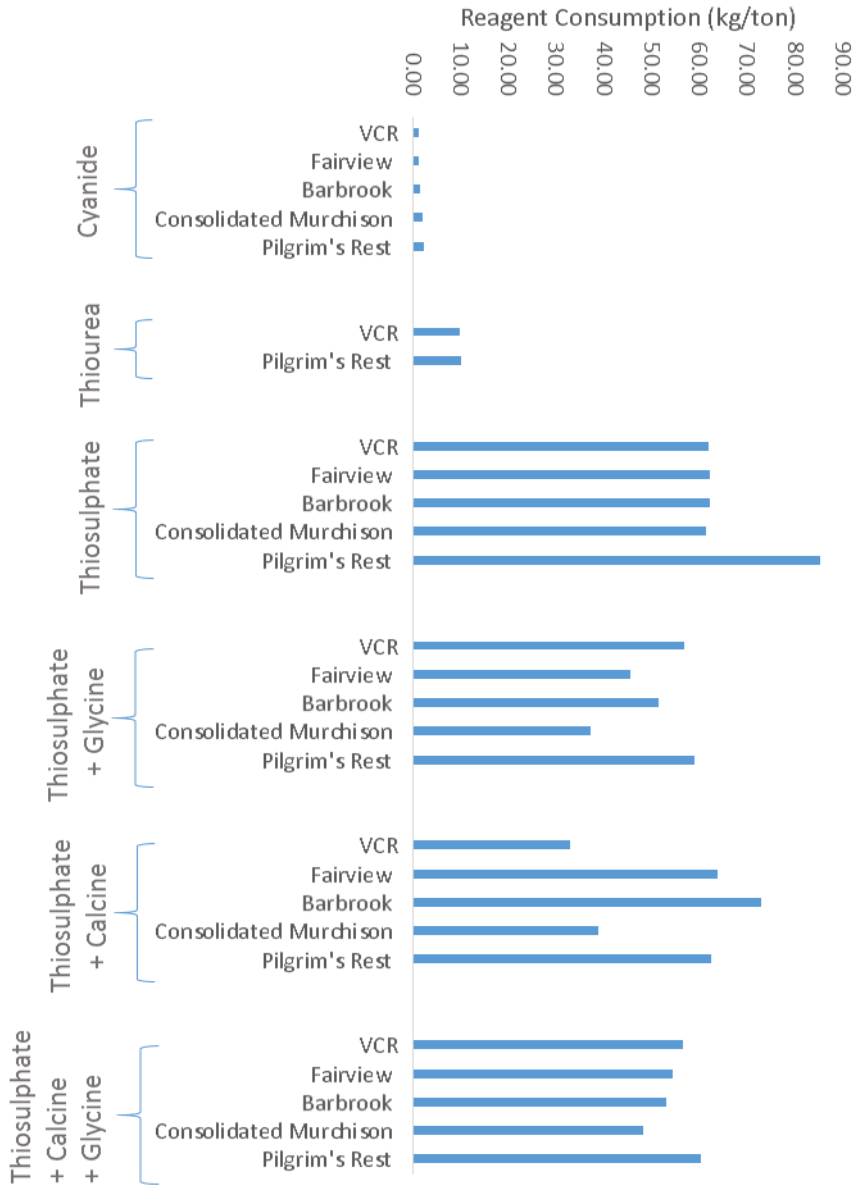


Figure 11-1 Graph of reagent consumption for the leach systems and deposits. Cyanide consumption is extremely low, approximately 3 kg/ton, relative to that of thiourea with 10 kg/ton and thiosulphate, with up to 80 kg/ton.

From these results it is evident that the reagent consumptions tend to reflect a correlation with the concentrations required for the leach.

11.4. Reagent Price Comparison

The pricing of reagents for respective leaching systems is a vital element in order to completely evaluate a technique. The bulk prices for the reagents for both the South African as well as international markets were compared and showed little variation between countries. The data below is based on South African availability (an average based on data obtained from Protea Chemicals and Marlyn Chemicals). The pricing of the respective reagents (Table 11-10) and whole leach systems (Table 11-11) have been represented as ratios relative to cyanide or cyanide leaching, for ease of reporting and the sake of consistency. Additionally, these ratios should stay approximately the same after inevitable price increases.

Table 11-10 emphasises the high cost of cyanide per unit relative to other reagents, which are generally less than half the price of cyanide. However, in order to evaluate the leaches in a consistent manner it is important to consider costs within the context of the leach. In order to do so, all reagents along with their respective concentrations must be incorporated to determine total costs (Table 11-11). The exact quantities of reagent required for leaching were estimated for low consuming ores.

Therefore, despite the relatively high price of cyanide per unit, the cyanide leach is overall the cheapest to run. The low costs associated with cyanide leaching are due to the low concentration requirements for cyanide and lime.

For thiourea leaching, ferric iron and significant amounts of sulphuric acid are required in addition to the thiourea, which in total makes this leach system 5.29 times more costly than cyanide.

The chloride-hypochlorite leach required large concentrations of sodium chloride, sodium hypochlorite and hydrochloric acid which makes the leaching reagents for this system 19.87 time more expensive than the cyanide system.

The thiosulphate leaching required sodium thiosulphate, copper sulphate and ammonia, making the leach an average of 4.8 times more expensive than leaching by cyanide.

While there may be scope for the reduction in the amount of the reagent required for both thiosulphate and thiourea, the focus of this research was not to optimise the reagent concentrations and therefore these values are based on excess leaching conditions.

Table 11-10 Reagents required for leaching and their average price as a fraction relative to cyanide. Cyanide is by far the most expensive reagent, with most reagents less than half that of cyanide, with the exception of thiourea and copper sulphate.

Reagent	Cost (Relative to Cyanide)
Cyanide	1
Thiourea	0.807
Sodium thiosulphate	0.058
Sodium chloride	0.046
Lime	0.230
Copper sulphate	0.790
Ferric sulphate	0.080
Sulphuric acid	0.156
Ammonia	0.133
Hypochlorite	0.150
Hydrochloric acid	0.102

Table 11-11 Average cost of all the reagents required for individual leaching systems relative to cyanide (values below assume negligible gangue reagent consumption and no reagent recycling). Cyanide is by far the most economic leaching system, while chloride-hypochlorite leaching is approximately 20 times more.

Leaching System	Cost Relative to Cyanide
Cyanide	1.00
Thiosulphate	4.81
Thiourea	5.29
Chloride-hypochlorite	19.87

11.5. Detoxification Considerations

The detoxification required for the components used in cyanide leaching can be expensive and thus the requirements for detoxification for the alternatives may offer a financial advantage in this regard. There are a number of well understood methods available for cyanide detoxification, utilising SO_2/air, hydrogen peroxide, alkaline chlorination, in addition to biodegradation and natural degradation (Logsdon et al., 1999; Botz and Mudder, 2002; Akcil and Mudder, 2003). There has been relatively little research into the detoxification of the alternative lixiviants. The considerations into detoxification for the leaching systems in

this thesis have therefore been based exclusively on the components that exceed the values recommended by the South African Standards for Effluent Discharge (Environmental Protection Regulations).

A major factor for both thiourea and chloride-hypochlorite leaching is the acidity of the solution. Acceptable discharge levels for pH range between 5 and 9 and therefore both the chloride-hypochlorite and thiourea leach solution would require an increase in pH before they could be legally discharged. Increasing pH for effluent is generally implemented by the addition of lime (Kuyucak and Sheremata, 1995). Thiourea-based leaching would require significantly more lime in order to achieve neutralisation, due to its lower operating pH. The amount of sulphate and iron in the thiourea leach (due to the requirements for ferric sulphate) must be reduced before it can be safely discharged. The iron concentration is reduced by bubbling oxygen through the solution (to ensure all ferrous iron is oxidised into ferric iron), and subsequently raising the pH sufficiently to ensure precipitation (Hove et al., 2007). The sulphates (a potential product of thiourea degradation) can be removed by barium chloride (Kefeni et al., 2015).

The chloride-hypochlorite leach liquor contains hypochlorous acid which will be neutralised with the pH adjustment previously mentioned. Further removal of the chlorine is possible, with the use of reverse osmosis or ion exchange membranes, however this is likely to be a prohibitively expensive and slow process particularly at such high concentrations (Heinen et al., 1978; Strathmann, 2004). There is however the capacity for solution recycling which would significantly reduce the expenses for both reagent requirements as well as detoxification (Ximing et al., 1992).

Thiosulphate would theoretically not require detoxification due to its non-toxic nature. However, the pH of the solution would be slightly too high for discharge and therefore would need to be adjusted to a safer pH using an acid, such as hydrogen peroxide or sulphuric acid. In addition, the copper concentration in the thiosulphate leach would be much higher than the 0.5 mg/L permitted for discharge. A resin-based recovery system for copper has been suggested by Jeffrey, (2001) where both gold and other metals are absorbed onto the resin. The copper is subsequently eluted and can be recycled for use in leaching. After the copper and other non-precious metals are eluted, gold can subsequently be recovered. Thiosulphate is commonly used for fertiliser (Langhans et al., 1992), which could allow for the discharge to

be sold to farms. The sale of thiosulphate-bearing solution could reduce the costs of detoxification and even add an additional source of revenue. However, the copper required for leaching would have to be removed from the solution to avoid high copper concentrations in soils. Elevated copper concentrations in soils are known to negatively affect plant growth (Adrees et al., 2015) and therefore this association in fertiliser is unfavourable and would have to be addressed. In addition, the degradation of thiosulphate throughout leaching would have to be limited in order for the discharge to be a useful fertiliser.

11.6. Discussion

The acid-based leaches are extremely reactive, consuming significant quantities of gangue minerals. While the consumption of gangue is not fundamentally a problem, the use of reagent for materials other than gold highlights inefficiencies and wasted reagent within the leach.

The acid consumptions incurred for the chloride-hypochlorite and thiourea leaches are well beyond acceptable limits of reagent use, with consumptions reaching more than 400 kg/ton. For this reason, some ores were fundamentally uneconomic even to test. Only the results for the VCR sample were presented for the chloride-hypochlorite leach, while the VCR and Pilgrim's Rest thiourea leaches are also available. In the context of cyanide leaching, the Pilgrim's Rest sample was the only one that has significant lime consumption due to the presence of acid forming sulphides.

In addition to the consumption of pH adjusting reagents, the consumptions of the main lixiviant must be taken into consideration. The consumption of thiosulphate and thiourea are of foremost concern due to their instability and tendency to undergo oxidative decomposition in the presence of sulphides. The stability of these reagents is also sensitive to pH and redox conditions, while cyanide is stable in a greater pH and redox range. Cyanide consumptions were moderate for the VCR, Fairview and Barbrook samples, and higher for the Consolidated Murchison and Pilgrim's Rest samples. These consumptions are consistent with the mineralogy of the ores.

The thiourea consumptions were significantly higher than cyanide, however this is to be expected due to the much higher concentrations required for targeted leaching conditions. At the completion of the 8-hour leach, there was less than 1% of the initial thiosulphate

remaining. This highlights the instability of the system and suggests consumptions would be higher, given sufficient reagent throughout the leach.

The thiosulphate consumptions are somewhat variable for the different ores and leaching variants; however, consumptions are all significantly higher than that determined for cyanide and thiourea. However, the concentrations required for thiosulphate leaching are also much higher than cyanide and thiourea. Consumptions for thiosulphate alone are consistently high, while the calcined ores and addition of glycine generally reduces the consumptions, but inconsistently so. The addition of glycine significantly improved the consumptions of the reactive ores (Pilgrim's Rest, Consolidated Murchison, Fairview and Barbrook) but has limited effect on the VCR material for which the thiosulphate consumption remains consistently high. The calcined ores significantly improved the thiosulphate consumption of the VCR, Consolidated Murchison and Pilgrim's Rest samples, but increased the consumptions for Fairview and Barbrook material. The leaches that utilised both calcined ores and glycine additions generally exhibited increased thiosulphate consumptions, to varying degrees depending on the sample. The increase in reagent consumption may represent the increased ability of this leach system to effectively recover gold. However, these minor but consistent variations in consumption may relate more to the conditions present during leaching than the gangue components of the ores. Variations in pH and Eh may be responsible for reagent degradation due to its inherent instability. As the mineralogy in the respective leaching systems for individual samples would not vary, the gangue interaction would not account for the inconsistent consumptions of samples relative to one another. Slight variations within the leaching conditions are therefore much more likely to be responsible for these relative consumptions.

While the consumptions incurred by the leaching systems are relevant, of greater importance is the reagent pricing. When comparing the cost of individual reagents, cyanide is by far the costliest, however further consideration of factors such as reagent concentration and the number of reagents required for leaching is important. When incorporating these factors to provide a complete leaching system cost, cyanide is by far the most economic. The chloride-hypochlorite leach is by far the most expensive with a value of 20 times the overall cost of cyanide leaching, while the thiourea leach costs approximately 5 times more than cyanide. The cheapest alternative process is thiosulphate with costs of approximately 4.8 times

greater than that of cyanide. Despite the significant consumptions associated with thiosulphate, the unit prices for this reagent is moderate. If these consumptions or concentrations could be mitigated further with the use of stabilising agents or other optimisation processes, there is potential for competition with cyanide leaching in terms of cost. Unfortunately, except in very specific cases, gold dissolution by thiosulphate is still economically inferior to cyanide.

The detoxification processes for alternative systems are not well constrained. All leaching systems, including cyanide, require pH adjustment before discharge. Cyanide has the advantage of several well-defined detoxification processes. Thiosulphate and to a lesser extent thiourea are currently classified as sufficiently non-toxic that they would not require neutralisation. However, thiosulphate and thiourea leaching involve high concentrations of copper and iron respectively, which would both require removal before they could safely be discharged. The chloride-hypochlorite neutralisation process is somewhat less clear, however processes such as precipitation could reduce salinity. The use of ocean water for these leaches has the potential to reduce costs particularly due to the presence of sodium chloride and therefore some scope may exist for the recycling of leach solution into the ocean.

The instability of thiosulphate and its tendency to decay, particularly to tetrathionates, reduces the ability to utilise recycling in the context of the leach. Should this process be mitigated, the use of thiosulphate as a fertiliser is certainly a possibility. The recycling process for ammonia is well understood and easily implemented, while the consumption of copper is minimal, and therefore both of these reagents have the potential for recycling (Abbruzzese et al., 1995).

While the reagent expenses for the chloride leaching system is approximately 20 times more than cyanide, there has recently been research into improving the recyclability of the reagents. Recycling could make chloride-hypochlorite leaching more economic in terms of reagent purchasing and detoxification costs (Adams, 2016; Aylmore, 2016b). In addition to high reagent prices, some elements of capital expenditure for leaching procedures would be higher than in use for cyanide leaching. Material required for chloride-hypochlorite leaching would have to be resistant to corrosion, due to the very oxidising conditions at which this leach operates and would thus be more expensive. Conversely, due to the rapid leaching

rates associated with this system there would be potential to cut costs by reducing the size and or number of leaching vessels (Adams, 2016; Aylmore, 2016b).

Similar capital expenditure issues exist for processing using thiourea as they do for chloride-hypochlorite. These leaching systems both operate within an acidic pH range. Higher costs will be required for material capable of withstanding the leaches at low pH's, but more rapid leaching rates could reduce the amount of equipment required for the leaching process.

A diverse set of data have been reported in the literature regarding the consumptions of thiourea, ranging from 0.5 to 47 kg/ton (Schulze, 1984). The consumption for the completed thiourea leaches for the Pilgrim's Rest and VCR samples approached 10 kg/ton, which represents the total concentration of thiourea available for leaching. In order to allow for a better understanding of the consumptions associated with this leach, more testing would be required, with alternate conditions where excess thiourea is maintained. The remaining post-leach concentration should be evaluated in conjunction with the residue mineralogy.

There is scope for optimising the conditions required for alternative leaching, thus consumptions could well be less extreme in the future, while maintaining or improving gold dissolution. Processes such as recycling, or in the case of the chloride-hypochlorite leach, using untreated ocean water, may have implications for reduced operating costs. Despite low costs per unit of reagent, the concentrations and consumptions make these alternatives notably more expensive than cyanide. These increased economics are not mitigated by the detoxification costs and therefore currently the alternative leaching techniques of study are generally not financially feasible. While there is potential for improvement, until significant developments are made, the alternative extractive processes are currently, in most contexts, economically unfavourable.

Chapter 12

Discussion

12.1. Introduction

The environmental concerns surrounding cyanide-based extraction are due to a number of well publicised cyanide related accidents with widespread environmental impacts including the Baia Mare Cyanide Spill of 2000. However, the use of cyanide in well-maintained plants is generally of limited concern, while consistent and minor levels of exposure have no proven long-term effect. Extensive measures are generally taken to prevent cyanide exposure to surrounding wildlife. Whether the widespread concerns surrounding the effects of cyanide are over exaggerated or appropriate, the demands for increasingly stringent cyanide controls are very real. In addition, the negative connotation of cyanide and increasing amounts of community influence in local mining (and processing) mean increasing amounts of pressure on cyanide use.

There is also a notable decrease in the global amount of gold ore that is amenable to cyanide extraction. Much of the free milling material in certain mines has been exploited and what remains is the more refractory gold at greater depths. Additionally, the number of new free-milling gold deposits that are being discovered is also limited.

Extensive motivation for research into alternatives exists, and this is evident in relevant publications in the literature, in conference agendas and in commercial laboratory research, where numerous methods, ranging from Glycat™ to bromine leaching, have been studied. These methods have been reviewed by a number of writers (Sparrow and Woodcock, 1995; Hiskey, 2000; Aylmore, 2005; Cui and Zhang, 2008; Aylmore, 2016a). Unfortunately, of these alternative methods, only a limited number of techniques show a chemical or financial feasibility. The chemical feasibility of a method is evaluated on the basis of the reagents ability to successfully dissolve gold and maintain gold in solution for a certain period. The limited chemical reactivity of gold, while a part of the appeal of the element, is a major difficulty in terms of hydrometallurgical recovery. Gold is reactive with cyanide but is generally an order of magnitude less stable when interacting with non-cyanide lixiviants. The limited understanding of the solution chemistry of alternatives relative to cyanide is a major

disadvantage. Fortunately in recent years there have been major advancements in the understanding of the chemistry of these alternatives, particularly thiourea and thiosulphate (Senanayake, 2004; Li and Miller, 2006). In addition to the chemical concerns there are also financial limitations relating to high reagent concentrations, reagent prices and alternative processing plant requirements.

In order to evaluate alternatives gold extraction techniques, three of the more feasible leaching methods were selected for testing in this thesis; thiosulphate, thiourea and chloride-hypochlorite. These methods span a variety of conditions from extremely oxidising low pH environments to moderate oxidation and basic leaching conditions. The selected leaching methods also include lixiviants that operate using mechanisms and processes that have long been understood with proven, if outdated, technology (chloride-hypochlorite), while others have only been discovered more recently (thiourea).

The selected methods were then assessed using several gold deposits. In order to evaluate these techniques properly, it was important to locate and characterise a mineralogically diverse series of gold samples. Samples were collected from the Witwatersrand Basin, the Barberton Greenstone Belt, the Murchison Greenstone Belt and the Transvaal Basin. The Witwatersrand VCR sample represented a high grade, free milling ore with a limited number of reactive components. The material collected from Barberton comprised two samples, from Fairview and Barbrook, both of which had a certain amount of gold locked in solid solution within arsenopyrite. In addition to the refractory component in the Barbrook sample, its mineralogy was further complicated by the presence of preg-robbing carbon. The Consolidated Murchison material had both native gold and gold within the aurostibite, in addition to several antimony minerals which are generally reactive as well as passivating. Finally, the Pilgrim's Rest sample was a free- milling and high-grade ore but was also host to a moderate amount of consuming/passivating minerals. These ores were well characterised using several mineralogical methods including optical microscopy, XRD, ICP-MS, XRF and MLA.

In addition to characterising the mineralogies of these samples in the context of leaching, an effort was made to understand the conditions that facilitated the formation of the metallurgically complex deposits. This was done in order to gain a better understanding of the environments in which they might form and to understand the controls on the formation

of refractory deposits. This understanding may have implications for targeted gold exploration in the future. The low metamorphic grade associated with, and occurring subsequent to, the Barberton mineralisation event is responsible for the gold bound within arsenopyrite as solid solution, whereas higher metamorphic conditions may have promoted the occlusion of the gold. Gold deposits associated with mafic or ultramafic intrusive systems either directly or hydrothermally are likely to have abundant pyrrhotite. Furthermore, some reactive components (arsenic and antimony) tend to be upgraded in sedimentary processes. Using the understanding of the formation of these deleterious components (as described above) in conjunction with mineralisation targeting in regions with well-defined geological history would help avoid time and effort spent on less feasible regions.

The ability to predict mineral responses to cyanidation based on deposit formation and geological history is a major advantage in early exploration stages, but it also emphasises the degree to which we understand the interaction between cyanide and gangue minerals and the lack of similar knowledge for alternative lixiviants. This gives rise to the major objective of this study, to understand the interaction of the gold and gangue mineral phases during leaching using alternative lixiviants.

In order to understand these interactions, the leaching systems were tested against well characterised, homogenised, milled samples, using bottle roll leaches at ambient temperature and pressure conditions. The appropriate conditions required for individual leaching systems (as based on literature) were achieved and adjusted approximately once an hour until stable conditions were maintained. Once the leach was completed, the residue and solution were separated. The residue was washed, dried and submitted for fire assay and XRD analyses. In certain circumstances, where the gold mineralogy of the residue needed to be studied, material was also submitted for MLA characterisation. The solutions were titrated to determine reagent consumption as well as submitted for fire assay, to evaluate the degree of successful gold dissolution. The leaching methods, their dissolution results, consumptions and subsequent implications have been subdivided below.

12.2. Chloride-Hypochlorite Leaching

The ability of halides to leach gold have long been known, with chlorine gas and bromine/bromide leaching implemented as early as the 1800s (Marsden and House, 2006).

This technology has been surpassed by leaching methods developed particularly since the discovery of cyanide, and a subsequent competitive method using halides as an alternative to cyanide has yet to be developed. The use of chloride-hypochlorite as a lixiviant has gained research momentum in the past twenty years (Puvvada and Murthy, 2000; Jeffrey et al., 2001; Ferron, 2003; Baghalha, 2007; Nam et al., 2008; Hasab et al., 2013; Hasab et al., 2014; Adams, 2016).

In order to better understand the chloride-hypochlorite technique, leaching was conducted using the conditions described in the literature, on the variety of ores used in this study. The chloride-hypochlorite leach had limited results, as the targeted conditions could only be stabilised sufficiently to complete leaching for the least reactive of the ores (the VCR sample). During the leaching attempts it was evident that reagent consumptions, particularly of the hydrochloric acid required to stabilise the leach solution pH, were extremely high. Consumptions of more than 400 kg/ton have been estimated for the more carbonate-rich ores. Some of the minerals consumed in this leach include calcite, pyrrhotite, pyrite, chlorite and muscovite. Halogens are also thought to be extremely reactive with sulphides according to literature (Jeffrey et al., 2001; Hasab et al., 2014; Adams, 2016), which was certainly evident in the leaches. In addition to the consumption of carbonates and sulphides, there was the unexpected reactivity with somewhat more stable minerals such as phyllosilicates. The consumptions and unstable pH and redox conditions are not described in literature, thus this understanding, in addition to the extreme gangue reactivity is a novel contribution to the understanding of the chloride-hypochlorite leach.

Furthermore, the total gold dissolution achieved by this system for the VCR sample was approximately 40%, whereas almost complete gold dissolution is possible with the use of cyanide. The VCR sample is free-milling and therefore the limited gold dissolution is due to the unstable nature of the leach and the inability to maintain optimal conditions.

The rate of gold dissolution is understood to be significant in the chloride-hypochlorite leach, with some authors suggesting rates almost two orders of magnitude higher than that of cyanide (Tran, 1992; Jeffrey, 2001; Jeffrey et al., 2001). The rapid kinetics associated with this leach would seem to suggest that the limited results incurred in the VCR leach are not due to insufficient time.

The gold chloride compound has been described as unstable and is likely to re-precipitate on contact with sulphides or other reducing agents (Stace, 1984). However, the gold-chloride compounds generated in the VCR leach were found in solution more than a week after leaching this sulphide-bearing ore. The sulphides in the sample had been consumed in the leach and this may account for the increased stability of gold complex. Some research has been done to reduce the reactivity of sulphides for the leaching process (Stace, 1984), however this is not a feasible consideration if the gold is hosted within, or on the surface of, sulphides such as in arsenopyrite in the Fairview and Barbrook samples.

Due to the high mineral reactivity found in the chloride-hypochlorite leaching system, this method has been suggested for completely oxidised ores. However, carbonates can still be found in oxidised ores and prohibitively high consumptions could therefore still be incurred due to the acidic pH required for leaching. In contrast to Aylmore (2016), Gos and Rubo (2001) have suggested that chloride-hypochlorite leaching is applicable to all ore types, and this may refer to the ability of this method to consume sulphides and therefore access refractory as well as exposed gold. While this system may be able to leach gold associated with all mineralogies given sufficient reagent concentration and control, this research has found that many mineralogies are extremely sensitive to this leaching process. The outcomes of this work would therefore suggest that the chloride-hypochlorite system is only applicable to oxidised, carbonate-free ore, or as has been suggested by Adams (2016), for concentrates.

12.3. Thiourea Leaching

The use of thiourea has been extensively studied by entities including Mintek, Canmet, Newmont and Barrick with a focus on treating refractory gold ores. However the potentially carcinogenic nature of thiourea has been a deterrent for further research with a limited number of exceptions (Zhang et al., 2001; Li and Miller, 2006; Poisot-Díaz et al., 2008).

Throughout the leach the redox conditions required moderate maintenance, however the pH was generally stable for the duration of the leach. Maintaining pH and Eh conditions within the optimal range is of paramount importance to ensure that sustained leaching occurs (Kenna and Moritz, 1991). While this only required moderate levels of intervention on a laboratory scale, considerations of control on a larger scale would be relevant to ensuring feasibility.

The thiourea leach was also highly reactive, requiring large amounts of acid, however target leaching conditions were reached and maintained during leaching for both VCR and Pilgrim's Rest samples (the two ores with the least carbonate content). The high reactivity of this leaching system is also evident in the depletion of minerals such as chlorite, muscovite, pyrite, pyrrhotite and calcite in the VCR material and the consumption of calcite, pyrrhotite and pyrite within the Pilgrim's Rest sample. The gold dissolution using thiourea was variable with 40% achieved for VCR and 14% for Pilgrim's Rest samples. At the end of the 8-hour leach there was no thiourea remaining for either sample. Absence of thiourea in solution at the end of an 8-hour leach (in the presence of both reactive and non-reactive or types) confirms the fundamental instability of thiourea in solution, regardless of the gangue mineralogy. Additionally, the absence of gold in solution, despite leaching clearly having occurred (based on residue assay) highlights the instability of the thiourea gold complex. These two factors – reagent degradation and the gold-thiourea complex instability- are the primary causes of ineffective leaching. Subsequent optimisations could have been implemented, some examples of which have been described below, however the high acid consumptions and the generally high degree of reactivity with many gangue components as determined in this study, in addition to the potential health concerns relating to thiourea, acted as a deterrent to further work.

Further optimisation could include the addition of a stabilising agent to improve the inherently unstable nature of thiourea, which tends to decompose rapidly into components incapable of gold dissolution (Tremblay et al., 1996). Examples of stabilising agents that have been implemented include ethylenethiourea, carboxyclic acids, sulphur dioxide, fluorides, and ethylenediaminetetraacetic acid (EDTA) to name a few (Schulze, 1984; Kenna and Moritz, 1991; Sparrow and Woodcock, 1995).

The kinetics of the thiourea leaching system are thought to achieve rates of up to 10 times that of cyanide, given excess conditions in which to operate (Schulze, 1984; Örgül and Atalay, 2002), but despite this, complete dissolution was not achieved. While targeted pH and redox conditions were maintained, the loss of available thiourea throughout the leach means that the rate of dissolution cannot be accurately accessed in this work, as it was unlikely that leaching took place throughout the entire 8-hour period. Additionally, the total consumption of the thiourea affects the ability to compare which ore would have consumed the most

reagent under ideal conditions. Thiourea is thought to be able to complex strongly with base metals (copper, iron, lead and zinc) (Deschênes and Prud'homme, 1997; Aylmore, 2016a) and therefore higher consumptions for the more base metal-bearing Pilgrim's Rest sample would have been expected. There would also have been higher consumption in the Pilgrim's Rest sample due to increased amounts of sulphides presents relative to the VCR sample, which are shown in this study to be highly reactive in thiourea leaching.

The high reagent consumptions associated with carbonate-bearing ores would suggest that this leaching system lends itself to concentrates, where much of the gangue would be removed and subsequently would not be capable of influencing leaching.

The leaching of gold using thiourea requires significant controls, as gold selectivity is limited, while there are several competing reactions incapable of leaching gold. The irreversible degradation of thiourea to components incapable of leaching gold (such as formamidine) also reduces the availability of gold leaching components.

Lapidus et al. (2008) has reported using thiourea in conjunction with bio-oxidised material which was found to significantly improve the leach results, due to increased rates of dissolution and decreased consumptions. The combination of thiourea and biological oxidation would be well suited for further study, as they both require acidic conditions in order to effectively operate. Thus the reagent used to increase the pH as required for cyanidation, or the acid required to lower the pH for thiourea leaching could be avoided (Aylmore, 2016). In the work by Lapidus et al., (2008) dissolution was tested on a large-scale using a pilot plant heap leach with biological oxidation pre-treatment over a period of 110 days, using reagent concentrations similar to that used in this experimental work. However the limited results obtained from this test were thought to be due to a combination of cold weather and coarse gold (Lapidus et al., 2008).

The unfavourable leaching results in conjunction with the concerns regarding the carcinogenic nature of thiourea acted as a deterrent for further work. These health concerns surrounding thiourea, while currently unconfirmed, will definitely reduce the number of researchers, and more importantly processing operations, willing to consider studying or utilising this process. The potentially carcinogenic nature of thiourea, as well as the high concentrations at which it is required for successful leaching, would equate to similar or

higher occupational health and safety considerations than that of cyanide. Despite the low toxicity of this system, appropriate measures would have to be implemented to ensure the long term safety of workers and the community (Gos and Rubo, 2001).

The high degree of reagent consumption, in conjunction with the gangue reactivity observed in this study, which is greater than has been described in previous literature (Deschenes and Ghali, 1988; Van Deventer et al., 1990), acts as a barrier to further development. Additionally the monitoring found in this study to be required for leaching when in the presence of reactive components, highlights the lack of feasibility of this leaching system.

12.4. Thiosulphate Leaching

Gold leaching using thiosulphate has a major environmental advantage due to its low toxicity, in addition it is relevant within industry when applied to preg-robbing ore. However, within the literature there are also some clearly defined disadvantages associated with this leach, those commonly cited include the chemical complexity of thiosulphate leaching and its relative instability particularly in the presence of sulphides. The difficulties around control have been implied by operators of this system at Barrick's Goldstrike operation, at an industrial scale (Braul, 2013). Braul (2013) has highlighted major differences in general processing when compared to cyanide as well as the steep learning curve involved in the implementation of thiosulphate leaching.

Despite these regularly cited disadvantages, thiosulphate was found by the experimental work in this study to show promise, particularly relative to the other non-cyanide lixiviants. The first thiosulphate leaches were able to maintain somewhat stable conditions, which allowed for the completion of leaching for all ores. The degree of input required to maintain conditions throughout leaching was significantly less than these alternatives. Furthermore, these leaches also had significantly lower degrees of consumption of the pH and redox controlling reagents required to achieve the appropriate conditions for thiosulphate leaching. In addition, the minerals consumed during this leach were insignificant, indicating a much more stable environment, and potential scope for improvements.

The oxidative decomposition of thiosulphate in solution is problematic, not only due to the loss of effective reagent but also due to the formation of passivating polythionates (tri- and tetra-tetrathionates). Research evaluating this major shortfall in thiosulphate leaching has

tested the use of several stabilising agents, including phosphates and EDTA (Aylmore, 2016b). However, the increased research attention (Van Deventer et al., 1990) and the low costs associated with glycine-bearing leaches encouraged the evaluation of glycine as a stabilising agent in the experimental work done for this thesis. The use of glycine significantly improved the dissolutions achieved for leaching on all samples, with the exception of the Consolidated Murchison sample. Gold dissolution for Barbrook and Fairview samples increased by more than 55% and 40% respectively, with the addition of a single low-cost reagent. Thiosulphate glycine-bearing leaching of the Barbrook sample achieved higher dissolutions than cyanide leaching with CIL and excess carbon. This increased recovery, despite the presence of preg-robbing components is one of the major advantages for thiosulphate and accounts for its only known use within industry. The dissolution of gold within the free milling ores was improved by the addition of glycine, but to a lesser extent than the refractory ores. The use of calcined material was subsequently implemented to understand the leaching response when the refractory nature of the gold was eliminated. From these results it is evident that thiosulphate consistently achieved ~70% dissolution of the total available gold, with the addition of a stabilising agent. To understand the consistently limited dissolution, the results from the mineralogy and post-leach titrations must be considered. The results from the titrations suggest that there was sufficient (but not excess) reagent in the leach solution for the dissolution to still be active. The mineralogy of the leach shows a number of fine, well liberated gold grains, that were significantly smaller than was present in the feed. The texture of the residual gold grains suggests that these particles have undergone incomplete dissolution. The literature states that the thiosulphate leaching system can, under ideal conditions, achieve similar kinetics to cyanide. However, should conditions be non-ideal, notably slower dissolution will occur (Langhans et al., 1992; Breuer and Jeffrey, 2000; Jeffrey, 2001; Jeffrey et al., 2001; Muir and Aylmore, 2004; Aylmore, 2005). After 24 hours of leaching the pH and redox conditions were within the target range, and thiosulphate was still present in solution. However, the lower thiosulphate concentration in the solution may have significantly reduced the leach kinetics.

Thiosulphate leaching is already a significant improvement over cyanidation for preg-robbing ores, which is evident in the results attained for Barbrook ore, where thiosulphate leaching with glycine successfully leached almost 40% more gold than the CIL cyanide leach. The major

disadvantage for thiosulphate leaching is a consistent 70% dissolution of the total gold available. The limited dissolutions are thought to be due to the reduced kinetics associated with non-optimised conditions, specifically inadequate reagent concentration. While further confirmation is not possible from the data at hand, the novel observation made evident from this study is that the gangue mineralogy is not responsible for incomplete leaching. The deleterious effect of sulphides present in thiosulphate leaching is consistently emphasised throughout literature (Langhans et al., 1992; Abbruzzese et al., 1995; Aylmore and Muir, 2001). However, when implementing thiosulphate leaches on a variety of ores, including material that had either undergone roasting or not, there was no notable decrease in the consumption despite the difference between sulphide-rich and sulphide-absent ores. It should be noted that while there was an improvement in leaching after calcining, this is due to an increase in the total gold exposure rather than to the removal of sulphides from the gangue. The fact that the total gold dissolution in the VCR sample decreased with calcining, is confirmation that the presence of sulphides in the feed have no effect on the thiosulphate leaching capacity. Leach tests involved numerous sulphides including less reactive pyrite and chalcopyrite to highly reactive, antimony sulphides and pyrrhotite, none of which affected the leaching ability of thiosulphate at the conditions used. This further concludes that any shortfalls in the leaching results are a product inherent to the lixiviant and not the lixiviant-gangue interaction. This has interesting implications for the degree to which the thiosulphate leaching systems can be applied to sulphide-bearing ores. This is a conclusion that has not previously been drawn by other researchers and is a novel contribution to the understanding of thiosulphate leaching.

Further experimental work with a metallurgical focus should be done on optimisation to understand if complete dissolution could be incurred, given increased time or additions of thiosulphate with periodic monitoring to ensure ideal conditions are maintained.

While the thiosulphate leaching system is fundamentally applicable to carbonaceous/preg-robbing ores, against which most research has been conducted, there are questions as to the scope of ore types to which this could be applied. While the ability to leach other, less complex ores are approximately equal to that of the preg-robbing ore, thiosulphate is likely to be used exclusively in the context of preg-robbing or sulphidic ores as it is the only circumstance where cyanide is not superior. There is also scope for the use of thiosulphate

leaching in the context of copper-rich ores where sustained gold dissolution is possible, and even aided by the presence of copper. This is in contrast to the difficulties associated with cyanidation of a copper-rich ore, due to the lack of selectivity incurred by cyanide when exposed to copper. Thus, despite this study establishing the advantages of thiosulphate, particularly its lack of reactivity with gangue and its high gold dissolution relative to other alternatives, for the moment it is only better than cyanidation under very specific circumstances.

12.5. General comments

A global movement to more sustainable practises has caused cyanide to be an effective but undesirable tool in gold processing. Alternatives to cyanidation are yet to have advanced sufficiently to be feasible at a global industrial scale, however significant research is making progress in advancing this field. This is most evident in the successful development of thiosulphate for commercial use in heap leaching at the Goldstrike operation of Barrick in Nevada (Aylmore, 2016a) and more recently, CLEAN Mining has acquired a patent for thiosulphate extraction from the CSIRO, after 10 years of intensive study by the Australian parastatal (Heselev, 2019).

In order to ensure that alternative leaching methods are comparable to cyanide, they must fulfil a number of criteria. Some relevant factors include gold dissolution and recovery, applicability (regarding different ore types), recyclability, detoxification, toxicity and environmental effect (Gos and Rubo, 2001). For the alternatives, many of these factors fall short of the standard that is cyanide.

The non-cyanide lixiviants, particularly thiosulphate, have scope for effective leaching in conjunction with well characterised, suitable ores. However due to the complexities of leaching, and the limited dissolution achieved even for the most effective of the alternative lixiviants, they do not, except in the context of specific (pre-robbing and copper bearing ores) surpass the results obtained by cyanide. Thus, without additional metallurgical optimisations are not applicable as a “one size fits all” that is cyanidation. However, that is not the main focus of this study but rather to allow a greater understanding of the alternative lixiviants suitability and interaction with common gangue mineralogy associated with gold mineralisation. The results of the cyanide leaching are entirely as expected based on

literature. The observations made in this study add no contribution to the understanding of cyanide, which is unsurprising due to the extent to which cyanide has been studied. It does however confirm the method and procedures undertaken in this study were appropriate. While cyanidation results were entirely in line with the literature, it is clear that this study has certainly contributed to the understanding of alternative lixiviants in the context of gangue interaction (Table 12-1). While much of the literature alludes to the deleterious effect of sulphides in the chloride-hypochlorite leach, no mention is made of reactivity to phyllosilicates and even carbonates. While it seems intuitive that a low pH leach is not appropriate for a carbonate ore, the inability of a leaching method to function in the presence of carbonates (a common mineral found in gold-bearing ores) is a major disadvantage that should be far more heavily emphasised. In addition, reactivity with phyllosilicates, a typically non-reactive mineral group, confirms that that leaching method is far too reactive to be a feasible alternative. This statement is similarly appropriate to thiourea which was shown to be reactive with most sulphides in addition to chlorite, muscovite and both dolomite and calcite (carbonates). In contrast to the chloride-hypochlorite and thiourea leaches which were found to be far more reactive than predicted by literature, the thiosulphate leach was found to be far less reactive than predicted by literature. While many researchers have suggested sulphides would be deleterious to the leach solution, a consistent lack of reactivity with any gangue mineral was evident in a varieties of thiosulphate leaching performed.

Table 12-1 A summary of the suspected deleterious phases present based on the literature in contrast to that found to be deleterious in this study.

Lixiviant	Deleterious Phases Based on Literature	Deleterious Phases Identified in this Study
Cyanide	Pyrrhotite	Pyrrhotite
	Arsenopyrite	Arsenopyrite
	Antimony Sulphides	Antimony Sulphides
	Carbon	Carbon
Chloride-Hypochlorite	Pyrrhotite	Pyrrhotite
	Arsenopyrite	Pyrite
	Antimony Sulphides	Calcite
	Pyrite	Chlorite
	Chalcopyrite	
Thiourea	Pyrrhotite	Pyrrhotite
	Arsenopyrite	Pyrite
	Antimony Sulphides	Chalcopyrite
	Pyrite	Chlorite
	Chalcopyrite	Muscovite
		Carbonates
Thiosulphate	Pyrrhotite	None observed
	Arsenopyrite	
	Antimony Sulphides	

The reactivity of these ores is particularly relevant to the understanding of feasibility and the scope of optimisation for these leaching systems as the reactivity will affect two important factors; 1) reagent consumption, 2) purity of the leach solution. The greater amounts of reagent that are capable of reacting with gangue, the less efficient the use of the reagent. This means not only higher costs in terms of reagent purchasing, but also for neutralising the reagent once leaching has been completed. The purity of the leach solution dictates the simplicity and costs associated with the recovery of gold from the leach solution.

Chloride-hypochlorite and thiourea leaching, which operates within the acidic range, tend to be very reactive, even with minerals that are not commonly considered susceptible to attack. Acidic media leaching is also likely to be capable of mobilising the metal components within the ore and will therefore have less selectivity for gold. The use of a reagent to complex with non-gold species is a misuse of reagent and also has subsequent complications for recovery, as other metals present in solution are competing with, or diluting the concentration of gold.

Acidic leaches seem to be limited in their relevance to most ores and further testing on concentrates or pre-oxidised ores would be the most relevant use for these leaching techniques in the future.

Thiosulphate leaching, while significantly less reactive than the acid based leaching systems and capable of significant gold dissolutions, still has the major disadvantage of high reagent consumptions and slow kinetics. For this reason, it is only in very specific circumstances where the use of thiosulphate currently has the advantage over cyanidation. As such considerations into altered flowsheets, etc. are not yet feasible. However, a relatively fast and cheap mineralogical study would help determine if there was scope for further study into the potential use of thiosulphate. As thiosulphate only outperforms cyanide in the context of preg-robbing carbons and abundant base metal sulphides, assays and XRD to determine the feed mineralogy would be a good initial study to determine if further, extensive metallurgical study maybe be required. While this extensive metallurgical study upfront may have cost implications, in a more environmentally and socially conscious future these costs may be offset.

In general, it is unsurprising that cyanide is currently still the global standard, not only is it necessary to consider its advantageous metallurgical factors, but consideration must also be given to health, environment and sustainability. While cyanide is highly toxic, it is only required in low concentrations and it is rapidly and readily detoxified (Gos and Rubo, 2001). Alternative reagents are required in high concentrations, therefore the amount of reagent required is less sustainable and the detoxification processes are not as well understood. The health implications of these alternatives are not entirely well-known as is evident in the somewhat recent classification of thiourea as a potential carcinogen.

While the interactions between lixiviants and gangue have been established in this research, there are several research avenues, particularly with a more metallurgical focus, that could be tested in the future to optimise the results of leaching with alternative lixiviants. One area of lixiviant gangue interaction that was unfortunately not studied in this work is oxidised gold deposits. Testing on this ore type was not possible due to lack of access. Further study focused on thiosulphate leaching would likely be the most productive. The use of stabilisers for thiosulphate leaching was found to be successful and further research into optimising conditions may be extremely advantageous. While the addition of glycine had no effect on

the lack of reactivity with the gangue during thiosulphate leaching, this may not hold true for other stabilisers. A recommendation for future optimisation would involve adjustments made to leaching conditions, particularly the concentrations of reagents, by reducing initial concentrations but maintaining these values throughout the leach. Research focusing on reducing consumptions while maintaining dissolution throughout the leach should be made the highest priority.

Chapter 13

Conclusions

There exists substantial environmental concern surrounding the use of cyanide in the gold industry. Whether the extent of concern surrounding cyanide is overestimated or real, there is certainly increasing legislation regarding the use of this reagent. Possibly of more relevance to the minerals industry is the decreasing number of gold deposits that are amenable to cyanidation. In order to address the increasing need for alternatives to cyanide, extensive research into substitute methods are being explored globally. Unfortunately, much of this work occurs either in industry where the information is closely guarded or in the metallurgical sector where the effort to study and correlate the mineralogy with gold dissolution is limited.

In order to properly evaluate the interaction between alternative leaching methods and gangue mineralogy, a variety of ore types were collected and characterised. The VCR material was selected as an example of high-grade free milling ore, where processing tends to be easy and compromising minerals (such as pyrrhotite) are only present in low quantities. The gangue was predominantly quartz, with minor amounts of chlorite, sericite and carbonates present in the sample. Sulphides included pyrite, pyrrhotite and chalcopyrite, which were associated with gold grades of approximately 23 g/ton. The Pilgrim's Rest sample represented a free milling ore with high gold grades (29 g/ton), in addition to notable concentrations of reactive components such as significant pyrite, lesser pyrrhotite and chalcocite. Fairview and Barbrook both represented refractory deposits, with solid solution gold in arsenopyrite at grades of 12 g/ton and 3 g/ton respectively. However, both ores are uniquely important to this study as Fairview represents a single refractory gold deposit, while the presence of preg-robbing carbon within Barbrook makes this ore double refractory, which is significantly more difficult to process. Consolidated Murchison represented moderate gold grades (2 g/ton) present as both native gold and aurostibite. The gold from the Consolidated Murchison sample had associations with potentially reactive and passivating minerals such as stibnite, berthierite as well as arsenopyrite. The well characterised samples were subsequently exposed to several leaching methods with varying results.

The chloride-hypochlorite leaches had consistently poor results (Table 13-1), with only one leach run to completion; the VCR sample, was the least reactive of all the samples. Both the complete and incomplete leaches exhibited high acid consumptions, high mineral reactivity and poor dissolutions. This study found that in addition to the expected reactivity to mineral phases such as sulphides, reaction with carbonates and phyllosilicates were also extensive. In addition, the conditions of the chloride-hypochlorite leach were extremely difficult to maintain, and constant monitoring was required.

The thiourea leach was slightly more promising than chloride-hypochlorite as the acid consumption, while still very high, was better than that of the chloride-hypochlorite leaches. The mineral reactivity was significant and dissolutions rather poor. The poor dissolutions were likely due to the instability of the thiourea in solution and re-precipitation of the gold. The combination of the high acid consumption and the concerns surrounding the carcinogenic effect of thiourea acted as a deterrent for further study.

Thiosulphate leaching had fundamentally better results than both thiourea and chloride-hypochlorite, with mineral reactivity during leaching almost non-existent. Furthermore, no major reagent additions were required in order to maintain the targeted Eh and pH conditions. While initial gold dissolution results for all samples were still poor, there was scope for improvement. The introduction of a stabilising agent significantly improved gold dissolution. In the instance of the preg-robbing Barbrook sample, a greater degree of gold dissolution was achieved than was possible using cyanidation (with excess carbon-in-leach). The incorporation of roasting to remove the sulphides and expose refractory gold showed a significant improvement in dissolution. However, in spite of these optimisations consistently only ~70% of the total available gold was successfully leached. For all leaches involving thiosulphate there was reagent present in solution after the 24 hour leach, however concentrations were reduced and therefore kinetics are likely responsible for these limited results. In order to confirm the cause of limited dissolution, further research would be required, either by exploring increased leaching time or reagent additions at certain intervals to maintain kinetics.

Table 13-1 A brief summary of the results obtained in the study. The primary focus of this study, the deleterious phases identified have been subdivided into those well known in literature (grey) and those determined by this study (blue).

	Consumptions				Deleterious phases*	Dissolution (Min- Max)
Lixiviant	Reagent	kg/ton	Reagent	kg/ton	-	%
Cyanide	Cyanide	1.2 (VCR) - 2.4 (Pilgrim's Rest)	Lime	1 (VCR) - 18 (Pilgrm's Rest)	Preg robbing carbon Pyrrhotite Arsenopyrite Antimony Sulphides	21% (Consolidated Murchison) - 96% (VCR)
Chloride-Hypochlorite	N/A		HCl	2 (VCR) - 460 (Barbrook)	Pyrrhotite Pyrite Calcite Chlorite	40% (VCR)
Thiourea	Thiourea	10 (VCR & Pilgrim's Rest)	H_2SO_4	4 (VCR) - 620 (Barbrook)	Pyrrhotite Pyrite Chalcopyrite Chlorite Muscovite Calcite Dolomite	15% (Pilgrim's Rest) - 30% (VCR)
Thiosulphate	Thiosulphate	45 (Fairview) - 65 (VCR)	N/A		None observed	43% (Fairview) - 60% (VCR)‡

*Deleterious mineral phases, expected based on literature
*Deleterious mineral phases, established in this study

The reactivity of leaching systems is particularly relevant to understand which ores these leaching mechanisms may be appropriate for. In addition to the degree of dissolution, the reactivity is a very relevant consideration for the leach feasibility. Solutions with high degrees of reactivity will consume greater degrees of leaching reagent and reduce solution purity, complicating subsequent recovery. While literature has outlined the disadvantages of the thiourea and chloride-hypochlorite leaches, the degree of reactivity with gangue mineral phases has not previously been defined to the extent observed in this study. The reactivity with not only the least reactive sulphides but even typically non-reactive phases such as

phyllosilicates highlights this reactive nature. Conversely thiosulphate was found to be consistently non-reactive with even the most reactive phases, a concept not previously described in research. The degree to which the deleterious phases identified in cyanidation correlate to the phases identified in the literature emphasises the extent to which cyanide has been studied and understood. This stands in stark contrast to the results of the alternative lixiviants.

Based on the deleterious phases established in this study, including those in line with published literature and those that are novel to this study, a brief summary of the ore types applicable to the lixiviants of this study is possible (Table 13-2). Cyanide leaching is moderately reactive and applicable to most ore types (except carbon bearing, abundant base metal sulphides). Thiourea and chloride-hypochlorite are both highly reactive with extremely limited applicability and only potential for concentrates. Thiosulphate is non-reactive and applicable to all ores. However, these definitions are in the context of gangue mineralogy and reactivity and do not apply to gold dissolution. While thiosulphate is consistently less reactive than cyanide, consideration will have to be given to the degree of gold dissolution that is possible relative to cyanide.

Table 13-2 A brief summary of the lixiviants and their relative degrees of reactivity

Lixiviant	**Degree of Reactivity**	**Comments on Relevant Ore Types**
Cyanide	Moderately reactive	Applicable to most ore types (except carbon bearing or abundant base metal sulphides)
Thiourea	Highly reactive	Extremely limited scope, potential only for concentrates
Chloride - Hypochlorite	Highly reactive	Extremely limited scope, potential only for concentrates
Thiosulphate	Non-reactive	Applicable to all ores*

** while these definitions are in the context of mineralogy, they do not hold true to gold dissolution. Thiosulphate is consistently less reactive than cyanide, consideration will have to be given to the degree of gold dissolution that is possible relative to cyanide*

This study has confirmed that the scope for thiourea and chloride-hypochlorite leaching are extremely limited. However, thiosulphate has been shown to be even more feasible than previously suggested by the literature. While for the moment, cyanide remains the standard for gold processing, this study has established ore components that indicate instances where thiosulphate may have better leaching results than thiosulphate. Thus, instead of completing

extensive metallurgical testing, a brief mineralogical/chemical characterisation would act as an early indication of possible feasibility of thiosulphate leaching. Thiosulphate would have the potential to be a superior recovery method in the presence of preg-robbing carbon or in conditions where abundant reactive sulphides (for example pyrrhotite, antimony sulphides and arsenopyrite) or copper-bearing minerals are present. This knowledge could be determined by relatively cheap assays or XRD analyses to characterise ore types.

While significant research of alternative lixiviants have been found in the literature, the comparative, mineralogical nature of this study has never before been addressed in any public space. The ability to compare the results of multiple alternative lixiviants (at consistent conditions) on a vast range of well characterised ores was the key objective set in the design of this study and the main contribution in terms of results. While there are a number of further questions raised in this research, the project results highlight the importance of increasingly mineralogically-focused studies of this nature to further contribute to the current body of knowledge.

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Appendix A - Assay Data

All assay results for the feed material have been included in the spreadsheet below, this data includes repeats, standards and blanks used to determine the precision and accuracy of the data, which was generally found to be high. The standards and blanks were consistently within 10% of the expected values. These assays were performed by SGS analytical division on sub-samples that had undergone crushing, milling and pulverising, as well as several stages of splitting throughout comminution, to ensure representative results. The assay data are discussed in Chapter 5.3.

Assay Methods

Various methods were used for assay, the abbreviations of which can be found in the spreadsheet for the respective elements, however their full descriptions have been outlined below.

FAA303- refers to gold determination using collection by Pb fusion fire assay, followed AAS (atomic absorption spectroscopy).

CSA06V- determines the total sulphur present using Leco Combustion Infrared Detection

CSA08V- determines the total amount of sulphur present as sulphides, using Leco Combustion

CSA03V- determines the total amount of carbon present using Leco Combustion Infrared Detection

CSA03V- determines the total amount of carbon present as organic carbon Leco Combustion Infrared Detection

XRF79V- major element oxide determination using borate fusion and XRF

AAS11C- arsenic determination using aqua regia digestion followed by AAS

ICP90A- minor element determination using sodium peroxide fusion, followed with inductively coupled plasma atomic emission spectrometry (ICP-AES) or inductively coupled plasma mass spectrometry (ICP-MS)

Assay Results for Feed Samples for VCR and Fairview (FV) as well as standards, blanks and repeats to ensure high quality data

WITS - 001	WtRec	Au	S	S2-_S	C	Al2O3	SiO2	Fe2O3	CaO	MgO	MnO	K2O	Na2O	TiO2	P2O5	V2O5	Cr2O3	LOI	As	Cu	Co	Ni	Zn	Pb
METHOD	WGH79	FAA303	CSA06V	CSA08V	CSA01V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	AAS11C	ICP90A	ICP90A	ICP90A	ICP90A	ICP90A
LDETECTION	0.01	0.02	0.01	0.01	0.01	0.05	0.05	0.01	0.01	0.05	0.01	0.01	0.05	0.01	0.01	0.01	0.01	-50	0.01	10	10	10	10	20
UDETECTION	0	100	100	30	73.8	100	100	100	100	100	100	100	100	100	100	100	100	100	5	100000	100000	100000	100000	100000
UNITS	G	G/T	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%	PPM	PPM	PPM	PPM	PPM
VCRA	106	22.2	1.19	1.09	0.31	3.52	87.9	4.59	0.82	1.5	0.04	0.73	0.1	0.2	0.033	<0.01	0.11	1.75	0.09	139	43	228	75	193
VCRB	104.2	25.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
VCRC	111.4	23	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
FVA	95.4	12.4	2.86	2.76	1.27	14	53	10.5	1.01	6.59	0.22	3.72	0.3	0.7	0.096	0.03	0.18	7.91	0.81	84	50	491	124	<20
FVB	87.6	12.5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
FVC	87.6	12.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*STD-MP-1B	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	2.32	-	-	-	-	-
*BLK-BLANK	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	<0.01	-	-	-	-	-
*REP-VCRA	-	-	-	-	0.31	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*BLK-BLANK	-	-	-	<0.01	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*REP-VCRA	-	20.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*STD-OREAS 224	-	2.13	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*BLK-BLANK	-	<0.02	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*REP-VCRC	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*STD-GBM398-4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	4140	2120	4040	5640	12400
*BLK-BLANK	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	<10	<10	22	<10	<20
*REP-VCRA	-	-	-	-	-	3.52	87.7	4.62	0.81	1.49	0.05	0.73	0.09	0.19	0.037	<0.01	0.1	1.75	-	-	-	-	-	-
*STD-SARM 5	-	-	-	-	-	4.19	50.6	12.7	2.62	25.2	0.23	0.1	0.37	0.2	0.019	0.04	3.51	-0.23	-	-	-	-	-	-
*BLK-BLANK	-	-	-	-	-	<0.05	<0.05	<0.01	<0.01	<0.05	<0.01	<0.01	<0.05	<0.01	<0.01	<0.01	<0.01	100	-	-	-	-	-	-
*REP-VCRA	-	-	-	-	0.31	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*STD-STD-1%	-	-	-	1.09	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*REP-VCRA	-	-	-	1.09	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*REP-VCRA	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*REP-VCRA	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.08	-	-	-	-	-
*BLK-BLANK	-	-	<0.01	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*REP-VCRA	-	-	1.18	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*REP-VCRA	-	-	1.27	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*BLK-BLANK	-	-	-	-	<0.01	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*STD-GGC-08	-	-	1.51	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*STD-GGC-08	-	-	-	-	1.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*REP-VCRA	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	143	41	218	74	176
*STD-PTM-1A	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.23	-	-	-	-	-

Assay Results for Feed Samples for Barbrook (BK), Consolidated Murchison (CM) and Pilgrim's Rest (PG) as well as standards, blanks and repeats to ensure high quality data

WITS-003	WtRec	Au	S	C	ORG C	Al2O3	SiO2	CaO	Cr2O3	Fe2O3	K2O	MgO	MnO	Na2O	TiO2	P2O5	V2O5	LOI	Total	As	Cu	Fe	Ni	Sb	Pb	Zn
METHOD	WGH79	FAA303	CSA06V	CSA01V	CSA03V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	XRF79V	AAS11C	ICP90A	ICP90A	ICP90A	ICP90A	ICP90A	ICP90A
LDETECTION	0.01	0.02	0.01	0.01	0.05	0.05	0.05	0.01	0.01	0.01	0.01	0.05	0.01	0.05	0.01	0.01	0.01	-50	0.01	0.01	10	0.01	10	50	20	10
UDETECTION	0	100	100	73.8	40	100	100	100	100	100	100	100	100	100	100	100	100	100	120	5	100000	30	100000	100000	100000	100000
UNITS	G	G/T	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%	PPM	%	PPM	PPM	PPM	PPM
BK	91.4	1.67	8.06	3.94	0.23	5.94	40	6.87	0.11	21.8	0.77	3.62	0.44	0.2	0.21	0.071	0.01	12.25	92.31	0.23	597	15.3	472	65	579	2330
CM	68.4	3.12	6.29	3.55	0.13	8.58	32.7	2.12	0.21	13.7	0.35	9.44	0.19	0.63	0.32	0.053	0.02	17.54	85.79	2.96	164	10.2	700	70100	87	98
PG	86.8	30.3	29.6	0.7	0.14	0.56	35.7	1.3	0.09	36.5	0.18	0.82	0.05	<0.05	0.03	<0.01	<0.01	18.7	93.9	0.35	7230	25.5	84	931	108	47
*REP-BK	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.23	-	-	-	-	-	-
*STD-PTM-1A	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.22	-	-	-	-	-	-
*BLK-BLANK	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	<0.01	-	-	-	-	-	-
*REP-BK	-	-	-	3.97	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*BLK-BLANK	-	-	-	<0.01	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*REP-BK	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*STD-SARM 33	-	-	-	-	0.19	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*BLK-BLANK	-	-	-	-	<0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*REP-BK	-	-	8.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*BLK-BLANK	-	-	<0.01	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*REP-BK	-	1.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*STD-AMIS0441	-	2.17	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*BLK-BLANK	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*REP-BK	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	600	15.4	543	82	669	2400
*STD-GBM398-4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	3760	5.11	3950	-	12100	4960
*BLK-BLANK	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	<10	0.01	<10	<50	<20	<10
*REP-BK	-	-	-	3.99	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*REP-BK	-	-	-	-	0.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*STD-SARM 33	-	-	-	-	0.16	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*REP-BK	-	-	-	-	-	5.99	40.4	6.91	0.11	22	0.77	3.63	0.44	0.2	0.21	0.074	0.01	12.28	92.94	-	-	-	-	-	-	-
*STD-SARM 5	-	-	-	-	-	4.12	50.4	2.59	3.56	12.6	0.09	25.4	0.22	0.33	0.19	0.018	0.05	-0.3	99.23	-	-	-	-	-	-	-
*BLK-BLANK	-	-	-	-	-	<0.05	<0.05	<0.01	0.04	<0.01	<0.01	<0.05	<0.01	<0.05	<0.01	<0.01	<0.01	100	99.89	-	-	-	-	-	-	-
*STD-OREAS 906	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	2950	5.75	-	<50	47	156
*STD-SARM 33	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*STD-SARM 33	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*STD-GGC-08	-	-	-	1.01	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*STD-GGC-09	-	-	-	2.91	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*REP-BK	-	-	8.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*STD-GS903-4	-	-	11.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
*STD-GS309-7	-	-	29.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Appendix B – Electron Microscopy Data from Feed

The full data from the gold deportment, as discussed in Chapter 6, for all samples have been included below. The data of relevance includes grain size distribution, liberation, exposure, modal mineralogy etc. MLA analyses were performed by Dr Brad Guy while the EPMA analyses were set up by Dr Christian Reinke and performed by the PhD candidate, both of these analyses were performed at Spectrum (The Central Analytical Facility of the Faculty of Science, University of Johannesburg). The EMPA work was undertaken using a CAMECA SX100, while the MLA is a FEI XL40 ESEM with two EDAX Sapphire Si(Li) EDS detectors. The MLA identification was performed using spectra matching, as is typical for this autoSEM. The electron microscopy was performed on gravity concentrated feed in the form of polished section, due to the highly concentrated nature of the samples, it was generally only necessary to analyse two blocks per sample to ensure statistically significant results (greater than 50 gold grains per sample).

MLA Grain Size Distribution

The grain size distribution of the gold within respective samples. The grain size distribution is an important factor for consideration in processing as it will affect the milling size and leach times required for extraction.

VCR: Total Gold			
Sieve Size	Retained Wt%	Cum. Retained Wt%	Cum. Passing Wt%
0.73	0.04	99.97	0.03
0.87	0.01	99.97	0.03
1	0.02	99.95	0.05
1.2	0.04	99.93	0.07
1.45	0.04	99.89	0.11
1.75	0.05	99.85	0.15
2	0.09	99.80	0.20
2.4	0.10	99.71	0.29
2.9	0.08	99.61	0.39
3.4	0.12	99.54	0.46
4.1	0.13	99.41	0.59
4.8	0.14	99.28	0.72
5.7	0.22	99.14	0.86
6.8	0.22	98.92	1.08
8.1	0.32	98.69	1.31
9.6	0.32	98.37	1.63
11.4	0.48	98.05	1.95
13.5	0.42	97.57	2.43
16	0.82	97.15	2.85
19	0.68	96.33	3.67
22	1.72	95.65	4.35
27	1.65	93.93	6.07
32	2.17	92.28	7.72
38	3.29	90.10	9.90
45	2.50	86.81	13.19
53	8.79	84.31	15.69
63	4.51	75.53	24.47
75	3.03	71.02	28.98
90	0.00	67.99	32.01
106	2.01	67.99	32.01
125	3.04	65.98	34.02
150	8.47	62.94	37.06
180	0.00	54.47	45.53
212	8.97	54.47	45.53
250	22.45	45.51	54.49
300	0.00	23.06	76.94
355	23.06	23.06	76.94
425	0.00	0.00	100.00

Fairview: Total Gold			
Sieve Size	Retained Wt%	Cum. Retained Wt%	Cum. Passing Wt%
0.73	0.04	99.96	0.04
0.87	0.09	99.87	0.13
1	0.02	99.85	0.15
1.2	0.45	99.40	0.60
1.45	0.37	99.03	0.97
1.75	0.55	98.47	1.53
2	1.31	97.16	2.84
2.4	0.97	96.19	3.81
2.9	1.51	94.68	5.32
3.4	1.86	92.83	7.17
4.1	1.83	91.00	9.00
4.8	2.40	88.60	11.40
5.7	1.46	87.14	12.86
6.8	3.34	83.80	16.20
8.1	3.69	80.10	19.90
9.6	3.09	77.01	22.99
11.4	2.78	74.23	25.77
13.5	3.58	70.65	29.35
16	8.34	62.31	37.69
19	0.08	62.24	37.76
22	0.44	61.80	38.20
27	0.16	61.65	38.35
32	0.00	61.65	38.35
38	0.66	60.99	39.01
45	13.36	47.63	52.37
53	0.61	47.02	52.98
63	0.00	47.02	52.98
75	0.00	47.02	52.98
90	0.00	47.02	52.98
106	0.00	47.02	52.98
125	0.00	47.02	52.98
150	0.00	47.02	52.98
180	5.88	41.15	58.85
212	0.00	41.15	58.85
250	0.00	41.15	58.85
300	0.00	41.15	58.85
355	0.00	41.15	58.85
425	41.15	0.00	100.00

Barbrook: Native Gold			
Sieve Size	Retained Wt%	Cum. Retained Wt%	Cum. Passing Wt%
0.73	0.00	100.00	0.00
0.87	0.01	100.00	0.00
1	0.02	99.99	0.01
1.2	0.17	99.97	0.03
1.45	0.07	99.80	0.20
1.75	0.04	99.73	0.27
2	0.23	99.70	0.30
2.4	0.00	99.47	0.53
2.9	0.17	99.47	0.53
3.4	0.59	99.30	0.70
4.1	1.49	98.71	1.29
4.8	0.81	97.22	2.78
5.7	2.68	96.41	3.59
6.8	5.28	93.72	6.28
8.1	4.94	88.44	11.56
9.6	1.64	83.50	16.50
11.4	10.57	81.86	18.14
13.5	5.97	71.29	28.71
16	10.62	65.32	34.68
19	6.77	54.71	45.29
22	15.63	47.94	52.06
27	0.00	32.31	67.69
32	0.00	32.31	67.69
38	0.00	32.31	67.69
45	0.00	32.31	67.69
53	32.31	32.31	67.69
63	0.00	0.00	100.00

Cons Murch: Native Gold			
Sieve Size	Retained Wt%	Retained Wt%	Cum. Passing Wt%
1	0.00	100.00	0.00
1.2	0.75	100.00	0.00
1.45	0.00	99.25	0.75
1.75	1.43	99.25	0.75
2	1.31	97.82	2.18
2.4	0.20	96.50	3.50
2.9	0.00	96.31	3.69
3.4	15.59	96.31	3.69
4.1	4.39	80.71	19.29
4.8	3.01	76.33	23.67
5.7	4.48	73.31	26.69
6.8	7.09	68.84	31.16
8.1	26.45	61.75	38.25
9.6	19.09	35.30	64.70
11.4	0.00	16.22	83.78
13.5	0.00	16.22	83.78
16.0	3.80	16.22	83.78
19.0	0.00	12.42	87.58
22.0	0.00	12.42	87.58
27.0	12.42	12.42	87.58
32.0	0.00	0.00	100.00

Cons Murch: Aurostibite			
Sieve Size	**Retained Wt%**	**Retained Wt%**	**Cum. Passing Wt%**
0.73	0.00	100.00	0.00
0.87	0.00	100.00	0.00
1	0.00	100.00	0.00
1.2	0.00	100.00	0.00
1.45	0.00	100.00	0.00
1.75	0.00	100.00	0.00
2	0.02	100.00	0.00
2.4	0.27	99.98	0.02
2.9	0.00	99.70	0.30
3.4	0.19	99.70	0.30
4.1	0.96	99.51	0.49
4.8	1.16	98.55	1.45
5.7	2.51	97.38	2.62
6.8	3.63	94.87	5.13
8.1	2.13	91.25	8.75
9.6	4.83	89.12	10.88
11.4	6.34	84.28	15.72
13.5	4.05	77.95	22.05
16.0	13.76	73.90	26.10
19.0	11.25	60.14	39.86
22.0	7.56	48.88	51.12
27.0	9.86	41.32	58.68
32.0	6.48	31.46	68.54
38	8.05	24.98	75.02
45	0.00	16.93	83.07
53	16.93	16.93	83.07
63	0.00	0.00	100.00

Cons Murch: Total Gold			
Sieve Size	Retained Wt%	Retained Wt%	Cum. Passing Wt%
0.73	0.00	100.00	0.00
0.87	0.00	100.00	0.00
1	0.00	100.00	0.00
1.2	0.01	100.00	0.00
1.45	0.00	99.99	0.01
1.75	0.11	99.99	0.01
2	0.32	99.87	0.13
2.4	0.33	99.55	0.45
2.9	0.00	99.23	0.77
3.4	1.25	99.23	0.77
4.1	1.68	97.98	2.02
4.8	1.58	96.30	3.70
5.7	2.98	94.72	5.28
6.8	4.44	91.74	8.26
8.1	1.81	87.30	12.70
9.6	9.37	85.48	14.52
11.4	4.90	76.11	23.89
13.5	3.13	71.21	28.79
16	12.90	68.08	31.92
19	8.93	55.18	44.82
22	3.96	46.25	53.75
27.0	11.36	42.29	57.71
32.0	6.37	30.92	69.08
38	7.91	24.55	75.45
45	0.00	16.63	83.37
53	16.63	16.63	83.37
63	0.00	0.00	100.00

PG I: Native Gold			
Sieve Size	Retained Wt%	Cum. Retained Wt%	Cum. Passing Wt%
0.73	0.00	100.00	0.00
0.87	0.00	100.00	0.00
1	0.00	100.00	0.00
1.2	0.02	100.00	0.00
1.45	0.02	99.98	0.02
1.75	0.03	99.95	0.05
2	0.08	99.93	0.07
2.4	0.03	99.85	0.15
2.9	0.19	99.82	0.18
3.4	0.23	99.63	0.37
4.1	0.33	99.40	0.60
4.8	0.27	99.08	0.92
5.7	0.56	98.81	1.19
6.8	1.53	98.25	1.75
8.1	1.58	96.72	3.28
9.6	3.74	95.15	4.85
11.4	2.81	91.40	8.60
13.5	8.71	88.59	11.41
16	9.20	79.88	20.12
19	11.25	70.68	29.32
22	10.99	59.44	40.56
27	8.16	48.45	51.55
32	5.21	40.29	59.71
38	14.50	35.08	64.92
45	8.97	20.58	79.42
53	11.61	11.61	88.39
63	0.00	0.00	100.00

MLA Mineral Association

Full mineral association from MLA generated data.

Minerals that are associated with the gold particles from respective deposits. Mineral associations will significantly affect the response to certain processing methods, for example the response to flotation, gravity separation and leaching.

VCR												
Mineral	Quartz	Pyrite	Other Sulphides	Other Silicate	Chlorite	Pyrrhotite	Other	Mica	Brannerite	Coffinite	Carbonates	Free Surface
Native gold	6.44	1.39	2.30	4.25	1.60	4.32	1.25	0.39	0.17	0.57	1.99	75.34

Fairview										
Mineral	Quartz	Pyrite	Other Sulphides	Arsenopyrite	Chlorite	Pyrrhotite	Fe(Mn)-oxide	Other	Mica	Free Surface
Native gold	1.07	54.59	0.41	15.68	1.54	1.44	0.23	1.84	2.49	20.71

Barbrook								
Mineral	Chlorite	Pyrite	Other Sulphides	Arsenopyrite	Mica	Ankerite	Siderite	Free Surface
Native gold	1.53	4.11	0.40	3.06	0.26	1.97	0.76	87.91

Cons Murch						
Mineral	Arsenopyrite	Pyrite	Other Sulphides	Stibnite(Fe)	Magnesite	Free Surface
Native gold	23.32	2.67	1.32	2.39	5.86	64.44
Aurostibite	0.81	0.00	0.00	0.00	0.00	99.19
Gold bearing	12.07	1.33	0.66	1.19	2.93	81.82

Pilgrim's Rest							
Mineral	Quartz	Pyrite	Other Sulphides	Arsenopyrite	Biotite	Other	Free Surface
Native gold	0.35	7.71	0.98	0.02	0.05	0.68	90.20

MLA Mineral Liberation

Liberation is defined by the area % of the mineral of interest relative to the area of the total host particle. Mineral liberation is an important characteristic to consider when ensuring effective processing procedures. Mineral liberation will control the ability of the ore to undergo successful flotation and gravity separation.

	Locked						Middlings										Liberated				
Distribution of Mineral (%)	**0-5%**	**5-10%**	**10-15%**	**15-20%**	**20-25%**	**25-30%**	**30-35%**	**35-40%**	**40-45%**	**45-50%**	**50-55%**	**55-60%**	**60-65%**	**65-70%**	**70-75%**	**75-80%**	**80-85%**	**85-90%**	**90-95%**	**95-100%**	**100%**
VCR	0.05	0.18	0.02	0.02	0.06	0.01	0.00	0.00	0.51	0.06	0.04	0.03	0.00	0.02	0.40	0.11	0.17	2.21	1.58	65.34	29.19
FV	10.37	3.05	2.85	0.44	0.21	7.99	0.20	0.00	1.43	0.62	0.04	0.00	0.00	0.47	0.00	0.34	0.01	0.00	0.00	64.53	7.46
Barbrook	1.41	0.00	0.20	0.48	0.00	0.03	0.46	0.00	0.00	0.30	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	7.99	3.40	85.73
CM Native gold	6.07	6.26	0.35	0.00	0.00	0.00	0.00	0.00	1.11	2.50	0.00	0.00	0.00	0.00	0.00	28.52	0.00	0.00	20.70	0.00	34.48
CM Aurostibite	0.14	0.00	0.00	0.00	0.00	0.00	0.00	0.04	0.00	0.88	0.00	0.00	0.00	0.00	0.00	2.28	1.48	0.00	7.30	3.28	84.59
CM Total	0.42	0.13	0.10	0.00	0.00	0.00	0.00	0.04	0.30	0.88	0.00	0.00	0.00	0.00	0.00	2.74	1.46	0.00	7.30	3.23	83.41
PG	0.32	0.44	0.16	0.16	0.18	0.00	0.04	0.00	0.04	0.81	0.00	0.00	0.00	0.00	0.00	0.58	0.00	0.17	0.00	33.68	63.41

MLA Mineral Exposure

Mineral exposure can be described as the percentage of the grain that has a free, exposed, surface. The portion of the grain with an exposed surface is available for interaction with reagents in leaches or flotation.

			VCR																				
Native gold Free Surface of Particle (%)	0% (barren)	0% (not exposed)	0-5%	5-10%	10-15%	15-20%	20-25%	25-30%	30-35%	35-40%	40-45%	45-50%	50-55%	55-60%	60-65%	65-70%	70-75%	75-80%	80-85%	85-90%	90-95%	95-100%	100%
No. of Particles	4312.00	21.00	13.00	9.00	6.00	2.00	5.00	1.00	2.00	1.00	1.00	5.00	2.00	0.00	4.00	3.00	4.00	2.00	4.00	5.00	10.00	13.00	153.00
Mean Phase ECD (µm)	0.00	2.80	2.44	4.69	5.11	6.78	4.90	6.48	14.82	3.59	1.56	9.95	17.18	0.00	18.25	26.21	30.04	34.92	45.53	29.61	38.21	43.53	17.57
Mean Phase Max Span (µm)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Particle Distribution (%)	82.58	1.07	0.55	0.21	0.06	0.01	0.22	0.01	0.21	0.02	0.00	0.08	1.59	0.00	0.31	0.36	3.68	0.24	1.49	0.32	0.78	1.76	4.44
Mean Density	4.74	4.52	4.70	6.06	5.54	6.05	8.13	7.03	11.37	8.30	9.23	11.19	15.60	0.00	14.95	15.87	17.66	17.97	18.77	17.52	19.04	19.16	19.28
Distribution of Mineral (%)	0.00	0.15	0.03	0.15	0.11	0.03	0.70	0.02	1.22	0.11	0.00	0.45	10.04	0.00	1.93	2.27	23.61	1.55	9.70	2.03	5.12	11.56	29.22
Cum. Distn. of Mineral (%)	0.00	100.00	99.85	99.82	99.67	99.56	99.53	98.83	98.81	97.59	97.48	97.48	97.03	86.99	86.99	85.05	82.78	59.18	57.63	47.93	45.89	40.78	29.22
Modal Analysis (%)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Quartz (%)	10.51	10.31	11.44	5.01	5.03	0.30	0.00	0.00	10.97	0.00	0.00	3.52	1.60	0.00	0.63	0.41	0.20	0.91	0.10	0.01	0.11	0.00	0.00
Chlorite (%)	1.74	5.32	0.56	1.58	0.00	34.32	0.41	34.72	0.00	0.00	21.67	0.35	0.34	0.00	1.81	0.00	0.01	0.00	0.00	0.00	0.02	0.00	0.00
Other Sulphides (%)	4.10	5.13	0.00	63.48	8.94	2.75	17.14	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.30	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00
Orthoclase (%)	0.03	0.00	0.00	0.92	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.52	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.04	0.00	0.00
Plagioclase (%)	0.07	1.99	0.04	0.14	0.85	6.28	0.00	0.00	0.00	0.00	0.00	0.00	0.23	0.00	0.36	0.00	0.00	0.00	0.00	1.22	0.00	0.02	0.00
Other Silicates (%)	2.05	2.83	0.00	5.09	8.36	0.00	0.31	0.00	0.00	0.03	0.00	0.00	0.02	0.00	1.15	3.85	0.00	0.00	0.01	0.00	0.00	0.00	0.00
Arsenopyrite (%)	0.78	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.11	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Pyrite (%)	27.36	47.72	53.26	2.15	1.27	0.00	0.48	0.00	0.00	0.00	0.00	0.00	0.09	0.00	0.00	0.00	0.27	0.00	0.12	0.00	0.01	0.00	0.00
Pyrrhotite (%)	22.44	9.94	16.33	0.07	47.26	0.00	20.09	0.00	0.00	0.00	0.00	15.16	0.06	0.00	0.05	0.00	1.90	0.04	0.32	0.72	0.07	0.00	0.00
Oxides (%)	18.97	8.75	13.88	8.63	0.00	0.00	3.49	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.05	0.00	0.00	0.00
Fe(Mn)-oxides (%)	6.16	0.12	0.69	0.00	0.00	0.00	0.00	0.00	0.00	17.05	0.00	0.00	0.39	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00
Phosphates (%)	0.12	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.09	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Other (%)	0.45	0.00	0.21	1.46	0.00	0.00	0.00	0.00	0.00	4.77	0.00	0.00	0.00	0.00	0.00	0.00	0.04	0.00	0.00	0.00	0.00	0.00	0.00
Mica (%)	0.43	0.29	0.21	0.06	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.06	0.24	0.03	0.02	0.00	0.05	0.00
Brannerite (%)	0.24	0.18	0.04	0.00	0.03	0.00	0.44	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.15	0.00	0.00	0.00	0.00
Coffinite (%)	4.36	5.31	2.46	0.69	0.82	0.00	9.66	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.06	0.00	0.00	0.00	0.00
Native gold (%)	0.00	2.09	0.84	10.71	27.43	56.35	47.80	65.28	89.03	78.15	78.33	80.97	95.86	0.00	93.66	95.63	97.52	98.81	99.23	97.99	99.75	99.87	100.00

Fairview																							
Total Gold Free Surface of Particle (%)	0% (barren)	0% (not exposed)	0-5%	5-10%	10-15%	15-20%	20-25%	25-30%	30-35%	35-40%	40-45%	45-50%	50-55%	55-60%	60-65%	65-70%	70-75%	75-80%	80-85%	85-90%	90-95%	95-100%	100%
No. of Particles	979.00	158.00	37.00	6.00	3.00	1.00	2.00	0.00	0.00	0.00	0.00	1.00	0.00	0.00	0.00	2.00	0.00	0.00	2.00	0.00	0.00	0.00	12.00
Mean Phase ECD (µm)	0.00	2.67	3.33	7.06	5.04	6.17	4.38	0.00	0.00	0.00	0.00	7.92	0.00	0.00	0.00	20.20	0.00	0.00	3.75	0.00	0.00	0.00	8.79
Mean Phase Max Span (µm)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Particle Distribution (%)	68.59	26.10	4.14	0.59	0.11	0.02	0.01	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.00	0.31	0.00	0.00	0.00	0.00	0.00	0.00	0.12
Mean Density	4.45	5.11	5.24	5.66	6.47	7.65	6.86	0.00	0.00	0.00	0.00	13.18	0.00	0.00	0.00	16.12	0.00	0.00	16.86	0.00	0.00	0.00	19.28
Distribution of Mineral (%)	0.00	22.80	15.38	11.84	3.27	1.24	0.48	0.00	0.00	0.00	0.00	0.68	0.00	0.00	0.00	31.50	0.00	0.00	0.49	0.00	0.00	0.00	12.34
Cum. Distn. of Mineral (%)	0.00	100.00	77.20	61.83	49.99	46.71	45.48	45.00	45.00	45.00	45.00	45.00	44.32	44.32	44.32	44.32	12.83	12.83	12.83	12.34	12.34	12.34	12.34
Modal Analysis (%)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Total Gold (%)	0.00	0.83	3.52	19.11	28.97	46.84	36.41	0.00	0.00	0.00	0.00	78.75	0.00	0.00	0.00	96.27	0.00	0.00	96.75	0.00	0.00	0.00	100.00
Quartz (%)	4.56	1.11	0.47	3.78	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Muscovite (%)	2.55	0.51	0.60	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.48	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Biotite (%)	0.90	0.64	0.15	0.18	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	2.56	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Chamosite (%)	2.06	1.73	0.47	0.93	0.00	0.22	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.68	0.00	0.00	2.34	0.00	0.00	0.00	0.00
Zircon (%)	0.02	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Kaolinite (%)	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Pyrite (%)	50.90	63.72	72.87	66.75	64.28	52.94	63.59	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.91	0.00	0.00	0.00	0.00
Pyrrhotite (%)	2.05	1.26	0.31	0.77	0.26	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Chalcopyrite (%)	0.36	0.17	0.19	0.02	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Arsenopyrite (%)	19.96	29.42	19.06	7.77	6.49	0.00	0.00	0.00	0.00	0.00	0.00	21.25	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Gersdorffite (%)	0.76	0.02	0.02	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Sphalerite (%)	0.04	0.02	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Galena (%)	0.20	0.01	0.03	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Fe(Mn)-oxides (%)	12.58	0.07	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Rutile (%)	0.04	0.06	0.19	0.05	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Uraninite (%)	0.08	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Brannerite (%)	0.02	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Calcite (%)	0.00	0.00	0.10	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Ankerite (%)	0.20	0.13	0.56	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Magnesite (%)	0.21	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Siderite (%)	0.23	0.03	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Apatite (%)	0.01	0.02	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Lollingite (%)	0.02	0.04	1.33	0.63	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Fe-oxide (Sb) (%)	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Unknown (%)	2.09	0.16	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00

Barbrook																							
Total Gold Free Surface of Particle (%)	0% (barren)	0% (not exposed)	0-5%	5-10%	10-15%	15-20%	20-25%	25-30%	30-35%	35-40%	40-45%	45-50%	50-55%	55-60%	60-65%	65-70%	70-75%	75-80%	80-85%	85-90%	90-95%	95-100%	100%
No. of Particles	2098.00	4.00	4.00	1.00	0.00	1.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	6.00
Mean Phase ECD (µm)	0.00	3.95	3.44	7.55	0.00	1.81	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	20.22
Mean Phase Max Span (µm)	0.00	6.58	5.51	10.88	0.00	2.53	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	30.88
Particle Distribution (%)	99.17	0.26	0.26	0.03	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.28
Mean Density	4.64	5.29	5.62	2.85	0.00	3.18	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	19.28
Distribution of Mineral (%)	0.00	2.05	2.97	1.43	0.00	0.08	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	93.46
Cum. Distn. of Mineral (%)	0.00	100.00	97.95	94.98	93.55	93.55	93.46	93.46	93.46	93.46	93.46	93.46	93.46	93.46	93.46	93.46	93.46	93.46	93.46	93.46	93.46	93.46	93.46
Modal Analysis (%)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Total Gold (%)	0.00	2.35	3.43	16.27	0.00	28.83	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	100.00
Quartz (%)	3.70	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Plagioclase (%)	0.05	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Muscovite (%)	1.34	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Biotite (%)	0.14	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Chamosite (%)	2.53	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Actinolite (%)	0.05	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Zircon (%)	0.03	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Pyrite (%)	72.11	49.24	55.20	83.08	0.00	71.17	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Pyrrhotite (%)	2.44	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Chalcopyrite (%)	0.04	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Arsenopyrite (%)	3.20	44.88	35.42	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Sphalerite (%)	0.74	0.00	0.00	0.39	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Galena (%)	7.21	0.04	5.93	0.27	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Ullmannite (%)	0.20	0.00	0.03	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Fe(Mn)-oxides (%)	1.27	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Rutile (%)	0.02	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Uraninite (%)	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Brannerite (%)	0.08	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Calcite (%)	0.09	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Ankerite (%)	3.36	0.35	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Siderite (%)	0.67	3.13	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Apatite (%)	0.09	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Monazite-(Ce) (%)	0.04	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Fe-oxide (Sb) (%)	0.02	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Tungsten (%)	0.02	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Carbon (%)	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Unknown (%)	0.47	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00

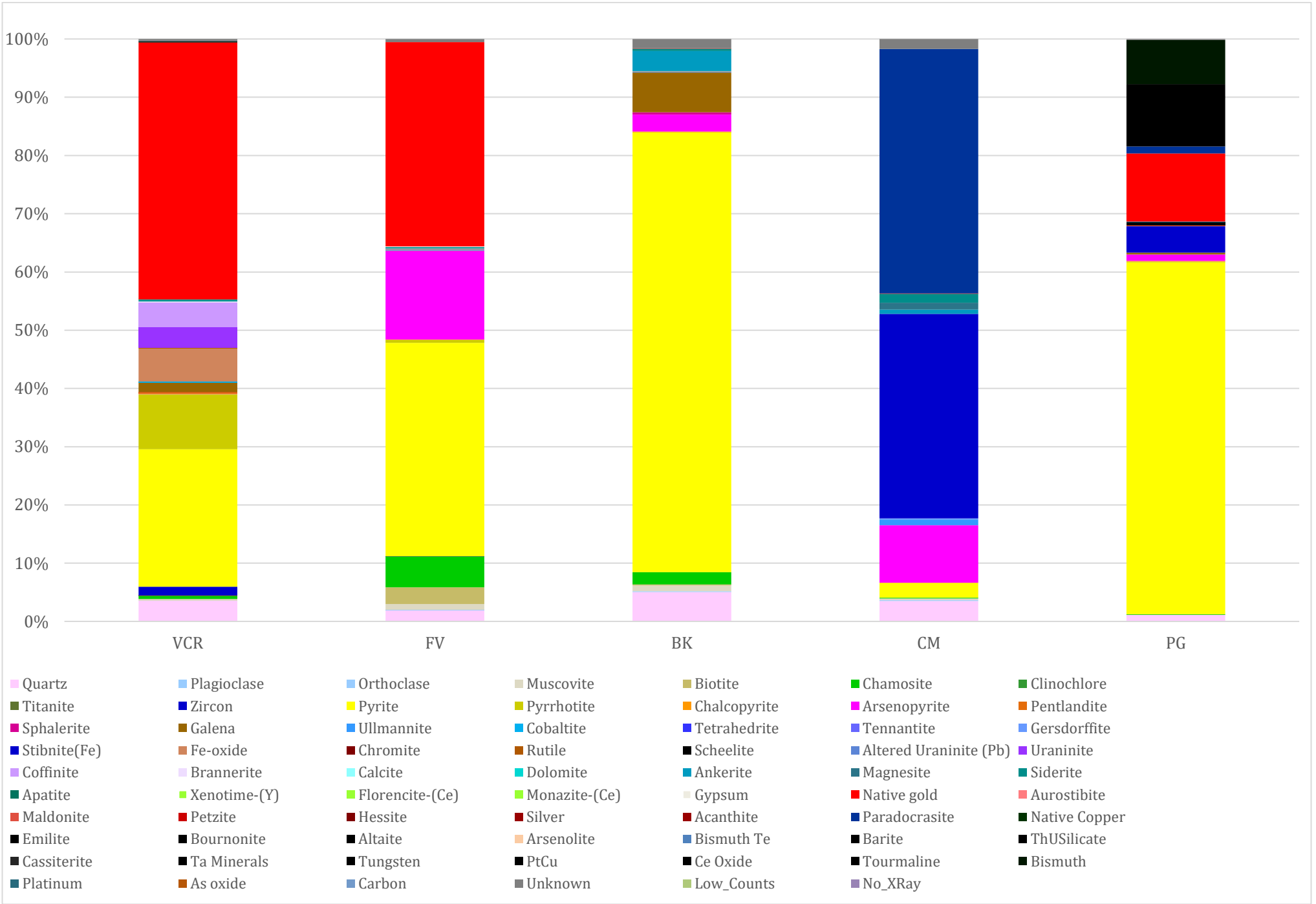
Consolidated Murchison Native Gold																							
Native gold Free Surface of Particle (%)	0% (barren)	0% (not exposed)	0-5%	5-10%	10-15%	15-20%	20-25%	25-30%	30-35%	35-40%	40-45%	45-50%	50-55%	55-60%	60-65%	65-70%	70-75%	75-80%	80-85%	85-90%	90-95%	95-100%	100%
No. of Particles	57672.00	0.00	1.00	1.00	0.00	1.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	8.00
Mean Phase ECD (µm)	0.00	0.00	2.39	1.81	0.00	3.50	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	5.60
Mean Phase Max Span (µm)	0.00	0.00	4.00	2.26	0.00	5.06	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	8.26
Particle Distribution (%)	99.98	0.00	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01
Mean Density	4.61	0.00	2.06	2.17	0.00	1.59	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	19.28
Distribution of Mineral (%)	0.00	0.00	1.96	1.06	0.00	3.32	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	93.66
Cum. Distn. of Mineral (%)	0.00	100.00	100.00	98.04	96.99	96.99	93.66	93.66	93.66	93.66	93.66	93.66	93.66	93.66	93.66	93.66	93.66	93.66	93.66	93.66	93.66	93.66	93.66
Modal Analysis (%)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Quartz (%)	3.61	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Chlorite (%)	0.51	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Other Sulphides (%)	2.55	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Orthoclase (%)	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Plagioclase (%)	0.27	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Other Silicates (%)	0.06	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Tourmaline (%)	2.65	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Arsenopyrite (%)	3.12	0.00	97.37	89.89	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Pyrite (%)	1.42	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Pyrrhotite (%)	0.18	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Gersdorffite (%)	0.14	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Stibnite (%)	49.27	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Berthierite (%)	5.84	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Oxides (%)	0.10	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Fe(Mn)-oxides (%)	0.15	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Phosphates (%)	0.04	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Other (%)	0.33	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Mica (%)	0.62	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Brannerite (%)	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Ankerite (%)	1.30	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Magnesite (%)	1.52	0.00	0.00	0.00	0.00	56.06	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Siderite (%)	2.39	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Other Carbonates (%)	0.02	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Native Antimony (%)	23.06	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Other Native Elements (%)	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Native gold (%)	0.00	0.00	2.63	10.11	0.00	43.94	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	100.00
Aurostibite (%)	0.03	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Sb-oxide/sulphate (%)	0.79	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00

Pilgrim's Rest																							
Native gold Free Surface of Particle (%)	0% (barren)	0% (not exposed)	0-5%	5-10%	10-15%	15-20%	20-25%	25-30%	30-35%	35-40%	40-45%	45-50%	50-55%	55-60%	60-65%	65-70%	70-75%	75-80%	80-85%	85-90%	90-95%	95-100%	100%
No. of Particles	1959.00	10.00	10.00	2.00	1.00	2.00	0.00	1.00	1.00	0.00	0.00	0.00	0.00	0.00	1.00	0.00	0.00	1.00	2.00	0.00	1.00	2.00	64.00
Mean Phase ECD (µm)	0.00	3.55	3.61	5.62	2.21	6.21	0.00	8.36	7.08	0.00	0.00	0.00	0.00	0.00	9.38	0.00	0.00	14.75	16.15	0.00	36.69	19.36	15.85
Mean Phase Max Span (µm)	0.00	5.00	5.60	6.96	3.58	7.93	0.00	15.58	9.92	0.00	0.00	0.00	0.00	0.00	14.18	0.00	0.00	17.12	22.07	0.00	63.61	28.57	23.56
Particle Distribution (%)	87.21	2.16	4.94	0.09	0.01	0.04	0.00	0.08	0.08	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.05	0.12	0.00	0.29	0.18	4.73
Mean Density	4.71	5.06	4.97	4.07	2.66	3.77	0.00	3.87	5.34	0.00	0.00	0.00	0.00	0.00	7.09	0.00	0.00	9.05	9.22	0.00	9.55	9.49	19.27
Distribution of Mineral (%)	0.00	0.55	0.71	0.37	0.02	0.32	0.00	0.65	1.16	0.00	0.00	0.00	0.00	0.00	0.34	0.00	0.00	0.84	2.14	0.00	5.21	3.15	84.54
Cum. Distn. of Mineral (%)	0.00	100.00	99.45	98.74	98.37	98.35	98.03	98.03	97.38	96.22	96.22	96.22	96.22	96.22	96.22	95.88	95.88	95.88	95.04	92.90	92.90	87.69	84.54
Modal Analysis (%)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Quartz (%)	10.37	0.00	0.51	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.22	0.00
Chlorite (%)	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Other Sulphides (%)	11.15	0.06	0.01	0.00	0.00	0.00	0.00	0.00	18.72	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Plagioclase (%)	0.05	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Other Silicates (%)	0.02	0.00	0.07	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Amphibole (%)	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Arsenopyrite (%)	0.26	0.02	0.19	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Pyrite (%)	64.71	98.27	97.16	77.21	91.93	54.58	0.00	52.13	3.14	0.00	0.00	0.00	0.00	0.00	12.62	0.00	0.00	2.28	1.59	0.00	0.34	0.05	0.03
Pyrrhotite (%)	0.71	0.22	1.18	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Stibnite (%)	0.11	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Berthierite (%)	0.25	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Fe(Mn)-oxides (%)	0.22	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Other (%)	1.11	0.00	0.08	0.34	0.00	0.00	0.00	0.03	0.24	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Mica (%)	0.17	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Ankerite (%)	0.58	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Native Antimony (%)	2.32	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Other Native Elements (%)	7.93	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Native gold (%)	0.00	1.43	0.80	22.45	8.07	45.42	0.00	47.85	77.89	0.00	0.00	0.00	0.00	0.00	87.38	0.00	0.00	97.72	98.41	0.00	99.66	99.73	99.97
Sb-oxide/sulphate (%)	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00

MLA Modal Mineralogy

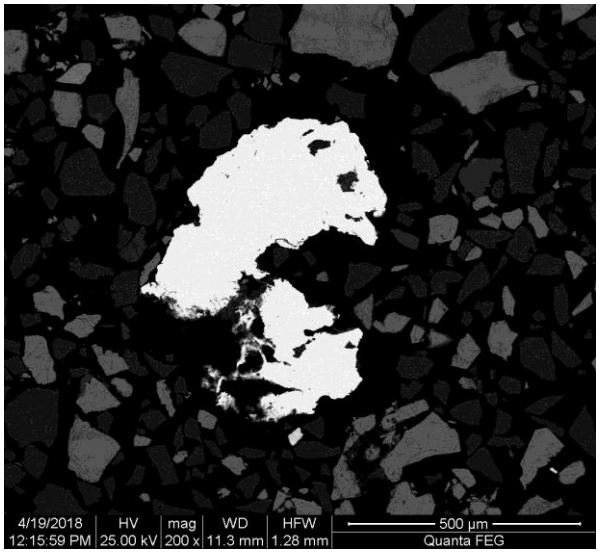
The modal mineralogy of the surface of the gravity concentrate block. The modal mineralogy is determined using rapid identification and quantification of all constituent minerals.

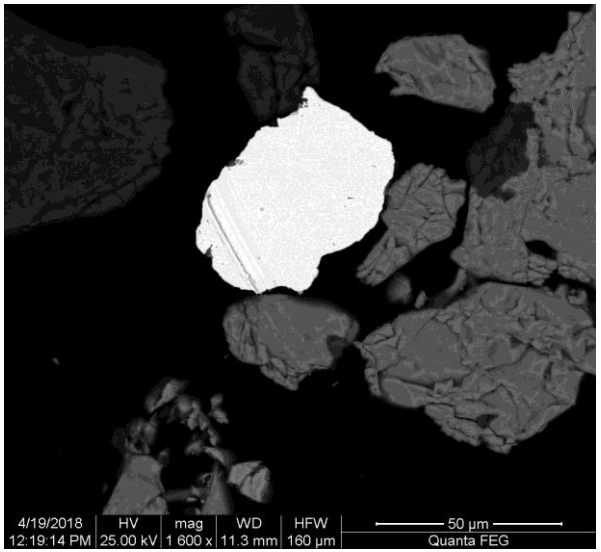
	VCR	FV	BK	CM	PG
Mineral	Wt%	Wt%	Wt%	Wt%	Wt%
Quartz	3.55	1.87	5.05	3.49	1.13
Plagioclase	0.01	0.06	0.12	0.13	0.00
Orthoclase	0.00	0.05	0.00	0.00	0.00
Muscovite	0.24	1.00	1.02	0.27	0.00
Biotite	0.07	2.89	0.16	0.00	0.01
Chamosite	0.44	5.25	2.10	0.16	0.08
Clinochlore	0.00	0.04	0.00	0.00	0.00
Titanite	0.15	0.00	0.00	0.00	0.00
Zircon	1.50	0.02	0.00	0.00	0.00
Pyrite	23.60	36.65	75.51	2.52	60.42
Pyrrhotite	9.47	0.49	0.01	0.00	0.13
Chalcopyrite	0.01	0.08	0.14	0.02	0.12
Arsenopyrite	0.04	15.19	2.92	9.80	1.13
Pentlandite	0.27	0.00	0.00	0.00	0.00
Sphalerite	0.01	0.01	0.37	0.00	0.00
Galena	1.63	0.07	6.85	0.00	0.32
Ullmannite	0.00	0.00	0.09	0.97	0.00
Cobaltite	0.18	0.00	0.00	0.00	0.00
Tetrahedrite	0.00	0.00	0.00	0.00	0.00
Tennantite	0.00	0.00	0.00	0.00	0.00
Gersdorffite	0.00	0.17	0.00	0.17	0.00
Stibnite(Fe)	0.00	0.00	0.01	34.82	4.50
Fe-oxide	5.64	0.04	0.05	0.00	0.19
Chromite	0.00	0.00	0.00	0.00	0.01
Rutile	0.19	0.10	0.05	0.00	0.00
Scheelite	0.00	0.00	0.00	0.00	0.52
Altered Uraninite (Pb)	0.06	0.00	0.00	0.00	0.00
Uraninite	3.43	0.00	0.00	0.00	0.00
Coffinite	4.32	0.00	0.00	0.00	0.00
Brannerite	0.12	0.00	0.00	0.00	0.00
Calcite	0.01	0.02	0.02	0.00	0.00
Dolomite	0.00	0.00	0.00	0.00	0.00
Ankerite	0.00	0.06	3.46	0.74	0.10
Magnesite	0.00	0.05	0.00	1.18	0.00
Siderite	0.32	0.23	0.38	1.49	0.00
Apatite	0.03	0.04	0.02	0.01	0.00
Xenotime-(Y)	0.00	0.01	0.00	0.00	0.00
Florencite-(Ce)	0.00	0.00	0.00	0.00	0.00
Monazite-(Ce)	0.01	0.00	0.00	0.00	0.00
Gypsum	0.00	0.00	0.00	0.00	0.00
Native gold	44.09	35.05	0.05	0.04	11.67
Aurostibite	0.00	0.00	0.00	0.03	0.00
Maldonite	0.00	0.00	0.00	0.00	0.00
Petzite	0.00	0.00	0.00	0.00	0.00
Hessite	0.00	0.00	0.00	0.00	0.00
Silver	0.00	0.00	0.00	0.00	0.00
Acanthite	0.00	0.00	0.00	0.00	0.00
Paradocrasite	0.00	0.00	0.00	41.62	1.21
Native Copper	0.00	0.00	0.00	0.00	0.00
Emilite	0.07	0.00	0.00	0.00	10.71
Bournonite	0.00	0.00	0.00	0.00	0.00
Altaite	0.00	0.00	0.00	0.00	0.00
Arsenolite	0.00	0.00	0.00	0.00	0.00
Bismuth Te	0.00	0.00	0.00	0.00	0.00
Barite	0.00	0.00	0.00	0.00	0.00
ThUSilicate	0.00	0.00	0.00	0.00	0.00
Cassiterite	0.00	0.00	0.00	0.00	0.00
Ta Minerals	0.00	0.00	0.00	0.00	0.00
Tungsten	0.00	0.00	0.00	0.00	0.00
PtCu	0.00	0.00	0.00	0.00	0.00
Ce Oxide	0.00	0.00	0.00	0.00	0.00
Tourmaline	0.00	0.00	0.00	0.00	0.00
Bismuth	0.15	0.00	0.00	0.00	7.59
Platinum	0.00	0.00	0.00	0.00	0.00
As oxide	0.00	0.00	0.00	0.00	0.00
Carbon	0.02	0.00	0.00	0.00	0.00
Unknown	0.37	0.54	1.61	1.69	0.16
Low_Counts	0.00	0.00	0.00	0.00	0.00
No_XRay	0.00	0.00	0.00	0.00	0.00
Mineral	0.00	0.00	0.00	0.84	0.00
Total	100.00	100.00	100.00	100.00	100.00

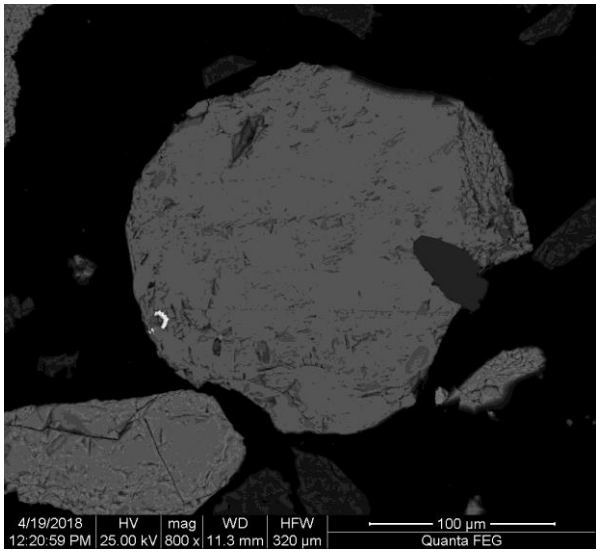

100%
90%
80%
70%
60%
50%
40%
30%
20%
10%
0%
VCR
FV
BK
CM
PG
Quartz
Plagioclase
Orthoclase
Muscovite
Biotite
Chamosite
Clinochlore
Titanite
Zircon
Pyrite
Pyrrhotite
Chalcopyrite
Arsenopyrite
Pentlandite
Sphalerite
Galena
Ullmannite
Cobaltite
Tetrahedrite
Tennantite
Gersdorffite
Stibnite(Fe)
Fe-oxide
Chromite
Rutile
Scheelite
Altered Uraninite (Pb)
Uraninite
Coffinite
Brannerite
Calcite
Dolomite
Ankerite
Magnesite
Siderite
Apatite
Xenotime-(Y)
Florencite-(Ce)
Monazite-(Ce)
Gypsum
Native gold
Aurostibite
Maldonite
Petzite
Hessite
Silver
Acanthite
Paradocrasite
Native Copper
Emilite
Bournonite
Altaite
Arsenolite
Bismuth Te
Barite
ThUSilicate
Cassiterite
Ta Minerals
Tungsten
PtCu
Ce Oxide
Tourmaline
Bismuth
Platinum
As oxide
Carbon
Unknown
Low_Counts
No_XRay

MLA BSE (Back-scattered electron) Images taken as part of the gold deportment

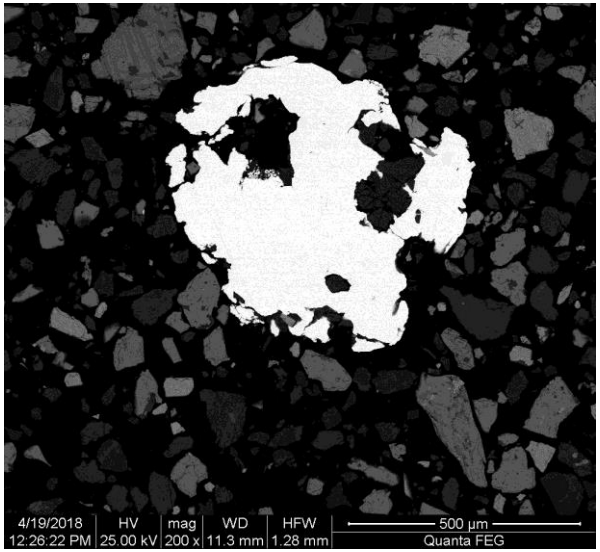
VCR Gold Grains

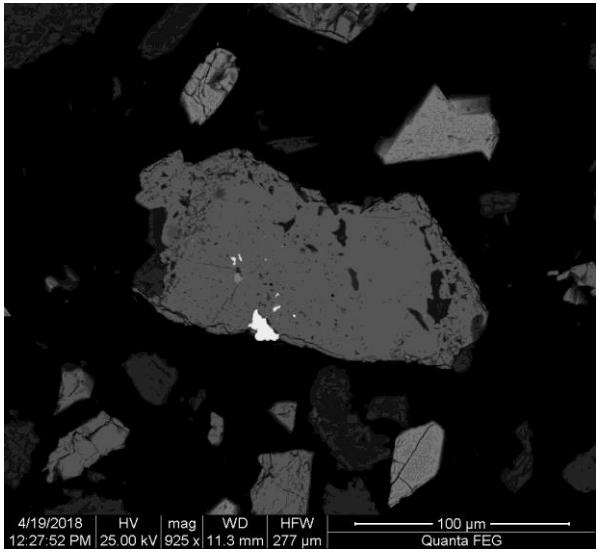


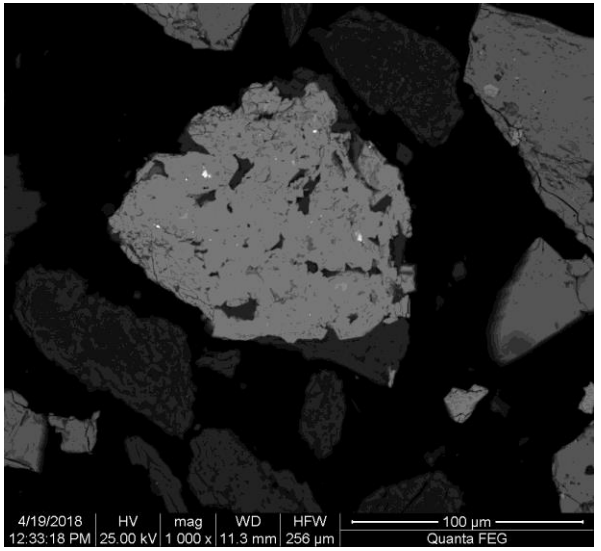




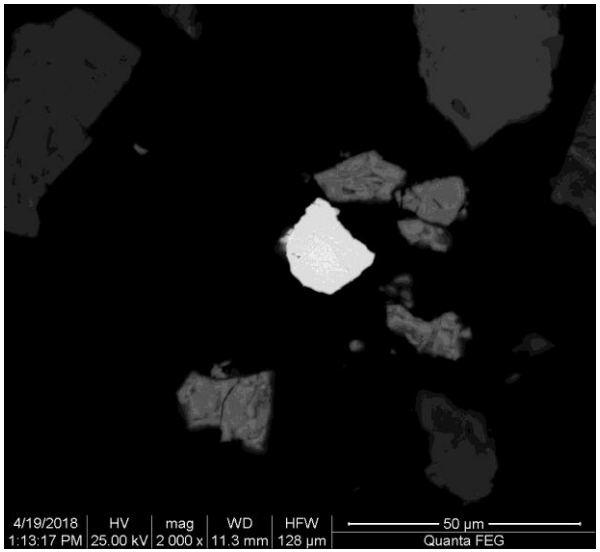
Fairview Gold Grains

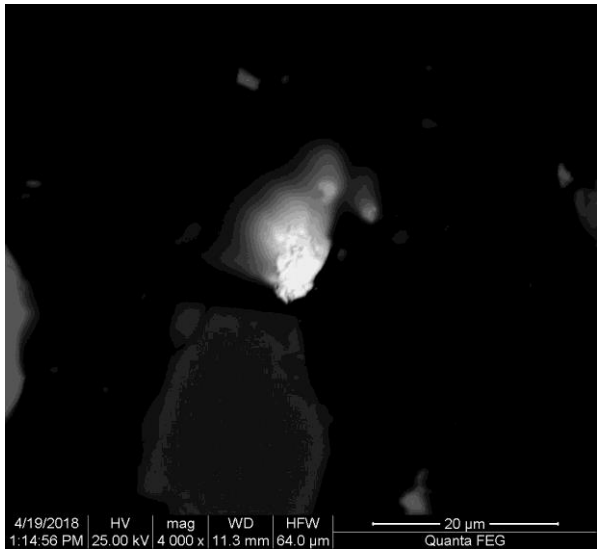


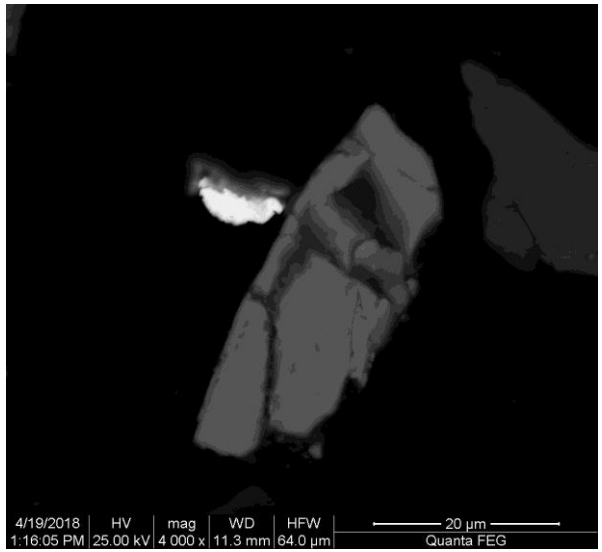




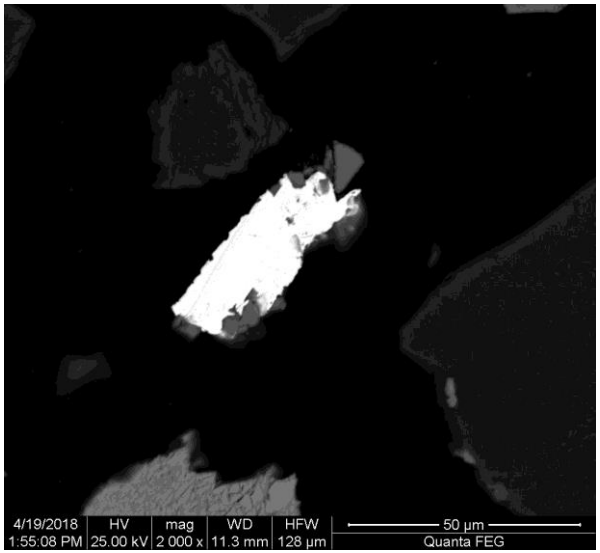
Barbrook Gold Grains

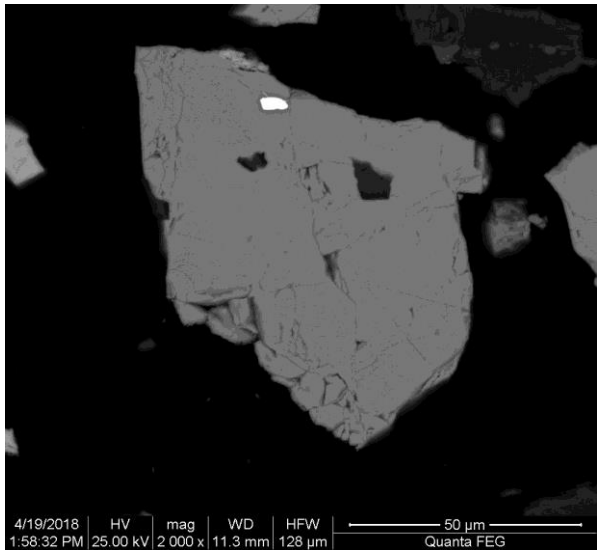


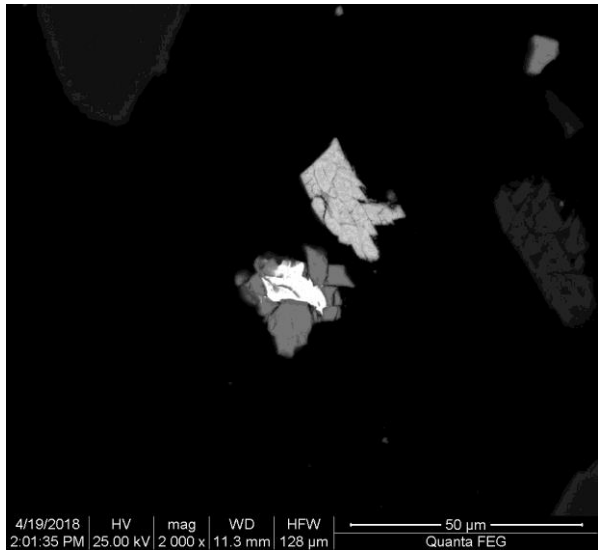




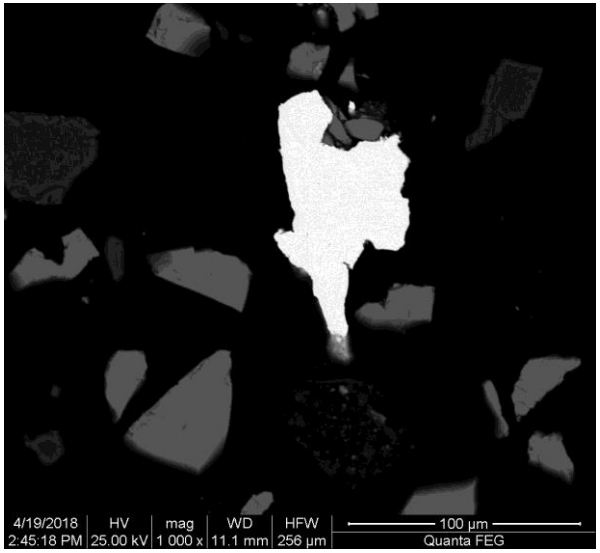
Consolidated Murchison

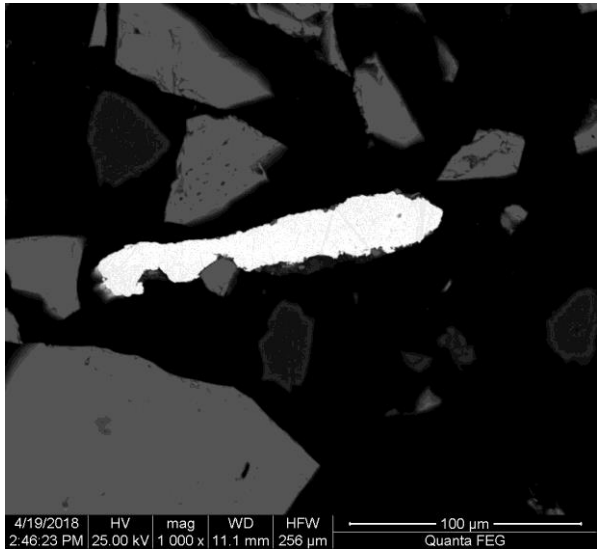


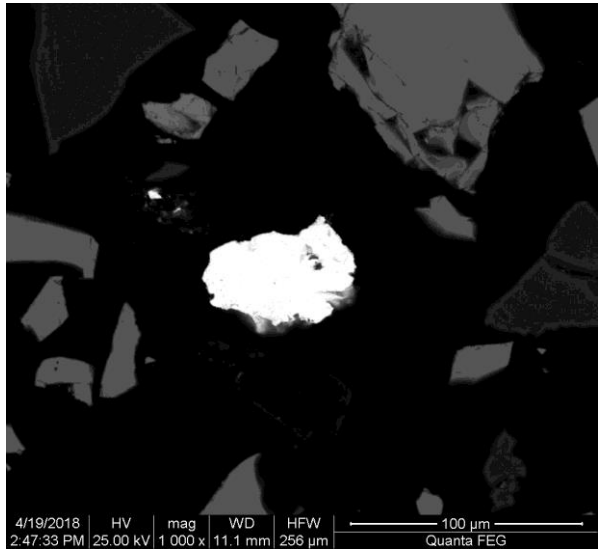




Pilgrim's Rest

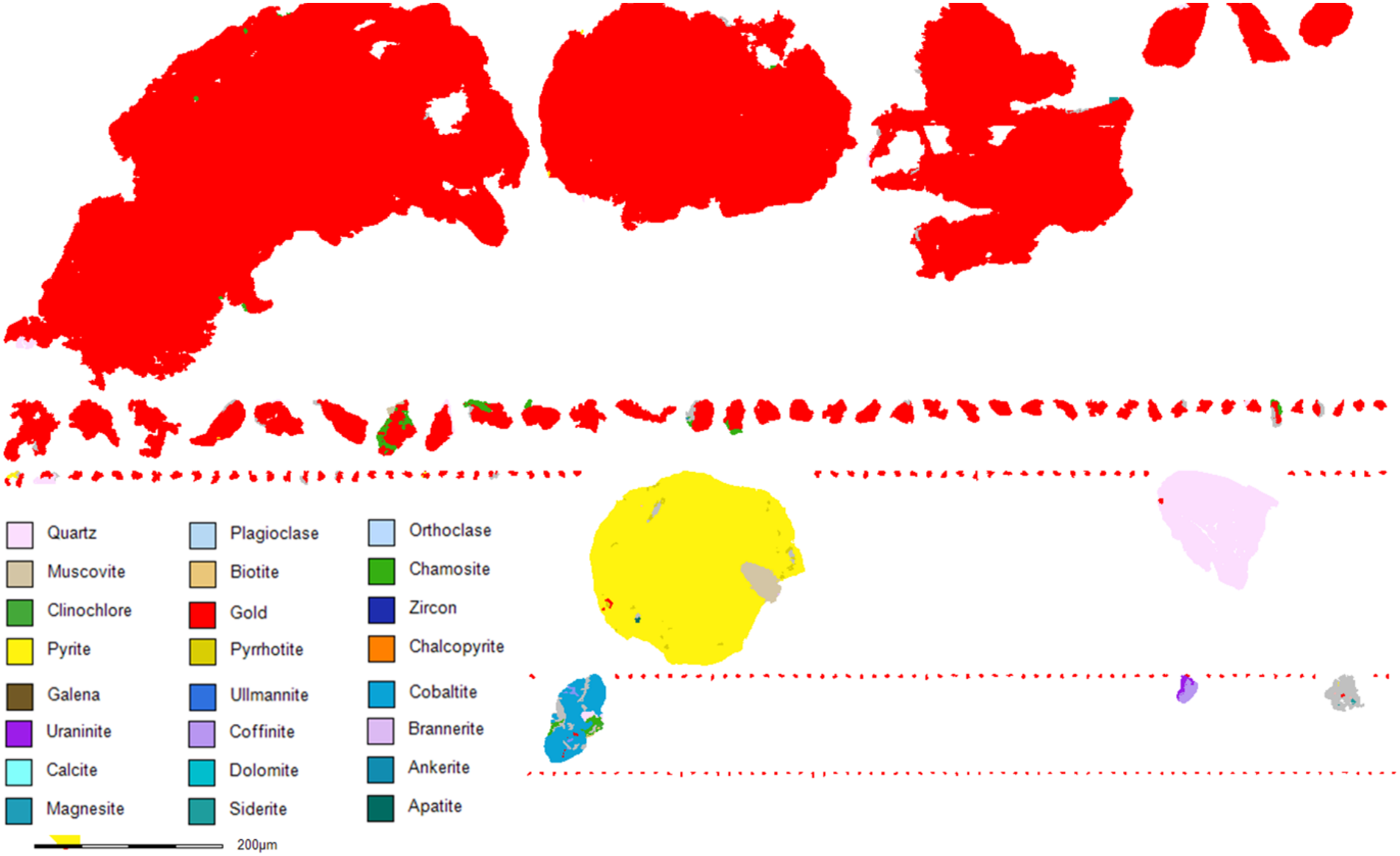




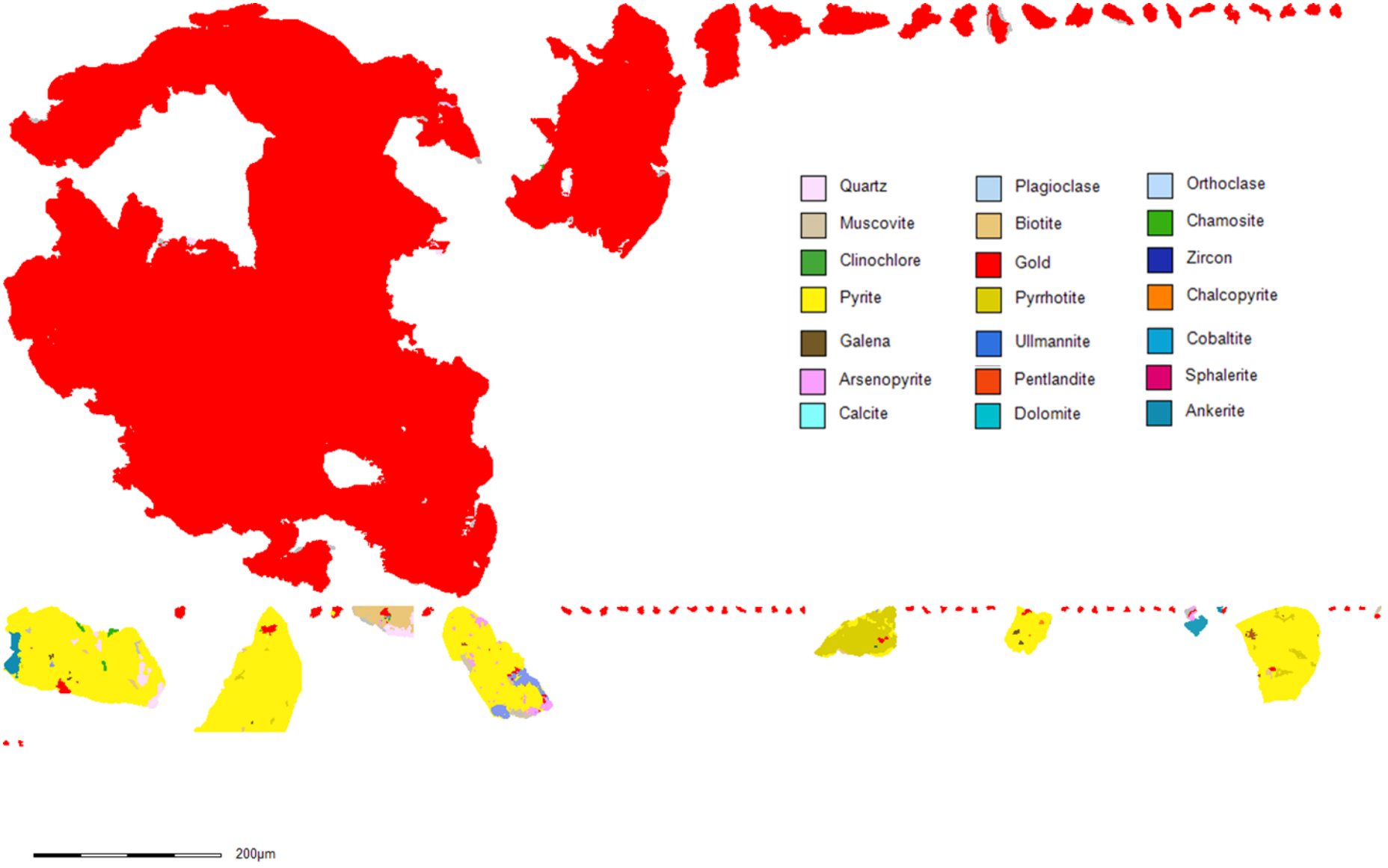


MLA False Colour Images taken as part of the gold deportment

VCR False Colour Images



Fairview False Colour Images


Quartz
Plagioclase
Orthoclase
Muscovite
Biotite
Chamosite
Clinochlore
Gold
Zircon
Pyrite
Pyrrhotite
Chalcopyrite
Galena
Ullmannite
Cobaltite
Arsenopyrite
Pentlandite
Sphalerite
Calcite
Dolomite
Ankerite
200µm

Barbrook False Colour Images



Consolidated Murchison False Colour Images



Pilgrim’s Rest



EPMA Data Points

Electron Probe Micro Analysis (EPMA) was undertaken to locate and better understand the nature of the "invisible gold" thought to be present in the refractory gold ores of the Fairview, Barbrook and Consolidated Murchison samples. Thus grains with BSE's similar to that of arsenopyrite and pyrite were analysed in an attempt to located gold hosted within these sulphides.

Fairview EPMA points														
DataSet	S	Fe	Cu	As	Sb	Au	Total	X	Y	Z	Distance (µ)	Mean Z	Point#	Date
1 / 1 .	22.842	35.934	0	40.247	0.033	0.015	99.071	10236	9967	62	0	26.304	1	2018-07-24 09:21
2 / 1 .	52.545	46.375	0	0.237	0.025	0.007	99.188	10353	9975	60	117.29	20.525	2	2018-07-24 10:04
3 / 1 .	23.422	35.55	0	39.332	0.02	0.014	98.338	10055	9862	66	209.29	25.967	3	2018-07-24 10:20
4 / 1 .	52.263	45.725	0.014	0.329	0	0	98.331	9975	10130	66	307.74	20.355	4	2018-07-24 10:34
5 / 1 .	22.616	35.141	0	41.174	0.024	0.019	98.974	10057	10169	66	269.93	26.348	5	2018-07-24 10:43
6 / 1 .	22.314	34.81	0.079	41.577	0.006	0.015	98.801	10053	10178	66	279.33	26.379	6	2018-07-24 10:54
7 / 1 .	52.341	45.872	0	0.915	0	0	99.128	9743	9896	64	498.09	20.588	7	2018-07-24 11:04
8 / 1 .	22.459	35.505	0	41.67	0.017	0.003	99.653	9605	9684	64	691.56	26.553	8	2018-07-24 11:16
9 / 1 .	52.786	46.246	0	0.549	0.013	0.004	99.597	9728	9277	64	856.84	20.647	9	2018-07-24 11:26
10 / 1 .	52.968	46.402	0.018	0.501	0	0.001	99.89	9696	9342	64	825.97	20.694	10	2018-07-24 11:33
11 / 1 .	0	7.191	0	0.09	0.072	0.001	7.354	9932	9242	64	786.16	1.916	11	2018-07-24 11:42
12 / 1 .	24.011	35.294	0	39.829	0.061	0.007	99.203	10193	9209	64	759.22	26.163	12	2018-07-24 11:51
13 / 1 .	23.597	35.202	0	39.929	0.032	0	98.761	10097	9459	64	526.68	26.095	13	2018-07-24 12:00
14 / 1 .	22.913	34.912	0	41.1	0.046	0.031	99.003	10470	9525	64	500.12	26.347	14	2018-07-24 12:08
15 / 1 .	22.751	35.156	0	40.623	0.037	0.007	98.573	9351	9583	67	964.73	26.206	15	2018-07-24 12:17

Barbrook EPMA points														
DataSet	S	Fe	Cu	As	Sb	Au	Total	X	Y	Z	Distance (µ)	Mean Z	Point#	Date
16 / 1 .	53.264	45.9	0	0.025	0.019	0.003	99.21	9612	-21046	47	31019.28	20.463	16	2018-07-24 12:39
17 / 1 .	22.118	34.774	0	41.632	0.038	0.007	98.568	10072	-20975	47	30942.44	26.338	17	2018-07-24 12:46
18 / 1 .	53.005	45.777	0.032	0.097	0.027	0.001	98.94	9676	-21577	47	31548.97	20.439	18	2018-07-24 13:14
19 / 1 .	53.375	46.052	0	0.102	0	0.001	99.53	9676	-21577	47	31548.97	20.496	19	2018-07-24 13:22
20 / 1 .	53.187	46.171	0	0.25	0	0	99.608	9313	-21627	47	31607.48	20.582	20	2018-07-24 13:30
21 / 1 .	39.387	58.208	0.044	0	0.003	0.003	97.646	9124	-21238	47	31224.81	21.443	21	2018-07-24 13:38
22 / 1 .	52.571	45.327	0	0	0.006	0.002	97.906	9550	-21207	47	31181.55	20.173	22	2018-07-24 13:45
23 / 1 .	23.103	34.691	0	40.545	0	0.005	98.345	8768	-22011	47	32011.68	26.087	23	2018-07-24 13:56
24 / 1 .	53.008	45.939	0.003	0	0	0.006	98.957	8657	-20966	47	30973.28	20.407	24	2018-07-24 14:04
25 / 1 .	52.556	45.99	0.001	0.189	0	0.003	98.74	8700	-20608	47	30613.56	20.429	25	2018-07-24 14:12
26 / 1 .	21.736	33.267	0	41.488	0.025	0	96.517	8821	-20805	47	30804.52	25.829	26	2018-07-24 14:20

Cons Murch EPMA points														
DataSet/	S	Fe	Cu	As	Sb	Au	Total	X	Y	Z	Distance (µ)	Mean Z	Point#	Date
27 / 1 .	15.685	26.49	0	0.462	57.209	0.003	99.85	12630	10859	76	2554.82	38.706	27	2018-07-24 14:57
28 / 1 .	21.403	33.974	0.052	42.69	0.151	0	98.269	12908	11049	76	2882.79	26.436	28	2018-07-24 15:05
29 / 1 .	27.78	0.121	0	0.395	71.333	0.005	99.633	13090	10785	76	2968.95	40.978	29	2018-07-24 15:14
30 / 1 .	52.82	45.56	0	0.125	0.045	0.001	98.551	13157	10797	76	3036.67	20.353	30	2018-07-24 15:27
31 / 1 .	0	0.156	0	5.021	93.484	0.004	98.665	13213	10787	75	3087.9	49.371	31	2018-07-24 15:36
32 / 1 .	0.028	0.092	0.051	4.684	92.667	0.004	97.526	12877	10729	75	2748.76	48.852	32	2018-07-24 15:44
33 / 1 .	21.748	34.559	0.017	42.23	0.172	0.005	98.731	12595	11427	75	2774.28	26.498	33	2018-07-24 15:53
34 / 1 .	27.895	0.502	0	0.394	71.473	0.016	100.281	13563	11426	75	3632.88	41.162	34	2018-07-24 16:02
35 / 1 .	27.845	0.063	0.038	0.354	71.167	0	99.466	13654	10931	75	3551.37	40.894	35	2018-07-24 16:12
36 / 1 .	0.012	0.141	0	4.808	94.014	0.006	98.98	13650	11020	75	3572.73	49.541	36	2018-07-24 16:19
37 / 1 .	28.359	0.244	0	0.339	71.361	0.021	100.325	12843	10476	75	2656.26	41.098	37	2018-07-24 16:28
38 / 1 .	15.412	26.776	0	0.444	58.702	0.007	101.341	12273	10658	75	2151.05	39.506	38	2018-07-24 16:37
39 / 1 .	20.599	34.255	0	43.796	0.175	0.001	98.827	12208	10938	75	2198.13	26.742	39	2018-07-24 16:44
40 / 1 .	20.936	34.329	0	43.385	0.175	0.005	98.83	11986	11033	75	2049.15	26.658	40	2018-07-24 16:53

Appendix C – Leaching Data

Leaching Conditions and Preparations

A summary of the ideal leaching conditions for alternative gold extraction techniques as discussed in Chapter 8.

Thiosulphate preparation

The leach was prepared by weighing out the prepared sample aliquot (ranging between 400 and 500 g, depending on the spilt of the sample), the equivalent mass of tap water is added to ensure a 50% solid solution is maintained. The appropriate masses of reagent are added, the concentrations of which can be found in Table 1, they have been represented and must be adjusted to reflect the true value of water added). An important consideration when determining the amount of reagent required, is the molar mass difference between the component required and the reagent the component is present in (i.e. the difference between thiosulphate and hydrated sodium thiosulphate). All the reagents used in this experimentation were recently opened and thus the crystal waters were as represent on the label. However, it should be noted that older products are capable of containing greater quantities of crystal water than indicated and should a certain level of precision be required, chemical analyses would need to be performed to determine the precise concentration.

The pH was monitored and maintained using sulphuric acid. For the thiosulphate leaching system the reduction potential and pH are controlled using $CuSO_4$ and NH_3 respectively. Where necessary the pH was also adjusted using H_2SO_4 if the initial pH (due to the ore) was too high. Ammonia is important for pH adjustment but is also a required reagent for an effective leach.

Table 1 The concentrations and additions used to prepare leaching solution for thiosulphate leaching

Component addition (g/L)	Reagent addition (g/L)
40 g/L S_2O_3	88.53 g/L $Na_2S_2O_3(H_2O)_5$
6 g/L Cu	23.43 g/L $CuSO_4(H_2O)_5$

Thiourea preparation

The leach was prepared by adding the required sample aliquot, the equivalent mass of water, as well as the required reagent masses (the values for which can be found in Table). The conditions (reduction potential and pH) are monitored and maintained using Fe_2SO_4 and H_2SO_4 respectively.

Table 2 The concentrations and additions used to prepare leaching solution for thiourea leaching.

Component addition (g/L)	Reagent addition (g/L)
10 g/L CH_4N_2S	10 g/L CH_4N_2S
5 g/L Fe (III)	17.90 Fe2(SO4)3

Chloride-Hypochlorite preparation

The leach was prepared by adding the required sample aliquot, the equivalent mass of water as well as the required reagent masses (the values for which can be found in Table) which must be adjusted to reflect the true value of water added. The conditions, reduction potential and pH, are monitored and maintained using NaClO and HCl respectively.

Table 3 The concentrations and additions used to prepare leaching solution for chloride – hypochlorite leaching.

Component addition (g/L)	Reagent addition (g/L)
80 g/L Cl^-	100g/L NaCl
10 g/L $NaClO^-$	14.47 g/L NaClO

Titration Preparations

A summary of the conditions and preparations required for titration

Thiosulphate consumption titration reagent preparation

The starch indicator solution was prepared daily as this solution tends to rapidly decompose (due to interaction with both light and bacteria). The indicator was prepared by adding 1g of starch to 10 mL of boiling water, once this has been thoroughly mixed, forming a white opaque solution a further 90 mL of room temperature water can be added which will ensure the solution goes clear and is subsequently ready for use (Harris, 2010).

Iodine solution can be difficult to prepare due to limited solubility of iodine in water. Therefore, iodine is prepared in situ using a reaction of iodate and iodide in the presence of sulphuric acid, operating by the following mechanism;

$$KIO_3 + 5KI + 3H_2SO_4 \rightarrow 3I_2 + 3H_2O + 3K_2SO$$

The iodine titrant was prepared daily, to prevent solution degradation. In order to generate an iodine solution of approximately a 0.020 M, a combination potassium iodate (0.15 g) and

potassium iodide (4 g) as well as sulphuric acid (1.5 mL or more as required for complete dissolution), can then be made up to 100 mL.

Titration procedure: the titration was conducted by transferring a known volume of thiosulfate containing leach solution into a flask, adding 2 mL of starch indicator, and titrating with iodine until the solution turned a deep blue/black colour.

Thiourea consumption titration reagent preparation

The starch indicator solution was prepared daily due to rapid decomposition (due to interaction with both light and bacteria). The indicator was prepared by adding 1g of starch to 10 mL of boiling water, once this has been thoroughly mixed forming a white opaque solution a further 90 mL of room temperature water can be added which will ensure the solution goes clear and is ready for use (Harris, 2010).

The iodine titrant was prepared daily, to prevent solution degradation. In order to generate an iodine solution of approximately a 0.020 M, a combination potassium iodate (0.15 g) and potassium iodide (4 g) as well as sulphuric acid (1.5 mL or more if required for complete dissolution), which can then be made up to 100 mL.

Titration procedure: a known volume of thiourea containing leach solution is transferred into a flask along with 2 ml of starch indictor. Iodine titrant is added until the solution turns into a deep blue.

Cyanide Leaching Data

Leaching Conditions, Reagent Addition and Consumption

Sample ID	Residence Time	Initial Mass added	Mass Water added	Oxygen Sparging	Reagent Addition							Residual NaCN				Residual CaO			Reagent Consumption		Final pH
					Lime	CaO	CaO	Carbon		NaCN		Titre	Aliquot	NaCN	NaCN End	Titre	Conc	CaO End	NaCN (kg/t	CaO (kg/t	
	Hrs	(g)	(ml)		(ml)	(g)	(kg/t)	(g)	(g/l)	(g)	(ppm)	(ml)	(ml)	(g/l)	(g)	(ml)	(g/l)	(g)	Feed)	Feed)	
VCR	24	433.7	434.0	No	1.3	0.13	0.30	8.68	20.0	0.729	1680.4	6.6	10	0.48	0.21	0.10	0.03	0.01	1.20	0.27	10.41
FV	24	478.1	479.5	No	1.2	0.12	0.25	9.58	20.0	0.829	1734.0	5.5	10	0.40	0.19	0.10	0.03	0.01	1.33	0.22	10.30
CM	24	509.8	510.6	No	2.2	0.22	0.43	10.21	20.0	0.921	1807.2	4.5	10	0.33	0.17	0.10	0.03	0.01	1.47	0.40	10.29
BK	24	457.3	458.0	No	1.9	0.19	0.42	9.15	20.0	1.118	2445.3	3.7	10	0.27	0.12	0.10	0.03	0.01	2.18	0.39	10.38
PG	24	453.8	454.4	No	17.6	1.76	3.88	9.09	20.0	1.112	2451.2	0.6	10	0.04	0.02	0.10	0.03	0.01	2.41	3.85	10.19

Final Conditions, Assays and Calculated Gold Dissolutions

Sample ID	Final Mass (g)	Mass of Bottle (g)	Final Mass of Slurry + (g)	Final Volume (g)	Mass Adjust	Carbon Assayed (g)	Head	Cyanidation			Calculated Head Au (g/t)	Au Dissolution based on Residue (%)	Au Dissolution based on Carbon and (%)	Au Dissolution Based on Calc Head (%)	Accountability Au (%)
							Assay Au (g/t)	Residue Au (g/t)	Carbon Au (g/t)	Solution Au (mg/l)					
VCR	432.54	1370.05	2249.59	438.3315	0.997279	8.6685	23.6	0.93	1190		24.7112819	96.07004324	100.778865	96.24677587	104.7088218
FV	477.9	1354.65	2321.46	479.3918	0.999603	9.5182	12.5	3.98	406		12.0613926	68.17265368	64.66379468	67.01526587	96.491141
CM	505.9	1348.12	2385.7	521.3239	0.992389	10.3561	3.05	2.42	20.3		2.81397236	21.25963691	13.52102578	14.65512925	92.26138887
BK	453.87	1357.02	2279.05	458.9193	0.992521	9.2407	2.08	1.59	29.3		2.17018918	24.12939231	28.46541056	27.28243903	104.3360183
PG	450.45	1354.58	2287.95	473.8304	0.992552	9.0896	29.4	3.27	1350		30.2843168	88.96038794	91.96826826	89.28274999	103.0078803

Metal Assay of Cyanide leached residue, including standards, blanks and repeats

	Cu	Fe	Ni	S	Zn	Co	Mn	Sb	As
METHOD	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B
LDETECTION	0.02	0.02	0.05	0.6	0.17	0.12	0.5	0.31	0.34
UNITS	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L
VCR	11.8	4.72	0.17	126	3.2	0.1	0.6	<0.31	0.7
FV	5.33	17.1	1.27	194	2.9	0.3	2.7	<0.31	14.1
BK	47.4	37.7	0.88	819	28.6	0.3	6.9	1.5	1.1
CM	3.45	6.78	0.21	552	1.3	<0.12	1.2	63.4	16.7
PG	266	21.6	0.09	1650	<0.17	<0.12	<0.5	<0.31	<0.34
*REP-CM	3.44	6.74	0.22	556	1.3	<0.12	1.2	63.4	16.9
*STD-STANDARD	-	50.4	-	-	-	-	-	-	-
*STD-STANDARD	5.05	-	4.91	-	5.2	4.9	5.1	5	5.1
*BLK-BLANK	<0.02	<0.02	<0.05	<0.6	<0.17	<0.12	<0.5	<0.31	<0.34

Cyanide Titration

Sample ID	Residual NaCN				Residual CaO			Reagent Consumption		Final pH
	Titre (ml)	Aliquot (ml)	NaCN (g/l)	NaCN End (g)	Titre (ml)	Conc (g/l)	CaO End (g)	NaCN (kg/t Feed)	CaO (kg/t Feed)	
VCR	6.6	10	0.48	0.21	0.10	0.03	0.01	1.20	0.27	10.41
FV	5.5	10	0.40	0.19	0.10	0.03	0.01	1.33	0.22	10.30
CM	4.5	10	0.33	0.17	0.10	0.03	0.01	1.47	0.40	10.29
BK	3.7	10	0.27	0.12	0.10	0.03	0.01	2.18	0.39	10.38
PG	0.6	10	0.04	0.02	0.10	0.03	0.01	2.41	3.85	10.19

Chloride-Hypochlorite Leaching VCR

Leach Conditions				
Target Conditions			R Required	R Received
Solids	%	50		
pH		4-7		
Eh		1000.0		
NaCl	g/l	100	43.63	43.68
NaOCl	g/l	21.705 (144.78 g soln)	63.17	63.15

Bottle Mass	Sample + Bottle	Sample + Bottle + Solution
1355.32	1791.35	2227.49
	Sample	Water
	436.1	436.14

Sample ID : VCR

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	NaCl	NaOCl	HCl	Bottle Mas
Hours	Minutes		mV	°C		mV	°C	g	g	g	g
	-10	11.38	501	21.5				43.68	63.15		2338.57
0	0				5.22	1114	23.6		24.55		2363.12
0	30	4.27	280	22.1	4.4	1106	22.6		9.45		2372.57
1	0	4.37	341	21.5	5.32	1057	21.9	8.75	97.41		2385.82
1	30	4.69	661	21.5	5.81	1077	21.7		8.44		2392.12
2	0	4.77	688	21.3	5.74	1037	21.6		5.37		2395.95
2	30	4.85	640	21.2	5.53	1108	22.1		14.37	0.43	2407.84
3	0	4.5	696	21.4	5.16	1103	21.5		4.59	0.16	2412.59
3	30	4.55	702	21.1	5.97	1073	21.9		5.63	3.86	2419.89
4	0	5.04	506	21.6	7.76	509	22.7		19.38	16.23	2452.76
4	30	6.73	567	22.4	6.76	627	23.7		35.07	16.14	2503.97
5	0	6.58	643	22.8	5.36	648	23.5	14.05	40.07	5.23	2542.47
5	30	7.49	639	23.2	6.93	1061	23.6		4.92	21.59	2581.44
6	0	6.44	691	22.4	6.79	1058	23.3	7.28	6.13	18.48	2611.69
6	30	7.03	873	22.5	6.98	925	23.2		18.19	25.75	2652.78
7	0	6.72	922	22.6	5.98	1055	23.2		12.96	6.37	2670.36
8	0	6.42	938	22.6							

Chloride-Hypochlorite Assay Results for Leach Solution

	Au	Cu	Fe	Ni	S	Zn	Co	Mn	Sb	As
LDETECTION	0.001	0.02	0.02	0.05	0.6	0.17	0.12	0.5	0.31	0.34
UDETECTION	500	0	0	0	0	0	0	0	0	0
METHOD	SOL81X	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B
UNITS	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L
SCLVCR	4.35	38.9	4930	16.5	3530	31.6	4.8	60.2	0.8	131

Chloride-Hypochlorite Assay Results for Leach Residue

	WtRec	Au
METHOD	WGH79	FAA303
LDETECTION	0.01	0.01
UDETECTION	0	100
UNITS	G	G/T
CLVCR	222.3	15

Thiourea Leaching Data

Thiourea Leach Conditions – VCR

Leach Conditions				
Target Conditions			R Required	R Received
Solids	%	50		
pH	Sulphuric	1.5		
Eh		420.0		
Thiourea	g/l	10	4.4052	4.4021
Fe_2SO_4	g/l	17.9	7.8854	7.8902

Bottle Mass	Sample + Bottle	Sample + Bottle + Solution
1353.89	1794.39	2243.65
	Sample	Bottle
	440.6	440.4

Sample ID : VCR

		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	Thiourea	Fe_2SO_4	H_2SO_4	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g
		7.15	51.4	19.8	2.17	228	22.2	4.4021	7.8902	5.095	2233.86
		1.6	205	22.7	1.75	288	22.8		8.5661	0.5505	2242.43
0	0	3.7	234	22.6	1.78	254	23.7		10.2568	1.7696	2225.49
0	20	2.03	254	22.9	1.46	395	23.1		18.0063	0.7789	2242.44
0	40	1.44	410	22	1.42	413	22.8		2.7383		2243.52
1	40	1.43	407	21	1.4	417	21.6		10.6509		2252.91
2	40	1.42	422	20.7	1.4	424	21.2		0.5674		2253.82
3	40	1.48	420	21	1.41	422	21.2		0.6607		2254.9
4	40	1.55	420	21	1.4	422	21.4		1.0536		2254.75
5	40	1.47	418	20.6	1.39	422	21.5		1.8141		2255.75
6	40	1.53	418	21.5	1.4	421	21.8		2.0893		2256.72
7	40	1.46	418	22	1.39	421	22.8		1.1431		2256.51
8	0	1.5	415	22.3							2256.17

Thiourea Leach Conditions – Pilgrim's Rest

Leach Conditions				
Target Conditions			R Required	R Received
Solids	%	50		
pH	Sulphuric	1.5		
Eh		420.0		
Thiourea	g/l	10	4.525	4.5506
Fe_2SO_4	g/l	17.9	7.69	7.6761

Bottle Mass	Sample + Bottle	Sample + Bottle + Solution
1362.41	1814.32	2266.46
	Sample	Bottle
	452.11	452.12

Sample ID : PG

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	Thiourea	Fe_2SO_4	H_2SO_4	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g
		3.66	272	20.2	1.92	210	21.8	4.5506	7.6761	5.9107	
					1.74	288	22.6		8.1851	1.8027	2288.28
0	0	1.89	340	22	1.57	408	22.6		4.692	1.4321	2291.19
0	30	1.75	398	21.4	1.51	411	21.1		2.2333	0.7726	2281.02
1	30	1.58	394	20.7	1.44	414	20.6		6.3375		2273.13
2	30	1.54	408	21.4	1.42	415	21.6		3.638		2274.23
3	30	1.53	405	21	1.44	418	22.1		8.0707		2280.15
4	30	1.51	416	20.8	1.47	421	21.9		3.0237		2280.53
5	30	1.48	412	21	1.46	421	21.8		4.3408		2282.67
6	30	1.52	415	21.9	1.52	422	22.6		6.4825		2287.42
7	30	1.61	418	21.2	1.51	420	22.4		1.4096		2287.06
8	0	1.62	417	21.4							2286.07

Titration Data for VCR

Titrant Volume	Titrant Concentration	Moles
L	mol/L	Moles
0.00025	0.0225	0.0000056

Moles of TU	TU Mass	TU Conc Remaining	% Remaining
Moles	g	g/L	
0.0000056	0.000427	0.0855	0.971

Titration Data for PG

Titrant Volume	Titrant Concentration	Moles
L	mol/L	Moles
0.00015	0.0225	0.0000034

Moles of TU	TU Mass	TU Conc Remaining	% Remaining
Moles	g	g/L	
0.0000034	0.000256	0.0513	0.563

Thiourea Assay Results for Leach Solution

	Au	Cu	Fe	Ni	S	Zn	Co	Mn	Sb	As
LDETECTION	0.001	0.02	0.02	0.05	0.6	0.17	0.12	0.5	0.31	0.34
UDETECTION	500	0	0	0	0	0	0	0	0	0
METHOD	SOL81X	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B
UNITS	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L
Pilgrim's Rest	0.01	1050	32300	3.41	51300	6.6	<0.12	255	1.7	72.8
VCR	0.01	17.9	38200	10.6	53600	4.4	2.6	120	1.1	22.7

Thiourea Assay Results for Leach Residue

GEO13136	WtRec	Au
METHOD	WGH79	FAA303
LDETECTION	0.01	0.01
UDETECTION	0	100
UNITS	G	G/T
CLVCR	222.3	15
PGTU	229.7	26.9
TUVCR	241.3	17
*REP-CLVCR	-	13.7
*STD-AMIS0420	-	0.41

Thiosulphate Leaching Data

Thiosulphate Leach Conditions – VCR

Leach Conditions			
Target Conditions			Actual Reagent required
Solids	%	50	
pH		9.5	
Eh		> -5 (<195)	
Thiosulphate	g/L	88.53	39.53
Copper Sulphate	g/L	23.4	10.45

Bottle Mass	Sample + Bottle	Sample + Bottle + Solution
1355.3	1801.63	2248.55
	Sample	Water
	446.3	446.64

Sample ID : VCR

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	$CuSO_4$	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g
0	0	8.18	102.1	17.1	10.04	158.2	14.7	39.5802	0.3837	10.482	2298.67
	10	9.72	108.3	16.5	10	99.9	16.5		0.1989		2297.76
1	10	10.05	89.3	19.2							2296.89
3	10	10.26	79.3	19.9							2296.15
3	56	10.22	94.5	19.6	10.1	83.2	19.8				2295.97
5	10	10.24	79.5	19.6	10.08	94.7	19.7				2295.48
7	10	10.28	76	19.6	9.88	100.8	19.8				2295.37
24		10.16	80.4	18.5							2293.2

Thiosulphate Leach Conditions – Fairview

Leach Conditions				
Target Conditions			Actual Reagent required	
Solids	%	50		
pH		9.5		
Eh		> -5 (<195)		
Thiosulphate	g/L	88.53	39.381	39.3481
Copper Sulphate	g/L	23.4	10.409	10.415

Bottle Mass	Sample + Bottle	Sample + Bottle + Solution
1433.59	1878.23	2322.91
	Sample	Water
	444.68	444.68

Sample ID : FV

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	$CuSO_4$	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g
0	0	9.04	197.4	17.1	9.92	4.5	16.6	39.3481	1.5519	10.415	2371.07
0	10	9.93	109.7	15.8							2370.16
1	0	9.9	96.7	16.5							2369.28
2	0	9.72	90.7	17.1	9.97	77.2	17.4		0.3984		2368.92
4	0	9.9	78.5	17.3							2367.78
6	0	9.89	75.6	17.3							2366.89
8	0	9.88	81.4	17.6							2365.85
24	0	10.1	66.8	17.5							2364.95

Thiosulphate Leach Conditions – Barbrook

Leach Conditions			
Target Conditions			Actual Reagent required
Solids	%	50	
pH		9.5	
Eh		> -5 (<195)	
Thiosulphate	g/L	88.53	41.95
Copper Sulphate	g/L	23.4	11.09

Bottle Mass	Sample + Bottle	Sample + Bottle + Solution
1405.01	1878.92	2352.84
	Sample	Water
	473.93	473.92

Sample ID : BK

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	$CuSO_4$	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g
0	0	6.72	166.8	18	9.01	151.7	16.3	41.9081	1.0137	11.0702	2404.53
	10	9.45	90	17							2404.01
1	0	9.56	91.7	18.4							2403.95
2	0	9.57	71.1	18.8							2403.22
4	0	9.52	67.2	19.8							2402
6	0	9.51	59.5	20							2401
8	0	9.45	58.7	20							2401.56
10	0	9.48	63.6	19							2401.22
24	0	9.47	49.4	19.7							2400.65

Thiosulphate Leach Conditions – Consolidated Murchison

Leach Conditions			
Target Conditions			Actual Reagent required
Solids	%	50	
pH		9.5	
Eh		> -5 (<195)	
Thiosulphate	g/L	88.53	46.57
Copper Sulphate	g/L	23.4	12.309

Bottle Mass	Sample + Bottle	Sample + Bottle + Solution
1363.58	1887.09	2412.65
	Sample	Water
	523.49	525.28

Sample ID : CM

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	$CuSO_4$	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g
0		7.96	149.1	16.8	9.88	225	15.3	46.5883	0.5426	12.3069	2465.98
	10	8.87	148.8	16.5	9.95	73.5	16.7		0.89		2465.78
1		9.72	98	18.8	9.96	57.8	19.1		0.4476		2464.41
2		9.8	80.7	18.6	9.97	56.5	19.1		0.9829		2464.59
4		9.71	69.3	18.7	9.96	53.7	19		0.6481		2463.36
6		9.8	79.1	18.5	9.97	41.9	18.9		0.5882		2461.64
8		9.91	48.5	18.1	9.95	35.2	18.6		0.2198		2461
24		9.93	51.3	16.9							2459.38

Thiosulphate Leach Conditions – Pilgrim's Rest

Leach Conditions			
Target Conditions			Actual Reagent required
Solids	%	50	
pH		9.5	
Eh		> -5 (<195)	
Thiosulphate	g/L	88.53	39.24
Copper Sulphate	g/L	23.4	10.372

Bottle Mass	Sample + Bottle	Sample + Bottle + Solution
1360.76	1804.09	2247.49
	Sample	Water
	443.29	443.32

Sample ID : PG

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	$CuSO_4$	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g
0	0	6.8	333	16.6	89.43	-220	15.9	39.2381	7.9829	10.3434	2301.52
	10	9.21	-78.2	16.2	9.42	-114.8	16.3		0.7598		2301.37
1	10	9.41	-9.2	16.3	9.7	-28.2	16.5		1.4246		2301.97
3	10	9.68	3	16.6	9.81	-20.9	16.9		1.8601		2303.24
5	10	9.78	-10.33	18.4	9.93	-29.1	186		83.22		2303.24
7	10	9.9	-5.3	17.7							2303.11
9	10	9.91	2.6	17.5							2302.01
24		10.58	45.7	19.8							2302.25

Titration Data – VCR

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.0175	0.0306	0.000536

Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Remaining reagent	% Remaining
Moles	g	g/L	g/L	
0.000536	0.0601	12.02	26.6	33.6

Titration Data – Fairview

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.0174	0.0306	0.000533

Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Remaining reagent	% Remaining
Moles	g	g/L	g/L	
0.000533	0.0597	11.947	26.473	31.499

Titration Data – Barbrook

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.0183	0.0288	0.000528

Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Remaining reagent	% Remaining
Moles	g	g/L	g/L	
0.000528	0.0592	11.841	26.238	31.304

Titration Data – Consolidated Murchison

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.0179	0.0306	0.0005480

Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Remaining reagent	% Remaining
Moles	g	g/L	g/L	
0.000548	0.0615	12.2903	27.234	29.228

Titration Data – Pilgrim's Rest

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.0021	0.0306	0.0000643

Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Remaining reagent	% Remaining
Moles	g	g/L	g/L	
0.0000643	0.00721	1.44	3.20	4.07

Assay Results for Thiosulphate Leach Residue

	WtRec	Au
METHOD	WGH79	FAA303
LDETECTION	0.01	0.01
UDETECTION	0	100
UNITS	G	G/T
VCR	243.5	13.9
Fairview	246	15
Barbrook	221.5	1.98
Consolidated Murchison	261	3.16
Pilgrim's Rest	207.5	16.5

Assay Results for Leach Solution

	Au	Cu	Fe	Ni	S	Zn	Co	Mn	Sb	As
METHOD	SOL81X	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B	ICP84B
LDETECTION	0.001	0.02	0.02	0.05	0.6	0.17	0.12	0.5	0.31	0.34
UNITS	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L	MG/L
VCR	0.005	5460	<0.02	<0.05	25900	42.8	<0.12	<0.5	0.8	<0.34
FairviewV	0.003	4760	<0.02	<0.05	24300	38.3	<0.12	<0.5	<0.31	4.3
Barbrook	0.004	5090	67.4	0.18	34100	0.5	<0.12	0.6	<0.31	<0.34
Consolidated Murchison	0.002	4520	<0.02	<0.05	27700	35.6	<0.12	<0.5	12	63.2
Pilgrim’s Rest	0.143	1120	0.8	0.08	21300	9.1	<0.12	<0.5	0.5	<0.34

Thiosulphate Calcined Ore Leaching Data

Thiosulphate Leach Conditions – Calcined VCR

Leach Condition			
Target Conditions			Actual Reagent required
Solids	%	50	
pH		9.5	
Eh		> -5 (<195)	
Thiosulphate	g/L	62	26.49
Copper Sulphate	g/L	7.5	3.2
Glycine	g/L	3.0	

Bottle Ma	Sample + Bottle	Sample + Bottle + Solution
1357.19	1803.8	2250.72
	Sample	Water
	446.72	446.74

Sample ID : Calcined VCR

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	$CuSO_4$	H2SO4	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g	g
0		7.89	123.5	19	9.47	84.3	18.4	26.4341	64.2267	3.2281		2278.58
0	20	9.62	91.2	18.7	9.52	91.7	18.7				0.010047	2277.62
1	0	9.58	92.4	18.9								2276.69
2	0	9.55	103.5	19								2275.42
3	0	9.5	103	19.4								2274.48
4	0	9.49	98.4	20.4								2273.7
6	0	9.52	104.1	19.6								2273
8	0	9.51	100	19.6								2271.88
24		9.33	110.8	19.4								2270.22

Thiosulphate Leach Conditions – Calcined Fairview

Leach Conditions			
Target Conditions			Actual Reagent required
Solids	%	50	
pH		9.5	
Eh		> -5 (<195)	
Thiosulphate	g/L	88.53	39.381
Copper Sulphate	g/L	23.4	10.409

Bottle Mass	Sample + Bottle	Sample + Bottle + Solution
1433.59	1878.23	2322.91
	Sample	Water
	444.68	444.68

Sample ID : Calcined FV

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	$CuSO_4$	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g
0	0	9.04	197.4	17.1	9.92	4.5	16.6	39.3481	1.5519	10.415	2371.07
0	10	9.93	109.7	15.8							2370.16
1	0	9.9	96.7	16.5							2369.28
2	0	9.72	90.7	17.1	9.97	77.2	17.4		0.3984		2368.92
4	0	9.9	78.5	17.3							2367.78
6	0	9.89	75.6	17.3							2366.89
8	0	9.88	81.4	17.6							2365.85
24	0	10.1	66.8	17.5							2364.95

Thiosulphate Leach Conditions – Calcined Barbrook

Leach Conditions				
Target Conditions			Actual Reagent required	
Solids	%	50		
pH		9.5		
Eh		> -5 (<195)		
Thiosulphate	g/L	88.53	38.391	38.4565
Copper Sulphate	g/L	23.4	10.147	10.1136

Bottle Mass	Sample + Bottle	Sample + Bottle + Solution
1404.87	1838.29	2261.51
	Sample	Water
	433.44	433.57

Sample ID : Calcined BK

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	$CuSO_4$	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g
0	0	6.28	79.1	21.7	9.87	99.2	17.2	38.4565	4.0279	10.1136	2324.15
	10	9.96	109.9	16.6							2323.12
1	0	9.61	96.4	16.5	9.85	50.8	17.1		1.0556		2322.23
3	0	9.89	69.3	16.9							2321.62
5	0	9.92	75.7	17.2							2320.97
7	0	9.97	79.3	17							2319.97
24	0	10.34	76.4	17.4							2319.18

Thiosulphate Leach Conditions – Calcined Consolidated Murchison

Leaching Conditions				
Target Conditions			Actual Reagent required	
Solids	%	50		
pH		9.5		
Eh		> -5 (<195)		
Thiosulphate	g/L	62	28.0518	28.0518
Copper Sulphate	g/L	7.5	3.9	3.3869
Glycine	g/L	0.0		

Bottle Mass	Sample + Bottle	Sample + Bottle + Solution
1435.56	1886.69	2338.01
	Sample	Water
	451.32	451.25

Sample ID : Calcined CM

Eh

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	$CuSO_4$	H2SO4	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g	g
0	0	9.5	113.6	21.8	9.51	-6	20.8	28.0518	0.2626	3.3869	0.6442	2371.48
0	10	9.55	58.3	20.6	9.6	71.2	21.1		0.494		0.9223	2370.9
0	38	9.59	58.5	20.9								2370.2
1	38	9.57	23.6	22.1								2369.35
2	40	9.66	35	21.9	9.57	34	22.3				0.0873	2367.67
4	40	9.67	30.6	21.9								2366.82
6	40	9.75	24.5	21.7	9.55	25.8	22.6				0.7362	2365.43
8	40	9.77	-10.9	21								2363.72
10	40	9.76	20	22.1	9.51	18.7	22.6				0.7214	2363.72
24	0	9.8	-40.8	20.9								2362.73

Thiosulphate Leach Conditions – Calcined Pilgrim's Rest

Leaching Conditions			
Target Conditions			Actual Reagent required
Solids	%	50	
pH		9.5	
Eh		> -5 (<195)	
Thiosulphate	g/L	62	24.8
Copper Sulphate	g/L	7.5	3.0
Glycine	g/L	0.0	

Bottle Mass	Sample + Bottle	Sample + Bottle + Solution
1437.94	1833.07	2228.29
	Sample	Water
	395.13	395.14

Sample ID : Calcined PG

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	$CuSO_4$	H2SO4	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g	g
0	0	5.6	471	26.5	8.75	-166.7	25.2	24.8265	3.3913	3.0292		2257.52
0	10	5.71	281	24.1	8.82	-25	24.8		13.7501			2268.94
0	30	9.13	-74.1	24.5								2268.23
1	15	9.42	-4.8	24.7								2267.32
2	2	9.4	55.4	24.9								2266.87
3	4	9.47	64.1	24.8								2266.05
5	0	9.51	64.8	25.4								2265.53
7	2	9.52	76.4	25.5								2264.85
9	0	9.78	82.4	25.2	9.53	132.2	26.3				2.5584	2264.2
10	55	9.52	138	23.5								2263.76
12	50	9.89	133	24.4	9.22	158.8	25.6				1.5906	2263.78
22	50	9.64	22.3	168.2								2262.59
24	0	1047	22.4	190								2262.09

Titration Data - Calcined VCR

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.0194	0.0273	0.000530

Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Reagent Remaining	% Remaining
Moles	g	g/L	g/L	
0.000530	0.0594	11.9	26.3	49.8

Titration Data – Calcined Fairview

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.0154	0.0321	0.000494

Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Reagent Remaining	% Remaining
Moles	g	g/L	g/L	
0.000494	0.0554	11.1	24.6	31.2

Titration Data – Calcined Barbrook

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.0103	0.0306	0.000315

Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Reagent Remaining	% Remaining
Moles	g	g/L	g/L	
0.000315	0.0354	7.07	15.7	20.4

Titration Data – Calcined Consolidated Murchison

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.0152	0.0308	0.000468

Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Reagent Remaining	% Remaining
Moles	g	g/L	g/L	
0.000468	0.0524	10.5	23.2	41.4

Titration Data – Calcined Pilgrim's Rest

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.0004	0.021812982	0.0000087

Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Reagent Remaining	% Remaining
Moles	g	g/L	g/L	
0.0000087	0.000978356	0.195671175	0.43358636	0.873232956

Thiosulphate Glycine Leaching Data

Thiosulphate Glycine Leach Conditions – VCR

Leach Condition			
Target Conditions			Actual Reagent required
Solids	%	50	
pH		9.5	
Eh		> -5 (<195)	
Thiosulphate	g/L	62	26.95
Copper Sulphate	g/L	7.5	3.26
Glycine	g/L	3.0	1.3

Bottle Ma	Sample + Bottle	Sample + Bottle + Solution
1408.19	1843.09	2278.02
	Sample	Water
	434.9	434.92

Sample ID : VCR

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	H2SO4	$CuSO_4$	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g	g
0	0	8.57	182	20.9	9.72	-48.5	17.2	26.9025	0.7901		3.2536	2309.07
0	10	9.13	-23.1	17.9	9.32	-83.1	18.2		4.9277			2311.53
0	26	9.24	-75.2	18.5	9.39	-76.2	18.6		1.8729			2312.5
0	43	9.54	-54.9	18.7								2312.03
2	0	10.22	-17.3	20.4	10.04	-18.2	20.5			0.196719		
3	0	10.24	-13.4	20	9.8	-13.6	20.1			0.350558		2309.5
4		10.14	-9.5	20.1	9.8	-11	20.2			0.681958		2334.92
5		9.92	-6.4	21	9.65	-7.3	21.4			0.237832		2342.61
6		9.89	-4.3	21.7	9.6	-6.2	22			0.229822		2350.26
8		10.04	-1.6	21.4	9.49	11.3	21.9			2.30584		2354.13
10		9.53	8.6	21.2	9.43	9.1	21.4			0.462856		2342.27
24		10.21	43.7	20.7								2341.56

Thiosulphate Glycine Leach Conditions – Fairview

Leach Condition			
Target Conditions			Actual Reagent required
Solids	%	50	
pH		9.5	
Eh		> -5 (<195)	
Thiosulphate	g/L	62	29.38
Copper Sulphate	g/L	7.5	3.55
Glycine	g/L	0.0	1.42

Bottle Mass	Sample + Bottle	Sample + Bottle + Solution
1409.73	1883.68	2360.48
	Sample	Water
	473.91	476.74

Sample ID : FV

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	$CuSO_4$	H2SO4	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g	g
0	0	8.1	76.8	21.3	9.4	-24	20.2	29.3693	1.3722	3.5547		2394.52
0	10	9.3	25	21.4	9.55	-54.5	21.4		0.8632			2393.9
1	10	9.59	-7.8	23								2393.37
3	10	9.82	-9.7	22.9	9.4	-10.4	23.5				0.297	2392.96
5	10	9.64	-8.6	23.3								2392.18
7	10	9.72	-11.2	23.3	9.4	-10.5	23.9				0.386	2329.06
24	0	10.03	-9.6	23.5								2392.01

Thiosulphate Glycine Leach Conditions – Barbrook

Leach Conditions			
Target Conditions			Actual Reagent required
Solids	%	50	
pH		9.5	
Eh		> -5 (<195)	
Thiosulphate	g/L	62	28.7
Copper Sulphate	g/L	7.5	3.47
Glycine	g/L	3.0	1.38

Bottle Mass	Sample + Bottle	Sample + Bottle + Solution
1436.2	1898.53	2361.34
	Sample	Water
	462.32	462.58

Sample ID : BK

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	$CuSO_4$	H2SO4	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g	g
0	0	8.67	92	21.6	9.62	-20.2	20.4	28.7447	1.5305	3.4823		2394.75
0	10	9.22	-38.6	21.6	9.43	-63.8	21.7		1.6443			2394.85
0	33	9.57	-31.5	23								2394.15
1	55	9.68	-24.4	24.3	9.55	-26.2	24.9				0.1674	2393.4
2	48	9.62	-20.3	23.4								2392.9
4	53	9.66	-19.1	23.5								2392.68
6	48	9.74	-19.6	23.5	9.45	-18.5	24.1				0.4267	2391.92
8	48	9.68	-13.6	23.6	9.39	-13	24.2				0.4251	2391.54
24	0	10.27	-2.6	23.4								2390.67

Thiosulphate Glycine Leach Conditions – Consolidated Murchison

Leach Conditions			
Target Conditions			Actual Reagent required
Solids	%	50	
pH		9.5	
Eh		> -5 (<195)	
Thiosulphate	g/L	62	31.96
Copper Sulphate	g/L	7.5	3.86
Glycine	g/L	3.0	1.55

Bottle Mass	Sample + Bottle	Sample + Bottle + Solution
1412.4	1926.42	2440.32
	Sample	Water
	514.15	514.26

Sample ID : CM

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	$CuSO_4$	H2SO4	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g	g
0	0	7.49	104.2	19.7	8.97	-20	18.5	31.9556	12.5444	3.8576		2488.8
0	10	8.82	-228	19.2								2488.05
1	10	8.85	-234	20.4	9.02	-181.5	20.7		1.7031			2489.17
3	11	8.84	-116.2	20.8								2489.21
5	11	8.85	-168.7	20.7								2488.53
7	11	8.97	-157.7	21.4								2488.08
9	11	8.93	-145.6	20								2487.63
24	0	9.65	-63.2	21.4								2488.12

Thiosulphate Glycine Leach Conditions – Pilgrim's Rest

Leach Conditions			
Target Conditions			Actual Reagent required
Solids	%	50	
pH		9.5	
Eh		> -5 (<195)	
Thiosulphate	g/L	62	29.38
Copper Sulphate	g/L	7.5	3.55
Glycine	g/L	0.0	1.42

Bottle Mass	Sample + Bottle	Sample + Bottle + Solution
1357.43	1831.6	2305.36
	Sample	Water
	474.16	474.23

Sample ID : PG

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	$CuSO_4$	H2SO4	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g	g
0	0	4.71	382	22.5	9.52	-82	21.4	29.3566	10.9752	3.553		2349.18
0	10	9.32	-200	21.9								2348.16
1	0	9.52	-58.8	22.7								2347.84
2	0	9.54	-53.1	23.3								2346.27
2	50	9.4	-44.3	23.1								2345.7
3	40	9.43	-43.5	23.1								2345.81
4	40	9.49	-39.2	23.4								2345.28
6	35	9.58	-36.1	23.7								2344.64
8	40	9.59	-30.3	23.4								2344.48
10	57	9.47	-24.8	23								2343.45
12	45	9.54	-17.5	24.1								2342.87
14	45	9.49	-19.9	25.9								2342.2
24	0	10.17	-5.7	24.3								2342.6

Titration Data – Glycine Thiosulphate VCR

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.0037	0.0273	0.000101

Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Reagent Remaining	% Remaining
Moles	g	g/L	g/L	
0.000101	0.0113	2.27	5.02	9.34

Titration Data – Glycine Thiosulphate Fairview

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.008	0.0407	0.000326

Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Reagent Remaining	% Remaining
Moles	g	g/L	g/L	
0.000326	0.0365	7.31	16.2	27.6

Titration Data – Glycine Thiosulphate Barbrook

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.0053	0.0407	0.000216

Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Reagent Remaining	% Remaining
Moles	g	g/L	g/L	
0.000216	0.0242	4.84	10.7	18.7

Titration Data – Glycine Thiosulphate Consolidated Murchison

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.0163	0.0308	0.000502

Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Reagent Remaining	% Remaining
Moles	g	g/L	g/L	
0.000502	0.0562	11.2	24.9	39.0

Titration Data – Glycine Thiosulphate Pilgrim's Rest

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.0025	0.0218	0.0000545

Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Reagent Remaining	% Remaining
Moles	g	g/L	g/L	
0.0000545	0.00611	1.22	2.71	4.62

Thiosulphate Glycine Leaching Data, Calcined Ores

Thiosulphate Glycine Leach Conditions – Calcined VCR

Leach Conditions			
Target Conditions			Actual Reagent required
Solids	%	50	
pH		9.5	
Eh		> -5 (<195)	
Thiosulphate	g/L	62	25.72
Copper Sulphate	g/L	7.5	3.11
Glycine	g/L	3.0	1.244

Bottle Mass	Sample + Bottle	Sample + Bottle + Solution
1367.99	1782.81	2197.72
	Sample	Water
	414.86	414.89

Sample ID : Calcined VCR

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	$CuSO_4$	H2SO4	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g	g
0		7.38	68.6	19.6	9.39	-48.7	18.1	25.7764	50.6752	3.1317	0.034231	2236.07
0	10	9.45	-29	18.5							0.169339	
1	0	9.62	-10.8	19							0.463056	2234.5
2		9.57	-11.7	20.2							0.567199	2234.02
3		9.56	-10.8	20.7								2230.79
4		9.63	-10.9	20.4	9.55	-11.1	21					2231.23
5		9.68	-10.3	20	9.55	-10.8	20.5					2235.74
7		9.64	-7.7	20	9.51	-6.2	20.5					2236.1
9		9.68	-5.1	19.5	9.42	-2.3	20					2237.73
24		10.28	35.1	20.1								2235.93

Thiosulphate Glycine Leach Conditions – Calcined Fairview

Leach Condition			
Target Conditions			Actual Reagent required
Solids	%	50	
pH		9.5	
Eh		> -5 (<195)	
Thiosulphate	g/L	62	27.68
Copper Sulphate	g/L	7.5	3.35
Glycine	g/L	3.0	1.34

Bottle Ma	Sample + Bottle	Sample + Bottle + Solution
1362.2	1807.79	2253.4
	Sample	Water
	445.56	445.58

Sample ID : Calcined FV

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	$CuSO_4$	H2SO4	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g	g
0	0	9.08	15.2	22.4	9.49	37.7	21.7	27.6927	2.5565	3.384	0.6273	2286.94
0	10	9.53	8.5	22.1								2286.06
1	10	9.38	4.8	23								2285.39
2	0	9.34	0.3	24.1	9.39	-5	24.6		0.2729			2284.12
4	5	9.29	-3.6	24.7	9.4	-24.2	2		2.3492			2285.02
6	0	9.39	-7.4	24.7								2284.32
8	0	9.35	-6.2	24.8	9.64	-34	25.2		4.7076			2286.72
24	0	9.64	3.6	24.4								2285.92

Thiosulphate Glycine Leach Conditions – Calcined Barbrook

Leach Conditions			
Target Conditions			Actual Reagent required
Solids	%	50	
pH		9.5	
Eh		> -5 (<195)	
Thiosulphate	g/L	62	27.19
Copper Sulphate	g/L	7.5	3.29
Glycine	g/L	3.0	1.32

Bottle Ma	Sample + Bottle	Sample + Bottle + Solution
1414.25	1853.48	2292.76
	Sample	Water
	439.23	439.21

Sample ID : Calcined BK

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	$CuSO_4$	H2SO4	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g	g
0	0	5.35	79.4	24.4	9.46	-46.1	23	27.2425	1.7684	3.299		2325.43
0	10	8.3	50.2	23.6	9.33	-41.2	23.6		3.9947			2326.5
0	25	9.27	-39.3	23.2	9.52	-42.4	23.7		1.1846			2326.03
1	12	9.12	-4.2	23.6	9.47	-8.2	23.9		3.5225			2325.26
2	40	9.4	3.4	23.2								2327.05
5	34	9.31	7.2	23	9.42	-20.3	23.7		3.8861			2326.04
7	34	5.1	22.7									2360.15
9	34	10.4	22.7									2325.68
24	0	9.54	23.2	22.4								2325.17

Thiosulphate Glycine Leach Conditions – Calcined Consolidated Murchison

Leach Conditions			
Target Conditions			Actual Reagent required
Solids	%	50	
pH		9.5	
Eh		> -5 (<195)	
Thiosulphate	g/L	62	27.5
Copper Sulphate	g/L	7.5	3.33
Glycine	g/L	3.0	1.33

Bottle Mass	Sample + Bottle	Sample + Bottle + Solution
1408.51	1853.11	2297.92
	Sample	Water
	444.6	444.64

Sample ID : Calcined CM

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	$CuSO_4$	H2SO4	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g	g
0		9.68	-4.7	21	9.45	-20.1	20.3	27.551	0.8857	3.3383	1.3825	2329.42
0	10	9.66	2.7	20.3	9.55	12.9	21				0.3685	2328.63
1	5	9.57	-11.1	21.1								2328.15
2	5	9.6	2	20.8								2327.56
4	5	9.62	-1.8	21								2327.09
6	5	9.77	0.9	20.4	9.57	-0.2	20.9				0.6644	2326.15
8	5	9.76	0.5	20.1								2325.81
10	5	9.8	-7.3	20.4	9.51	1.2	21.4				0.7364	2325.62
24	0	9.9	10.4	22.6								2325.83

Thiosulphate Glycine Leach Conditions – Calcined Pilgrim's Rest

Leach Conditions			
Target Conditions			Actual Reagent required
Solids	%	50	
pH		9.5	
Eh		> -5 (<195)	
Thiosulphate	g/L	62	23.22
Copper Sulphate	g/L	7.5	2.8
Glycine	g/L	3.0	1.12

Bottle Mass	Sample + Bottle	Sample + Bottle + Solution
1411.23	1784.74	1784.58
	Sample	Water
	373.5	409.13

Sample ID : Calcined PG

Time		Initial pH	Initial Eh	Initial T	Final pH	Final Eh	Final T	$Na_2S_2O_3$	NH_3	$CuSO_4$	H2SO4	Bottle Mass
Hours	Minutes		mV	°C		mV	°C	g	g	g	g	g
0	0	6.53	488	26.8	7.82	190	26	23.2826	19.3894	2.8316		2237.98
0	10	8.05	-156	25.6	8.68	-160	26.2		3.0164			2239.02
0	20	8.85	-101.3	24.8								2238.37
1	20	8.85	9.1	24.9	9.21	-70	23.6		14.2347			2250.28
2	20	9.4	-30.6	23.4								2249.89
3	5	9.51	-9.5	23.4								2249.75
4	7	9.54	5.7	23.6								2249.03
6	3	9.59	7.8	23.7								2248.44
8	5	9.59	16.4	23.6								2247.95
10	10	9.72	21.9	23.3	9.58	23.4	24				1.4744	2247.48
12	3	9.64	33.9	23.2	9.54	40.5	23.9				0.8386	2247.6
14	3	9.74	39	24.7	9.22	122.8	27.9				4.4229	2250.88
24	0	10.1	22.7	121.7								2251.1

Titration Data - Calcined Thiosulphate Glycine VCR

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.004	0.027322302	0.0001093

Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Reagent Remaining	% Remaining
Moles	g	g/L	g/L	
0.0001093	0.012254599	2.450919799	5.430975677	10.53478313

Titration Data – Calcined Fairview Thiosulphate Glycine

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.0038	0.0407	0.000155

Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Reagent Remaining	% Remaining
Moles	g	g/L	g/L	
0.000155	0.0174	3.47	7.69	13.9

Titration Data – Calcined Barbrook Thiosulphate Glycine

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.0044	0.0407	0.000179

Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Reagent Remaining	% Remaining

Moles	g	g/L	g/L	
0.000179	0.0201	4.02	8.90	16.3

Titration Data – Calcined Consolidated Murchison Thiosulphate Glycine

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.009	0.0308	0.000277

Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Reagent Remaining	% Remaining
Moles	g	g/L	g/L	
0.000277	0.0311	6.211	13.8	25.0

Titration Data – Calcined Pilgrim's Rest Thiosulphate Glycine

Titrant Volume	Titrant Concentration	Moles of Titrant
L	mol/L	Moles
0.0018	0.0218	0.0000393

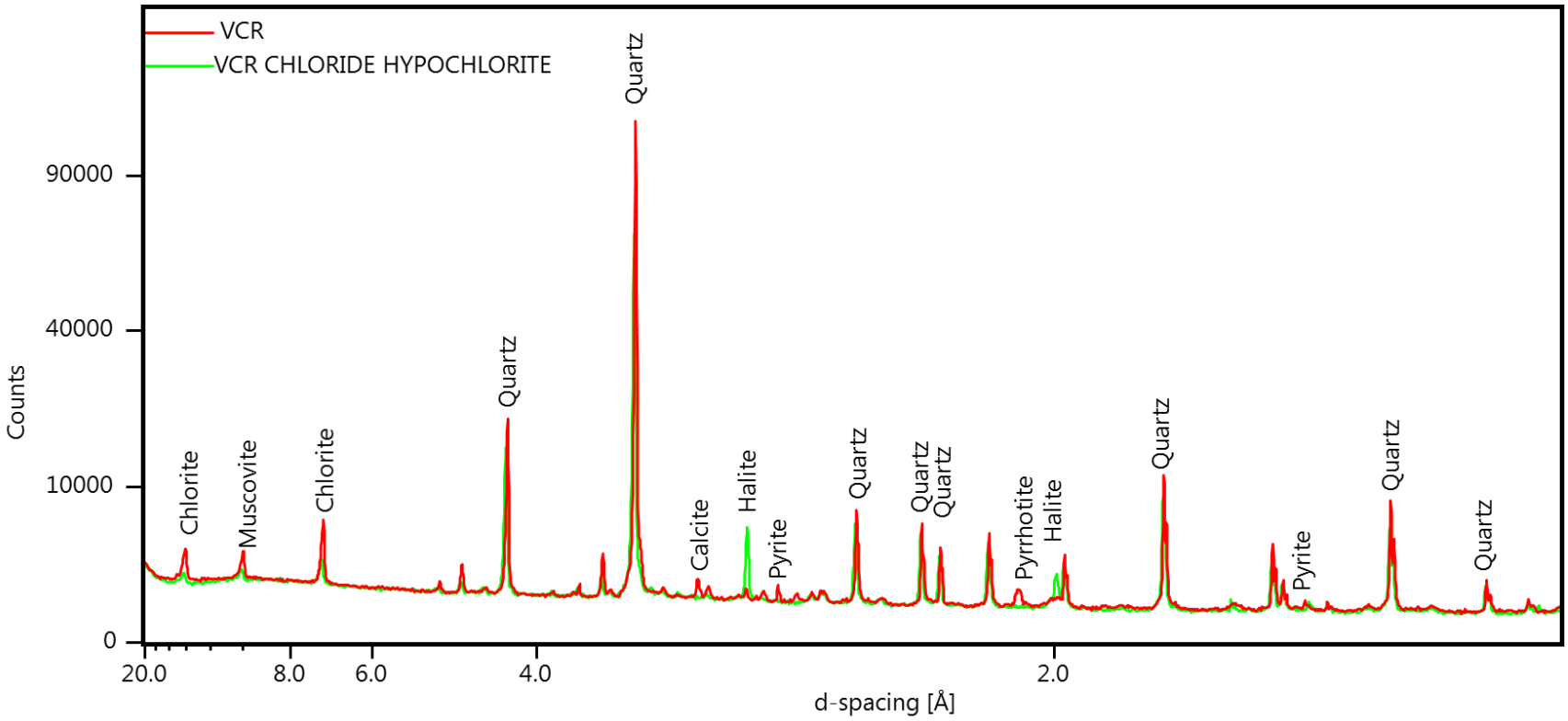
Moles of Thiosulphate	Thiosulphate Mass	Thiosulphate Remaining	Reagent Remaining	% Remaining
Moles	g	g/L	g/L	
0.0000393	0.00440	0.881	1.95	4.190121849

Appendix D – Residue Mineralogy

The comparison of the residue mineralogy, relative to that of the feed, is an important method used to evaluate the leach, as well to understand the processes occurring throughout, this is discussed in Chapter 10.

Chloride-Hypochlorite

XRD data show the mineralogical difference between the VCR feed and chloride-hypochlorite leach residue



Gold Deportment for Chloride-Hypochlorite VCR residue, using data obtained from MLA

Mineral	Wt%	Area%	Area (micron)	Particle Count	Grain Count
Native gold	2.65	0.48	248.96	9	11

Grain size distribution

Sieve Size	Retained Wt%	Cum. Retained Wt%	Cum. Passing Wt%
16	0	0	100
13.5	61.44	61.44	38.56
11.4	0	61.44	38.56
9.6	0	61.44	38.56
8.1	0	61.44	38.56
6.8	0	61.44	38.56
5.7	0	61.44	38.56
4.8	8.23	69.67	30.33
4.1	7.2	76.86	23.14
3.4	5.14	82.01	17.99
2.9	15.42	97.43	2.57
2.4	0	97.43	2.57
2	1.54	98.97	1.03
1.75	1.03	100	0
1.45	0	100	0

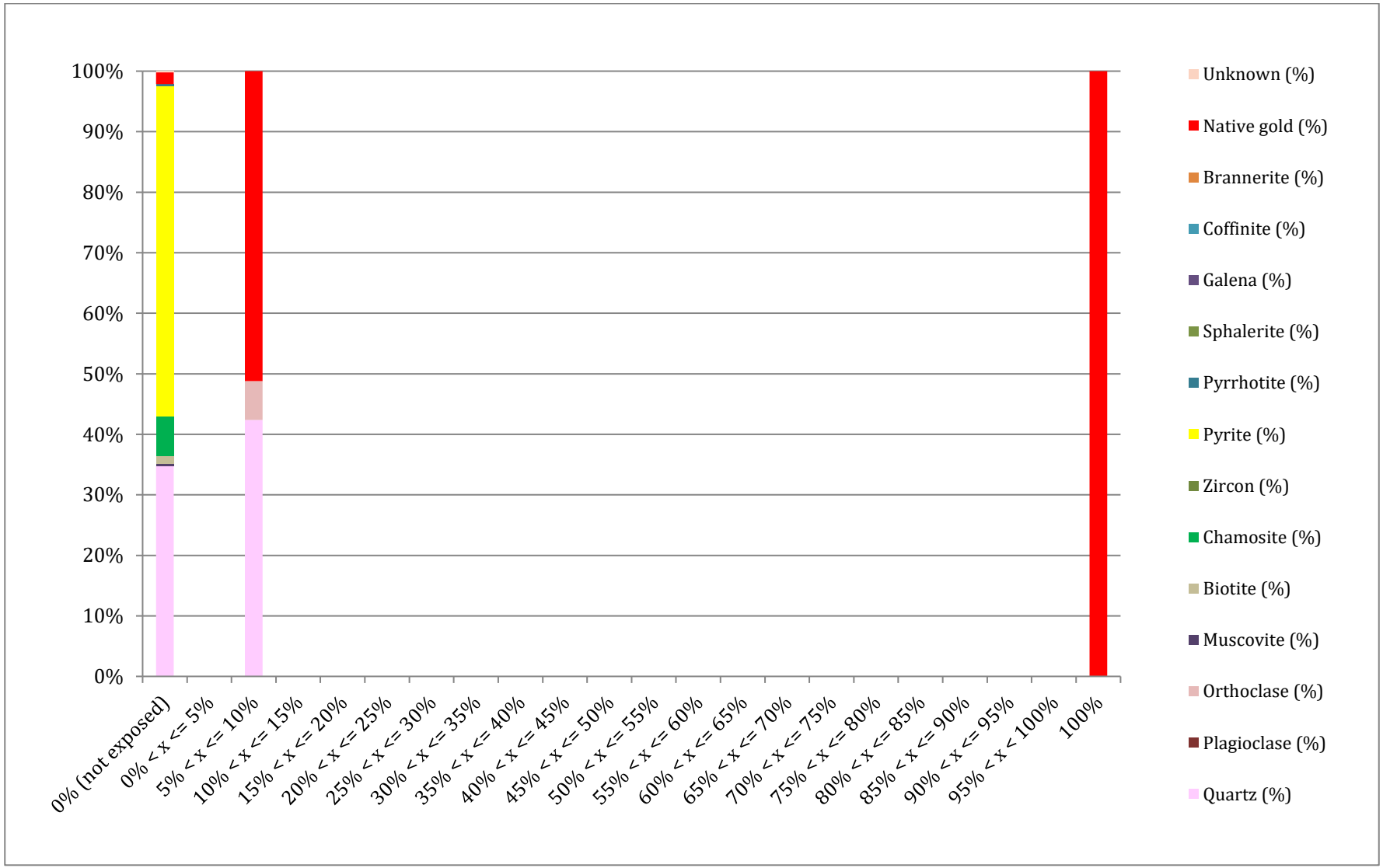
False colour images of gold found in leach residue



Exposure Data

The exposure (percentage of the grain that has a free, exposed, surface) of the gold grains in leach residue

	Barren	0%	0-5%	5-10%	10-15%	15-20%	20-25%	25-30%	30-35%	35-40%	40-45%	45-50%	50-55%	55-60%	60-65%	65-70%	70-75%	75-80%	80-85%	85-90%	90-95%	95-100%	100%
No. of Particles	14	6	0	2	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1
Mean Phase ECD (µm)	0	3.28	0	8.08	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	5.11
Mean Phase Max Span (µm)	0	4.99	0	13.1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	7.55
Particle Distribution (%)	56.77	39.76	0	3.25	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0.22
Mean Density	3.35	3.69	0	4.69	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	19.28
Distribution of Mineral (%)	0	28.79	0	62.98	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	8.23
Cum. Distn. of Mineral (%)	0	100	71.21	71.21	8.23	8.23	8.23	8.23	8.23	8.23	8.23	8.23	8.23	8.23	8.23	8.23	8.23	8.23	8.23	8.23	8.23	8.23	8.23
Modal Analysis (%)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Quartz (%)	40.72	34.7	0	42.36	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Plagioclase (%)	0.1	0.05	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Orthoclase (%)	1.37	0.01	0	6.41	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Muscovite (%)	0.08	0.38	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Biotite (%)	0.01	1.26	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Chamosite (%)	1.2	6.57	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Zircon (%)	2.79	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Pyrite (%)	50.08	54.45	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Pyrrhotite (%)	0.2	0.38	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Sphalerite (%)	0.01	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Galena (%)	0.4	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Gersdorffite (%)	0	0.08	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Coffinite (%)	0.03	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Brannerite (%)	0.47	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Native gold (%)	0	1.92	0	51.23	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	100
Unknown (%)	2.54	0.22	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0


100%
90%
80%
70%
60%
50%
40%
30%
20%
10%
0%
0% (not exposed)
0% < x <= 5%
5% < x <= 10%
10% < x <= 15%
15% < x <= 20%
20% < x <= 25%
25% < x <= 30%
30% < x <= 35%
35% < x <= 40%
40% < x <= 45%
45% < x <= 50%
50% < x <= 55%
55% < x <= 60%
60% < x <= 65%
65% < x <= 70%
70% < x <= 75%
75% < x <= 80%
80% < x <= 85%
85% < x <= 90%
90% < x <= 95%
95% < x < 100%
100%
Unknown (%)
Native gold (%)
Brannerite (%)
Coffinite (%)
Galena (%)
Sphalerite (%)
Pyrrhotite (%)
Pyrite (%)
Zircon (%)
Chamosite (%)
Biotite (%)
Muscovite (%)
Orthoclase (%)
Plagioclase (%)
Quartz (%)

Association Data

Minerals that are associated with the gold particles found in the leach residue.

Mineral	Quartz	Plagioclase	Orthoclase	Muscovite	Biotite	Chamosite	Zircon	Pyrite	Pyrrhotite	Sphalerite	Galena	Gersdorffite	Coffinite	Brannerite	Native gold	Unknown	Free Surface
Quartz	0	0.39	0.61	0.87	0.33	10.74	0	1.79	1.03	0	1.12	0	0	0	1.99	4.76	76.38
Plagioclase	25.51	0	0	0	0	13.27	0	17.35	0	0	0	0	0	0	0	0	43.88
Orthoclase	12.04	0	0	2.47	0	0	0	32.72	0	0	0	0	0	0	3.7	0	49.07
Muscovite	34.36	0	4.91	0	0	0	0	9.2	0	0	0	0	0	0	0	0	51.53
Biotite	6.54	0	0	0	0	0	0	40.5	0	0	0	0	0	0	0	2.49	50.47
Chamosite	56.98	1.07	0	0	0	0	2.15	6.52	0	0	0	0	0	0	1.98	0	31.3
Clinochlore	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Titanite	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Zircon	0	0	0	0	0	12.62	0	0	0	0	0	0	0	0	0	0	87.38
Pyrite	4.03	0.6	3.71	0.53	4.55	2.77	0	0	0.67	0.28	1.47	0.7	0	0	1.89	3.57	75.24
Pyrrhotite	55.93	0	0	0	0	0	0	16.1	0	0	0	0	0	0	0	17.8	10.17
Chalcopyrite	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Arsenopyrite	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Pentlandite	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Sphalerite	0	0	0	0	0	0	0	100	0	0	0	0	0	0	0	0	0
Galena	56.25	0	0	0	0	0	0	32.81	0	0	0	0	0	0	0	10.94	0
Gersdorffite	0	0	0	0	0	0	0	100	0	0	0	0	0	0	0	0	0
Native gold	45.39	0	4.26	0	0	8.51	0	19.15	0	0	0	0	0	0	0	0	22.7
Unknown	34.08	0	0	0	0.89	0	0	11.36	2.34	0	1.56	0	1.56	6.01	0	0	42.2

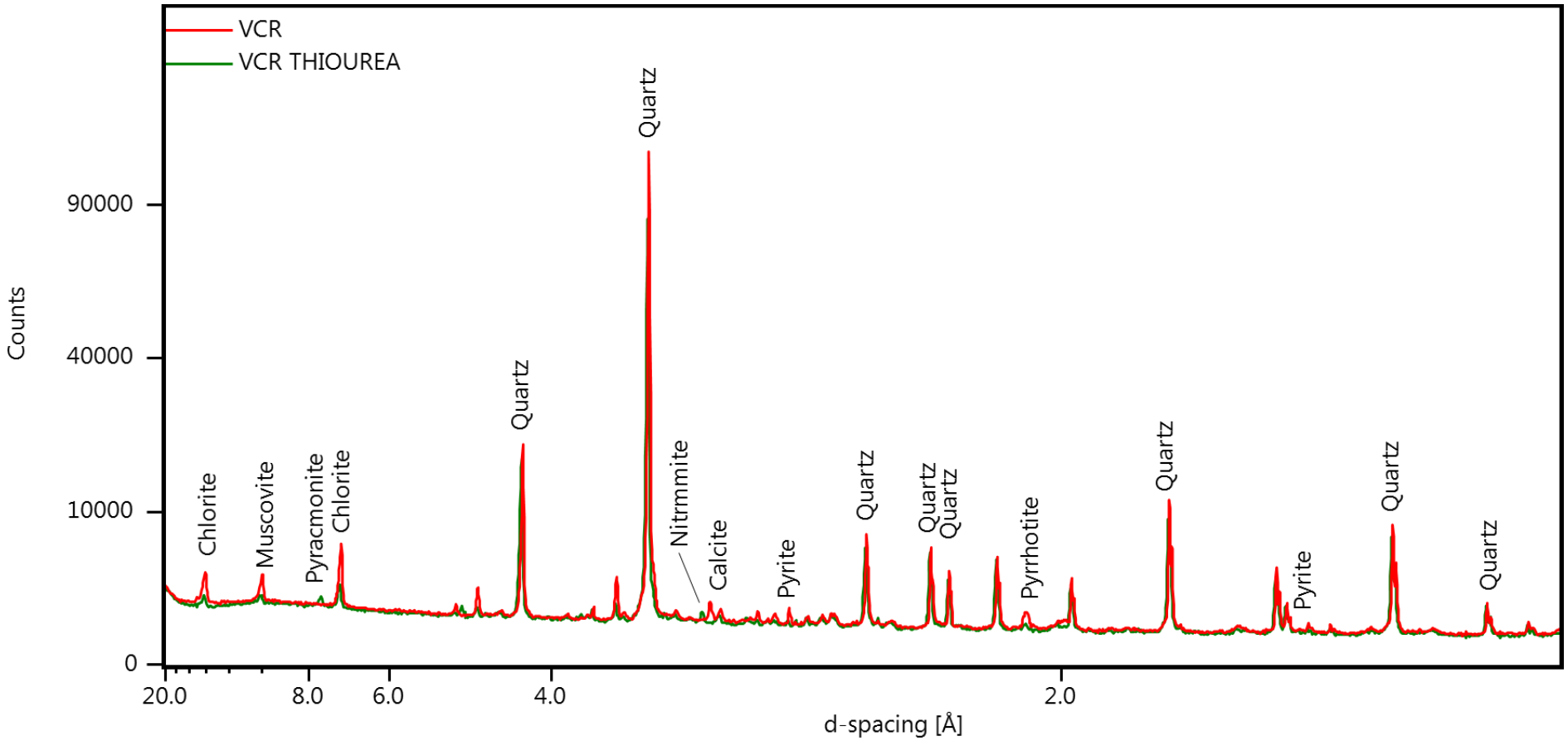
Liberation

Liberation is defined by the area % of the gold relative to the area of the total host particle within the leach residue.

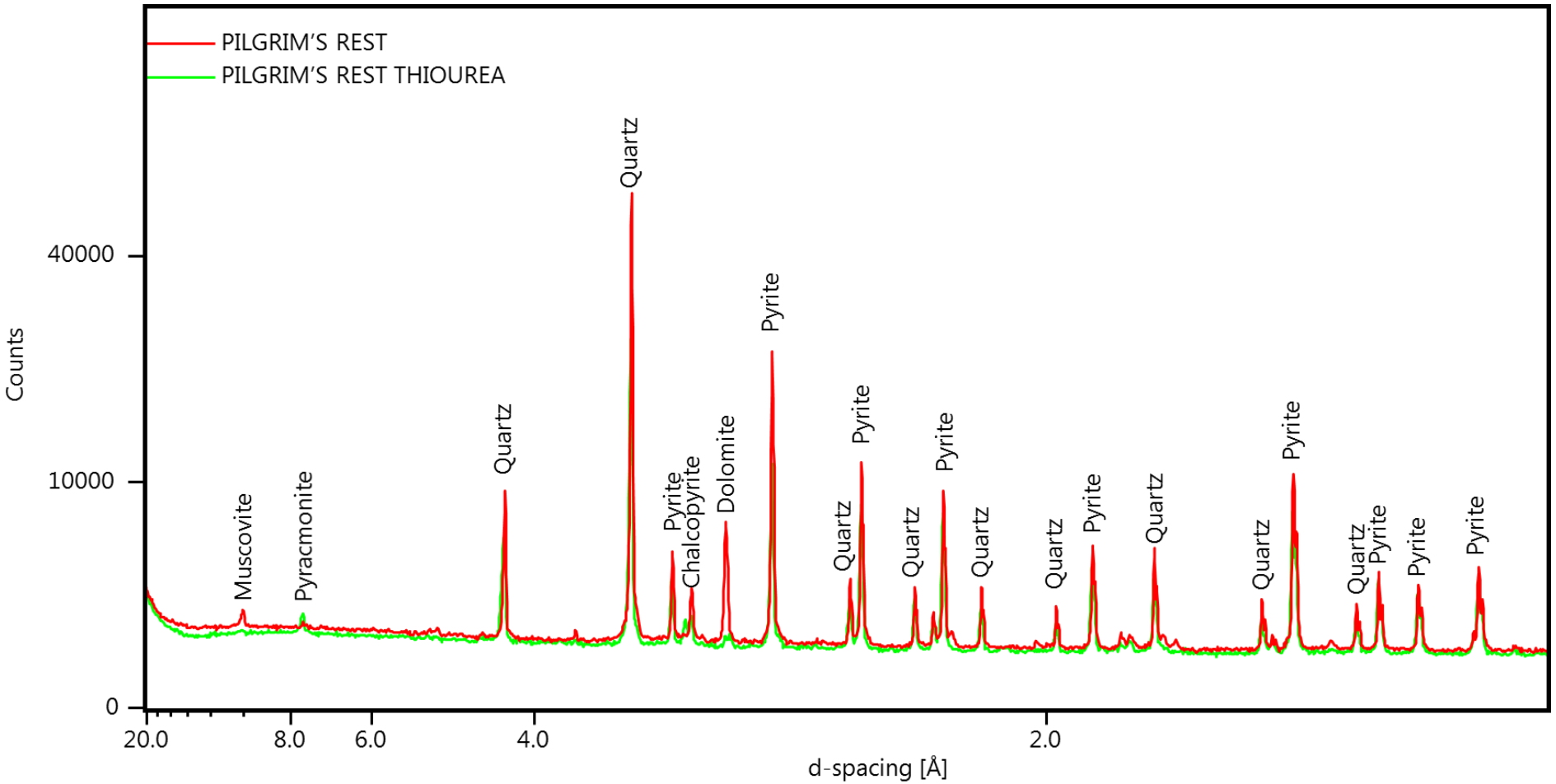
	0%	0-5%	5-10%	10-15%	15-20%	20-25%	25-30%	30-35%	35-40%	40-45%	45-50%	50-55%	55-60%	60-65%	65-70%	70-75%	75-80%	80-85%	85-90%	90-95%	95-100%	100%
No. of Particles	14	3	2	0	0	2	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0	1
Mean Phase ECD (µm)	0	2.89	3.52	0	0	3.49	0	0	0	0	0	13.96	0	0	0	0	0	0	0	0	0	5.11
Mean Phase Max Span (µm)	0	4.52	5.2	0	0	4.88	0	0	0	0	0	23.32	0	0	0	0	0	0	0	0	0	7.55
Particle Distribution (%)	56.77	36.72	2.19	0	0	1.02	0	0	0	0	0	3.08	0	0	0	0	0	0	0	0	0	0.22
Mean Density	3.35	3.74	3.19	0	0	3.26	0	0	0	0	0	4.81	0	0	0	0	0	0	0	0	0	19.28
Distribution of Mineral (%)	0	13.62	7.97	0	0	8.74	0	0	0	0	0	61.44	0	0	0	0	0	0	0	0	0	8.23
Cum. Distn. of Mineral (%)	0	100	86.38	78.41	78.41	78.41	69.67	69.67	69.67	69.67	69.67	69.67	8.23	8.23	8.23	8.23	8.23	8.23	8.23	8.23	8.23	8.23
Modal Analysis (Wt%)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Quartz (%)	40.72	34.5	21.35	0	0	77.4	0	0	0	0	0	40.48	0	0	0	0	0	0	0	0	0	0
Plagioclase (%)	0.1	0	0.82	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Orthoclase (%)	1.37	0.01	0	0	0	0	0	0	0	0	0	6.77	0	0	0	0	0	0	0	0	0	0
Muscovite (%)	0.08	0.41	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Biotite (%)	0.01	1.36	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Chamosite (%)	1.2	3.65	57.97	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Zircon (%)	2.79	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Pyrite (%)	50.08	58.93	0.64	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Pyrrhotite (%)	0.2	0	6.9	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Sphalerite (%)	0.01	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Galena (%)	0.4	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Gersdorffite (%)	0	0.09	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Coffinite (%)	0.03	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Brannerite (%)	0.47	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Native gold (%)	0	0.98	9.61	0	0	22.6	0	0	0	0	0	52.75	0	0	0	0	0	0	0	0	0	100
Unknown (%)	2.54	0.07	2.7	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0

Thiourea

XRD data show the mineralogical difference between the VCR feed and thiourea leach residue

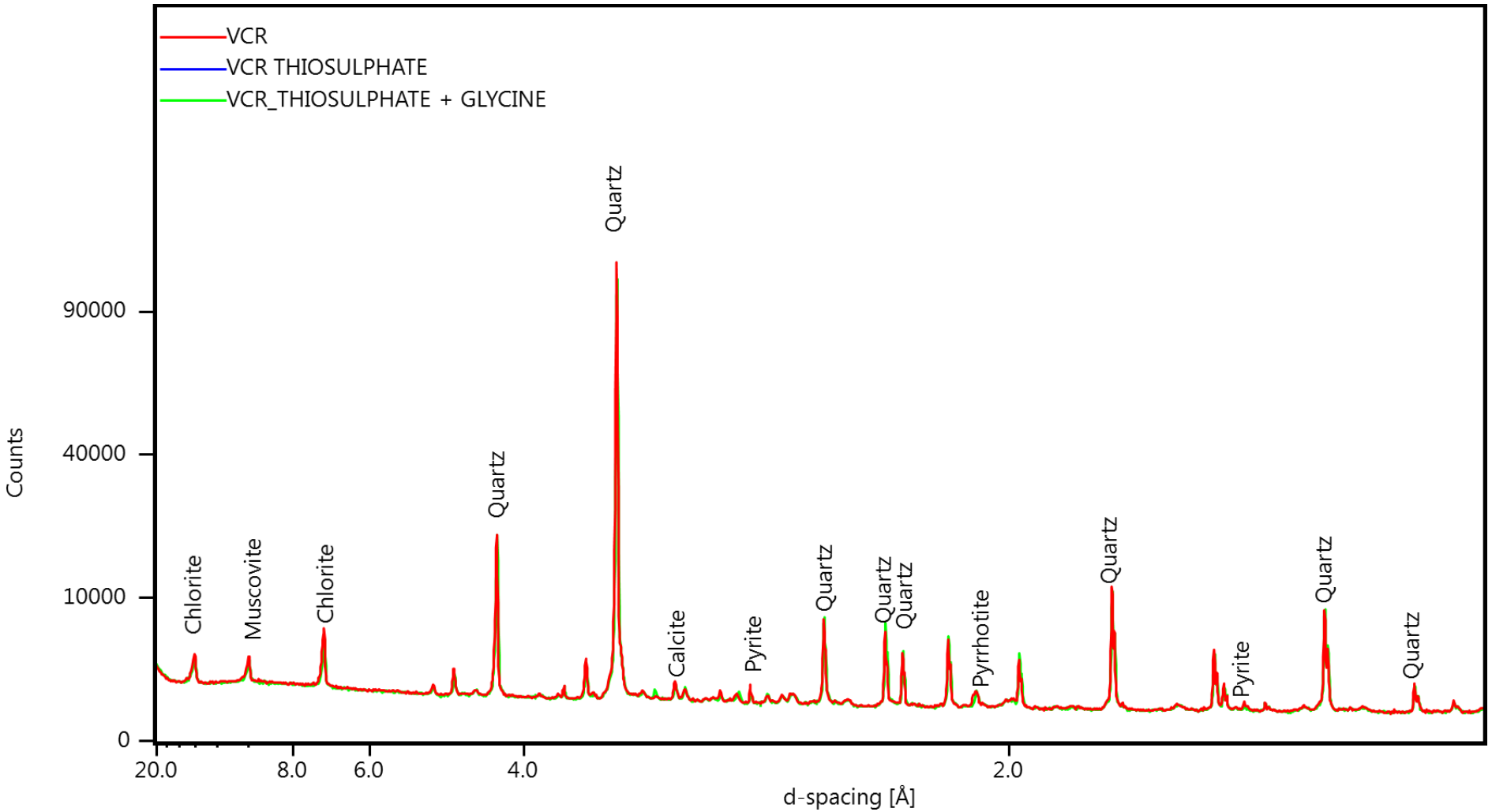


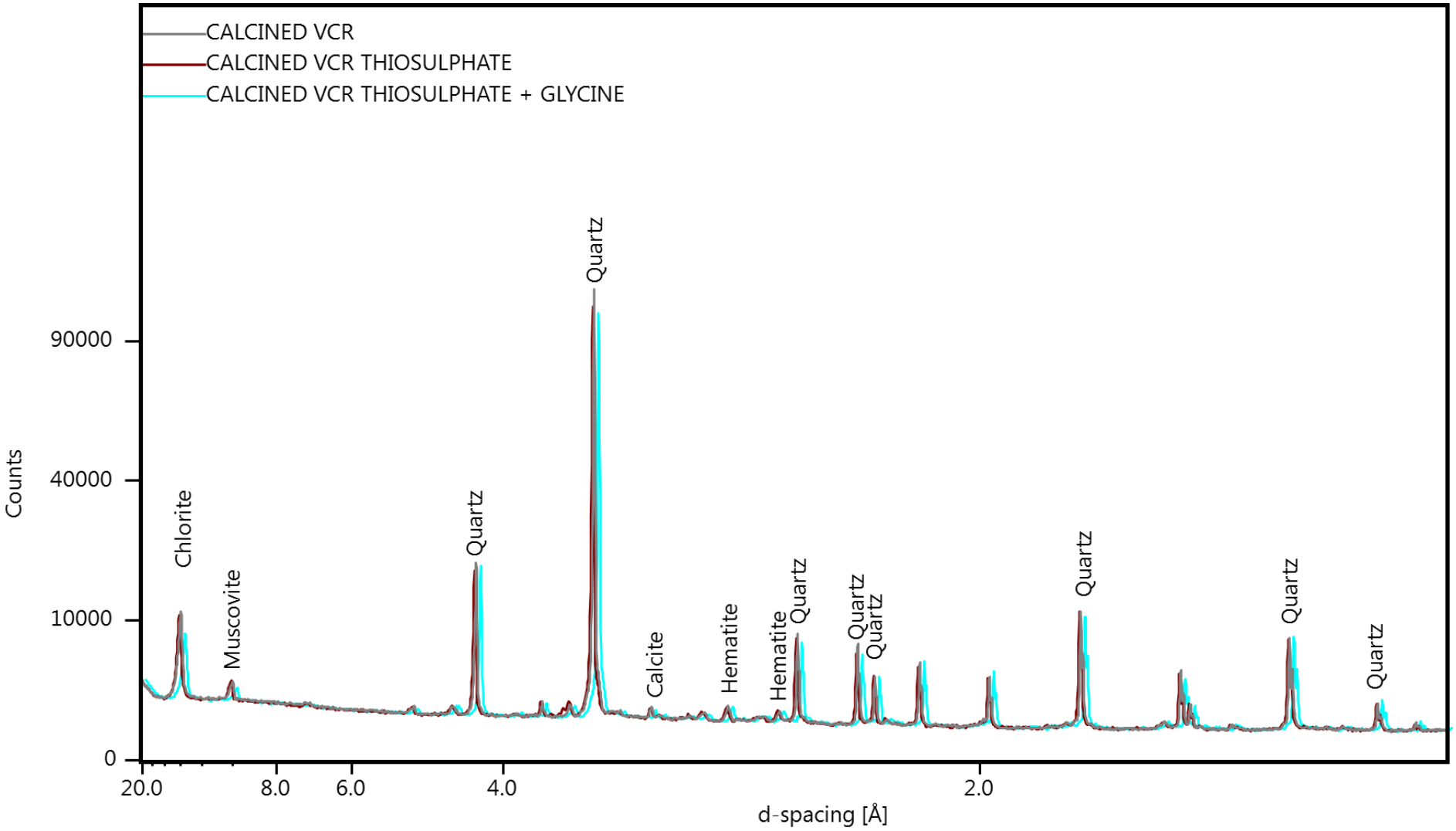
XRD data show the mineralogical difference between the Pilgrim's Rest feed and thiourea leach residue

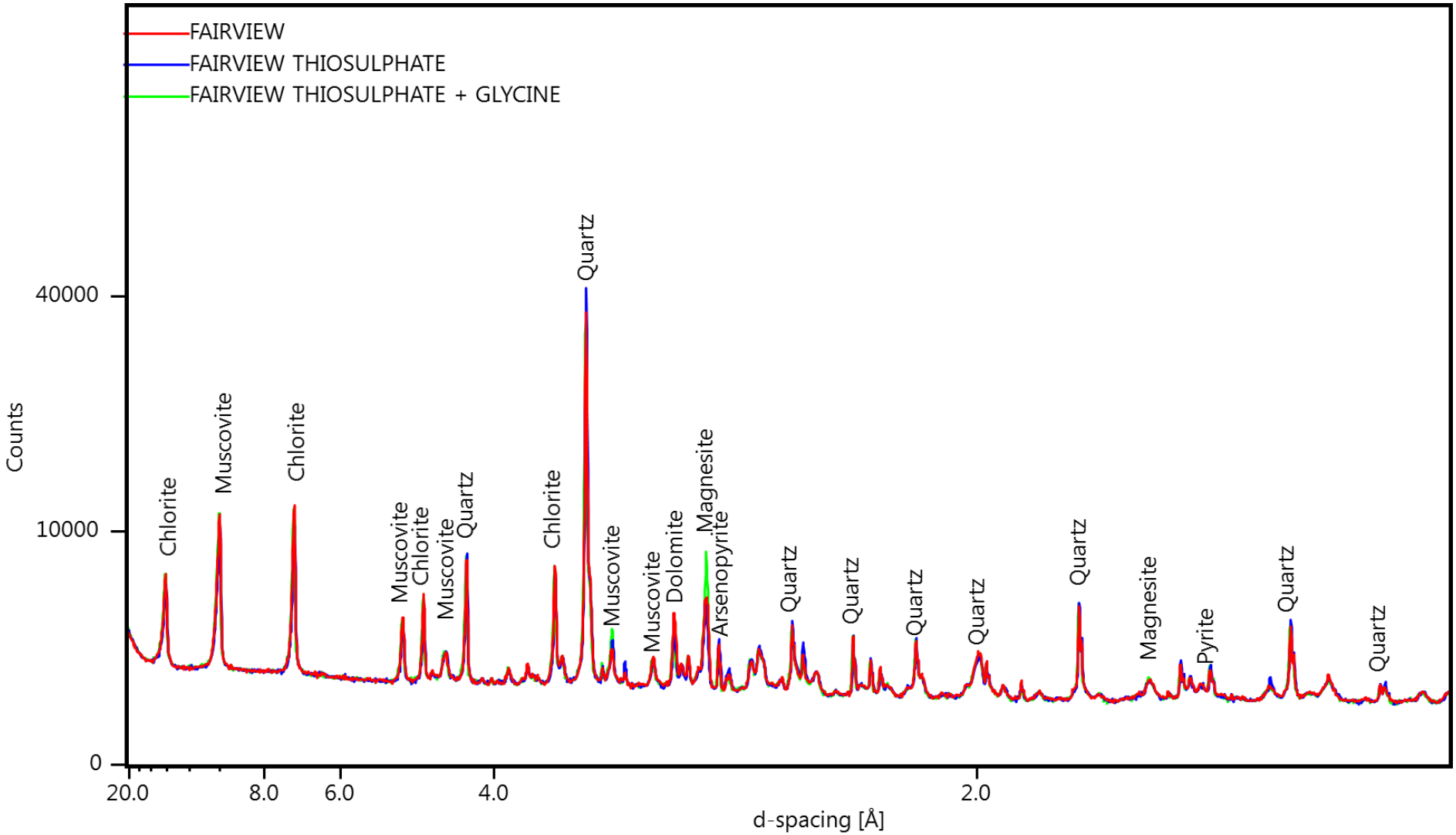


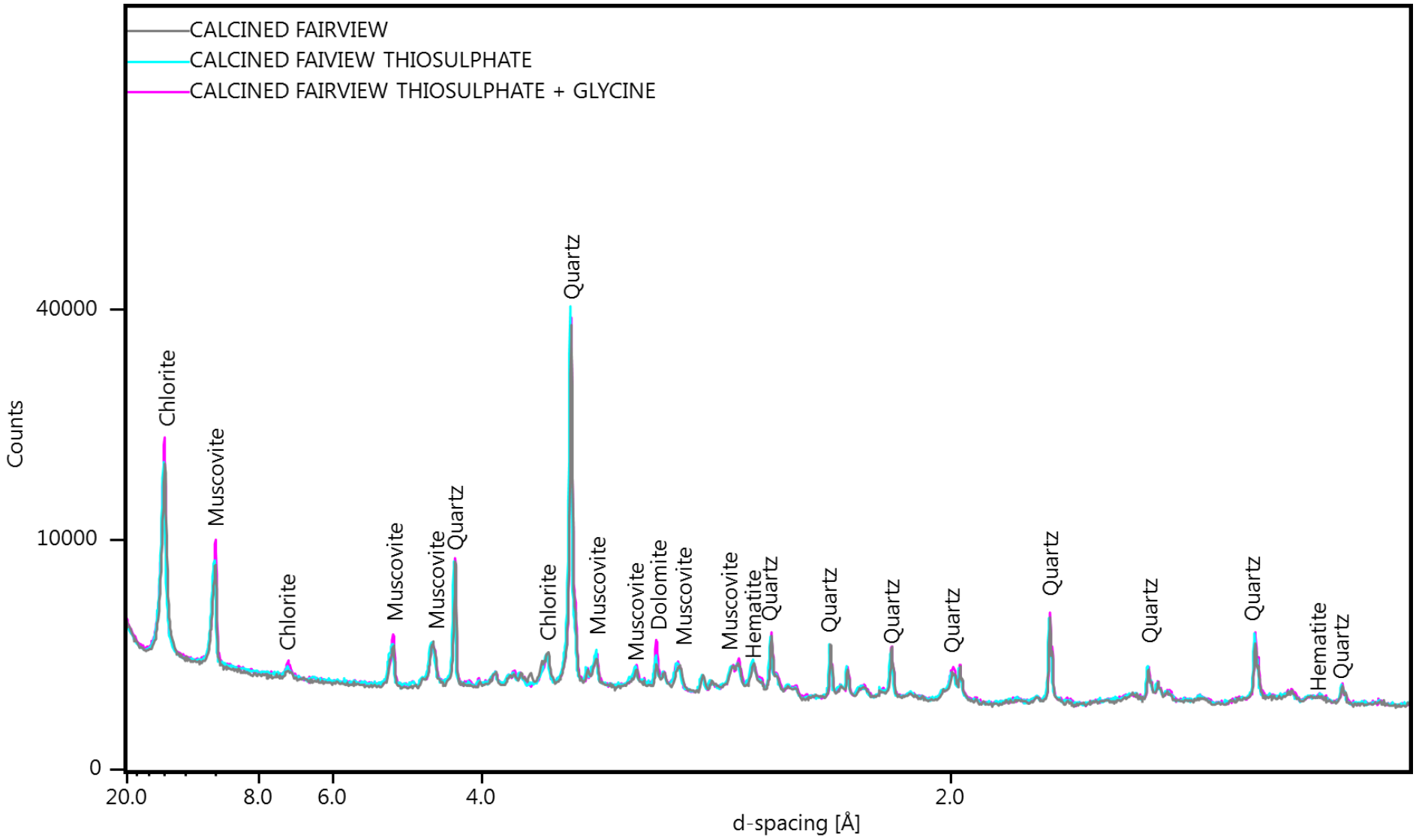
Thiosulphate

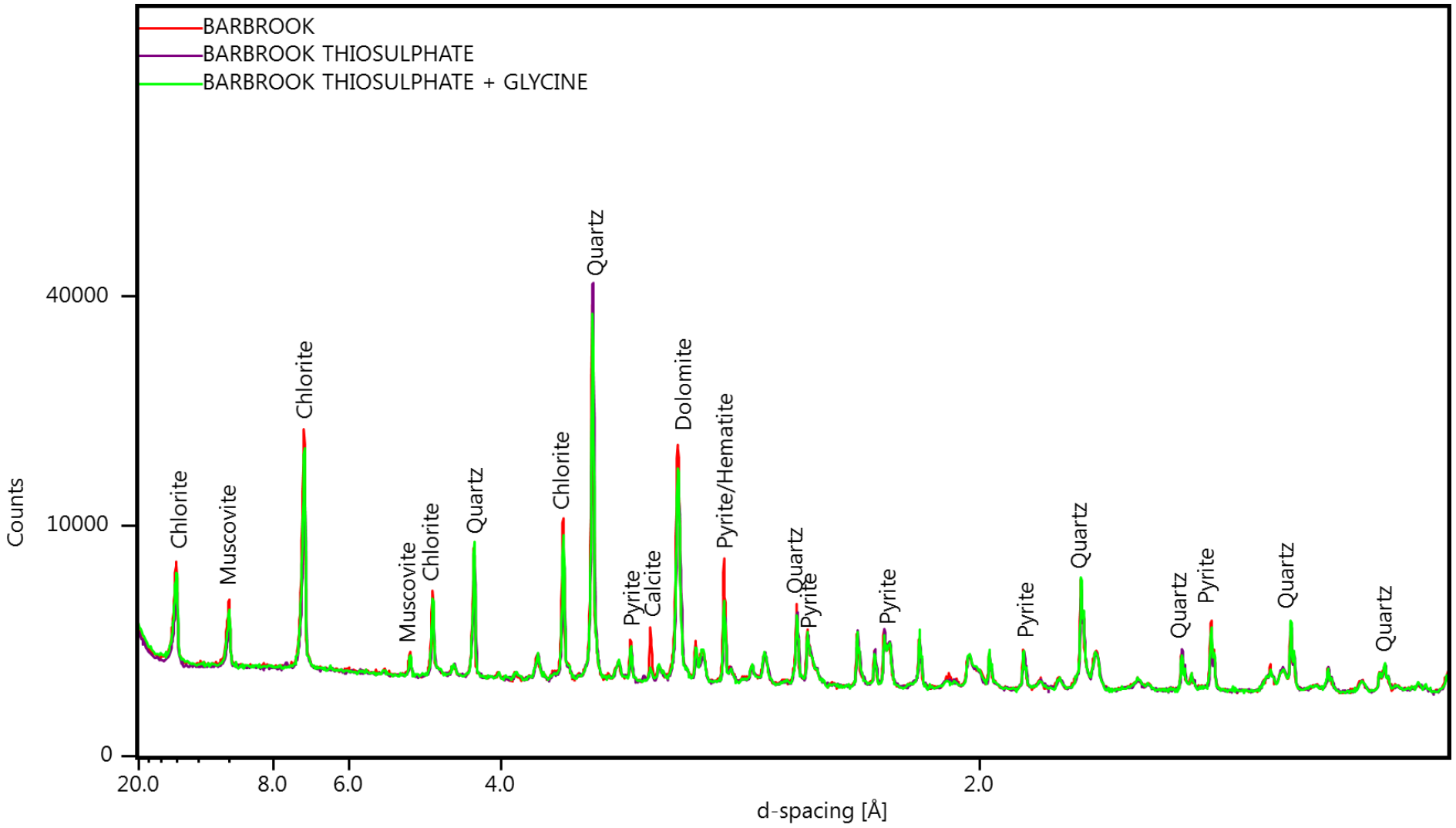
XRD data show the mineralogical difference between the feed and thiosulphate leach residue

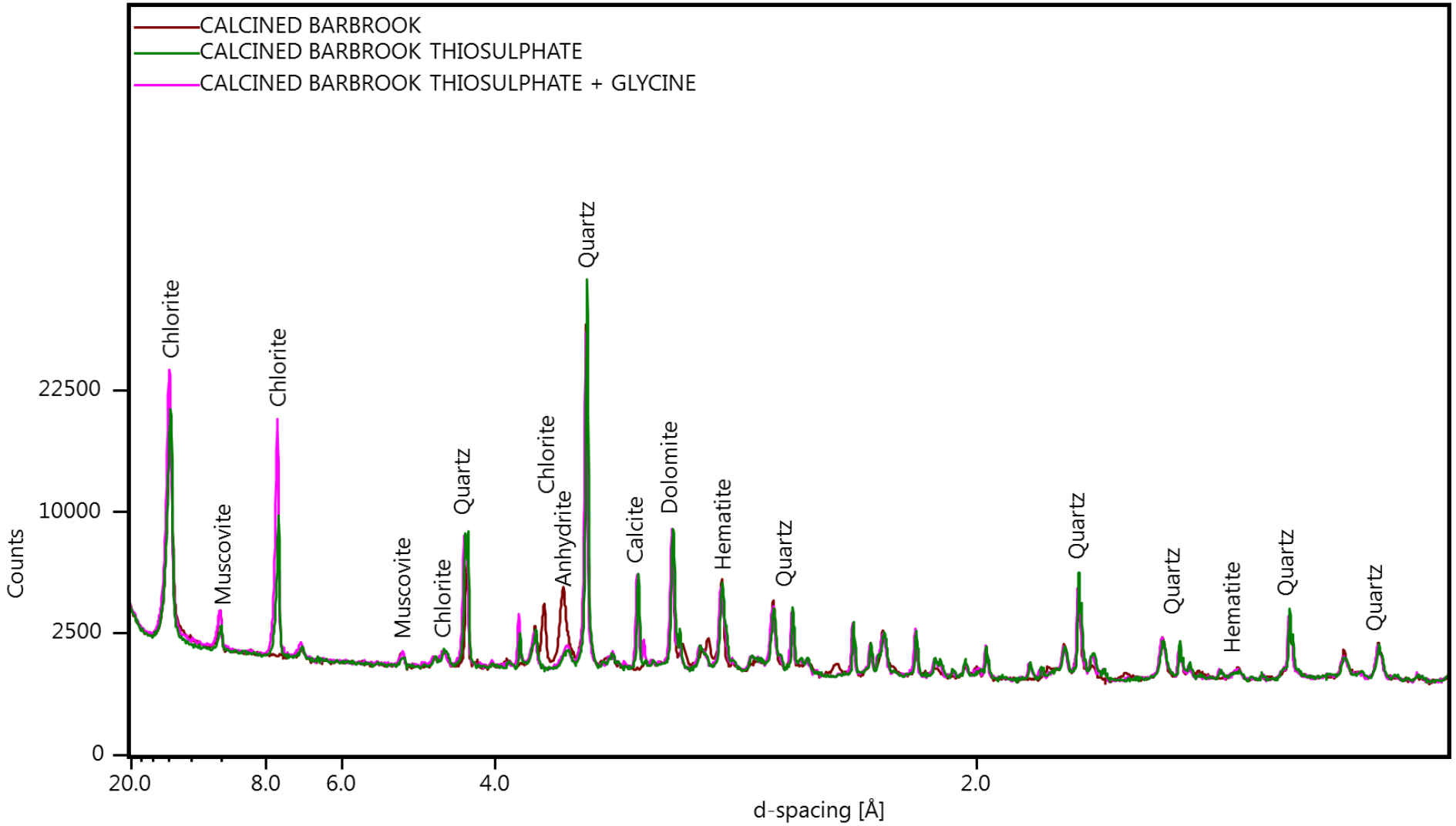


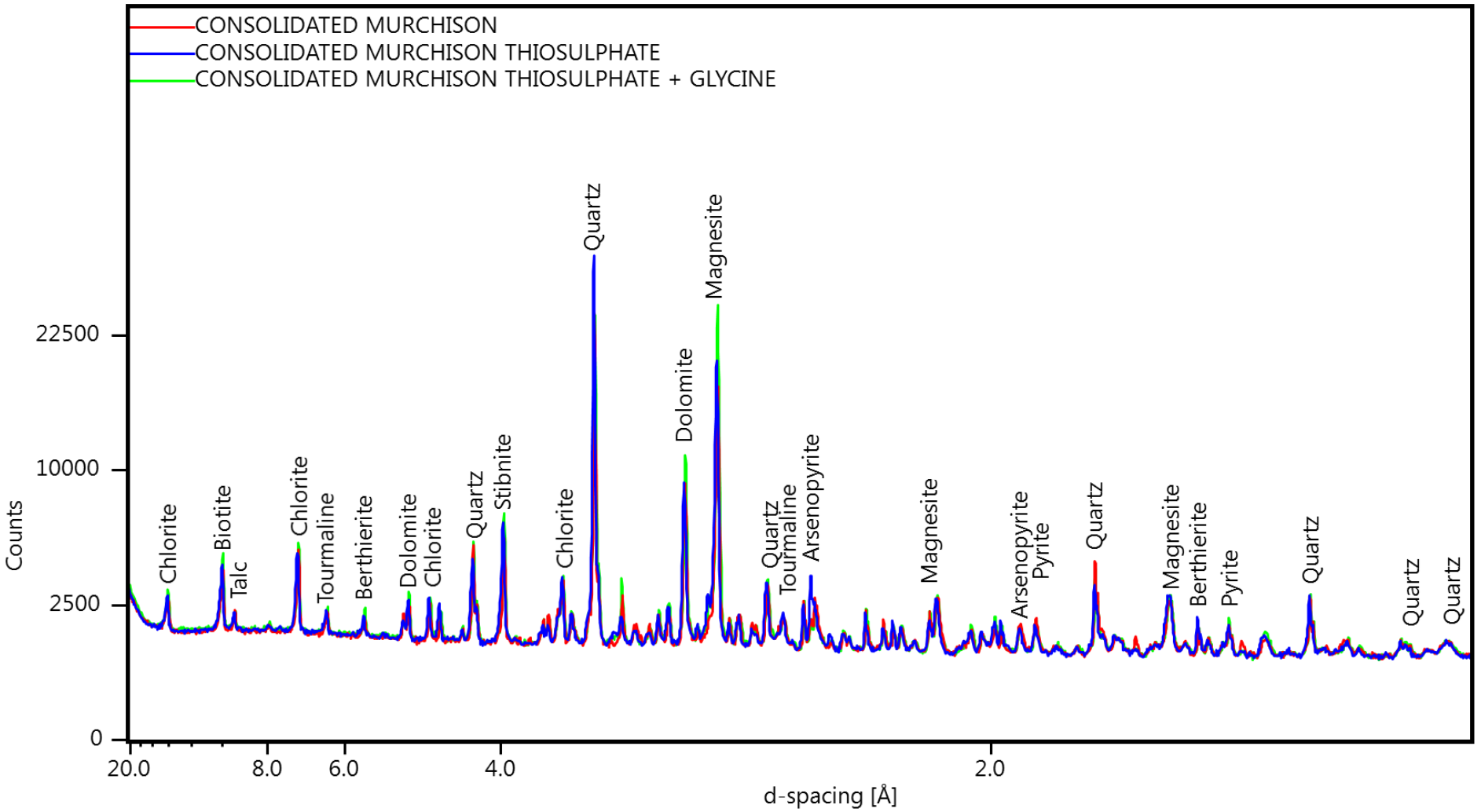

CALCINED VCR
CALCINED VCR THIOSULPHATE
CALCINED VCR THIOSULPHATE + GLYCINE
Counts
90000
40000
10000
0
20.0
8.0
6.0
4.0
2.0
d-spacing [Å]
Chlorite
Muscovite
Quartz
Quartz
Quartz
Calcite
Hematite
Hematite
Quartz
Quartz
Quartz
Quartz
Quartz
Quartz

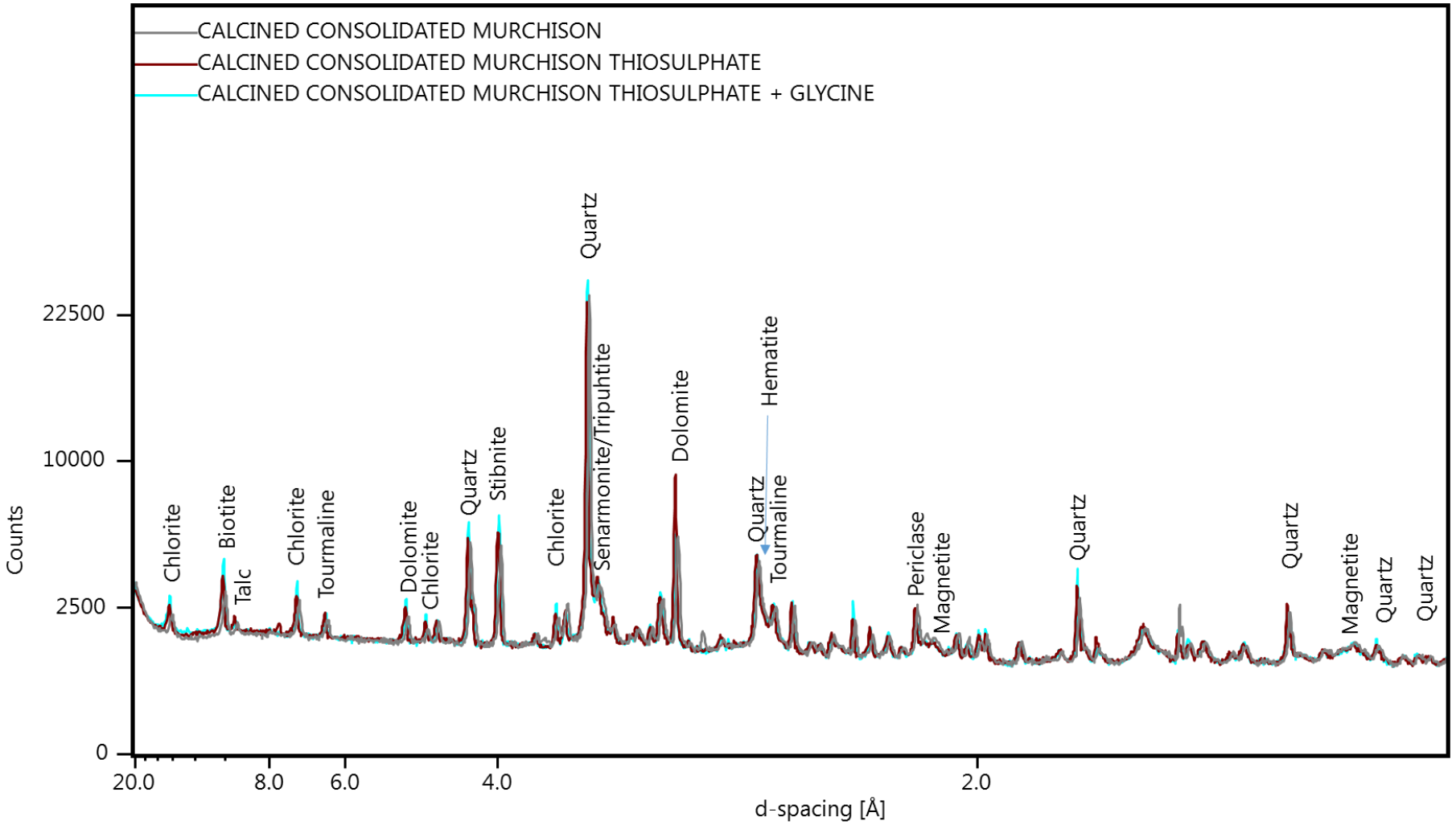

FAIRVIEW
FAIRVIEW THIOSULPHATE
FAIRVIEW THIOSULPHATE + GLYCINE
Counts
40000
10000
0
20.0
8.0
6.0
4.0
2.0
d-spacing [Å]
Chlorite
Muscovite
Chlorite
Muscovite
Chlorite
Muscovite
Quartz
Chlorite
Quartz
Muscovite
Muscovite
Dolomite
Magnesite
Arsenopyrite
Quartz
Quartz
Quartz
Quartz
Quartz
Magnesite
Pyrite
Quartz
Quartz

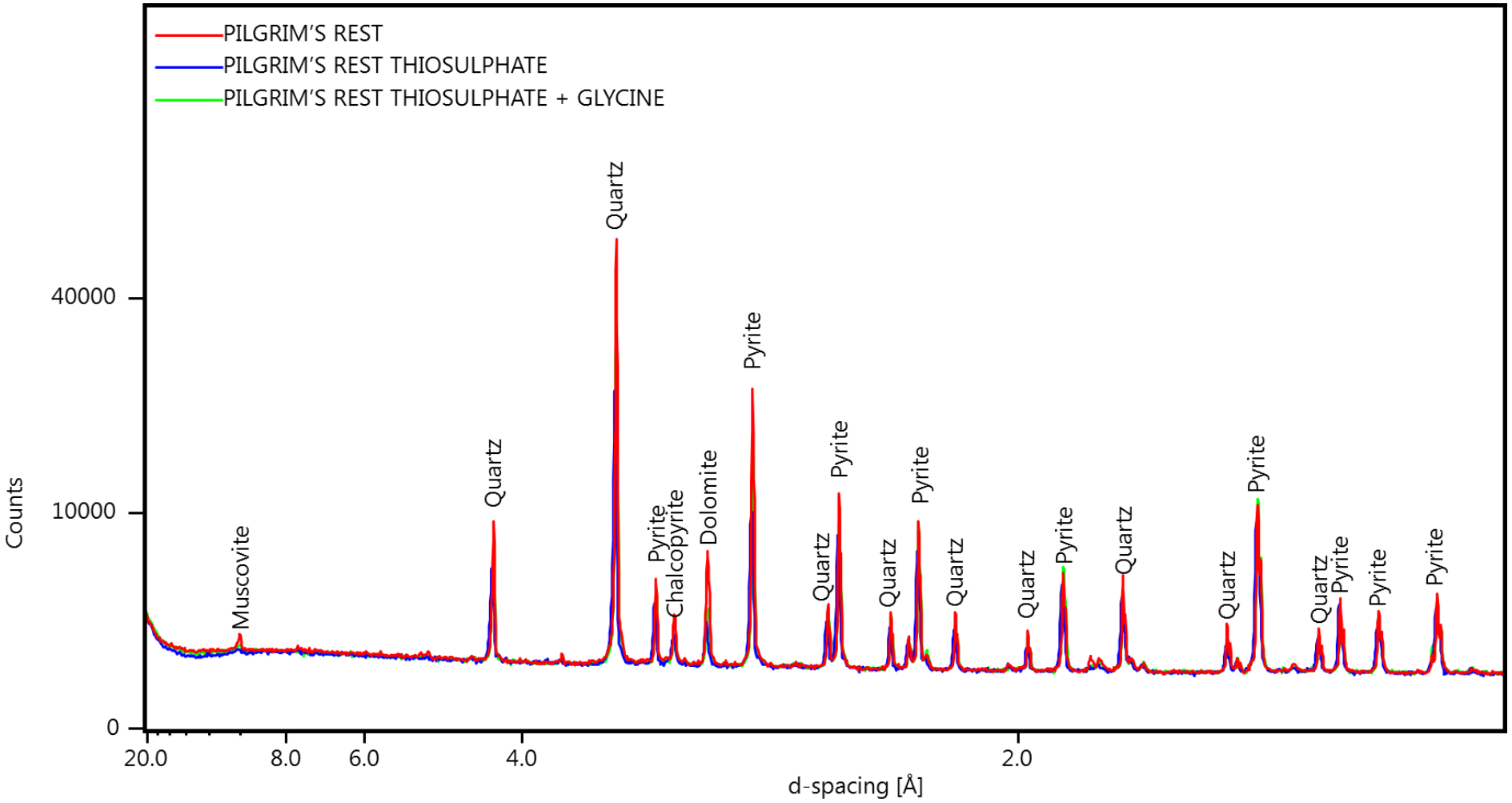

CALCINED FAIRVIEW
CALCINED FAIVIEW THIOSULPHATE
CALCINED FAIRVIEW THIOSULPHATE + GLYCINE
Counts
40000
10000
0
20.0
8.0
6.0
4.0
2.0
d-spacing [Å]
Chlorite
Muscovite
Chlorite
Muscovite
Muscovite
Quartz
Quartz
Chlorite
Muscovite
Muscovite
Dolomite
Muscovite
Muscovite
Hematite
Quartz
Quartz
Quartz
Quartz
Quartz
Quartz
Quartz
Hematite
Quartz

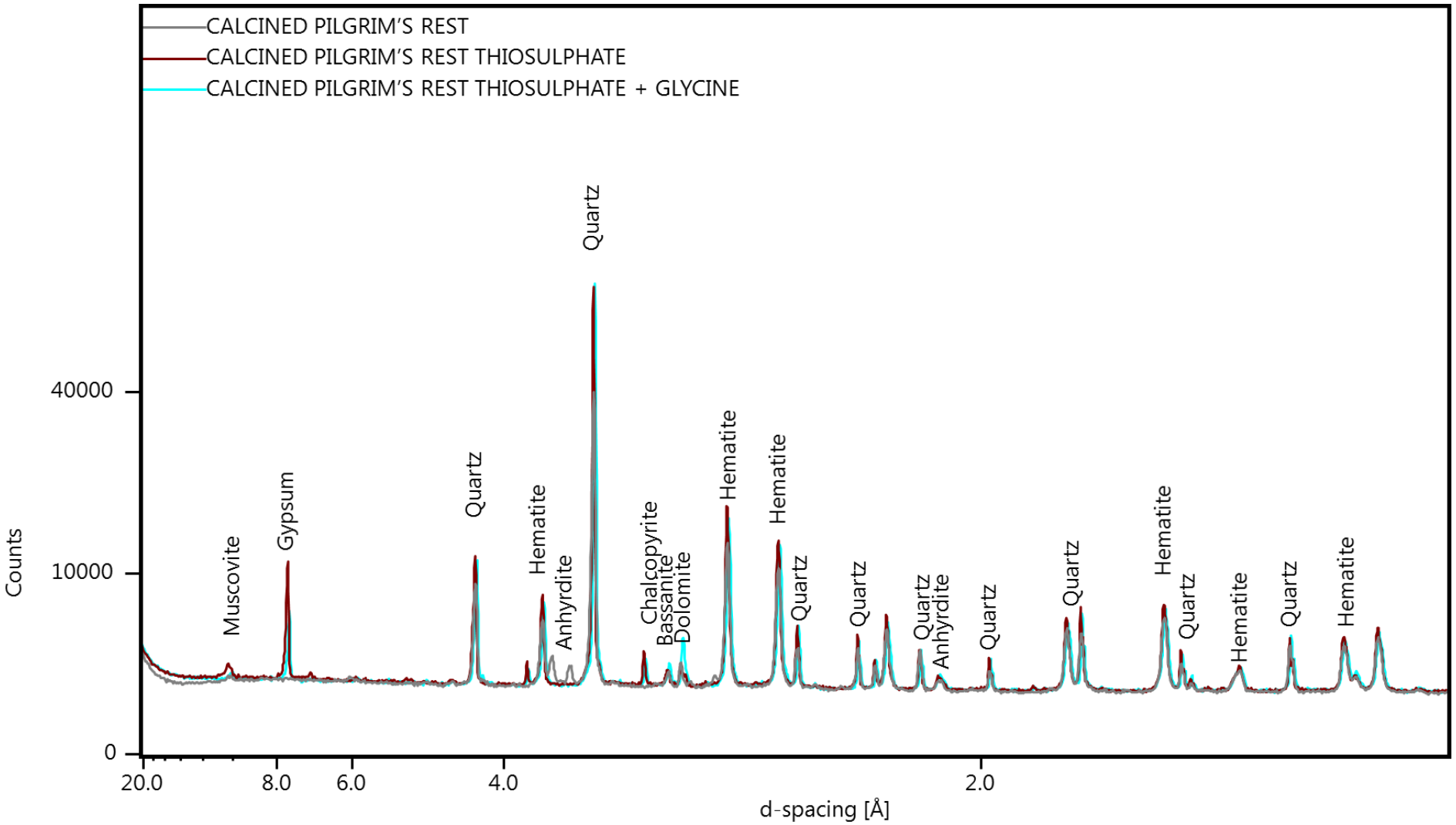

BARBROOK
BARBROOK THIOSULPHATE
BARBROOK THIOSULPHATE + GLYCINE
Counts
40000
10000
0
20.0
8.0
6.0
4.0
2.0
d-spacing [Å]
Chlorite
Muscovite
Chlorite
Muscovite
Chlorite
Quartz
Chlorite
Quartz
Pyrite
Calcite
Dolomite
Pyrite/Hematite
Quartz
Pyrite
Pyrite
Pyrite
Quartz
Quartz
Pyrite
Quartz
Quartz


CALCINED BARBROOK
CALCINED BARBROOK THIOSULPHATE
CALCINED BARBROOK THIOSULPHATE + GLYCINE
Counts
22500
10000
2500
0
20.0
8.0
6.0
4.0
2.0
d-spacing [Å]
Chlorite
Muscovite
Chlorite
Muscovite
Chlorite
Quartz
Chlorite
Anhydrite
Quartz
Calcite
Dolomite
Hematite
Quartz
Quartz
Quartz
Hematite
Quartz
Quartz


CONSOLIDATED MURCHISON
CONSOLIDATED MURCHISON THIOSULPHATE
CONSOLIDATED MURCHISON THIOSULPHATE + GLYCINE
Counts
22500
10000
2500
0
20.0
8.0
6.0
4.0
2.0
d-spacing [Å]
Chlorite
Biotite
Talc
Chlorite
Tourmaline
Berthierite
Dolomite
Chlorite
Quartz
Stibnite
Chlorite
Quartz
Dolomite
Magnesite
Quartz
Tourmaline
Arsenopyrite
Magnesite
Arsenopyrite
Pyrite
Quartz
Magnesite
Berthierite
Pyrite
Quartz
Quartz
Quartz


CALCINED CONSOLIDATED MURCHISON
CALCINED CONSOLIDATED MURCHISON THIOSULPHATE
CALCINED CONSOLIDATED MURCHISON THIOSULPHATE + GLYCINE
Counts
22500
10000
2500
0
20.0
8.0
6.0
4.0
2.0
d-spacing [Å]
Chlorite
Biotite
Talc
Chlorite
Tourmaline
Dolomite
Chlorite
Quartz
Stibnite
Chlorite
Quartz
Senarmonite/Tripuhtite
Dolomite
Hematite
Quartz
Tourmaline
Periclase
Magnetite
Quartz
Quartz
Magnetite
Quartz
Quartz


PILGRIM'S REST
PILGRIM'S REST THIOSULPHATE
PILGRIM'S REST THIOSULPHATE + GLYCINE
Counts
40000
10000
0
20.0
8.0
6.0
4.0
2.0
d-spacing [Å]
Muscovite
Quartz
Quartz
Pyrite
Chalcopyrite
Dolomite
Pyrite
Quartz
Pyrite
Quartz
Pyrite
Quartz
Quartz
Pyrite
Quartz
Quartz
Pyrite
Quartz
Pyrite
Pyrite
Pyrite


CALCINED PILGRIM'S REST
CALCINED PILGRIM'S REST THIOSULPHATE
CALCINED PILGRIM'S REST THIOSULPHATE + GLYCINE
Counts
40000
10000
0
20.0
8.0
6.0
4.0
2.0
d-spacing [Å]
Muscovite
Gypsum
Quartz
Hematite
Anhyrdite
Quartz
Chalcopyrite
Bassanite
Dolomite
Hematite
Hematite
Quartz
Quartz
Quartz
Anhyrdite
Quartz
Quartz
Hematite
Quartz
Hematite
Quartz
Hematite

MLA results - Glycine Thiosulphate VCR Residue

The MLA deportment, of the residue of the VCR sample that underwent glycine thiosulphate leaching, located 241 particles and 509 grains of gold.

Grain size distribution

The grain size distribution of the gold within the leach residue.

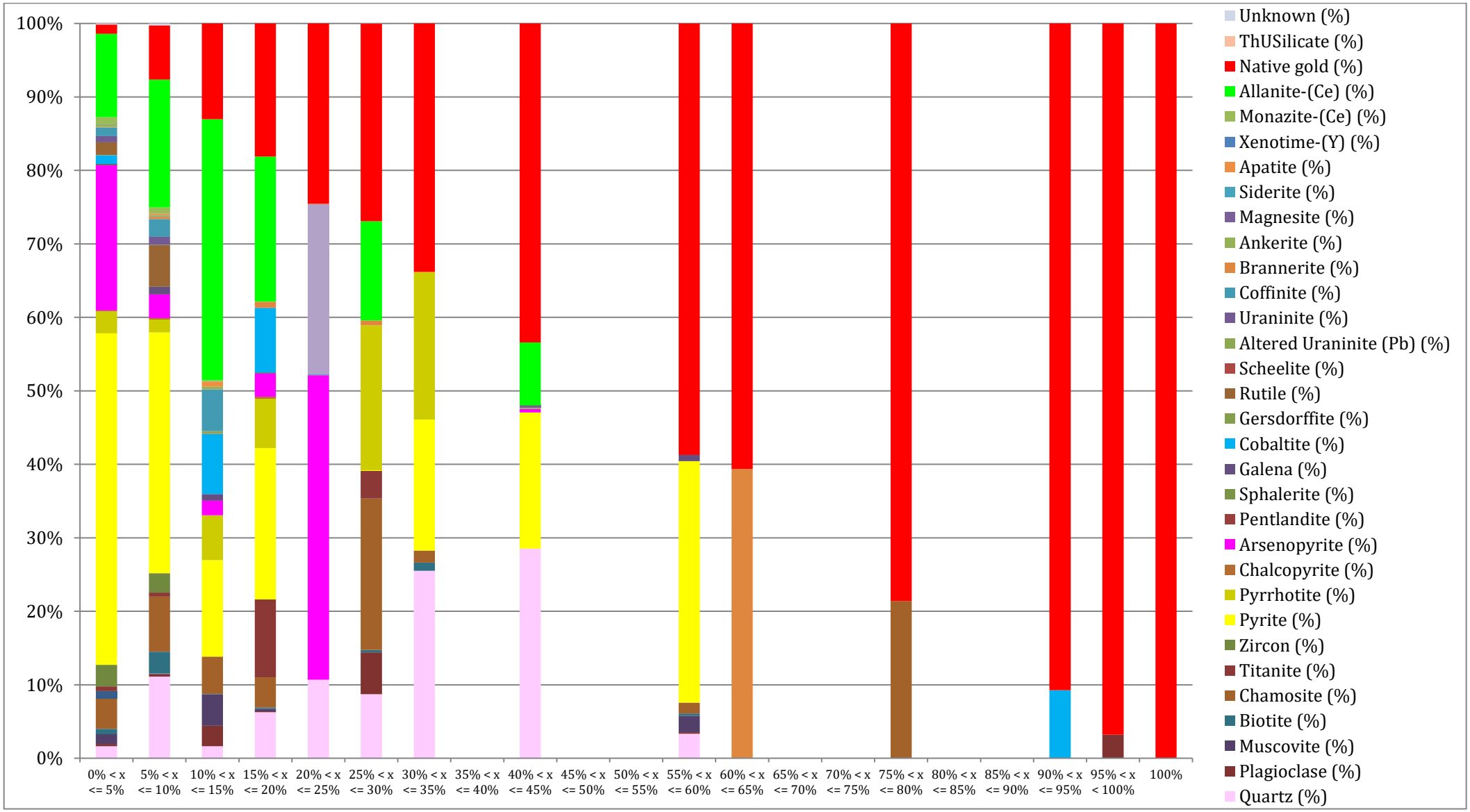
Sieve Size	Retained Wt%	Cum. Retained Wt%	Cum. Passing Wt%
1.2	0	100	0
1.45	0.95	100	0
1.75	1.47	99.05	0.95
2	1.97	97.58	2.42
2.4	2.62	95.61	4.39
2.9	2.15	92.99	7.01
3.4	3.56	90.84	9.16
4.1	3.14	87.28	12.72
4.8	3.56	84.15	15.85
5.7	3.03	80.59	19.41
6.8	4.34	77.56	22.44
8.1	6.37	73.22	26.78
9.6	9.46	66.85	33.15
11.4	5.36	57.39	42.61
13.5	9.19	52.03	47.97
16	5.02	42.84	57.16
19	9.42	37.82	62.18
22	18.03	28.4	71.6
27	10.37	10.37	89.63
32	0	0	100

Liberation data

Liberation is defined by the area % of the mineral of interest relative to the area of the total host particle.

Native gold Composition of Particle (Wt%)	0%	0-5%	5-10%	10-15%	15-20%	20-25%	25-30%	30-35%	35-40%	40-45%	45-50%	50-55%	55-60%	60-65%	65-70%	70-75%	75-80%	80-85%	85-90%	90-95%	95-100%	100%
No. of Particles	3035	151	25	18	9	2	7	1	0	2	0	0	1	1	0	0	1	0	0	1	1	21
Mean Phase ECD (µm)	0	2.6	3.87	3.52	4.24	6.84	7.65	10.93	0	12.34	0	0	24.65	7.32	0	0	9.33	0	0	13.91	13.88	16.92
Mean Phase Max Span (µm)	0	3.83	6.04	5.54	6.56	11.25	10.84	14.75	0	16.44	0	0	39.64	10.88	0	0	12	0	0	18.97	20.04	29.47
Particle Distribution (%)	89.93	7.12	0.97	0.82	0.26	0.05	0.27	0.02	0	0.07	0	0	0.05	0	0	0	0.01	0	0	0.04	0.02	0.34
Mean Density	4.8	4.7	4.38	4.53	4.88	2.99	4.53	2.47	0	5.87	0	0	4.03	4.8	0	0	4.64	0	0	8.1	8.05	19.28
Distribution of Mineral (%)	0	10.39	8.07	12.04	5.4	1.45	8.27	0.7	0	3.62	0	0	3.55	0.29	0	0	1.11	0	0	3.98	2.46	38.69
Cum. Distn. of Mineral (%)	0	100	89.61	81.54	69.5	64.1	62.65	54.39	53.69	53.69	50.07	50.07	50.07	46.52	46.24	46.24	46.24	45.13	45.13	45.13	41.15	38.69

Modal Analysis (Wt%)	0%	0-5%	5-10%	10-15%	15-20%	20-25%	25-30%	30-35%	35-40%	40-45%	45-50%	50-55%	55-60%	60-65%	65-70%	70-75%	75-80%	80-85%	85-90%	90-95%	95-100%	100%
Quartz (%)	3.16	1.64	11.08	1.63	6.27	10.69	8.71	25.5	0	28.52	0	0	3.33	0	0	0	0	0	0	0	0	0
Plagioclase (%)	0.1	0.32	0.35	2.8	0.12	0	5.69	0	0	0	0	0	0.22	0	0	0	0	0	0	0	3.21	0
Muscovite (%)	1	1.35	0.08	4.24	0.33	0	0	0	0	0	0	0	2.24	0	0	0	0	0	0	0	0	0
Biotite (%)	0.66	0.72	2.96	0.06	0.19	0	0.34	1.12	0	0	0	0	0.29	0	0	0	0	0	0	0	0	0
Chamosite (%)	4.48	4.17	7.5	5.1	4.1	0	20.6	1.65	0	0	0	0	1.48	0	0	0	21.41	0	0	0	0	0
Titanite (%)	1.48	0.64	0.55	0	10.59	0	3.76	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Zircon (%)	1.49	2.98	2.63	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Pyrite (%)	26.74	45.46	32.73	13.1	20.6	0	0.09	17.81	0	18.52	0	0	32.86	0	0	0	0	0	0	0	0	0
Pyrrhotite (%)	24.38	3.02	1.76	6.05	6.73	0	19.71	20.08	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Chalcopyrite (%)	0.34	0.14	0.22	0.06	0.26	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Arsenopyrite (%)	4.57	19.94	3.18	1.98	3.15	41.44	0	0	0	0.46	0	0	0	0	0	0	0	0	0	0	0	0
Pentlandite (%)	0.25	0.01	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Sphalerite (%)	0.17	0.02	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Galena (%)	5.7	0.14	1.06	0.92	0.14	0.09	0	0	0	0	0	0	0.85	0	0	0	0	0	0	0	0	0
Cobaltite (%)	0.1	1.17	0	8.16	8.78	0.08	0	0	0	0	0	0	0	0	0	0	0	0	0	9.27	0	0
Gersdorffite (%)	0.21	0.04	0.02	0.06	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Rutile (%)	1.9	1.72	5.63	0.08	0.11	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Altered Uraninite (Pb) (%)	0.22	0.01	0.03	0.23	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Uraninite (%)	11.64	0.85	1.09	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Coffinite (%)	5.63	1.2	2.42	5.69	0	23.09	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Brannerite (%)	1.06	0.07	0.32	0.04	0.74	0.09	0.67	0	0	0	0	0	0	39.38	0	0	0	0	0	0	0	0
Ankerite (%)	0.16	0.43	0.13	0.32	0.03	0	0	0	0	0.19	0	0	0	0	0	0	0	0	0	0	0	0
Magnesite (%)	0.22	0	0	0	0	0	0	0	0	0.32	0	0	0	0	0	0	0	0	0	0	0	0
Siderite (%)	0.35	0.01	0.08	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Apatite (%)	0.1	0	0.19	0.71	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Xenotime-(Y) (%)	0.05	0.01	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Monazite-(Ce) (%)	0.15	0.88	0.86	0.18	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Allanite-(Ce) (%)	1.21	11.43	17.37	35.51	19.73	0	13.49	0	0	8.56	0	0	0	0	0	0	0	0	0	0	0	0
Native gold (%)	0	1.3	7.38	13.01	18.11	24.53	26.89	33.83	0	43.42	0	0	58.73	60.62	0	0	78.59	0	0	90.73	96.79	100
ThUSilicate (%)	1.1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Unknown (%)	0.31	0.13	0.26	0	0	0	0.03	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0


100%
90%
80%
70%
60%
50%
40%
30%
20%
10%
0%
0% < x <= 5%
5% < x <= 10%
10% < x <= 15%
15% < x <= 20%
20% < x <= 25%
25% < x <= 30%
30% < x <= 35%
35% < x <= 40%
40% < x <= 45%
45% < x <= 50%
50% < x <= 55%
55% < x <= 60%
60% < x <= 65%
65% < x <= 70%
70% < x <= 75%
75% < x <= 80%
80% < x <= 85%
85% < x <= 90%
90% < x <= 95%
95% < x < 100%
100%
Unknown (%)
ThUSilicate (%)
Native gold (%)
Allanite-(Ce) (%)
Monazite-(Ce) (%)
Xenotime-(Y) (%)
Apatite (%)
Siderite (%)
Magnesite (%)
Ankerite (%)
Brannerite (%)
Coffinite (%)
Uraninite (%)
Altered Uraninite (Pb) (%)
Scheelite (%)
Rutile (%)
Gersdorffite (%)
Cobaltite (%)
Galena (%)
Sphalerite (%)
Pentlandite (%)
Arsenopyrite (%)
Chalcopyrite (%)
Pyrrhotite (%)
Pyrite (%)
Zircon (%)
Titanite (%)
Chamosite (%)
Biotite (%)
Muscovite (%)
Plagioclase (%)
Quartz (%)

Exposure Data

Mineral exposure can be described as the percentage of the grain that has a free, exposed, surface.

Free Surface of Particle (%)	0% (barren)	0%	0-5%	5-10%	10-15%	15-20%	20-25%	25-30%	30-35%	35-40%	40-45%	45-50%	50-55%	55-60%	60-65%	65-70%	70-75%	75-80%	80-85%	85-90%	90-95%	95-100%	100%
No. of Particles	3035	162	46	6	1	0	0	0	2	1	0	0	0	0	0	0	1	1	0	0	0	0	21
Mean Phase ECD (µm)	0	3.05	3.79	6.92	5.08	0	0	0	15.98	9.33	0	0	0	0	0	0	13.91	13.88	0	0	0	0	16.92
Mean Phase Max Span (µm)	0	4.63	5.76	9.95	8.4	0	0	0	25.26	12	0	0	0	0	0	0	18.97	20.04	0	0	0	0	29.47
Particle Distribution (%)	89.93	6.97	2.4	0.22	0.01	0	0	0	0.06	0.01	0	0	0	0	0	0	0.04	0.02	0	0	0	0	0.34
Mean Density	4.8	4.68	4.5	4.44	2.9	0	0	0	8.83	4.64	0	0	0	0	0	0	8.1	8.05	0	0	0	0	19.28
Distribution of Mineral (%)	0	20.58	22.68	6.52	0.15	0	0	0	3.83	1.11	0	0	0	0	0	0	3.98	2.46	0	0	0	0	38.69
Cum. Distn. of Mineral (%)	0	100	79.42	56.74	50.22	50.07	50.07	50.07	50.07	46.24	45.13	45.13	45.13	45.13	45.13	45.13	45.13	41.15	38.69	38.69	38.69	38.69	38.69

Modal Analysis (%)	0% (barren)	0%	0-5%	5-10%	10-15%	15-20%	20-25%	25-30%	30-35%	35-40%	40-45%	45-50%	50-55%	55-60%	60-65%	65-70%	70-75%	75-80%	80-85%	85-90%	90-95%	95-100%	100%
Quartz (%)	3.16	1.73	6.57	14.26	0	0	0	0	3.09	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Plagioclase (%)	0.1	0.36	1.03	6.97	0	0	0	0	0.21	0	0	0	0	0	0	0	0	3.21	0	0	0	0	0
Orthoclase (%)	0.15	0.04	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Muscovite (%)	1	1.36	1.51	0.75	0	0	0	0	2.08	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Biotite (%)	0.66	1.13	0.11	0.27	0	0	0	0	0.27	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Chamosite (%)	4.48	5.26	4.58	1.15	0	0	0	0	1.37	21.41	0	0	0	0	0	0	0	0	0	0	0	0	0
Titanite (%)	1.48	0.56	1.67	4.62	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Zircon (%)	1.49	1.54	5.47	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Pyrite (%)	26.74	39.06	41.78	1.7	81.73	0	0	0	30.47	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Pyrrhotite (%)	24.38	3.69	2.27	20.8	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Chalcopyrite (%)	0.34	0.12	0.18	0.32	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Arsenopyrite (%)	4.57	20.3	3.42	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Pentlandite (%)	0.25	0.01	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Sphalerite (%)	0.17	0.01	0.02	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Galena (%)	5.7	0.25	0.46	0.03	0	0	0	0	0.78	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Cobaltite (%)	0.1	1.25	3.61	0	0	0	0	0	0	0	0	0	0	0	0	0	9.27	0	0	0	0	0	0
Gersdorffite (%)	0.21	0.04	0.01	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Chromite (%)	0.46	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Rutile (%)	1.9	2.54	0.03	0.13	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Altered Uraninite (Pb) (%)	0.22	0.04	0.01	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Uraninite (%)	11.64	0.97	0.16	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Coffinite (%)	5.63	1.81	1.74	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Brannerite (%)	1.06	0.11	0.11	0.89	0	0	0	0	2.86	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Ankerite (%)	0.16	0.45	0.05	0.87	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Magnesite (%)	0.22	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Siderite (%)	0.35	0.02	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Apatite (%)	0.1	0.03	0.24	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Monazite-(Ce) (%)	0.15	1.02	0.06	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Allanite-(Ce) (%)	1.21	13.35	16.42	20.89	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Native gold (%)	0	2.62	8.41	26.34	18.27	0	0	0	58.87	78.59	0	0	0	0	0	0	90.73	96.79	0	0	0	0	100
ThUSilicate (%)	1.1	0	0.01	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Unknown (%)	0.31	0.15	0.03	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0

Modal Mineralogy

The modal mineralogy of the leach residue block.

Combined					
Mineral	Wt%	Area%	Area (micron)	Particle Count	Grain Count
Quartz	6.61	10.9	0	1058	1058
Plagioclase	0.23	0.38	0	36	36
Orthoclase	0.17	0.28	0	27	27
Muscovite	1.58	2.43	0	231	231
Biotite	1.03	1.44	0	139	139
Chamosite	2.33	3.16	0	301	301
Clinochlore	0.05	0.08	0	8	8
Titanite	0.89	1.12	0	106	106
Zircon	0.61	0.57	0	54	54
Pyrite	33.55	29.13	0	2775	2775
Pyrrhotite	41.61	39.21	0	3736	3736
Chalcopyrite	0.28	0.29	0	28	28
Arsenopyrite	4.28	3.07	0	291	291
Pentlandite	0.15	0.14	0	13	13
Sphalerite	0.08	0.09	0	8	8
Galena	0.09	0.05	0	5	5
Cobaltite	0.03	0.02	0	2	2
Tennantite	0.01	0.01	0	1	1
Gersdorffite	0.21	0.15	0	14	14
Stibnite(Fe,As)	0.01	0.01	0	1	1
Fe-oxide	0.65	0.53	0	51	51
Chromite	0.62	0.57	0	54	54
Rutile	0.71	0.72	0	69	69
Uraninite	0.42	0.21	0	20	20
Coffinite	0.1	0.08	0	8	8
Brannerite	0.25	0.2	0	19	19
Calcite	0.06	0.1	0	9	9
Ankerite	0.27	0.39	0	38	38
Magnesite	0.94	1.36	0	130	130
Siderite	1.19	1.3	0	125	125
Apatite	0.06	0.08	0	8	8
Xenotime-(Y)	0.01	0.01	0	1	1
Monazite-(Ce)	0.15	0.12	0	12	12
Allanite-(Ce)	0.34	0.39	0	37	37
Barite	0.01	0.01	0	1	1
ThUSilicate	0.07	0.03	0	3	3
Fe-oxide (Sb)	0.07	0.06	0	6	6
Scorodite	0.01	0.01	0	1	1
Tourmaline	0.01	0.01	0	1	1
Carbon	0.01	0.12	0	12	12
Unknown	0.25	1.08	0	103	103
Low_Counts	0	0.07	0	7	7
Total	100	100	0	9549	9549

Appendix E – Conference papers and publications

Conference Paper 1, Publication Manuscript 1

This conference paper was presented at the SAIMM Geometallurgy Conference in 2018, it was subsequently submitted as a paper to the Journal of the South African Institute of Mining and Metallurgy.

Depositional environment - the original control on gold processing

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The refractory nature of gold ores is controlled by a limited number of well understood mineralogical factors. The nature of the gold is a significant cause of complex gold processing. Solid solution gold, found in arsenopyrite or arsian pyrite, as well as certain gold-bearing minerals such as aurostibite, adversely affect extraction. Other sources of processing complications include preg-robbing ores containing carbon, which reduces gold recovery, and base metal minerals, which compete with gold during leaching. These factors, which critically influence gold extraction, are a product of the environment of formation.

Although a variety of deposit types host gold, refractory ores are typically restricted to a select few. The present study is an investigation of selected South African refractory/complex gold deposits, their mineralogical controls, and conditions of metallogenesis, to better understand the formational controls of problematic ores. This is relevant as the location and regional geology of a prospective deposit could act as an early warning system to future metallurgical complications. The deposits studied include Barberton (both Fairview and Barbrook), Pilgrim's Rest and Consolidated Murchison.

INTRODUCTION

Gold has been an important commodity throughout much of human history, and is thought to be one of the earliest metals mined (Kongolo and Mwema, 1998; Reardon, 2011). Evidence exists for gold mastery as early as 6 000 BC, and it continues to be highly sought after to the present day. Early gold mining involved manual or gravity-based separation methods from stream beds, beyond this, little advancement took place until underground mining was initiated, in Russia, in 1744, where gravity concentration and amalgamation were used for

gold extraction (Kongolo and Mwema, 1998). By 1848, a process to extract gold using chlorine gas was developed and extensively used, but was limited in its capacity to process ore of low gold concentration. Thus, when cyanidation was first patented in 1888, it was rapidly commercialised, first in New Zealand (in 1889) and subsequently in a number of countries including South Africa, USA, Mexico and France (Reardon, 2011). Cyanidation was widely adopted as it is extremely effective at processing more readily extractable/free milling gold. However, the number of ore bodies that are both easily accessible and amenable to cyanide is diminishing. Increasing attention has therefore been given to both the discovery and recovery of refractory ores (Fraser et al., 1991). Understanding the geological controls of the formation of the different types of refractory ores may be important to the future of gold exploration, mining and processing.

GOLD DEPOSITS

There are a number of important gold deposits within South Africa, accounting for significant gold production in the past (Janisch, 1986). Although gold production has declined in past decades, these deposits are still available for study.

Barberton, Fairview & Barbrook

Barberton Greenstone Belt is one of the oldest and best preserved supracrustal sequences in the world. The Barberton Greenstone Belt occurs near the town of Barberton, Mpumalanga, approximately 280 km east of Johannesburg. The mechanism of mineralisation is a contentious issue amongst workers in Barberton. In the past it was agreed that the mineralisation at Barberton had an orogenic genesis, linked to the accretion of the greenstone belt (de Ronde et al., 1994; Groves et al., 1998; Goldfarb et al., 2001). However, a number of more recent papers suggest that the mineralisation occurred later than the accretion, and likely associated with a subsequent extensional event (Dirks et al., 2009; Munyai et al., 2011; Dirks et al., 2013). Isotope studies suggest that the mineralising fluids originated from external sources, likely magmatic, taking advantage of brittle-ductile shear zones, fractures and faults (de Ronde et al., 1994; Ward, 1999; Munyai et al., 2011).

Consolidated Murchison

Consolidated Murchison represents an important antimony–gold deposit within the Murchison Greenstone Belt, an Archean volcano-sedimentary belt. Located in the Northern Kaapvaal Craton, near Gravelotte, 50 km west of Phalaborwa and 370 km northeast of Johannesburg (Davis, 1986; Anheusser, 2001). There has been uncertainty regarding the metallogenesis of the Antimony Line. Models for both syngenetic and epigenetic origins have been proposed. However the vast, complex, deformation undergone by this region makes accurate studies extremely difficult. The ores appear to be orogenic despite being associated with significant metamorphic remobilisation. The simultaneous occurrence of both gold and antimony (in their various forms), suggests that a single process was responsible for this enrichment (Pearton, 1982; Jaguin et al., 2012).

Pilgrim's Rest

The Sabie-Pilgrim's Rest, a gold district located in Mpumalanga, west of the Drakensberg, approximately 320 km east northeast of Johannesburg. The Sabie-Pilgrims rest district is an

epithermal deposit, hosted by the Proterozoic Transvaal Basin (Boer et al., 1995). The mineralisation is thought to be contemporaneous with the formation of the Bushveld complex and owing much of its mineralisation processes to this. The Bushveld Complex is likely responsible for the emplacement of the thrust system (exploited by the hydrothermal, mineralising fluids), but also the source of the fluids and heat required to form this epithermal gold deposit. Isotopic data, fluid inclusion studies and elemental fingerprints all support the Bushveld origin hypothesis (Tyler, 1986; Boer et al., 1995; Meyer and Robb, 1996; Tyler and Tyler, 1996).

Extractive Metallurgy

Gold, a noble metal, by definition is resistant to chemical action, therefore there are a limited number of solvents capable of dissolving it (Moffa, 1977). Systems where gold dissolution is effective, typically require an oxidant, capable of oxidising the gold, as well as a reactant/complexing agent able to form a reasonably stable gold complex post-oxidation (Kongolo and Mwema, 1998).

Cyanide has been used in the extraction of gold for over 100 years, as such the chemistry of its interaction with gold is well understood. Sodium (or potassium) cyanide is added to the ore in order to dissolve gold, which operates by the overall reaction mechanism (1) (Kongolo and Mwema, 1998);

$$4Au + 8NaCN + O_2 + 2H_2O \rightarrow 4AuNa(CN)_2 + 4NaOH \quad [1]$$

Oxygen acts as an oxidising agent, while CN^- dissociates from sodium, and complexes with the gold (Habashi, 1967).

Refractory Ores

The cyanidation process typically occurs quite readily, which accounts for its widespread use, however its capability is limited in the presence of certain interfering agents. The nature of these agents allows for the classification of gold ores, while the classification system may vary from author to author (Chryssoulis et al., 1987; Haque, 1987; Monhemius, 1987; Komnitsas and Pooley, 1989; Vaughan, 2004; Zhou and Cabri, 2004; Chryssoulis and McMullen, 2005; Miller et al., 2005; Karimi et al., 2010) the overall concept is the same. This description has been strongly based on Vaughn, 2004. Three distinct gold types exist in this system (Figure 1); free milling, complex and refractory (in increasing order of processing difficulty). Free milling ores encompass all ores that readily undergo gold extraction by cyanide, within these ores there are no factors which complicate or reduce extraction efficiency. A gold recovery of 90% or more defines free milling ores (La Brooy et al., 1994; Marsden and House, 2006). Complex ores refers to ores which can undergo gold extraction with a high recovery rate, but only under altered/more intense leaching conditions. Complex ores can be subdivided into two categories; preg-robbing and reactive. Preg-robbing ores contain material capable of absorbing gold-cyanide complexes during leaching. This material tends to be carbonaceous matter (specifically elemental and organic carbon), although clays are also thought to have a minor capacity for preg-robbing (Vaughan, 2004; Goodall et al., 2005; Miller et al., 2005). The effects of preg-robbing can be countered (to a limited degree) in a number of ways; 1) addition of kerosene to deactivate the carbonaceous material, 2) Carbon-In-Leach, where activated carbon is present throughout the leach and can absorb the gold onto its surface

more rapidly than the competing preg-robbing component (La Brooy et al., 1994; Vaughan, 2004). Reactive ore refers to any ore that contains material that react with reagents required for successful gold leaching. These reactive materials tend to be cyanicides (i.e. they react with and reduce the available concentration of cyanide), or oxygen consumers. Examples of these minerals include; pyrrhotite, copper minerals (e.g. chalcocite, malachite, azurite and cuprite) with the exception of chalcopyrite, as well as most base metal minerals (bearing iron, arsenic, antimony, zinc, nickel and cobalt) (Aylmore and Muir, 2001; Karimi et al., 2010). These ores can be dealt with by adding higher concentrations of reagents than would be required for standard cyanidation.

Refractory ores refer to deposits that cannot achieve acceptable gold recoveries using standard procedures. There are two types of refractory ores, physically and chemically locked gold, both of which make the gold inaccessible to lixiviants. Physically locked gold occurs when gold is present as very fine inclusions, typically within a sulphide mineral. Chemically locked gold can be found in gold minerals such as gold tellurides, aurostibite, aurocupride and maldonite (Harris, 1990; Vaughan, 2004). Chemically locked gold can also refer to gold present as solid solution within certain minerals, this means that atomic gold is present within crystal lattices (Chryssoulis et al., 1987; Haque, 1987; Monhemius, 1987; Komnitsas and Pooley, 1989; Vaughan and Kyin, 2004). Arsenopyrite and to a lesser extent pyrite act as host minerals to solid solution gold (Vaughan, 2004; Vaughan and Kyin, 2004; Barker et al., 2009). In order to ensure acceptable recoveries for refractory ores, it is usually necessary to implement a pre-treatment step in the extraction process, examples of pre-treatments include ultra-fine milling, pressure oxidation, biological oxidation (BIOX) and in older designs roasting. Pressure cyanidation, while not a pre-treatment, can also be an effective option (Davis, 1986; Haque, 1987; Monhemius, 1987; Lindström et al., 1992).

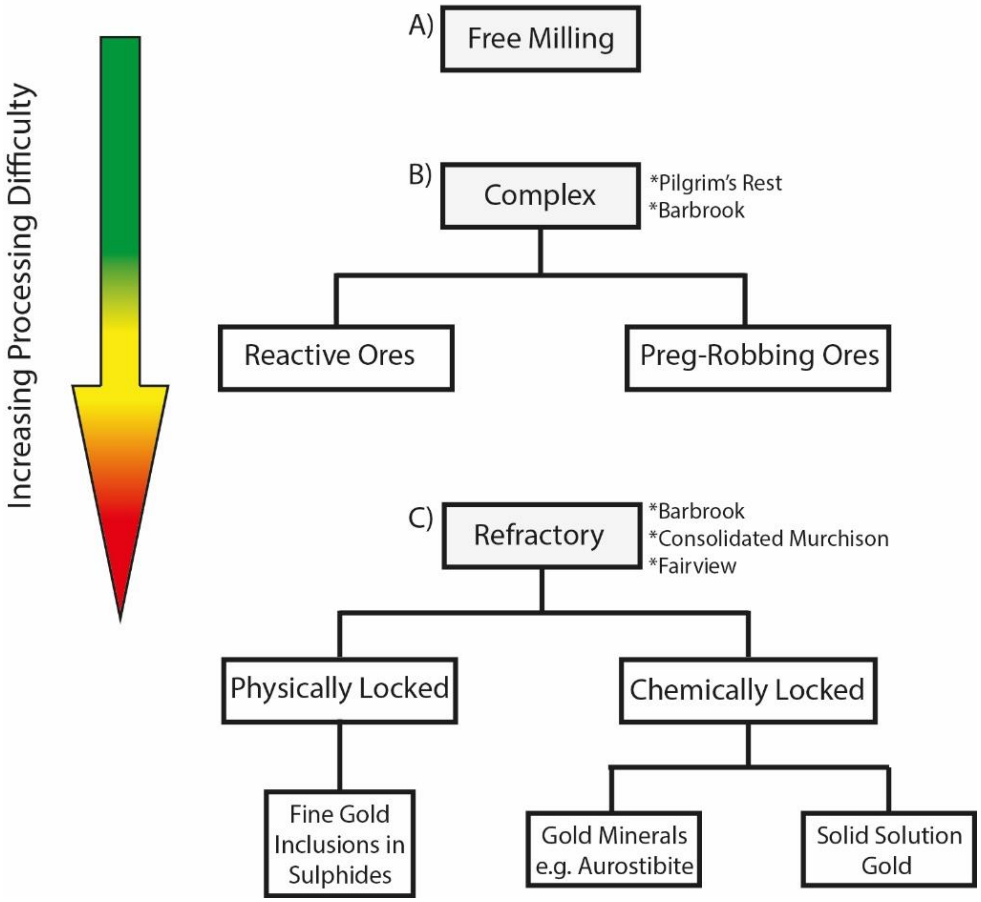


Figure 1 A diagram outlining the three main types of gold deposit types; 1) Free Milling, 2) Complex and 3) Refractory. While the deposits of study and their respective mineralogy will be outlined later, the deposits have been grouped with their respective deposit types.

DEPOSIT MINERALOGY/PROCESSING CONTROLS

The information provided on the respective deposits has been gathered using chemical assay, optical microscopy, XRD (X-ray Diffraction), and MLA (Mineral Liberation Analyser).

Barberton, Fairview

The mineralisation found in Barberton is hosted by a highly altered lithology comprising lithic quartz fragments with a notable alignment found within a finer grained matrix of chlorite, dolomite, magnesite and to a lesser extent muscovite. Ore minerals are either present in veins/fractures (where they tend to be large and euhedral), or as disseminations (where they tend to be fine and sub- to anhedral), these minerals include pyrite, arsenopyrite and minor visible gold (Figure 2A). Gold grades in the studied material are close to 12.5 g/ton (Table). At a grind of 80% -75 µm, visible native gold was 99.73% liberated, with a surface exposure/leach reactivity of 99.7%. Gold grains range in size from 0.5 µm to 425 µm (Figure 4A), with the most significant weight percent contribution to total gold in the range of 180 µm and 425 µm. The large range in grain size, as well as presence of coarse gold will affect the leach time. Pyrite (3.79%), arsenopyrite (3.26%), and quartz (0.87%) are commonly associated with gold (Figure 2B). The extent of visible gold present, relative to the grade (12.5 g/ton Au) suggests that there is gold invisible to both optical petrography and MLA. Based on

literature, this is likely due to solid solution gold present in arsenopyrite (Anheusser, 2001; Anhaeusser, 2015).

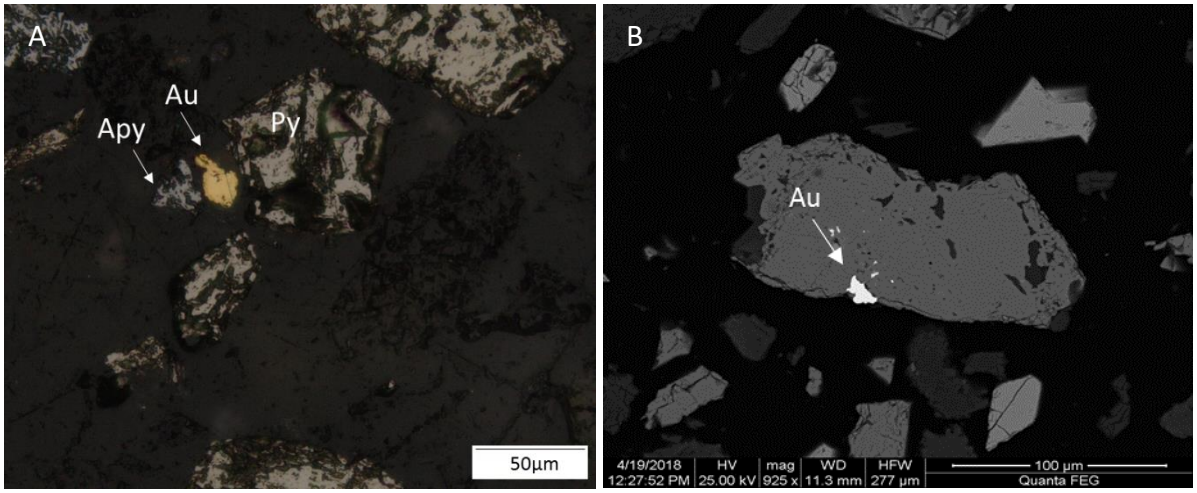


Figure 2 A) Example of the mineralisation found in Fairview, including arsenopyrite (Apy), pyrite (Py) and gold (Au). The gold grain has an equivalent circular diameter (ECD) of 46.5 µm. B) Example of a gold grain at Fairview (BSE), with limited liberation due to association with pyrite, but good exposure and an ECD of 10.7 µm.

Barberton, Barbrook

The host lithology tends to be highly altered carbonate-rich shale, with extensive veins. Interstial material present within the shale includes chlorite and muscovite and carbon. The extensive veination and range in orientation thereof give the appearance of a stockwork texture. The veins are either carbonate or quartz, with carbonate being the dominant type. The amount of iron-rich material present in the sedimentary units is variable, and can range from highly abundant to entirely absent, secondary iron oxides such as limonite and goethite are also present. Ore minerals evident in Barbrook include pyrite, arsenopyrite, with minor chalcocite, galena and pyrrhotite. Only a limited number of gold grains were identified using MLA, which reduces the scope of the conclusions that can be made. However, based on MLA data, the visible gold has a liberation of 93.21%, while 99.07% of gold is exposed for leaching thus increased grinding will have negligible positive impact on the gold recovery. Of the gold grains found, the grain size distribution ranges from 1 µm to 20 µm. The gold tends to only be associated with pyrite (10.02%), which is unlikely to significantly affect leaching conditions. A comparison of assayed (Table) and mineralogically calculated gold grades suggest that 40% of gold present is as solid solution in arsenopyrite, which is substantiated by literature (Anheusser, 2001; Pearton and Viljoen, 2017).

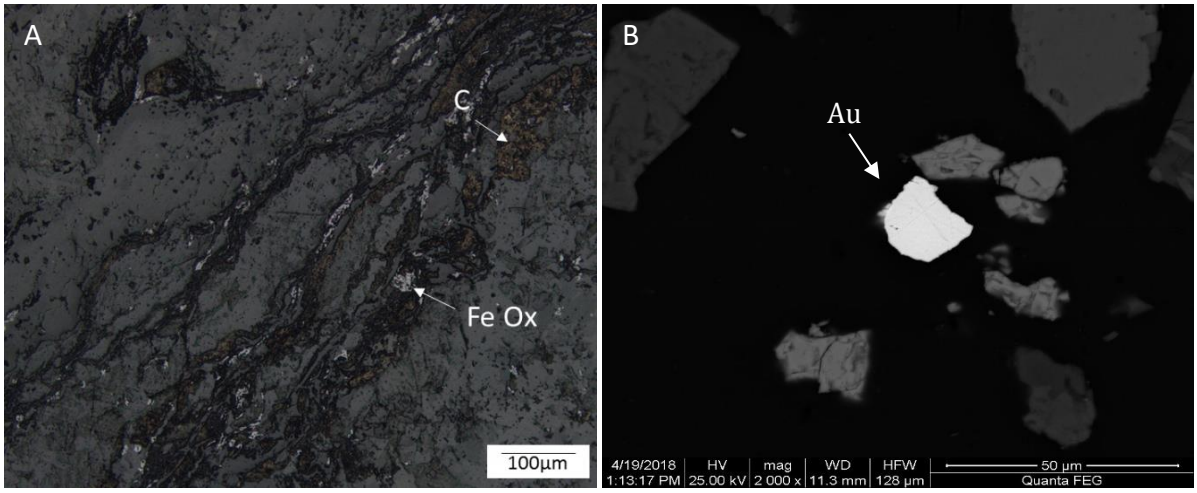


Figure 3 A) Example of carbon found in Barbrook, associated with various iron oxides. B) One of the larger Barbrook gold grains found using the MLA (BSE image) with an ECD of 16.9 µm.

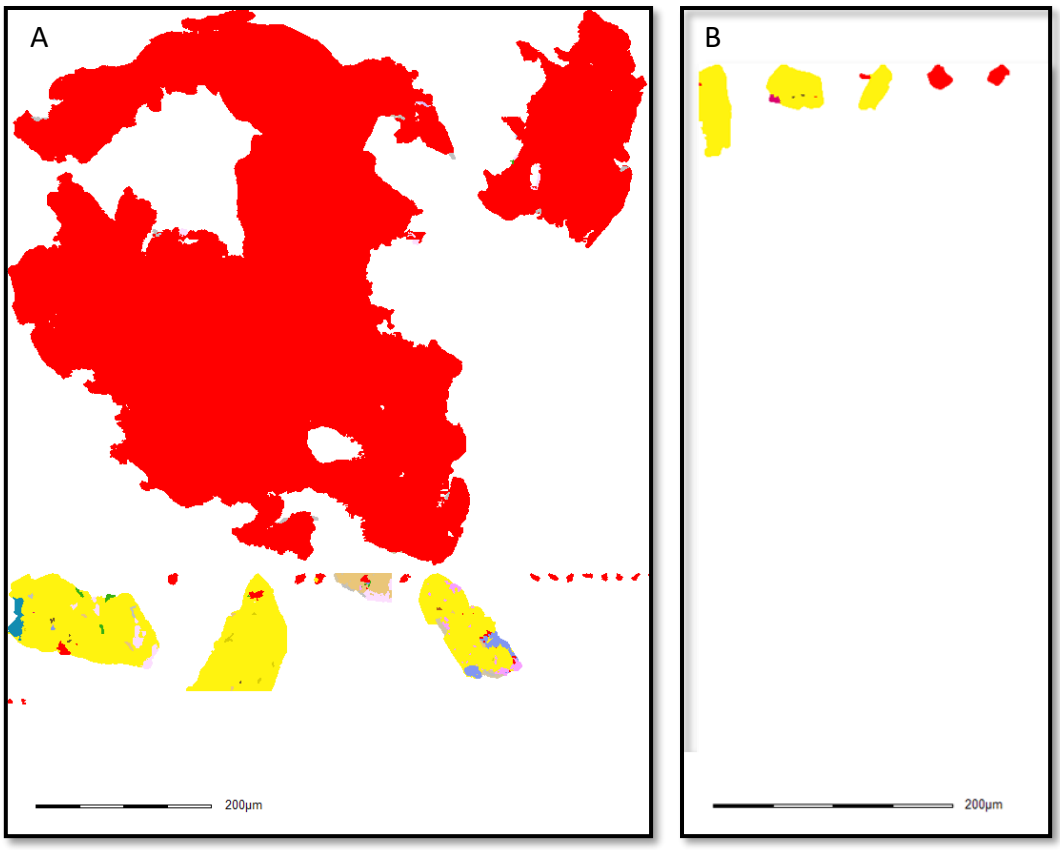


Figure 4 A) An example of the vast range in sizes of gold grains in Fairview, evident in this false colour MLA-generated image, where gold is represented by red. B) All the Barbrook ore gold grains, with pyrite association represented.

Consolidated Murchison

The mineralisation within Consolidated Murchison is hosted by altered schists, comprised dominantly of carbonate and quartz material, with lesser chlorite, biotite and magnesite,

locally abundant tourmaline. Ore minerals present include stibnite (Sb_2S_3), pyrite, berthierite ($FeSb_2S_4$), and lesser arsenopyrite (Figure 4-85A). The mineralisation tends to present as homogenous disseminations throughout the schist. Assays confirm there is gold mineralisation (3 g/ton, Table). MLA results, while restricted due to the number of gold grains, suggest that the gold has lower liberation (71.89%) than the other samples investigated in the present study (Figure 6A), however, 91.33% of the gold is available/exposed for leaching (Figure 5B). These grains range in size from 2 µm to 32 µm. Mineral associations include arsenopyrite (15.74%), pyrite (13.65), stibnite (11.57%). Associations with antimony-rich minerals, such as stibnite, will affect the gold leaching, due to both reagent consumption and passivation. The limited presence of visible, native gold, in conjunction with mineral associations and literature, suggest that gold is present in the Antimony Line in a number of forms varying from coarse free grains to solid solution in arsenopyrite, and aurostibite (Davis, 1986; Vearncombe et al., 1988; Marsden and House, 2006). However, aurostibite has yet to be detected in the present sample.

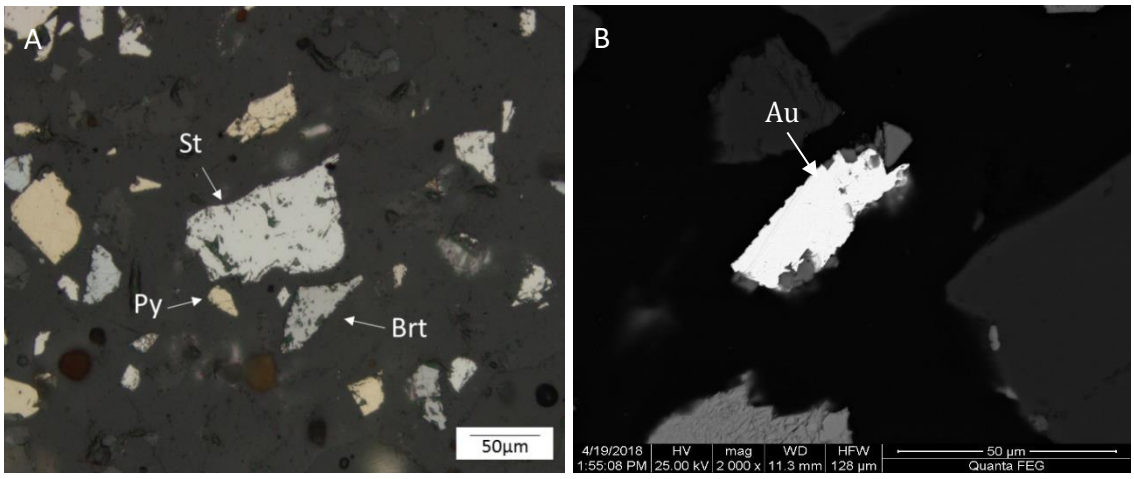


Figure 5 A) An example of mineralisation at Consolidated Murchison including stibnite (St), pyrite (Py) and berthierite (Brt). B) Gold grain associated with arsenopyrite, gold is well liberated and exposed, with an ECD of 25.5 µm.

Pilgrim's Rest

The host lithology is predominantly a fine-grained shale, crosscut by quartz carbonate veins, associated with the shale is fine muscovite. The ore mineralogy includes abundant pyrite with lesser pyrrhotite (Figure A), chalcocite, chalcopyrite and native gold (Figure 7B). Assay results show high concentrations of both gold (29.4 g/ton) and copper (7230 ppm) (Table 1). Pyrite represents approximately one third, by mass, of the feed. Mineralisation occurs extensively within veins, as disseminations and even as massive mineralisation. Gold occurs as discrete, fine grains, ranging in from 1.5 µm to 75 µm (Figure 6B). While the literature suggests that this ore typically contains significant proportions of gold less than 5 µm in size (Swiegers, 1948), the weight percent contribution of gold in this range is insignificant and more than 85 wt% of the gold is from grains between 10 µm to 75 µm. Therefore very fine grinding would not increase gold availability significantly. This assessment is supported by the MLA liberation and exposure data which show that the gold is well liberated (97.16%) and exposed (98.9%) with a grind size (80% -75 µm) that appears to be sufficiently fine for effective leaching. Gold

mineral associations are include pyrite (8.16%) and quartz (0.86%), which is unlikely to affect the gold extraction.

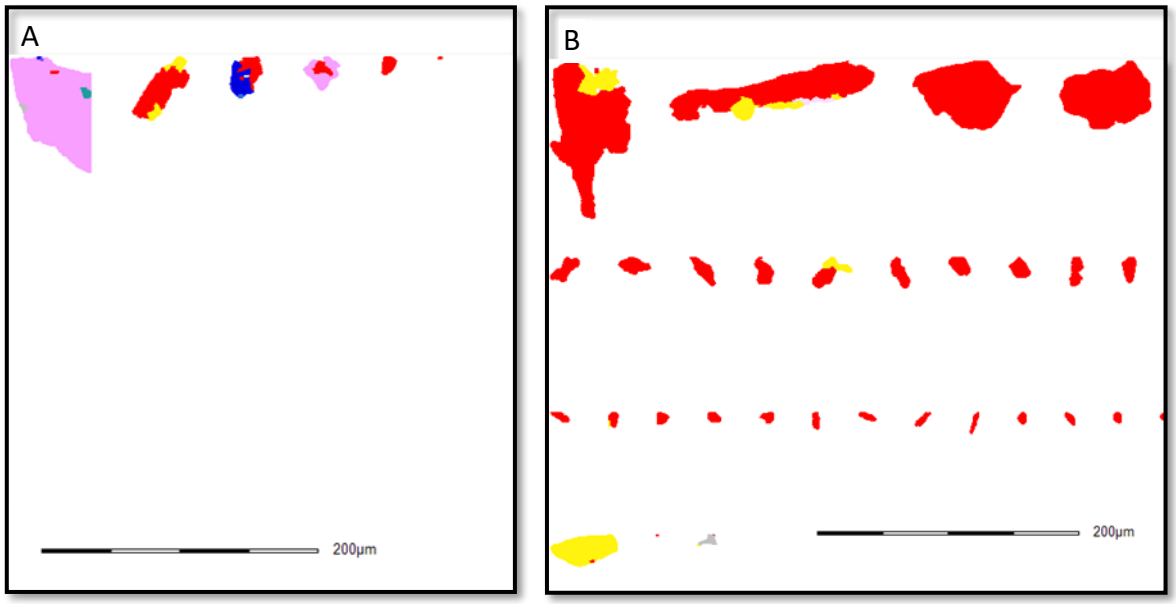


Figure 6 A) Consolidated Murchison gold grains, highlighting the lower degree of liberation. B) Pilgrim's Rest gold grains, highlighting the range in size, excellent liberation and exposure.

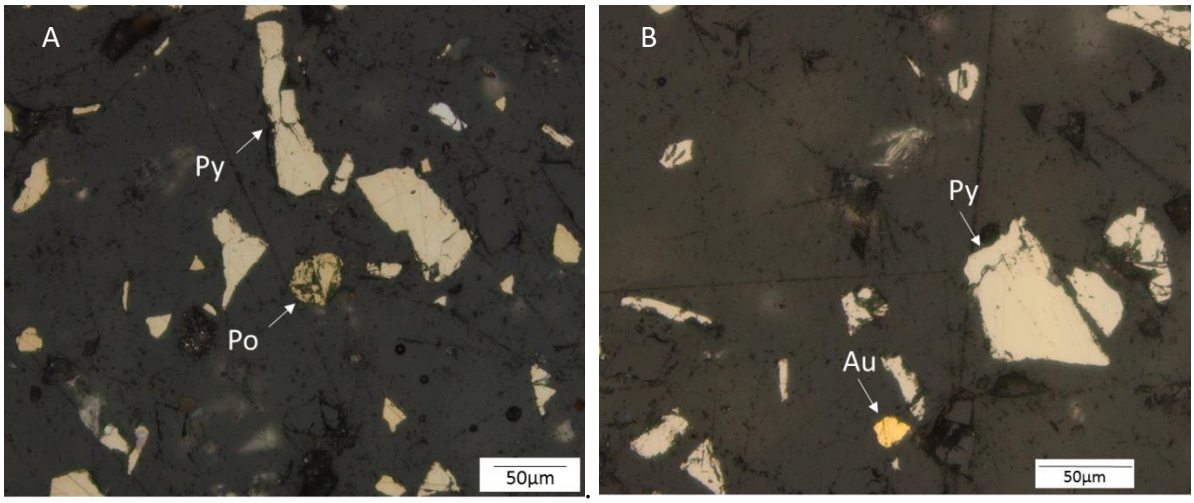


Figure 7 A) An example of a Pilgrim's Rest gold grain with an ECD of 16 µm. With significant pyrite mineralisation also evident. B) Mineralisation at Pilgrims Rest, extensive pyrite (py), and lesser pyrrhotite (po).

Table 1 Assay results of the head samples, elements which could affect gold extraction have been marked in red.

	Au	S	C	ORG C	As	Cu	Fe	Co	Ni	Sb	Zn	Pb
UNITS	G/T	%	%	%	%	PPM	%	PPM	PPM	PPM	PPM	PPM
Fairview	12.5	2.86	1.27		0.81	84		50	491		124	<20
Barbrook	2.08	8.06	3.94	0.23	0.23	597	15.3	-	472	65	2330	579
Consolidated Murchison	3.04	6.29	3.55	0.13	2.96	164	10.2	-	700	70100	98	87
Pilgrim's Rest	29.4	29.6	0.7	0.14	0.35	7230	25.5	-	84	931	47	108

DISCUSSION /CONCLUSION

There are a number of groups of gold deposit based on their genesis. Some of these groups are more likely to be refractory than others. Examples of deposits more likely to be free milling include deposits related to reduced intrusions, orogenic and low sulphidation epithermal systems. Conversely, Carlin, high sulphidation epithermal and importantly to this study, atypical greenstone deposits of disseminated stock work type, tend to be refractory (Rowe and Zhou, 2007).

In this study we have found a number of factors likely to affect the successful extraction of each respective deposit. This is further highlighted by the processing procedures of each deposit, adapted to account for these factors. Fairview, which is refractory due to sulphide-hosted gold, ensures the sulphides are destroyed using biological oxidation (BIOX) prior to leaching. Barbrook is refractory/complex due to sulphide-hosted gold and preg-robbing carbon respectively, and therefore undergoes a series of flotation steps, in order to concentrate the gold-bearing sulphides. Subsequently the sulphides undergo ultra-fine milling and pressure oxidation, which releases the gold and allows effective leaching. Cyanidation is then undertaken with the gold loading onto resin-in-leach (to reduce preg-robbing effects) (Grant, 2004).

Consolidated Murchison contains antimony which, during gold extraction, is thought to act as an oxygen scavenger (Monhemius, 1987), while increasing both cyanide and lime consumption, there is also evidence that the gold becomes passivated during normal cyanidation processes (Millard, 2005; Rohner and Millard, 2016). Gold extraction at Consolidated Murchison involved first separating out as much free gold as possible, using gravity concentration. The remaining ore was subjected to a flotation circuit, concentrating both gold and stibnite. The gold was subsequently recovered from the flotation concentrate using a low alkalinity, high pressure (8.8MPa) cyanidation process, in a 1.5 km pressure pipe reactor (Davies and Paterson, 1986; Millard, 2005).

A comparison of Pilgrim's Rest's mineralogical and assayed gold grades suggest there is no refractory gold, in addition the excellent liberation and exposure would suggest this ore is well disposed to cyanide leaching. However, significant copper concentrations (0.7%) will negatively affect leaching, as copper competes with gold to form cyanide complexes (Deschênes and Prud'homme, 1997). Copper will also reduce gold recovery by CIL/CIP as it will load onto the activated carbon (Dai et al., 2010). In addition, Pilgrim's Rest has notable quantities of pyrrhotite, which can cause cyanide consumption and reduced leaching kinetics

(Dai and Jeffrey, 2006). Pilgrim's Rest is therefore complex due to the presence of base metal sulphides and oxides (i.e. minerals containing copper and to a lesser degree iron). However, the presence of these complex minerals is heterogeneous and the extractive metallurgy of the Pilgrims Rest ore is variable. Free milling and complex ores have historically been kept separate and processed using two discrete processing circuits that only merge in the final cyanidation process (Congress, 1961).

Large proportions of gold contained at Fairview and Barbrook, are present within the crystal lattices of arsenopyrite. Submicroscopic gold is thought to be incorporated into these crystal lattices during precipitation at sub to lower greenschist facies. With an increase in metamorphic conditions, the gold is progressively ejected from the lattice, first into fractures and occlusions within the sulphides and finally into discrete minerals within the gangue (Vaughan and Kyin, 2004). Considering most South African orogenic belts, have only undergone low grade/greenschist metamorphism, the refractory nature of orogenic gold deposits is characteristic of this metamorphism (Foster and Piper, 1993).

Arsenic tends to be enriched within sedimentary, rather than igneous rocks. The arsenic concentration of metamorphic material is strongly dependant on that of the original protolith and is not significantly enriched by metamorphic processes(Tanaka, 1988). The enrichment of As in sedimentary material is thought to take place during the sedimentation process. Within sedimentary material, finer grained lithologies typically have higher concentrations (Tanaka, 1988). Another element with importance to the refractory nature of the gold deposits, specifically Consolidated Murchison, is antimony. The mechanisms that constrain antimony concentrations globally are not well understood, however from a series of analyses of a number of lithologies and localities it is apparent antimony enrichments occurs in sediments (notably shales and argillites), as well as andesites and rhyolites (Boyle and Jonasson, 1984).

The origin of potentially preg-robbing carbonaceous components may be due to both biotic and abiotic factors (Welte, 1969). Additionally, biotic factors may be effected by a number of controls including initial presence, oxygen conditions (both of which may vary on a scale of meters) iron concentration, burial rates, and subsequent metamorphism (Welte, 1969; Pedersen and Calvert, 1990; Meyers, 2007). This point is further emphasised by the example of Fairview and Barbrook; gold economic deposits formed from the same mineralising events and stratigraphic units, but with distinctly different carbonaceous natures.

Pyrrhotite, one of the most common base metal minerals with complex gold processing effects, is present in notable quantities in many ore deposits and specifically in this work in Pilgrims Rest. Pyrrhotite forms predominantly in mafic to ultramafic intrusions, which is not a typical primary gold deposit type, however, hydrothermal fluids are also commonly responsible for pyrrhotite formation (Vaughn, 1982; Gordon and McDonald, 2015). Unfortunately the source of hydrothermal fluids precipitating pyrrhotite cannot be constrained exclusively to mafic magmatic origins as the metal and sulphur may not necessarily be the same source (Reed and Palandri, 2006). Despite this, having an association with mafic magmatic material is still a good indication of a potential refractory nature, as demonstrated by Pilgrim's Rest (Kissin and Scott, 1982).

Utilising the characteristics of South African hydrothermal gold deposits that were highlighted, in conjunction with a desktop study of a region, could highlight the risk of complex or refractory ore.

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Mineralogical Controls on Gold Extraction

A Comparison of Leaching Techniques on South African Ores

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ABSTRACT

Mounting pressure, due to social and environmental concerns associated with cyanide extraction, as well as the decrease in available free milling gold deposits, continually complicates metallurgical processing of gold ores. The need for advancements in this field calls for research and the introduction of alternative extraction processes.

Samples of several different types of gold deposits, from various localities in South Africa, have been collected to evaluate their refractory nature, response to cyanide, and mineralogical controls on amenability to alternative extraction methods. These deposits; the Ventersdorp Contact Reef (VCR) at Mponeng Mine in the Witwatersrand Basin , Fairview and Barbrook, at Barberton, Pilgrim's Rest, and Consolidated Murchison, span a range of ore types and mineralogies. These include free milling, complex, and refractory ores. Complex mineralogies are defined by the presence of various reagent consuming and preg-robbing materials, while the presence of gold compound minerals, as well as sulphide-bound gold, results in refractory ore types.

Understanding of the mineralogy of each deposit and its influence on gold extraction is paramount to effective extraction. Examples of deleterious mineralogies include base metal sulphides, specifically significant levels of pyrite, chalcocite, and chalcopyrite in Pilgrim's Rest, and antimony minerals such as stibnite and berthierite in Consolidated Murchison. In addition, solid solution gold or discrete gold compound minerals, as found in Barberton deposits and Consolidated Murchison, reduce accessibility, and thus extraction.

In evaluation of each deposit, cyanide leaching was undertaken and maintained at marked conditions. Cyanidation was then used as a standard, against which, other leaching methods were evaluated. Under certain mineralogical conditions, alternatives to cyanide, which are typically uneconomic, may outperform the cyanide leaching method.

1. INTRODUCTION

Gold has been an important commodity throughout much of human history, and is thought to be one of the earliest metals mined (Kongolo and Mwema, 1998; Reardon, 2011). Evidence

exists for gold mastery as early as 6 000 BC, and it continues to be highly sought after to the present day. Early gold mining involved manual or gravity-based separation methods from stream beds. Beyond these methods, little advancement took place until underground mining was initiated in Russia, in 1744, where gravity concentration and amalgamation were used for gold extraction (Kongolo and Mwema, 1998). By 1848, a process to extract gold using chlorine gas was developed and extensively used but was limited in its capacity to process ore of low gold concentration. Thus, when cyanidation was first patented in 1888, it was rapidly commercialised, first in New Zealand and subsequently in a number of countries including South Africa, USA, Mexico and France (Reardon, 2011). Cyanidation was widely adopted as it is extremely effective at processing readily extractable/free milling gold. However, the number of ore bodies that are both easily accessible and amenable to cyanide is diminishing (Fraser et al., 1991).

Despite the importance of cyanide to the gold industry, increasing environmental concerns have been voiced in past decades, owing to a number of well publicised incidents involving cyanide in the mining industry (Donato et al., 2007; Laitos, 2012; Aylmore, 2016a). Ensuing public concern resulted in a number of countries having introduced varying degrees of legislation related to cyanide (Laitos, 2012). These restrictions, as well as an reduction in the number of cyanide amenable ores, have acted as a global initiative to establish economic and environmentally friendly alternatives to cyanide (Aylmore, 2005; Aylmore, 2016a).

2. DEPOSITS OF INTEREST

There are a number of important gold deposits within South Africa, accounting for significant gold production in the past (Janisch, 1986). Although gold production has declined in past decades, these deposits are still available for study. There are several deposits found throughout South Africa, ranging from world class to small scale deposits. The deposits of interest to this study are shown on the map in FIG 1 and include the Witwatersrand (Mponeng), two deposits from the Barberton granite and greenstone belt - Fairview and Barbrook, as well as Consolidated Murchison and Pilgrim’s Rest.

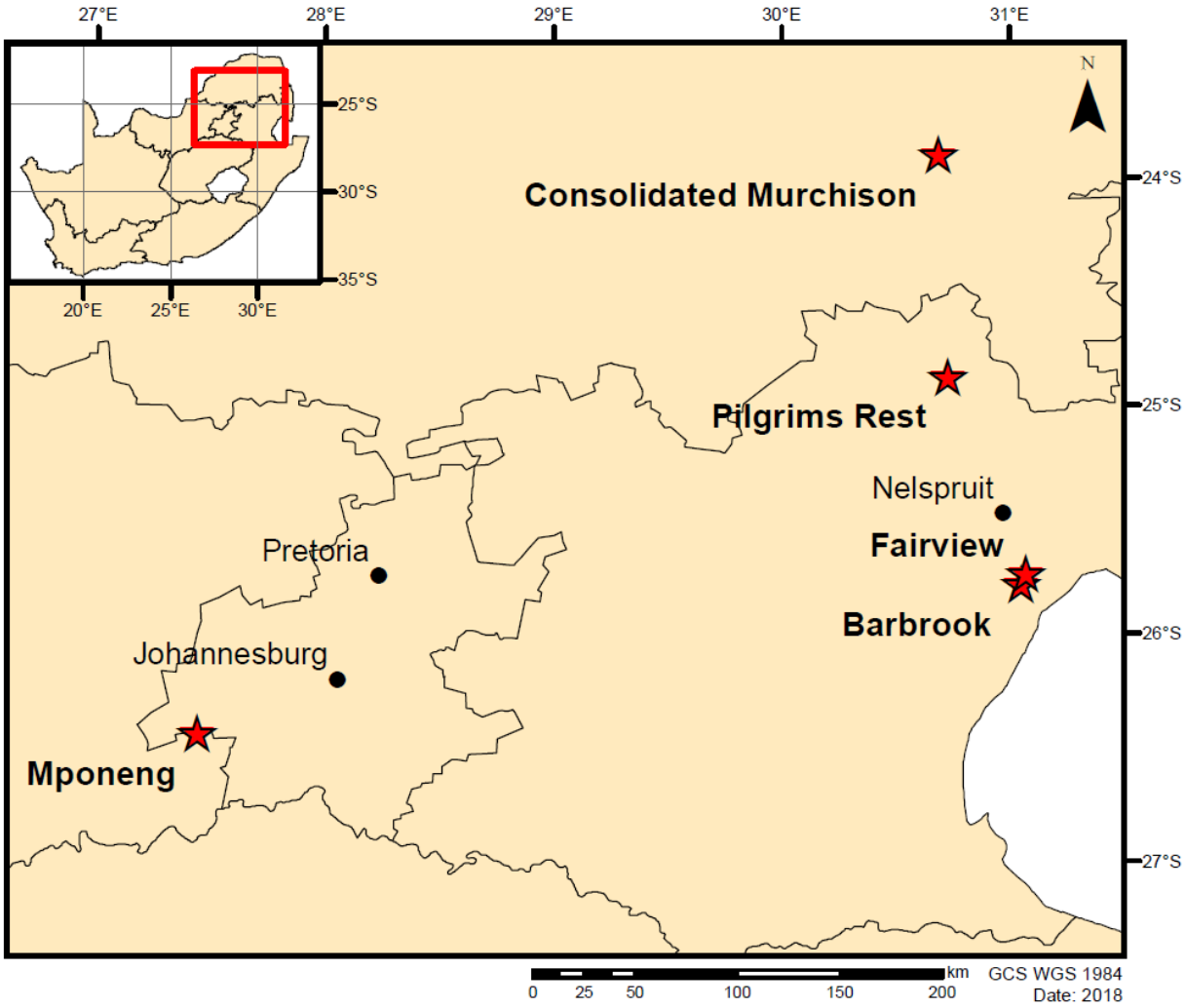


FIGURE 1 - Localities of the gold deposits studied.

3. EXTRACTIVE METALLURGY

Gold, a noble metal, by definition is resistant to chemical action, therefore there are a limited number of solvents capable of dissolving it (Moffa, 1977). Systems where gold dissolution is most efficient typically require an oxidant, capable of oxidising the gold, as well as a reactant/complexing agent that is able to form a reasonably stable gold complex (Kongolo and Mwema, 1998). Gold can be oxidised to two states; Au^{+1} and Au^{+3}, but only Au^{+3} forms the significantly stable, soluble complexes required for efficient gold leaching, therefore a strong oxidant is often required (Nam et al., 2008). This is the basis on which all gold dissolution and extraction is performed.

3.1. Cyanide

Cyanide has been used in the extraction of gold for over 100 years, so that the chemistry of its interaction with gold is well understood. Sodium (or potassium) cyanide is added to the ore to dissolve gold, which operates by the overall reaction mechanism (1) with intermediate steps (2) and (3) (Kongolo and Mwema, 1998);

$$4Au + 8NaCN + O_2 + 2H_2O \rightarrow 4AuNa(CN)_2 + 4NaOH \quad (1)$$

$$2Au + 4CN^- + O_2 + 2H_2O \rightarrow 2Au(CN)_2^- + 2OH^- + H_2O_2 \quad (2)$$

$$2Au + 4CN^- + H_2O_2 \rightarrow 2Au(CN)_2^- + 2OH^- \quad (3)$$

Oxygen acts as an oxidising agent, while CN^- dissociates from sodium, and complexes with the gold (Habashi, 1967). Cyanide-gold complexes are significantly more stable than alternative lixiviants, making it a useful tool in gold extraction (Chen et al., 1980; Aylmore, 2005; Aylmore, 2016a). For safety and efficiency this reaction is maintained at a basic pH of approximately 10-11 (Kongolo and Mwema, 1998).

3.1.1. Refractory Ores

The cyanidation process typically occurs quite readily, which accounts for its widespread use, but is limited in the presence of interfering agents. The presence of these agents allows for the classification of gold ores, while the classification system may vary from author to author (Chryssoulis et al., 1987; Haque, 1987; Monhemius, 1987; Komnitsas and Pooley, 1989; Zhou and Cabri, 2004; Chryssoulis and McMullen, 2005; Li and Miller, 2006; Karimi et al., 2010) the overall concept is the same. This description has been strongly based on Vaughan, 2004, which defines three distinct gold classifications exist in this system (FIG 2).

Free milling ores encompass all ores that readily undergo gold extraction by cyanide, within these ores there are no factors which complicate or reduce extraction efficiency. A gold recovery of 90% or more defines free milling ores (La Brooy et al., 1994; Marsden and House, 2006).

Complex ores are the next most difficult to process, this term refers to ores which can undergo gold extraction with a high dissolution rate, but only under altered/more intense leaching conditions. Complex ores can be subdivided into two categories; preg-robbing and reactive. Preg-robbing ores contain material capable of absorbing gold-cyanide complexes during leaching. This material tends to be carbonaceous matter (Vaughan, 2004; Goodall et al., 2005; Miller et al., 2005). The effects of preg-robbing can be countered (to a limited degree) by the addition of kerosene or similar binding agents to deactivate the carbonaceous material, by using Carbon-In-Leach, where activated carbon is present throughout the dissolution (La Brooy et al., 1994; Vaughan, 2004) or destroying the carbonaceous material via roasting. Reactive ore refers to any ore that contains material that react with reagents required for successful gold leaching. These reactive materials tend to be cyanicides or oxygen consumers. Examples of these minerals include; pyrrhotite, copper minerals (e.g. chalcocite, malachite, azurite and cuprite) with the exception of chalcopyrite, as well as most base metal sulphides (bearing iron, arsenic, antimony, zinc, nickel and cobalt) (Aylmore and Muir, 2001; Karimi et al., 2010). These ores can be dealt with by adding higher concentrations of reagents than would be required for standard cyanidation, although this may become economically unfeasible.

Refractory ores refer to deposits that cannot achieve acceptable gold dissolutions using standard procedures. There are two types of refractory ores, physically and chemically locked gold, both of which make the gold inaccessible to lixiviants. Physically locked gold occurs when gold is present as fine inclusions within a sulphide mineral. Chemically locked gold can be found in discrete gold minerals such as gold tellurides, aurostibite, aurocupride and maldonite (Harris, 1990; Vaughan, 2004). Chemically locked gold can also refer to gold present as solid solution within certain minerals, which means that atomic gold is present within crystal lattices (Chryssoulis et al., 1987; Haque, 1987; Monhemius, 1987; Komnitsas and Pooley, 1989; Vaughan and Kyin, 2004). Arsenopyrite and to a lesser extent pyrite act as host minerals to this solid solution gold (Vaughan, 2004; Vaughan and Kyin, 2004; Barker et al., 2009). To ensure efficient leaching of refractory ores, it is usually necessary to implement a pre-treatment step in the extraction process, examples of these pre-treatments include ultra-

fine milling, pressure oxidation, biological oxidation (BIOX) and, in older designs, roasting. Pressure cyanidation, while not a pre-treatment, may also be an effective option (Davies and Paterson, 1986; Haque, 1987; Monhemius, 1987; Lindström et al., 1992).

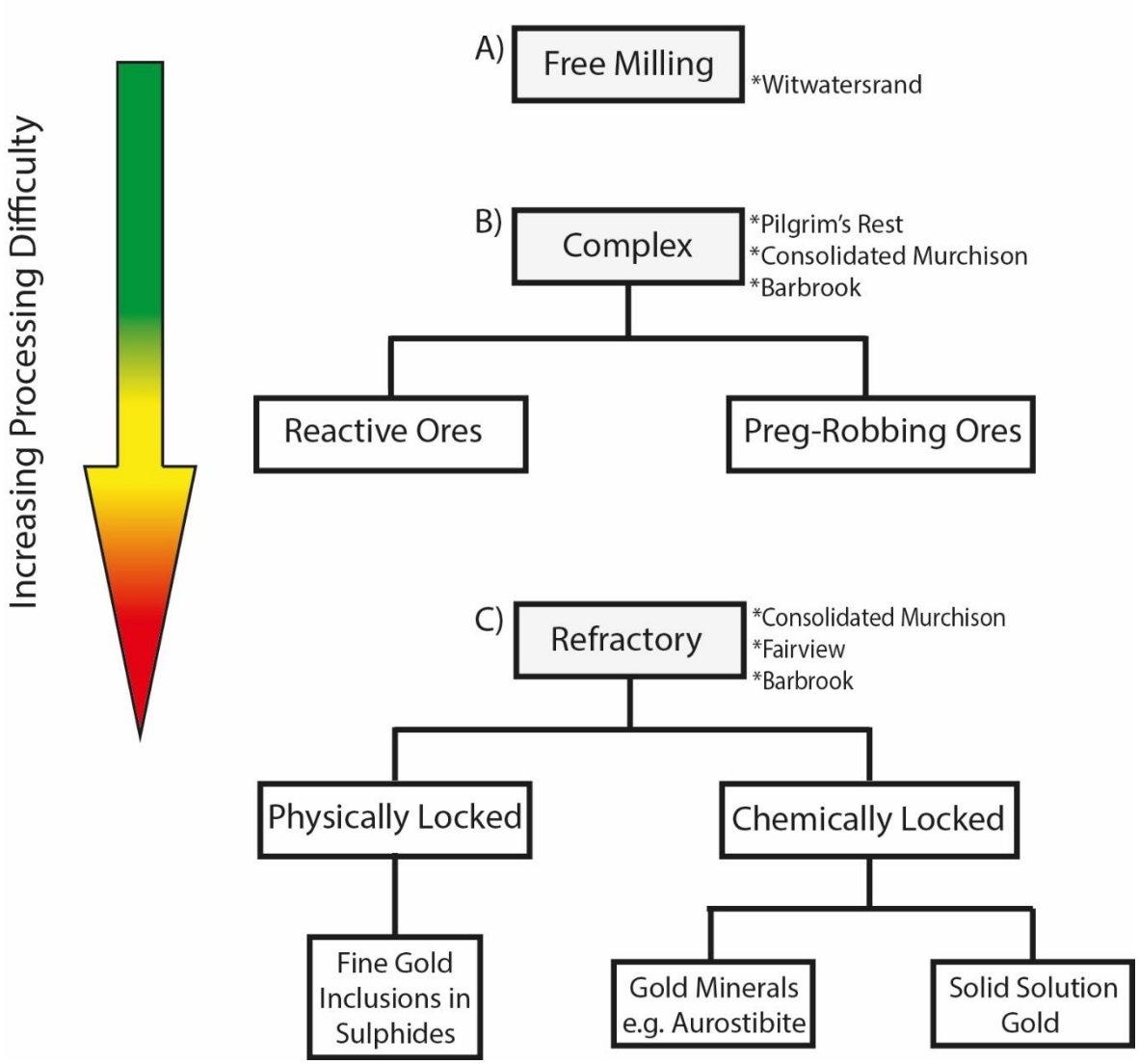


FIGURE 2 A diagram outlining the three main types of gold deposit types; 1) Free Milling, 2) Complex and 3) Refractory (in increasing order of processing difficulty). While the deposits of study and their respective mineralogy will be outlined, the deposits have been grouped with their respective deposit types.

3.2. Alternative Lixiviants

Alternative lixiviants have been extensively studied in recent years, and 27 possible systems have been investigated (Aylmore, 2005). Of these there are a limited number that show the potential to be financially and chemically feasible, these include; thiosulphate, thiourea and chloride. It should be noted that the commercial application of these systems is still in the developmental stages. All of these lixiviants form gold complexes that are orders of magnitude less stable than the gold cyanide complex, this complicates the chemistry further and requires more significant controls on the leaching conditions (FIG 3) (Chen et al., 1980; Jeffrey, 2001; Baghalha, 2007; Aylmore, 2016b).

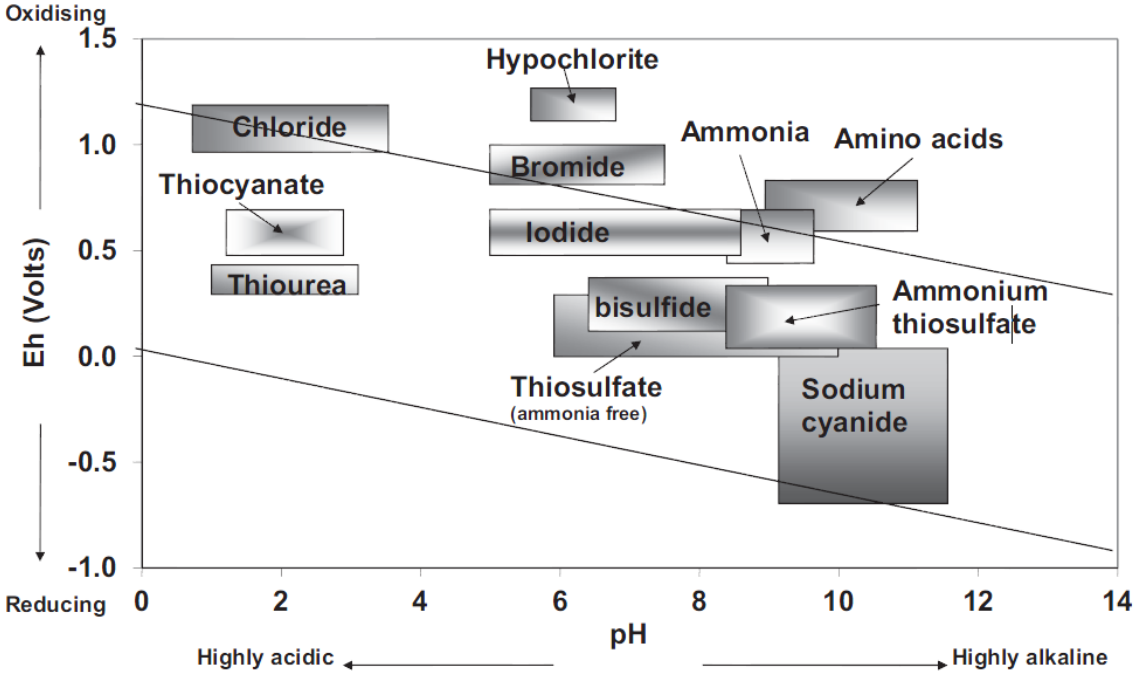


FIGURE 3 Stability fields (Eh and pH) of various gold lixiviants (Aylmore, 2016a). Note- the sodium cyanide operating conditions are significantly larger than any alternative lixiviant

3.2.1. Thiosulphate Leaching System

The thiosulphate leaching system is considerably more complicated than cyanide, as such the Eh, pH and reagent concentration must be precisely controlled, not only to promote gold leaching, but also to minimise the decomposition of thiosulphate (Hilson and Monhemius, 2006a). Despite these complications, the low toxicity, capacity to treat complex ores, and competitive dissolution rates make thiosulphate the best alternative lixiviant currently being researched (Aylmore, 2016a; Aylmore, 2016b). Thiosulphate dissolution of gold is slow without an oxidant more efficient than oxygen, as well as a catalyst (Abbruzzese et al., 1995). Extensive studies (Langhans et al., 1992; Breuer and Jeffrey, 2000; Jeffrey, 2001; Muir and Aylmore, 2004b; Aylmore, 2005) have found that the thiosulphate – copper – ammonia system is able to achieve kinetics of a similar rate to that of cyanidation. This reaction proceeds by the following mechanisms (4) and (5). Copper (II)-ammonia - thiosulphate complexes on to the surface of the gold, as the gold and thiosulphate oxidise. The copper acts as a catalyst.

$$Au + 5S_2O_3^{2-} + Cu(NH_3)_4^{2+} \rightarrow Au(S_2O_3)_2^{3-} + 4NH_3 + Cu(S_2O_3)_3^{5-} \quad (4)$$

$$4Cu(S_2O_3)_3^{5-} + 16NH_3 + O_2 + 2H_2O \rightarrow 4Cu(NH_3)_4^{2+} + 4OH^- + 12S_2O_3^{2-} \quad (5)$$

The thiosulphate can undergo oxidative decomposition if the conditions are not well controlled, which will negatively affect the leaching efficiency. Thiosulphate has been comprehensively researched for applications relating to carbonaceous and copper-gold deposits, as cyanide has limited suitability for these ores. Carbonaceous material is preg-robbing in a cyanide leaching system, while gold-thiosulphate complexes are not affected by this (Muir and Aylmore, 2004). Thiosulphate has been successfully developed for commercial use at Barrick's Goldstrike operation, Nevada (Aylmore, 2016a). Copper significantly increases cyanide consumption, consequently increasing the reagent cost (Kongolo and Mwema, 1998),

while in the thiosulphate system, copper is a necessary reagent and therefore its presence will reduce the amount and cost of reagent required.

3.2.2. Thiourea Leaching System

Thiourea has been actively studied for several years, although significant interest has waned slightly in recent years, as it has been classified as a potential carcinogen in certain animal species (Aylmore, 2005), despite this, research continues. Thiourea complexes with gold as can be seen in equation (6) (Li and Miller, 2006);

$$Au + 2NH_2CSNH_2 + Fe^{3+} \rightarrow [Au(NH_2CSNH_2)_2]^+ + Fe^{2+} \quad (6)$$

A number of oxidising agents could be used, but studies suggest that ferric ion is the most efficacious (Schulze, 1984; Ubaldini et al., 1998; Hilson and Monhemius, 2006b). The concentration of oxidising agents should be sufficient to oxidise 50% of thiourea to accelerate leaching conditions (Zhang et al., 2001). This leaching system is very sensitive to both Eh and pH conditions, as thiourea is inherently unstable and readily decomposes to substances incapable of leaching gold (Groenewald, 1976; Li and Miller, 2006). Despite these disadvantages, leaching rates of thiourea have been found to be up to 10 times faster than cyanidation (Hilson and Monhemius, 2006; Li and Miller, 2006), and good leaching conditions can be maintained if the Eh potential is between 0.42 and 0.45 V. Rapid leaching rates have major advantages for capital costs associated with plant construction, as leaching tanks can have a significantly lower capacity, as required capacity is proportional to leach time (Abbruzzese et al., 1995; Crundwell, 2005). Leaching takes place in acidic conditions, and as such it may be useful for refractory ores that have undergone oxidative pre-treatment and can undergo leaching without the neutralisation step required for cyanide leaching (Aylmore, 2016a). Gold associated with pyrite and chalcopyrite experience more rapid dissolution than standard leaching kinetics (Van Deventer et al., 1990), however, in the presence of certain base metals (especially copper), the thiourea consumption is significantly increased (Deschênes and Prud'homme, 1997). The presence of elemental sulphur and hydrogen sulphide will also affect leaching efficiency due to passivation and gold re-precipitation respectively.

3.2.3. Chloride/Hypochlorite Leaching System

The ability of chloride to extract gold has been known for over a century, and this was utilised considerably in the gold industry prior to the discovery of the cyanide system (Kongolo and Mwema, 1998), as well as in pre-treatment processes in refractory or carbonaceous ores (Marsden and House, 2006). For gold dissolution to take place in a chloride media, an oxidant is required. The oxidant used varies but at ambient temperature and pressure conditions, hypochlorite is thought to be the most effective reagent (Baghalha, 2007; Hasab et al., 2013; Hasab et al., 2014; Adams, 2016). The dissolution of gold in a chloride-hypochlorite leaching system operates by the following reaction mechanism (7) (Hasab et al., 2014);

$$2Au + 3HOCl + 3H^+ + 5Cl^- \rightarrow 2[AuCl_4]^- + 3H_2O \quad (7)$$

As with all leaching systems, chloride-hypochlorite has its own advantages and disadvantages; it has significantly faster kinetics than cyanidation, but the gold-chloride complex is unstable (Nam et al., 2008; Adams, 2016). Significant reagent consumption is a prohibitive factor in chloride usage, however in recent years much research has gone into

processes that allow for the reagents to be recycled (Aylmore, 2016a). The stability of $AuCl_4$ is strongly dependant on the solution pH as well as chloride and chlorine concentrations. The complex will also reprecipitate on contact with a reducing agent, and therefore the usefulness of this system will be extremely limited with regards to a sulphide-bearing ore (Adams, 2016; Aylmore, 2016a).

4. ORE CHARACTERISATION

Ore characterisation was undertaken using several methods including optical petrography, XRD (X-Ray Diffraction), chemical assay (using XRF, ICP-MS/AES and AAS), EPMA (Electron Probe Micro-Analyser) and MLA (Mineral Liberation Analyser).

The original run of mine material was studied to provide a holistic understanding of both the economic minerals as well as gangue, while gravity concentrated material was studied to fully understand the nature and characteristics of significant amounts of the ore mineral/s. The petrography has been combined with the MLA results. The gold was characterised using MLA, to obtain important data regarding grain size distribution, mineral association, liberation and exposure. EPMA was undertaken to locate and better understand the nature of the “invisible gold” thought to be present in the refractory gold ores of Fairview, Barbrook and Consolidated Murchison.

XRD was performed on the samples to determine the mineral compositions of each deposit, as an inexpensive and convenient way to identify major minerals present, especially those that might negatively affect the leaching process. The assays were undertaken to determine the presence of base metals, which may result in complex leach behaviour.

4.1. Ventersdorp Contact Reef (VCR)

The VCR is, in many ways a typical Witwatersrand ore; a conglomerate-hosted ore-bearing reef. The conglomeratic host tends to be matrix supported, however there is some variability in this throughout the composite ore. The conglomerate is polymict, with clasts of quartzites, vein quartz, and lesser chert pebbles. The matrix is dark green to black, and is composed of fine, rounded to sub-angular quartz grains, with interstitial chloritic and sericitic material, with a minor carbonate presence. Ore minerals present include minor pyrite, pyrrhotite, chalcopyrite as well as gold at an average grade of 23.6 g/ton.

The VCR ore has a range of gold grain sizes (FIG 4), from 0.5 to 425 μm. A significant proportion of total gold comes from the coarse gold fraction (212 to 425 μm), which will affect the leach times required to extract all available gold. At a grind of 80% -75 μm, the VCR sample had excellent exposure (99.8%), and liberation (98.49%), and its mineral associations are unlikely to significantly interfere with any extraction procedures, except for pyrrhotite. The native gold grains found in VCR sample are quite typical with approximately 10% silver present.

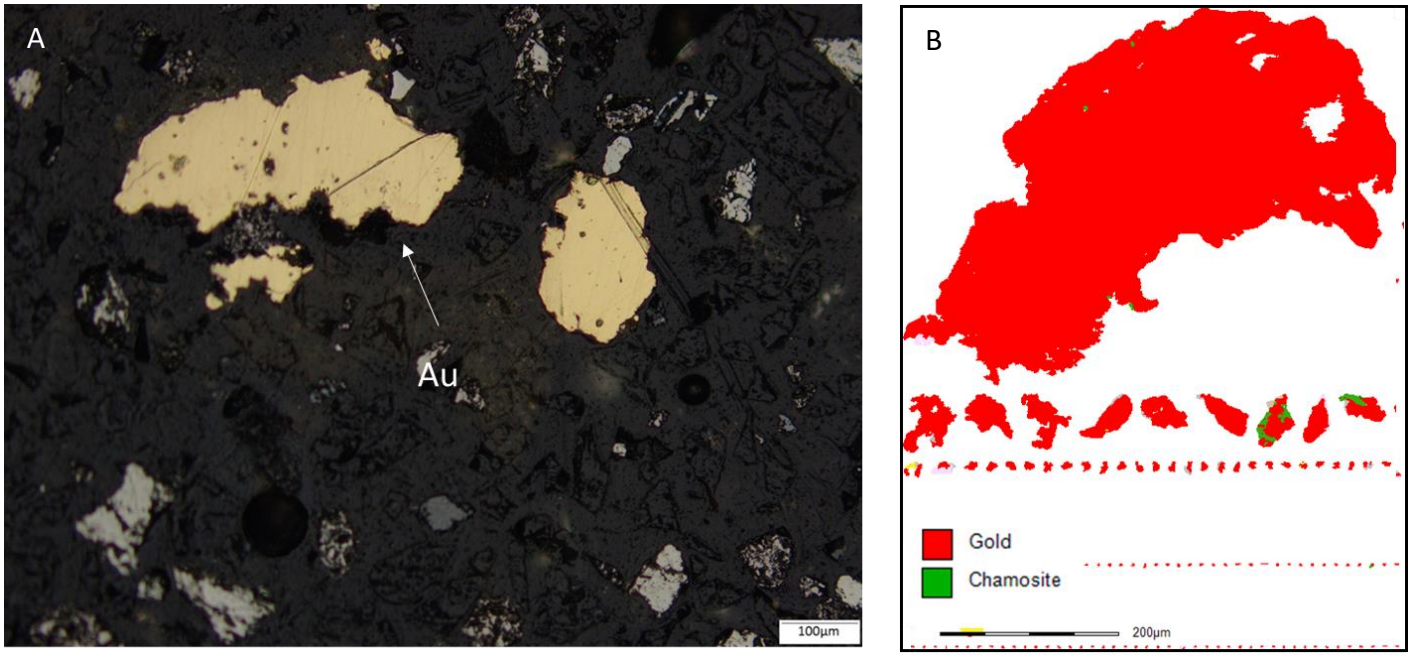


FIGURE 4 VCR gold mineralisation A) Gold grains with several surface exposures of gold, the largest of the gold grains (top left) has an equivalent circular diameter of greater than 250 µm. B) False colour image of a subset of gold grains found in the VCR, with excellent liberation and minor mineral associations.

4.2. Fairview

The mineralisation found in Barberton is hosted by an altered lithology comprising lithic quartz fragments found within a fine-grained matrix of quartz, chlorite, dolomite, magnesite and to a lesser extent muscovite. Ore minerals are either present in veins/fractures or as disseminations within the main lithology. Ore minerals (FIG 5A) include pyrite, arsenopyrite and minor visible gold (averaging 12.5 g/ton).

Fairview, at a grind of 80% -75 µm, has visible native gold that is 72% liberated, with a surface exposure/leach reactivity of 75%. Gold grains range in size from 0.5 µm to 425 µm (FIG 5B), with the most significant weight percent contribution to total gold in the range of 180 µm and 425 µm. The large range in grain size, as well as presence of coarse gold will affect the leach time. Pyrite, arsenopyrite, pyrrhotite and micas are commonly associated with gold. The limited extent of visible gold present, relative to the grade, suggests that there is gold invisible to both optical petrology and MLA. This has been confirmed by EMPA analyses that have determined the presence of gold hosted within arsenopyrite, as predicted by various sources of literature (Anheusser, 2001; Anhaeusser, 2015).

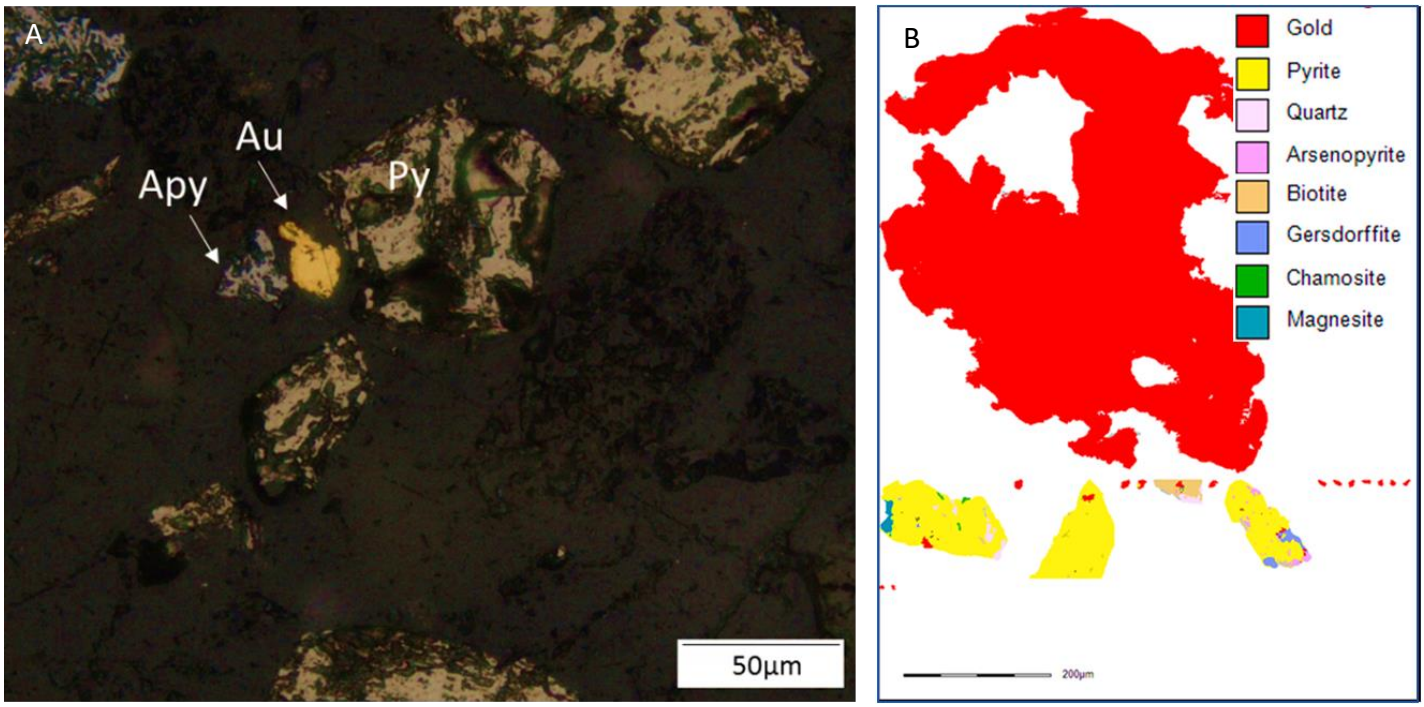


FIGURE 5 Fairview ore minerals A) Example of mineralisation, including arsenopyrite (Apy), pyrite (Py) and gold (Au). The gold grain has an equivalent circular diameter of 46.5 µm. B) The range in size of gold grains at Fairview is evident in this false colour MLA-generated image of polished blocks of panned concentrate.

4.3. Barbrook

The host lithology is predominantly an altered silicic carbonate, with extensive veins. The groundmass comprises predominantly amorphous dolomite, fine-grained quartz fragments with chlorite, muscovite and carbon (FIG 6A). The amount of iron-rich material present in the sedimentary units is variable, and can range from highly abundant to entirely absent, secondary iron hydroxides such as limonite and goethite are also present. Ore minerals evident in Barbrook include pyrite, arsenopyrite, with minor chalcocite, galena, pyrrhotite and sphalerite. The abundance of reactive minerals is mirrored in the elevated concentrations of copper, sulphur and iron.

In Barbrook, the limited number of gold grains, identified using MLA (FIG 6B) is likely due to the low grade (2.08 g/t average grade). This reduces the scope of the conclusions that can be made. However, based on all data obtained, the visible gold has a liberation of 97%, while 99% of gold is exposed for leaching. Of the gold grains found, the grain size distribution ranges from 1 µm to 53 µm. The gold tends to only be associated with pyrite, arsenopyrite, ankerite and chlorite which are unlikely to significantly affect leaching conditions. (Anheusser, 2001; Pearton and Viljoen, 2017). While it is likely that there is refractory gold present in this sample, none was located (again likely to be due to low grades/concentrations).

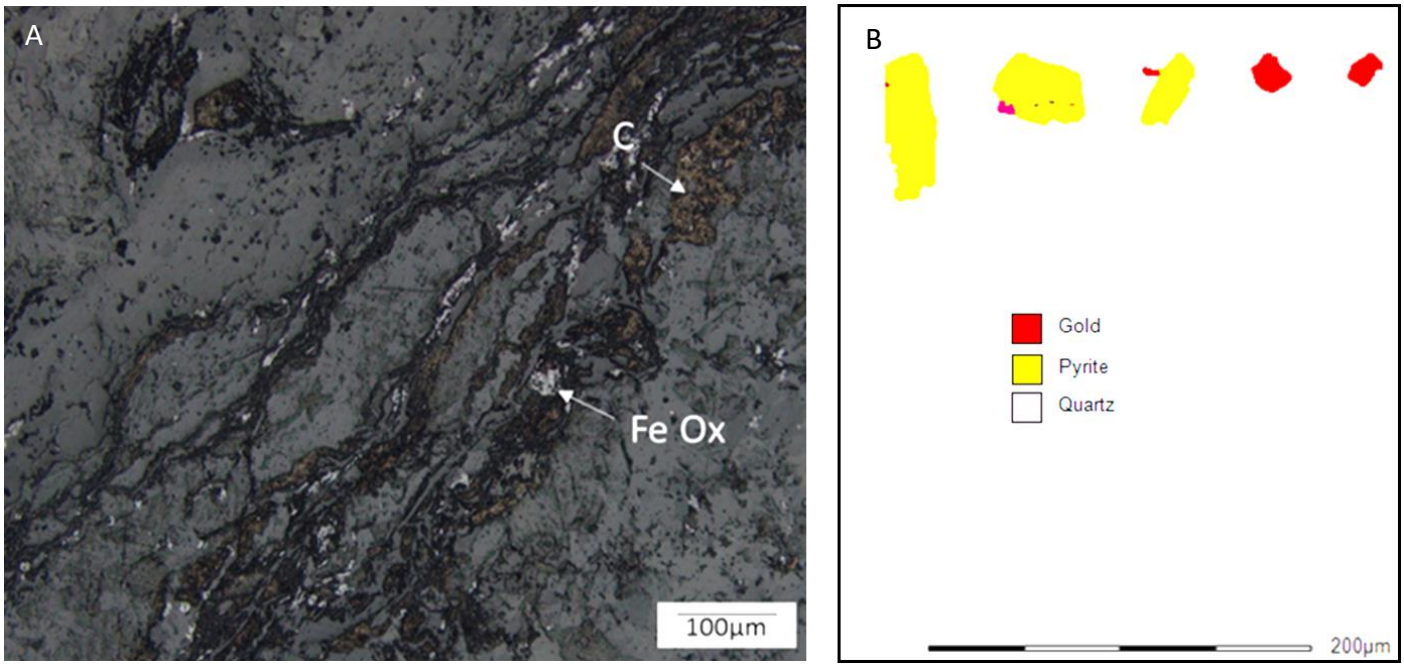


FIGURE 6 Features of interest from Barbrook ore A) Example of carbon associated with various iron oxides. B) Barbrook gold grains, with pyrite and minor quartz association represented.

4.4. Consolidated Murchison

The mineralisation within the Consolidated Murchison samples is hosted by a dominantly carbonate- and quartz-bearing rock, with chlorite, biotite, magnesite, talc and locally abundant tourmaline with a schistose texture. Ore minerals present include stibnite (Sb_2S_3), pyrite, berthierite ($FeSb_2S_4$), and lesser arsenopyrite (FIG 7A). The mineralisation tends to present as homogenous disseminations throughout the schist. The presence of Sb-rich minerals and high Sb concentrations suggest this ore is refractory.

While Consolidated Murchison is lower grade (typically 3.04 g/ton) the MLA results determined the presence of both native gold and aurostibite (FIG 7B). The gold has good liberation (88%) and exposure (85%), while the aurostibite has excellent liberation (96%) and 100% of the aurostibite was exposed. These grains range in size from 0.73 µm to 52 µm. Mineral associations include arsenopyrite, pyrite and stibnite. Associations with antimony-rich minerals, such as stibnite, will affect the gold leaching, due to both reagent consumption and passivation. While there is limited native gold, there is also aurostibite, as well as gold associated with stibnite identified from EPMA analyses. This confirms the consensus of literature which states that gold is present in the Antimony Line in a number of forms (Davis, 1986; Vearncombe et al., 1988; Marsden and House, 2006).

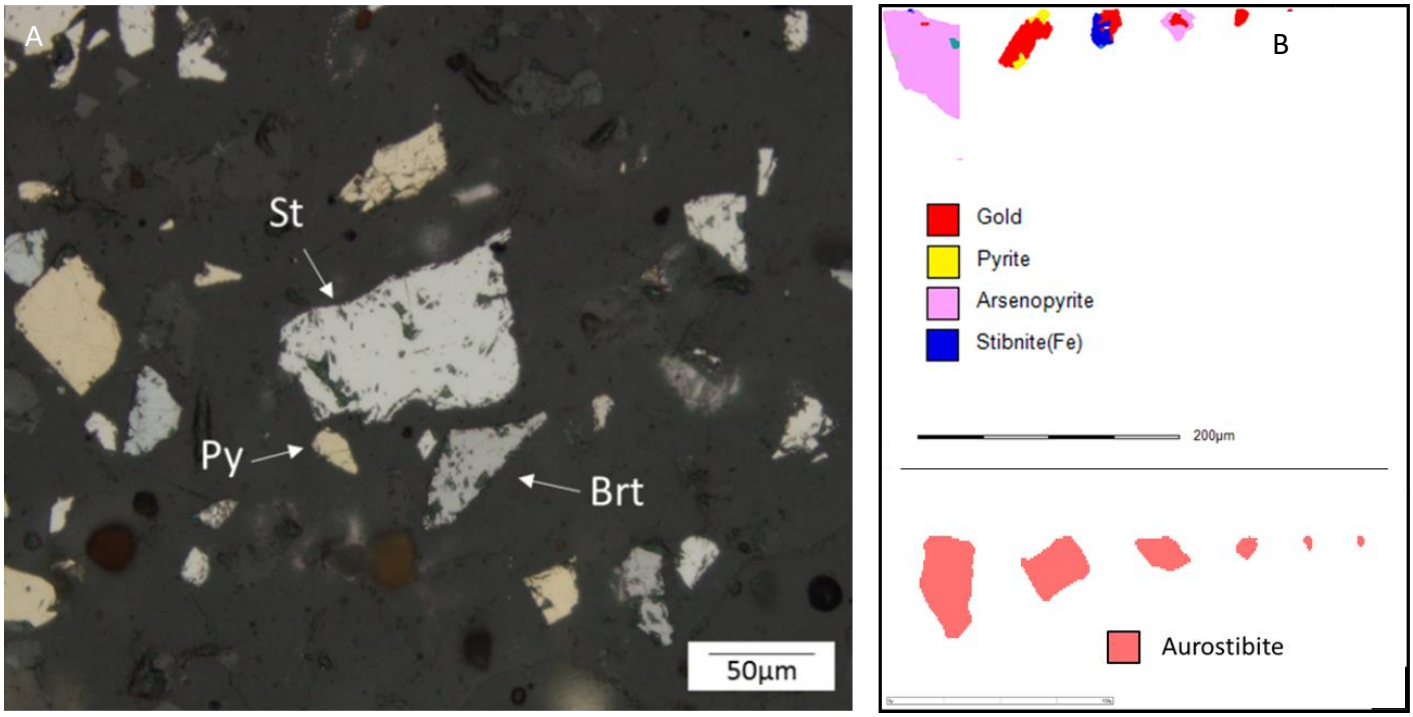


FIGURE 7 Mineralisation at Consolidated Murchison A) An example of mineralisation that includes stibnite (St), pyrite (Py), and berthierite (Brt). B) Gold and aurostibite grains, highlighting the lower degree of liberation in native gold, while aurostibite grains have excellent liberation and exposure.

4.5. Pilgrim's Rest

The host lithology varies between fine-grained siltstones and shales, commonly associated with fine calcite and muscovite. All original lithologies are crosscut by quartz-carbonate veins. In the region where the ore was sampled, the quartz veining and associated mineralisation are more extensive than the host lithologies. The ore mineralogy includes abundant pyrite (FIG 8A) with lesser pyrrhotite, chalcocite, chalcopyrite and native gold. Pyrite represents approximately one third, by mass, of the feed, accounting for the high iron and sulphur concentrations, while the presence of chalcocite and chalcopyrite account for notable copper concentrations. Mineralisation occurs extensively within veins, as disseminations and even as massive mineralisation.

The Pilgrim's Rest sample had a high grade of gold (averaging 29.4 g/ton). The gold occurs as discrete, fine grains, ranging in size from 0.75 µm to 53 µm (FIG 8B). The gold present appears to have small amounts of silver (5%), with a minor presence of electrum (with silver contributing more than 20% to a grain). While the literature suggests that this ore typically contains significant proportions of gold less than 5 µm in size (Swiegers, 1948), the weight percent contribution of gold within this range is insignificant and more than 85 wt% of the gold is from grains between 10 µm to 53 µm. Thus, a very fine grinding would not increase gold availability significantly. This assessment is supported by the MLA liberation and exposure data which show that the gold is well liberated (97%) and exposed (98%) with a grind size (80% -75 µm). Major gold mineral associations include pyrite and quartz which are unlikely to affect the gold extraction. However minor amounts of pyrrhotite and other sulphides may be deleterious.

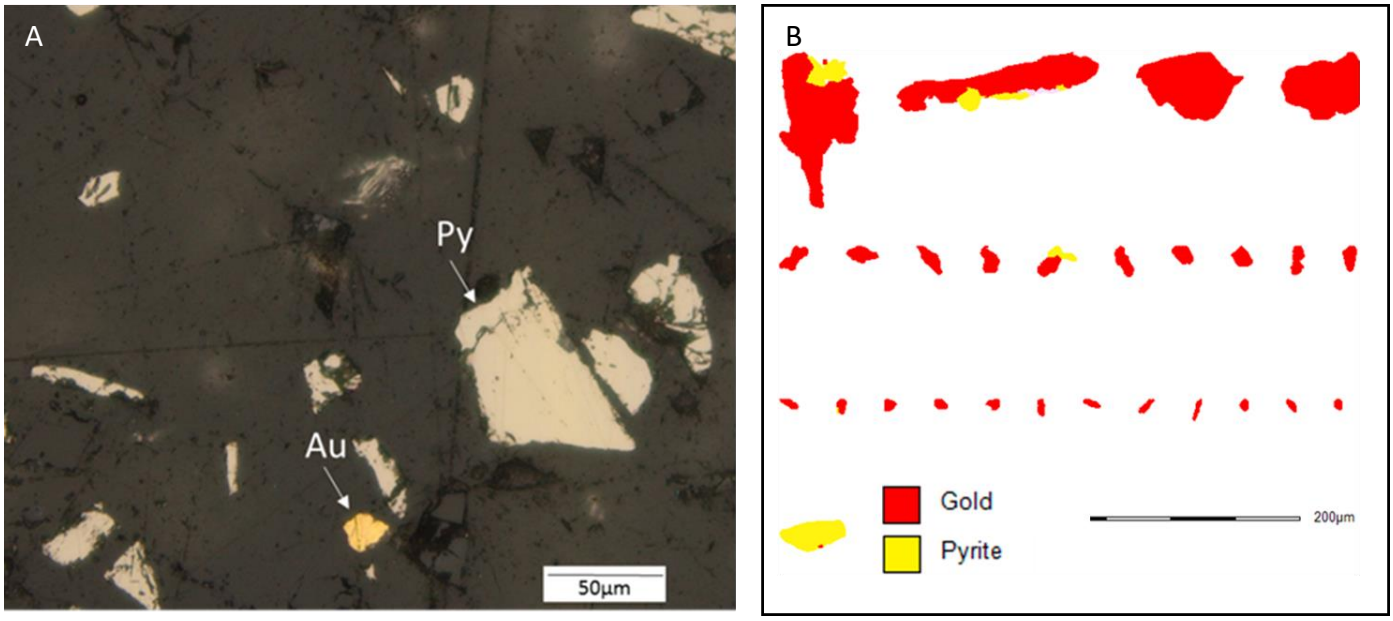


FIG 8 Pilgrim's Rest mineralisation A) A gold grain with an equivalent circular diameter of 16 µm. Significant pyrite mineralisation is also evident. B) MLA images of Pilgrim's Rest gold grains, highlighting the range in size, and excellent liberation and exposure.

5. LEACHING

The standardised leaching conditions selected for this test work are based on a number of resources for cyanide systems, which are well defined in literature (Schaeffer, 1882; Eisele et al., 1988; Dadgar, 1989; Sparrow and Woodcock, 1995; Jeffrey et al., 2001; Swanepoel, 2001).The leaching was conducted over a 24-hour period, at a NaCN concentration of 1 000ppm. The leach was maintained at ambient conditions, while the pH was adjusted, as is required for both safety and optimisation, to 10.5, with a solid/liquid ratio of 50%.

Conditions for each sample were monitored to maintain a consistent leaching environment, and as such, reagent addition for different samples was variable (TABLE 1). Increasing lime consumption is evident from Fairview, VCR, Barbrook, Consolidated Murchison to Pilgrim's Rest. The first 4 samples required only minor amounts of lime addition in order to maintain required pH conditions. However, the Pilgrim's Rest sample required an order of magnitude more lime. Cyanide consumption increased from VCR, Fairview, Consolidated Murchison, Barbrook and Pilgrim's Rest, ranging from 1.2 kg/t to 2.41 kg/t. Final concentrations of cyanide show that sufficient quantities were present for leaching to still have been occurring for samples VCR, Fairview, Consolidated Murchison and Barbrook.

The range in gold dissolution (TABLE 1) for the different ores emphasise the variability in ore types, ranging from just less than 100% to 14%. The VCR had almost complete dissolution, further emphasising its free milling nature. The Pilgrim's Rest sample, despite having the highest consumptions for both cyanide and lime, had a reasonable dissolution of approximately 90%. The Fairview sample had the next best gold dissolution; however, this still fell below the level of technical acceptable recovery, with an average of 66.6%. Both Barbrook and Consolidated Murchison samples attained very poor dissolutions, with 26.6% and 14% respectively.

TABLE 0-1 Summarised results from the test leach showing reagent consumptions, end of leach conditions and gold dissolution.

Sample ID	Reagent Addition			Reagent Consumption		Final pH	Au Dissolution
	CaO (kg/t)	Carbon (g/l)	NaCN (ppm)	NaCN (kg/t Feed)	CaO (kg/t Feed)		(%)
VCR	0.30	20.0	1680.4	1.20	0.27	10.41	96.2
FV	0.25	20.0	1734.0	1.33	0.22	10.30	64.7
CM	0.43	20.0	1807.2	1.47	0.40	10.29	13.5
BK	0.42	20.0	2445.3	2.18	0.39	10.38	28.5
PG	3.88	20.0	2451.2	2.41	3.85	10.19	92.0

6. DISCUSSION

Cyanide leaching may be impeded by several elements found in solution associated with the dissolution of sulphides. These metals; copper, zinc and iron, as well as nickel, cobalt, manganese, antimony and arsenic, can complex with cyanide which inhibits dissolution due to the reduced presence of reagents required for leaching (Safizadeh and Larachi, 2014). Increasing reagent consumption is likely to be related to the increasing presence of cyanide consumers. Remarkably high copper and iron concentrations in the Pilgrim's Rest sample, high iron and copper concentrations in Barbrook material and elevated antimony concentrations in Consolidated Murchison ore account for their high cyanide consumptions. These elevated elemental concentrations link to previously described mineralogy; e.g. the high reagent consumptions for Pilgrim's Rest is due to the presence of pyrrhotite, chalcocite and chalcopyrite. The deleterious components responsible for the poor performance of the Pilgrim's Rest ore may be the pyrite, chalcopyrite and sphalerite, while consumptions for Consolidated Murchison could be due any number of antinomy minerals.

Pilgrim's Rest is the only sample that has significant lime consumption, in addition to cyanide consumption. The elevated acidic nature of Pilgrim's Rest samples is likely to be due to the significant concentration of iron sulphide minerals, which tend to be oxidised during the cyanide leach, thus resulting in the formation of acid (Heinen et al., 1978).

Other factors that may be responsible for poor recovery relate to the presence of both chemically and physically locked gold identified in both Fairview and Consolidated Murchison samples, which would be inaccessible for cyanidation without a pre-treatment process.

Unfortunately, complete results are not currently available for alternative methods, however, preliminary investigations show very high reagent consumption, specifically for ores that contain high levels of sulphides. Despite high consumptions being evident, this may be outweighed by other factors under specific conditions. For example, the preg-robbing nature of Barbrook, which could account for the poor dissolution seen under excess cyanide conditions, would not be a deleterious factor when using thiosulphate leaching. Copper, a cyanide consumer found in Pilgrim's Rest (and Barbrook) ores, is a necessary component for effective thiosulphate leaching. The presence of copper in this ore, which would elevate costs during cyanide leaching would reduce costs in thiosulphate leaching. Thiourea, with its more rapid leaching rate could prove useful in conditions where coarse gold is present, and extended leach times would be required for complete dissolution using cyanide. This same advantage could also be applied to a chloride leach, although the chloride leach is likely to have very high reagent consumptions and may not be safer than cyanide. The acidic conditions under which thiourea leaching takes place could eliminate the need for pH adjustment after a pre-treatment process, which is commonly required for refractory ores such as those from Fairview.

7. CONCLUSIONS

Free milling ores are cheap and convenient to leach using cyanide therefore, it is unlikely that an alternative process would be economic or feasible for these. However, when ores are refractory, and either respond poorly to cyanide or consuming large amounts reagent, an alternative such as thiosulphate, even with high consumptions, may be feasible if good gold dissolution is achieved. Further investigations are warranted, possible leach tests that should be undertaken include thiosulphate leaching with Barbrook and Pilgrim's Rest ores, and thiourea and chloride leaching on the refractory hosted gold ores found in Barberton and Consolidated Murchison.

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Publication Manuscript 2

A Comparison of Thiourea and Chloride-Hypochlorite Leaching on South African Gold Ores

ABSTRACT

Non-cyanide leaching techniques are being extensively studied due to increasing environmental concerns, but another important consideration is the increasing number of metallurgically complex deposits. These alternatives need to be evaluated on the basis of their toxicity and environmental impact but also on their gold dissolution and operating expenses. The gold dissolutions obtained for the alternatives were of limited success due to instability of the leach and the resulting gold complexes. In addition, reagent costs for thiourea and chloride hypochlorite are significantly higher than cyanide leaches, due to the number of reagents, as well as high concentrations required. Reagent costs also drastically increase when the deposits are carbonate-bearing as these alternatives are most effective at low pH.

1. INTRODUCTION

Despite the importance of cyanide to the gold industry, increasing environmental concerns have been voiced in past decades, owing to a number of well publicised incidents involving cyanide in the mining industry (Donato et al., 2007; Laitos, 2012; Aylmore, 2016a). Ensuing public concern resulted in a number of countries having introduced varying degrees of legislation related to cyanide (Laitos, 2012). These restrictions, as well as a reduction in the number of cyanide amenable gold deposits, have acted as a global initiative to establish economic and environmentally friendly alternatives to cyanide (Aylmore, 2005; Aylmore, 2016a).

2. EXTRACTIVE METALLURGY

Systems where gold dissolution is most efficient typically require an oxidant, capable of oxidising the gold, as well as a reactant/complexing agent that is able to form a reasonably stable gold complex (Kongolo and Mwema, 1998). Gold can be oxidised to two states; Au^{+1} and Au^{+3}, but only Au^{+3} forms the significantly stable, soluble complexes required for efficient gold leaching, therefore a strong oxidant is often required (Nam et al., 2008). This is the basis on which all gold leaching is performed.

2.1. Cyanide

Cyanide has been used in the extraction of gold for over 100 years, such that the chemistry of its interaction with gold is well understood. Sodium (or potassium) cyanide is added to the ore to dissolve gold, which operates by the overall reaction mechanism (1) with intermediate steps (2) and (3) (Kongolo and Mwema, 1998);

$$4Au + 8CN^- + O_2 + 2H_2O \rightarrow 4Au(CN)_2^- + 4OH^- \quad (1)$$

$$2Au + 4CN^- + O_2 + 2H_2O \rightarrow 2Au(CN)_2^- + 2OH^- + H_2O_2 \quad (2)$$

$$2Au + 4CN^- + H_2O_2 \rightarrow 2Au(CN)_2^- + 2OH^- \quad (3)$$

Oxygen acts as an oxidising agent, while CN^- dissociates from sodium, and complexes with the gold (Habashi, 1967). Cyanide-gold complexes are significantly more stable than alternative lixiviants, making it a useful tool in gold extraction (Chen et al., 1980; Aylmore, 2005; Aylmore, 2016a). For safety and efficiency this reaction is maintained at a basic pH of approximately 10-11 (Kongolo and Mwema, 1998).

2.2. Alternative Lixiviants

Due to increasing restrictions and scrutiny on cyanide usage around the world, along with the increasing number of refractory/complex gold ores, there has been extensive study of alternative lixiviants in recent years. While many possible systems have been investigated, of these there are a limited number that show the potential to be financially and chemically feasible. In this work both thiourea and chloride-hypochlorite were tested and compared with cyanide in the context of several South African ores. Reduced stability fields, in pH-redox space, associated with alternatives as seen in Figure 1 emphasise key difference in stability between cyanide and alternatives (Chen et al., 1980; Jeffrey, 2001; Baghalha, 2007; Aylmore, 2016b).

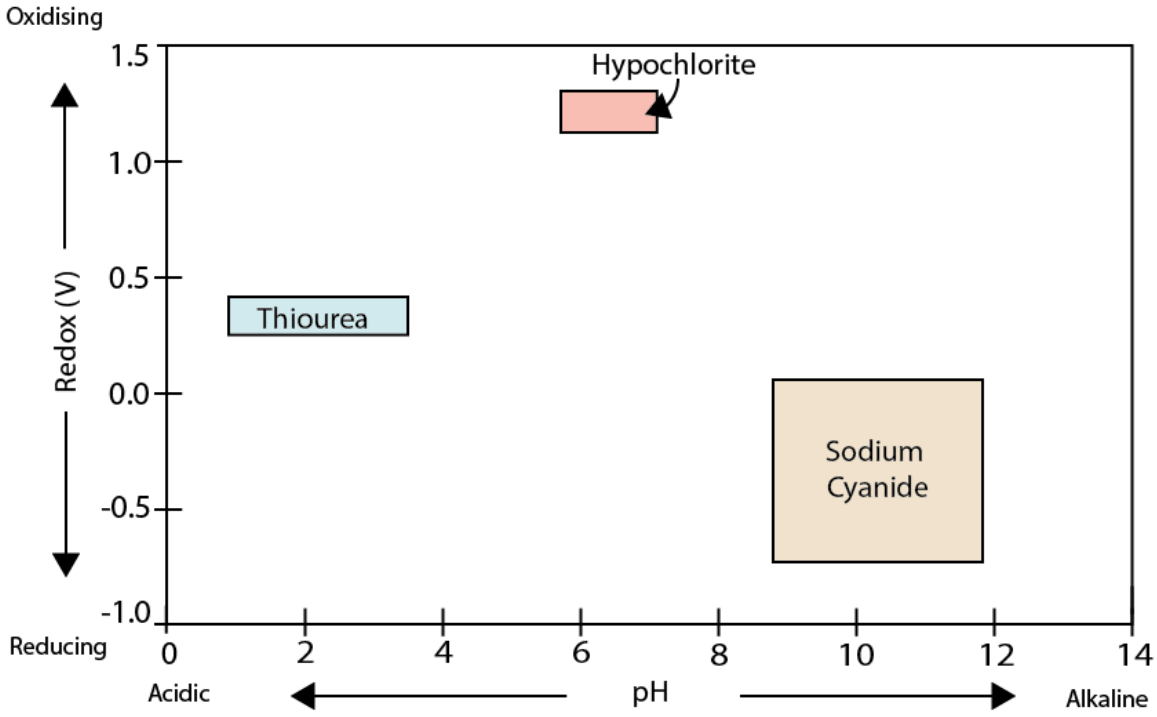


Figure 1 Stability fields (Eh and pH) of various gold lixiviants (Aylmore, 2016a). Note- the sodium cyanide operating conditions are significantly larger than any alternative lixiviant.

2.2.1. Thiourea Leaching System

Thiourea leaching is one of the more recently discovered gold extraction methods, proposed by Plaskin and Kozhukhova in 1941, and subsequently studied extensively. However significant interest has waned slightly in recent years, as it has been classified as a potential carcinogen in certain animal species (Aylmore, 2005). Thiourea complexes with gold as can be seen in equation (4) (Li and Miller, 2006);

$$Au + 2NH_2CSNH_2 + Fe^{3+} \rightarrow [Au(NH_2CSNH_2)_2]^+ + Fe^{2+} \quad (4)$$

This system requires ferric ion in order to facilitate the oxidation required for leaching. This leaching system is very sensitive to both Eh and pH conditions, as thiourea is inherently unstable and readily decomposes to substances incapable of leaching gold (Groenewald, 1976; Li and Miller, 2006). Despite these disadvantages, leaching rates can be an order of magnitude higher than cyanide should conditions be properly maintained (Hilson and Monhemius, 2006; Li and Miller, 2006). Leaching takes place in acidic media, and as such it may be useful for refractory ores that have undergone oxidative pre-treatment (such as pressure or bacterial oxidation) and can undergo leaching without the neutralisation step required for cyanide leaching (Aylmore, 2016a).

2.2.2. Chloride/Hypochlorite Leaching System

The ability of halides to extract gold has been known for over a century, and this was utilised considerably in the gold industry prior to the discovery of the cyanide system (Kongolo and Mwema, 1998) (Marsden and House, 2006). The chlorine-based extraction described in this work refers to chloride, oxidised by hypochlorite, allowing chlorine to be generated in-situ where it occurs in the aqueous form, reducing safety concerns. Hypochlorite is by far the most effective oxidant for this system at ambient conditions (Baghalha, 2007; Hasab et al., 2013; Hasab et al., 2014; Adams, 2016). The dissolution of gold in a chloride-hypochlorite leaching system operates by the following reaction mechanism (5) (Hasab et al., 2014);

$$2Au + 3HOCl + 3H^+ + 5Cl^- \rightarrow 2[AuCl_4]^- + 3H_2O \quad (5)$$

As with all leaching systems, chloride-hypochlorite has its own advantages and disadvantages; it has significantly faster kinetics than cyanidation, but the gold-chloride complex is unstable (Nam et al., 2008; Adams, 2016).

3. ORE CHARACTERISATION

There are several gold-bearing deposits found throughout South Africa, ranging from world class to small scale. The deposits of study include the Witwatersrand/VCR (Mponeng), two deposits from the Barberton granite and greenstone belt - Fairview and Barbrook, as well as Consolidated Murchison and Pilgrim's Rest. These represent a huge range in deposit types and mineralogies, against which the various techniques can be evaluated.

3.1. Gold Mineralogy

The mineralogy of the gold from these deposits is extremely variable. High grade deposits include VCR, Fairview and Pilgrim's Rest, while refractory gold is found in Fairview, Barbrook and Consolidated Murchison.

The VCR's sample had a gold grade of 23.6 g/ton with the vast majority of gold occurring in grains greater than 200 µm, which has implications for leach time requirements. Despite excellent exposure, gold had minor associations with minerals that likely to be of concern during processing such as pyrrhotite, chlorite and pyrite. Pyrrhotite may have implications for cyanide processing, while chlorite and pyrite may be reactive in alternative systems.

The Pilgrim's Rest sample had both high gold grades (29.3 g/ton) and fine grain sizes, making this ore more amenable to dissolution. Unfortunately, significant pyrite found in the ore is thought to have a deleterious effect on non-cyanide leaching techniques.

Although the Fairview sample was host to high gold grades (12.5 g/ton) and native gold, there was also a moderate component of the gold locked in solid solution in arsenopyrite. Much of the native gold in the Fairview sample occurred as coarse gold which would extend leach times. Significant amounts of the gold in the Fairview sample was associated with pyrite, arsenopyrite and pyrrhotite.

The Barbrook sample contained both native gold as well as gold locked in solid solution arsenopyrite, which combined amounted to gold grades of 2.08 g/ton. Furthermore, gold associated with arsenopyrite and pyrite may affect the results of leaching.

The Consolidated Murchison sample was host to both native gold and aurostibite, in addition to significant association with arsenopyrite, stibnite and berthierite which are reactive and passivating in cyanide leaches. The low grade (3.04 g/ton) and deleterious nature of the Consolidated Murchison ore has historically been mitigated by the recovery of antimony in addition to gold.

3.2. Gangue Mineralogy

The gangue mineralogy of the VCR sample was largely inert in all of the leaching systems considered in the present study due to large quantities of quartz, however chlorite (1.1%) and muscovite (1%) are present in minor amounts. While these components are not reactive in a cyanide leach, they may affect consumption in the leaches of the alternatives. The Fairview sample had large amounts of reactive gangue, of foremost concern is dolomite (10.8%) and magnesite (7.1%), while the presence of significant muscovite (24.8%) and chlorite (13.4%) is also unfavourable. The Barbrook sample, in addition to having refractory ores also has preg-robbing carbon which will affect cyanide leaches, while dolomite (58.0%) and to a lesser extent chlorite (6.4%) are likely to affect the acidic leaches. The Consolidated Murchison samples contains only magnesite (38%) as an acid consuming mineral, however the quantity is sufficient to be extremely unfavourable. While the Pilgrim's Rest sample only has moderate reactive components, dolomite (4.9%) and muscovite (4.9%), the major abundance of pyrite (32.6%) tends to make leaches unstable.

4. LEACHING

The extractive capacity of the various systems was tested using bottle roll leaching, which involved exposing ore to leach solution in a glass bottle, which was rolled at 72 rpm to induce agitation. This technique is an important first step in understanding the ores response to the selected leaching system on a small scale. Bottle roll leaching allows for a small-scale evaluation of important factors such as the total amount of leachable metal, reagent consumption and degree of monitoring required (Cetin et al., 2017).

The experimental work for this study was undertaken at ambient temperature and pressure conditions, on ore milled to 80% passing 75 µm and aliquots of between 400 and 500g, where they were leached with the equivalent mass of water (to ensure a 50% solid content), while the appropriate amount of reagent was added. Excess reagent conditions were selected to reduce the number of variables effecting dissolution. The leaches were run over a 24- and 8-hour period for cyanide and the alternatives respectively. The conditions were adjusted approximately once an hour until stable conditions were achieved.

4.1. Cyanide Leaching

The standardised leaching conditions selected for this test work are based on a number of well-defined resources for cyanide systems (Schaeffer, 1882; Eisele et al., 1988; Dadgar, 1989; Sparrow and Woodcock, 1995; Jeffrey et al., 2001; Swanepoel, 2001). Leaching was conducted over a 24-hour

period, at a NaCN concentration of 1 000 ppm. The leach was maintained at ambient conditions, while the pH was adjusted, as is required for both safety and optimisation, to 10.5.

Cyanide consumption increased from VCR, Fairview, Consolidated Murchison and Barbrook to the Pilgrim's Rest sample, ranging from 1.2 kg/t to 2.41 kg/t. Final concentrations of cyanide show that sufficient quantities of reagent were present for leaching to have occurred during the full 24 hours for samples from VCR, Fairview, Consolidated Murchison and Barbrook. However, the Pilgrim's Rest leach was likely to be cyanide starved, in spite of adding 2.4 kg/t of cyanide, over the duration of the leach, the final residual cyanide concentration was 0.04 kg/t.

The range in gold dissolution by cyanide for VCR (96.07%), Fairview (68.17%), Barbrook (24.13%), Consolidated Murchison (21.26%) and Pilgrim's Rest (88.96%) emphasises the variability in ore types within the studied sample (Table). The VCR sample had almost complete dissolution, further emphasising its free milling nature. The Pilgrim's Rest sample, despite having the highest consumptions for both cyanide and lime, had a reasonable dissolution. The Fairview sample had the next highest gold dissolution; however, gold dissolution was low for this sample. Both Barbrook and Consolidated Murchison samples attained very poor dissolutions.

Table 1 Summarised results from the leaches that were run to completion, showing conditions maintained, dissolutions achieved and reagent consumptions.

		Feed		Residue		Dissolution	Conditions		Consumption	
		Mass	Grade	Mass	Grade	Feed/Res	pH	Eh	Reagent	Unit
Lixiviant	Ore	g	g/t	g	g/t	%		mV		kg/ton
Cyanide	VCR	433.7	23.6	432.5	0.9	96.1	10.5	Natural	Cyanide	1.2
	Fairview	478.1	12.5	477.9	4.0	68.2	9.6	Natural	Cyanide	1.3
	Barbrook	457.3	2.1	453.9	1.6	24.1	10.4	Natural	Cyanide	2.2
	Consolidated Murchison	509.8	3.1	505.9	2.4	21.3	9.2	Natural	Cyanide	1.5
	Pilgrim's Rest	453.8	29.4	450.5	3.3	89.0	10.0	Natural	Cyanide	2.4
Chloride - Hypochlorite	VCR	436.1	23.6	417.1	15.0	39.2	4.0	1056.0	N/A	
Thiourea	VCR	440.6	23.6	426.7	17.0	30.2	1.7	415.0	Thiourea	9.9
	Pilgrim's Rest	452.1	29.6	425.4	26.9	14.5	1.6	605.0	Thiourea	10.0

4.2. Thiourea Leaching

The conditions for thiourea and ferric iron leaching system were selected based on various literature (Groenewald, 1976; Chen et al., 1980; Schulze, 1984; Eisele et al., 1988; Ubaldini et al., 1998; Li and Miller, 2006). The leaching was conducted over a period of 8 hours, due to rapid leaching kinetics, with conditions maintained at a pH of 1.5 and the redox at 420 mV. Initial reagent additions were made to achieve 10 g/L thiourea and 17 g/L ferric ion concentrations. The ferric iron maintained the redox conditions, while sulphuric acid was used to adjust the pH. However, the acid additions required for several of the ores made the leach unfeasible to continue, this applies to Fairview, Barbrook and Consolidated Murchison samples. Thiourea leaches were successfully completed for VCR and Pilgrim's Rest samples, while reagent consumption calculations were done on the remaining ores to highlight the prohibitive nature of this system.

The VCR sample achieved a maximum dissolution of 30%, based on feed and residue grades. Calculations based on solution grades were not possible due to the low stability of gold in solution in this system. It is possible that part of the reason for the high residue grades is that reprecipitation of

gold occurred during the leach, and therefore leaching may have been significantly more effective than represented in the dissolution percentage. Reagent consumption for this system was extremely high and only approximately 1 % of the thiourea remained. Consumptions were determined by complex formation titration using potassium iodate (KIO_3), in conjunction with a starch indicator.

The dissolution for the Pilgrim's Rest material was 14.5 %. This sample was also highly reactive, with 0.5% of the added thiourea remaining at the end of the leach.

The XRD results from feed and residue (Table 2) show that the major carbonates were almost entirely dissolved, as is evident in the calcite and the dolomite from VCR and Pilgrim's Rest samples respectively. In addition, the partial dissolution of minerals such as chlorite, pyrrhotite and chalcopyrite occurred. Note that the increase in minerals such as quartz from the feed to the residue represents an increase in the concentration of quartz resulting from the dissolution of other minerals. The total mass of quartz in the sample remained unchanged, while the mass of the sample reduced. In addition to the loss of certain minerals, it is interesting to note that new minerals have formed during the leach; pyracmonite [$(NH_4)_3Fe(SO_4)_3$] and nitrammite ($NH_4,K)(NO_3$). The formation of pyracmonite in this leach is understandable when considering the components of the leach – thiourea (CH_4N_2S) as well as ferric iron and oxygen incorporated into the leach during agitation. This can similarly be applied to nitrammite. The formation of these minerals has implications for the degradation of thiourea, which would have been required to produce sufficient nitrogen for the formation of these minerals.

Table 2 Data from XRD highlighting the mineralogical differences between feed and residue of the thiourea (TU) leaching (a) VCR feed (VCR) and residue (VCR-TU) as well as (b) Pilgrim's Rest feed (PG) and residue (PG TU).

Mineral	**VCR**	**VCR TU**	**PG**	**PG TU**
Units	**%**	**%**	**%**	**%**
Quartz	87.4	91.1	47.9	52.8
Chlorite	8.1	3.8	-	-
Muscovite	2.7	2.5	0.9	0.9
Pyrrhotite	0.8	0.4	-	-
Pyrite	0.4	0.3	33.4	41.3
Chalcopyrite	-	-	1.2	0.9
Calcite	0.6	-	-	-
Dolomite	-	-	16.6	0.7
Pyracmonite	-	1.2	-	1.5
Nitrammite	-	0.7	-	1.9
Total	100	100	100	100

4.3. Chloride – Hypochlorite Leaching

The chloride-hypochlorite leach conditions were selected based on those in literature (Puvvada and Murthy, 2000; Jeffrey et al., 2001; Baghalha, 2007; Nam et al., 2008; Hasab et al., 2013; Hasab et al., 2014; Adams, 2016). The leach duration was 8 hours, due to the rapid leaching kinetics, in addition to the tendency of the gold chloride complex to break down with time, and precipitate gold. This leach is highly unstable, with pH and redox conditions varying substantially, however an attempt was made to maintain the pH at approximately 4, by addition of hydrochloric acid, and the redox above 1000 mV using sodium hypochlorite. A minimum amount of 21.7 g/L of sodium hypochlorite was added initially in addition to 100 g/L of sodium chloride. Subsequent additions of sodium hypochlorite and hydrochloric acid were made to maintain conditions throughout the leaching period.

The reagent consumptions were prohibitively high for most ores. However, a leach using VCR sample was completed, where gold dissolution of only 39% was achieved. The limited degree of dissolution cannot be explained by instability of the gold-chloride complex in solution, as there was still gold was present during subsequent solution assays, and the accountability was good. Therefore, it was necessary to gain a better understanding of the characteristics of the unleached gold.

The remaining gold grains in the residue show low levels of exposure (Figure 2). They tend to be associated with minerals such as quartz, pyrite, chlorite (clinochlore) and pyrrhotite. However the MLA associations are significantly different than the feed deportment and may not be representative of the majority of the gold. The XRD data (Table 3) confirm that there is a decrease in the amount of pyrite, pyrrhotite and chlorite present. Successful leaching may have been limited due to the presence of minerals that were reactive with the leach solution, preventing optimal leach conditions and extensive leaching from occurring. The formation of halite in the residue (as evident in Table 3) is purely due to the reagents added to the leach.

Table 3 Data from XRD representing the mineralogical difference between VCR feed (VCR) and VCR residue (VCR-CL) after chloride- hypochlorite leaching.

Mineral	**VCR**	**VCR TU**
Units	**%**	**%**
Quartz	87.4	88.9
Chlorite	8.1	7.0
Muscovite	2.7	2.7
Pyrrhotite	0.8	-
Pyrite	0.4	-
Calcite	0.6	-
Halite	-	1.4
Total	100	100

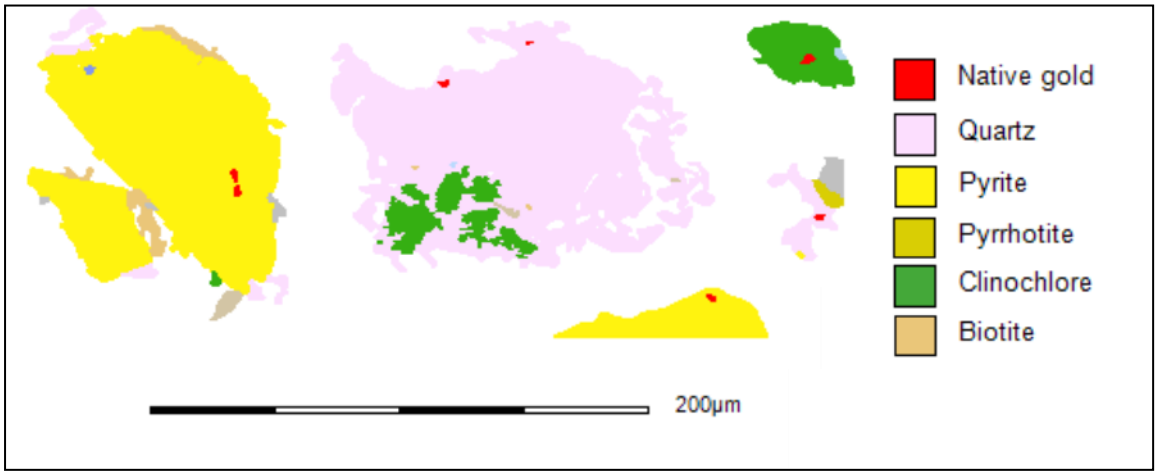


Figure 2 False colour images, produced by MLA, of the remaining gold grains in the residue from the chlorite-hypochlorite leach.

5. CONSUMPTION PREDICTIONS

Reagent consumptions presented in this section are based on calculations considering the mineralogy and mineral chemistry of each sample.

5.1. Chloride-Hypochlorite Leaching System

Because this leach is conducted under acidic conditions, carbonates present in the ore tend to be reactive. In the chloride -hypochlorite leaching system the pH controlling reagent is hydrochloric acid (HCl). HCl reacting with various carbonates is a well understood process and has been outlined for the predominant carbonates identified in the ores. Reactions with calcite, magnesite and dolomite are presented in equations 6-8 respectively;

$$CaCO_3 + 2HCl \rightarrow CaCl_2 + CO_2 + H_2O \quad (6)$$

$$MgCO_3 + 2HCl \rightarrow MgCl_2 + CO_2 + H_2O \quad (7)$$

$$MgCa(CO_3)_2 + 4HCl \rightarrow MgCl_2 + CaCl_2 + 2CO_2 + 2H_2O \quad (8)$$

In general carbonates have high dissolution rates compared to other minerals, which was supported by XRD results that show complete dissolution of these carbonates during the leach tests. Calculations for the rate of reaction of carbonates with hydrochloric acid, where M represents the metal ion, are presented in equations 9-11;

$$MCO_3 + H^+ \rightarrow M^{2+} + HCO_3^- \quad k1 \quad (9)$$

$$MCO_3 + H_2CO_3 \rightarrow M^{2+} + 2HCO_3^- \quad k2 \quad (10)$$

$$MCO_3 \rightarrow M^{2+} + CO_3^{2-} \quad k3 \quad (11)$$

As per the stoichiometry of the above equations as well as thermodynamic constrains the rate is expressed as:

$$R = k1\ \alpha_{H^+} + k2\ \alpha_{HCO_3^-} + k3 \quad (12)$$

The rate constants required to determine the rate were experimentally determined by (Chou et al., 1989)) and can be found in Table 4

Table 4 The rate constants for calcite, magnesite and dolomite reacting with hydrochloric acid. These constants correlate with the thermodynamic and calculated values (Chou et al., 1989).

	Calcite	Magnesite	Dolomite
k1	8.913E-05	2.5119E-09	2.60E-07
k2	5.012E-08	6.0256E-10	1.00E-08
k3	6.457E-11	4.4668E-14	2.20E-12

Additional factors that are required to determine the rate of the neutralisation of carbonates is the activity of H+ (α_{H^+}) and HCO_3^- ($\alpha_{HCO_3^-}$). The α_{H^+} is a factor that accounts for the deviations of substances from an ideal mixture. Deviations from the ideal are accommodated by modifying the activity coefficient. Thus, the activity defines the actual amount of acid available to maintain the pH The activity of H^+ can be calculated from the concentration of H^+ multiplied by the activity coefficient. The activity of H^+ can be determined using several equation including the calculation to determine pH (13), as well as Debye-Huckle (14) and Davis (16) equations (Harris, 2010; Haynes, 2014).

The extended Debye-Huckle formula (14) determines the activity coefficients using the ionic strength (I), the charge of the ion (Z), the hydrated radius of the ion in pm (α). To determine the ionic strength (15), the concentration of the ion (c) and charge (z) are required. The equation works well for I ≤ 0.1M. To find activity coefficients for ionic strengths above 0.1 M, more complicated equations are usually used.

$$pH = -\log(\alpha_{H^+}) \quad (13)$$

$$\log \gamma = \frac{-0.51\ zi^2\ (I)^{0.5}}{1 + [\alpha\ (I^{0.5})/305]} \quad (14)$$

$$I = 0.5\ \Sigma\ cz2 \quad (15)$$

The shortfalls of the extended Debye- Huckle equation with regards to the ionic strength require an alternative, the Davis equation (16);

$$\log \gamma = -0.51\ z^2 \left(\frac{(I)^{0.5}}{1 + I^{0.5}} - 0.3I\right) \quad (16)$$

The values for the activity constant of H have been determined using the above equations (15 &16), where Y= 0.2 and α_H+ = 1.43. The activity of H_2CO_3 has been assumed to be the saturation of carbonic acid (5.71×10^{-5})(Greenwood and Earnshaw, 2012). While this value may theoretically be higher, the rapid agitation associated with bottle rolling will reduce the extent to which this value could surpass saturation. The limitations to this assumption are mitigated by the fact that this value is only the second order component to the rate calculation.

Based on the above defined values and using equation (12) the reaction rates for calcite, dolomite and magnesite reacting with HCl can be determined (Table 6).

Using the reaction rates, the geometric surface area based on grind size, and the length of leach, the amount of carbonate minerals that could be leached in a specific leaching period can be determined (Table 6).

Table 6 The variables (surface area, leach duration, acid consumption and molar mass) used to calculate the total mass that could be consumed in the leaching conditions.

Minerals	Total SA (for 500 g)	Time	Consumption	Molar Mass	Mass
	cm^2	s	mol	g/mol	g
Calcite	208870	28800	362102.20	100.09	36242809.18
Magnesite	199309	28800	9.74	84.31	821.07
Dolomite	189946	28800	960.65	184	176759.08

As the total mass that could be consumed in an 8-hour leach is 36245 kg, 0.8 kg and 176 kg for calcite, magnesite and dolomite respectively. All of these masses are greater than the total amount of sample used in a single leach (~0.5 kg). Therefore, it can be assumed any exposed carbonates present in the ore would be dissolved.

Calculated acid consumptions ranged from 2.9 kg/ton to 461.5 kg/ton (Table 7). These consumptions have been calculated based on pure HCl (however the maximum concentration of hydrochloric acid is 37% due to solubility limits). Furthermore, these consumptions have been calculated based on carbonate content but there is evidence that other components (chlorite, muscovite etc.) have also been at least partially dissolved and would still contribute to the acid consumption.

Table 7 The values determined for the total amount of acid that could be consumed by each carbonate and in total the ore in the chloride-hypochlorite leach. This was calculated using the reaction rates as well as the total amount, (and) type of carbonates present in the respective sample.

			HCl Consumption / Mineral	HCl Consumption Total
Ore	Mineral	Mass %	kg/ton	kg/ton
VCR	Calcite	0.4	2.9	2.9
Fairview	Dolomite	10.8	85.4	146.8
	Magnesite	7.1	61.4	
Barbrook	Dolomite	58.0	458.6	461.5
	Calcite	0.4	2.9	
Pilgrim's Rest	Dolomite	4.9	38.7	38.7
Consolidated Murchison	Magnesite	38.0	328.6	360.2
	Dolomite	4.0	31.6	

5.2. Thiourea Leaching System

Sulphuric acid is used to control the pH in the thiourea leaching system and the reactions that occur with calcite, magnesite and dolomite respectively are described below;

$$CaCO_3 + H_2SO_4 \rightarrow CaSO_4 + CO_2 + H_2O \quad (17)$$

$$MgCO_3 + H_2SO_4 \rightarrow MgSO_4 + CO_2 + H_2O \quad (18)$$

$$MgCa(CO_3)_2 + 2H_2SO_4 \rightarrow MgSO_4 + CaSO_4 + 2CO_2 + 2H_2O \quad (19)$$

The degree of reactivity of these carbonates is related to several factors including acid concentration, carbonate particle size, and other metals in solution. Of these components the acid (H^+) activity is the most significant factor (Maree and Du Plessis, 1994{Chou, 1989 #219). Using the same free acid calculations described above (Equations 13-15) in addition to the stoichiometry (17-19), the rates for the respective carbonate minerals were determined (Table 8). Rate reactions were then used to determine the mass that could be consumed in an 8-hour leach – which is 43446 kg, 0.098 kg and 21 kg for calcite, magnesite and dolomite respectively (Table 9). The masses for calcite and dolomite are greater than the total masses of the sample and therefore it can be assumed that there was complete consumption of carbonates in the sample. However, for magnesite, any material that contains more than 98.46 g (i.e. 20 % of the sample) would be limited in its consumption. This constraint only applies to one of the samples in the group of study, Consolidated Murchison, which despite containing 38% magnesite will likely only consume 100 kg/ton of sulphuric acid. Acid consumption ranges from 3.92 to 620.90 kg/ton, most of these consumptions are far too high to be considered economic (Table 10). It should be noted that calcite and sulphuric acid may form a passivating layer of gypsum, which would reduce the degree of consumption (Booth et al., 1997). Calcite, however, is present in limited quantities and therefore this is not a major consideration in this work.

Table 8 The reaction rates determined for calcite, magnesite and dolomite when reacting with sulphuric acid. These reaction rates are specific to the reagent concentrations and pH conditions needed for the thiourea leaching.

Reaction Rates ($M^{-1} \cdot s^{-1}$)	
Calcite	7.22E-06

Magnesite	2.03E-10
Dolomite	2.11E-08

Table 9 The variables (surface area, leach duration, consumption and molar mass) used to calculate the total mass that could be consumed in the leaching conditions.

Minerals	Total SA (for 500 g)	Time	Consumption	Molar Mass	Mass
	cm^2	s	mol	g/mol	g
Calcite	208870	28800	43407.91	100.09	4344698.14
Magnesite	199309	28800	1.17	84.31	98.46
Dolomite	189946	28800	115.17	184	21191.72

Table 10 The values determined for the total amount of acid that could be consumed by each ore in the thiourea leach. This was calculated using the reaction rates as well as the total amount, (and) type of carbonates present in the respective sample.

			H_2SO_4 Consumption /Mineral	H_2SO_4 Consumption Total
Ore	Mineral	Mass %	kg/ton	kg/ton
VCR	Calcite	0.4	3.92	3.92
Fairview	Dolomite	10.8	114.89	197.48
	Magnesite	7.1	82.59	
Barbrook	Dolomite	58	616.98	620.9
	Calcite	0.4	3.92	
Pilgrim's Rest	Dolomite	4.9	52.12	52.12
Consolidated Murchison	Magnesite	38	84.73	100.54
	Dolomite	4	15.81	

6. COSTING REAGENTS AND DETOXIFICATION

The pricing of reagents for various leaching systems is vital in order to completely evaluate a technique. The prices for the various reagents were determined for both the South African markets as well as at international level, however there was little variation found. The data below is based on South African availability (data kindly given by Protea Chemicals and Marlyn Chemicals). The pricing of respective reagents (Table) and whole leach systems (Table 12) have been represented as ratios relative to cyanide for ease of reporting and the sake of consistency. Table 11 emphasises the high cost of cyanide relative to other reagents, which are all less than half the price of cyanide. However, the cyanide leaching system only requires cyanide and lime at low concentrations and is therefore overall a cheap leaching system (Table 11-1112). Thiourea leaching requires, in addition to thiourea, ferric iron and significant amounts of sulphuric acid, which in total makes this leach system 5.29 times more costly than cyanide, unless acid is already available from a peroxidation process (such as pressure or bacterial oxidation). The chloride-hypochlorite leach requires large concentrations of sodium chloride, sodium hypochlorite and hydrochloric acid which makes the leaching reagents for this system 19.87 more expensive than the cyanide system. The exact quantities of reagent required for leaching has been estimated for low consuming ores.

Table 11 Average reagent price per unit, as a fraction relative to cyanide

Reagent	Cost (Relative to Cyanide)
Cyanide	1
Thiourea	0.807
Sodium chloride	0.046
Lime	0.230
Ferric sulphate	0.080
Sulphuric acid	0.156
Hypochlorite	0.150
Hydrochloric acid	0.102

Table 120-2 Average cost of leaching systems relative to cyanide. The concentrations used for the total cost calculations are based on the average initial reagent additions suggested in literature, and not the high consumptions outlined previously which are specific to these samples of study.

Leaching System	Cost Relative to Cyanide
Cyanide	1.00
Thiourea	5.29
Chloride-hypochlorite	19.87

While the reagent cost is an important factor, the detoxification required for cyanide leaching can be expensive and the detoxification processes for the alternatives may offer an advantage in this regard. While there are a number of well understood methods available for cyanide detoxification, utilising a variety of methods including SO_2/air, hydrogen peroxide and even alkaline chlorination, in addition to biodegradation and natural degradation (Logsdon et al., 1999; Botz and Mudder, 2002; Akcil and Mudder, 2003), unfortunately there is little research into the detoxification of alternatives. Consideration into detoxification has therefore been based exclusively on the components that exceed the values recommended by the South African Standards for effluent discharge (Environmental Protection Regulations). A major factor for both thiourea and chloride-hypochlorite leaching systems is the acidic pH of the solution. Acceptable discharge levels for pH range between 5 and 9 and therefore both the chloride-hypochlorite and thiourea leach solution would require an increase in pH, generally implemented by the addition of lime (Kuyucak and Sheremata, 1995). The cost of neutralisation may be mitigated by the use of carbonate-bearing waste rock, where costs would be limited to milling and handling and therefore are specific to individual operations. Thiourea-based leaching would require significantly more lime in order to achieve neutralisation. The amount of sulphate and iron in the thiourea leach must be reduced before it can be safely discharged. Iron concentration can be reduced by bubbling oxygen through the solution (to ensure all ferrous iron is oxidised into ferric iron), and subsequently raising the pH sufficiently to ensure precipitation of iron (Hove et al., 2007). The sulphates (a potential product of thiourea degradation) can be removed by barium chloride, however this option is limited due to both health and safety, as well as financial considerations (Kefeni et al., 2015). The chloride-hypochlorite leach liquor contains hypochlorous acid which will be neutralised with the pH adjustment previously mentioned. Further removal of the chlorine is possible, with the use of reverse osmosis or ion exchange membranes, however this is likely to be an expensive and slow process particularly at such high concentrations (Heinen et al., 1978;

Strathmann, 2004). There is however the capacity for solution recycling which would significantly reduce the expenses for both reagents as well as detoxification (Ximing et al., 1992)

7. DISCUSSION

Despite the global movement to research non-cyanide lixiviants, cyanide remains the industry standard. Cyanide dissolution of gold in ores had varying results, from high (at 95%) to less than 22%. Despite some low recoveries, when compared to alternatives, cyanide is superior. While acceptable gold recoveries were not always achieved with the more refractory ores, although the alternatives consistently underperformed relative to cyanide, to the extent that many of the deposits were fundamentally uneconomic, even to test.

The prohibitive consumptions of these leaches generally relate to the presence of carbonates. While a number of gold deposits are associated with carbonate minerals, ores that are carbonate-free or that have undergone flotation would still be amendable. The spreadsheet used to calculate the consumptions is available and should serve to emphasise the importance of understanding ore mineralogy.

The thiourea leaching system is very sensitive to both Eh and pH conditions, as thiourea is inherently unstable and readily decomposes to substances incapable of leaching gold. This presents a problem to determining the total extent of dissolution, and has implications for recovery in a large scale system. The reduction of chlorite, muscovite, pyrrhotite, pyrite and calcite show the minerals vulnerable to this leach.

Chloride-hypochlorite while having a reasonably stable gold complex appears to have very unstable leaching conditions. The pH and redox conditions are generally difficult to maintain and interactions with sulphides have a further deleterious impact on the leach. The complete reaction of calcite, pyrite and pyrrhotite as well as a minor amount of chlorite highlight the reactive nature of this leach.

The economics of the alternative extraction processes to South African are not advantageous relative to cyanide. Despite the low unit prices, the high concentrations and numerous reagents required for the alternatives, mean cyanide is by far the most economic leaching system.

Processes such as recycling, or in the case of the chloride-hypochlorite leach, using untreated ocean water, may have implications for reduced operating costs. Alternative methods may have more scope when used in conjunction with pre-treatment and separation processes. For example flotation would increase the sulphide component and reject consuming carbonates. Subsequent processes such as pressure and bacterial oxidation could increase gold exposure as well as produce sulphuric acid required for leaching.

8. CONCLUSION

While processes such as recycling may have the ability reduce the costs of alternatives, this currently appears insufficient. With a combination of increased reagent requirements, high consumptions and limited gold dissolutions, it is clear that these processes are currently limited and not a feasible alternative for whole ore leaching.

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Thiosulfate Leaching of South African Ores

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Highlights

- Thiosulfate leaching on a spectrum of gold deposit types
- Thiosulfate mineral reactivity is found to be limited
- Use of glycine to improve limited dissolutions
- Thiosulfate achieved 70% dissolution of total gold

Abstract

The potential of thiosulfate as an environmentally friendly alternative to cyanidation when processing preg-robbing, copper rich gold ores has long been known. However understanding about the applicability of these leaches to other ore types, particularly with well constrained mineralogy, is lacking. Throughout these leaches it was evident that Thiosulfate suffered from stability issues, possibly enhanced by the presence of sulfides. The introduction of a low cost stabiliser, glycine, significantly improved and sustained gold dissolutions. Despite these optimisations the gold dissolutions were consistently ~ 70% of the total available gold, likely due to the slow kinetics associated with non-ideal leaching conditions.

Keywords

Gold leaching; Thiosulfate; Glycine; Mineralogy

1. Introduction

Thiosulfate is one of the foremost alternatives to cyanide-based gold extraction, and the only one to have successfully implemented commercial applications. The primary examples of this include the processing at Barrick's Goldstrike plant which has been extracting gold using thiosulfate as early as 2014, while more recently Clean Mining has released a thiosulfate - based gold extraction method. However the details and technicalities of these methods are closely guarded propriety information and significant work is still required in the public space to advance this technique to a competitive alternative to cyanide.

The thiosulfate leaching system is considerably more complicated than cyanide, as such variables as the Eh, pH and reagent concentration must be precisely controlled, not only to promote gold leaching, but also to minimise the decomposition of thiosulfate, which is a

metastable complex in aqueous leach conditions. The decomposition of thiosulfate is undesirable due to reagent consumption, reduced leaching capacity as well as passivation of gold surfaces (Hilson and Monhemius, 2006a). Passivation of gold can occur when thiosulfate degrades into tetrathionate and polythionate. The oxidative decomposition of thiosulfate is particularly prominent in the presence of catalytic copper.

Despite these complications, the low toxicity, capacity to treat complex ores and competitive dissolution rates make thiosulfate the most promising alternative lixiviant currently being researched (Aylmore, 2016a; Aylmore, 2016b). Thiosulfate dissolution of gold is slow without an oxidant more efficient than oxygen, as well as a catalyst (Abbruzzese et al., 1995). Extensive studies (Langhans et al., 1992; Breuer and Jeffrey, 2000; Jeffrey, 2001; Jeffrey et al., 2001; Muir and Aylmore, 2004b; Aylmore, 2005) have found that the thiosulfate – copper – ammonia system is able to achieve kinetics of a similar rate to that of cyanidation under specific reagent conditions. Elevated reagent concentrations increase leach kinetics, but also increase the degradation rates.

The high redox conditions required for thiosulfate leaching lead to significant reactivity with sulfides, increasing reagent consumptions in sulfide-bearing ores. However the extent of this can be controlled, while still maintaining sufficient gold extraction. Thiosulfate has been comprehensively researched for applications relating to carbonaceous and copper-gold deposits, as cyanide has a limited suitability for these ores. Carbonaceous material is preg-robbing in a cyanide leaching system, while gold-thiosulfate complexes are unaffected by this (Muir and Aylmore, 2004b).

Thiosulfate leaching proceeds via two mechanisms (Aylmore, 2016a). The gold dissolution occurs by the reaction in Equation 1, which requires that copper (II)-ammonia-thiosulfate complexes on to the surface of the gold. The gold and thiosulfate oxidise, while copper reduces. Equation 2 represents the regeneration of copper, Cu(II) from Cu(I), as well as thiosulfate, which is fundamental for continuous and sustainable leaching (as used in Equation 1). Oxygen is responsible for the regeneration of catalytic copper.

$$Au + 5S_2O_3^{2-} + Cu(NH_3)_4^{2+} \rightarrow Au(S_2O_3)_2^{3-} + 4NH_3 + Cu(S_2O_3)_3^{5-} \quad (1)$$

$$4Cu(S_2O_3)_3^{5-} + 16NH_3 + O_2 + 2H_2O \rightarrow 4Cu(NH_3)_4^{2+} + 4OH^- + 12S_2O_3^{2-} \quad (2)$$

The instability of non-cyanide lixiviants is a major drawback, the thiosulfate -gold complex is an order of magnitude less stable than cyanide-complexes (Aylmore, 2016a). Significant research has evaluated a variety of stabilising agents (Abbruzzese et al., 1995; Muir and Aylmore, 2004a; Feng and van Deventer, 2011) . Glycine (NH_2-CH_2-COOH), a simple amino acid, has several suggested uses for gold leaching; in conjunction with hydrogen peroxide as an alkaline leach (Oraby and Eksteen, 2015), in cyanide starved conditions in the GlyCat™ process (Oraby and Eksteen, 2019) or in association with thiosulfate (Michel and Frenay, 1998; Feng and van Deventer, 2011). Because the major disadvantage to the thiosulfate leaching system is its reagent instability, the use of various stabilising agents may improve the

efficiency and profitability of the leach. Glycine, when used with thiosulfate, can act as a low cost stabilising agent. The role of glycine within the thiosulfate is twofold, the glycine is capable of complexing directly with gold to form a stable compound as well as complexing with the copper, to allow it to continue acting as a catalyst, but reducing its degradation effect on thiosulfate (Michel and Frenay, 1998; Feng and van Deventer, 2011).

2. Ores

The ores used in this evaluation of thiosulfate in gold extraction represent a spectrum of different types of gold deposits and mineralogies from various localities in South Africa. These deposits include samples from the Ventersdorp Contact Reef (VCR) at Mponeng Mine in the Witwatersrand Basin and from the Fairview and Barbrook mines, at Barberton. A short description of the ores follows, this information can also be found summarised in Table 0-3.

The VCR material was selected as an example of high-grade free milling ore, where processing tends to be easy and compromising minerals are only present in low quantities. The gangue is predominantly quartz, with minor amounts of chlorite, sericite and carbonates. Sulfides are minor, but include pyrite, pyrrhotite and chalcopyrite, which can be found associated with gold of grades approximately 23 g/ton, of which 96% was found to be cyanide soluble at a grind of 80% -75 µm. For this sample, the percentage cyanide dissolution is considered a proxy for percentage of total gold exposed for leaching. The weighted mean gold grain size is 220 µm.

The material from Fairview and Barbrook both represent refractory deposits, with gold in solid solution in arsenopyrite and grades of 12 and 3 g/ton, and a weighted mean grain size of 180 µm and 32 µm respectively. However, both ores are uniquely important to this study as Fairview represents singly refractory gold, while the presence of preg-robbing carbon makes Barbrook a doubly refractory ore, which is significantly more difficult to process. The cyanide dissolution for the Fairview sample was 68%, which is considered to be the proportion of the total gold available for leaching. Barbrook achieved only 24% dissolution, however this cannot be assumed to be the total leachable gold due to the preg-robbing carbon component within the Barbrook sample. The gangue occurring in the Fairview sample was predominantly non-reactive to cyanide and thiosulfate with abundant quartz, muscovite, chlorite and dolomite. The Barbrook sample contained significant dolomite and quartz.

Table 0-3 A summary of the main characteristics of each sample,

Ore	Major Gangue	%	Accessory Minerals	%	Gold	
VCR	Quartz	96.9	Pyrrhotite	0.4	Native Gold	23.6 g/ton
	Muscovite	1.1	Pyrite	0.3	Grain size	220 µm
	Chlorite	1			CN Dissolution	96.07%
Fairview	Quartz	41.1	Pyrite	2.7	Native and Refractory gold	12.5 g/ton
	Muscovite	24.8	Arsenopyrite	0.2		
	Chlorite	13.4			Native gold grain size	180 µm
	Dolomite	10.8			CN Dissolution	68.17%
	Magnesite	7.1				
Barbrook	Dolomite	58	Pyrite	3.4	Native and Refractory gold	2.08 g/ton
	Quartz	29.8	Arsenopyrite	0.1		
	Chlorite	6.4			Native gold grain size	32 µm
	Muscovite	1.7			CN Dissolution	24.13%

3. Experimental Conditions

The experimental work for this study was undertaken using bottle roll leaches at ambient temperature and pressure conditions. The leaching took place on ore that had undergone milling to 80% -75 µm. Various leach conditions were tested for each ore, including the effect of using glycine on thiosulfate leaching of both calcined and uncalcined ores. Calcination was at a temperature of 540˚C overnight. The Thiosulfate leach conditions were based on those typically found in relevant literature (Chen et al., 1980; Schulze, 1984; Langhans et al., 1992; Abbruzzese et al., 1995; Ubaldini et al., 1998; Aylmore and Muir, 2001; Jeffrey et al., 2001; Aylmore, 2016b). Sample aliquots of between 400 and 500 g dry solids were used. These were leached at 50% solids, ~60 g/L hydrated sodium thiosulfate, 7.5 g/L copper sulphate and, for the tests where glycine was used, 3 g/L glycine. Excess reagent conditions were selected to reduce variables. The pH was maintained at approximately 9.5 using ammonia and sulphuric acid as required. The leaches were run over a 24-hour period, with the conditions adjusted approximately once an hour until stable conditions were achieved. After the leach was completed, the residue and solution were separated and submitted for fire assay.

The mineralogy of the residue was determined using X-ray Diffraction (XRD), while selected ores underwent Heavy Liquid Separation (HLS) in order to upgrade the remaining gold similar

to the method suggested by (Coetzee et al., 2011). This material was then studied using MLA to gain a better understanding of the processes the leach response of the ore.

4. Leaching Results

A summary of the thiosulfate leaching results can be found in Table 2 where the gold dissolution rate has been summarised and the various systems compared.

The dissolutions achieved for thiosulfate leaching (without calcination or glycine) were consistently low, approximately 40% for VCR and less than 10% for the refractory ores. It was observed that the gold did not remain in solution for long after completion of the leach. This observation, in combination with the knowledge of the instability of the Thiosulfate -gold complex, led the authors to suspect that the low dissolution rates being achieved were due to precipitation of the gold complexes. In order to test this, ore calcining and glycine addition were implemented, both separately and combined in order to gain a better idea of the mechanisms of the leach.

It is interesting to note that for the VCR material, the dissolution decreased when the ore was calcined, this was observed to be consistent, whether or not glycine was used. The decreased dissolution could be due to the formation of oxidation products from sulfides, which may have been capable of enclosing previously exposed gold. The addition of glycine however, is a significant advantage as in calcined ores it increased the gold dissolution by more than 30% (more than double the original value) while in untreated ores the dissolution increased by 23%. However, the highest dissolution achieved for VCR using the thiosulfate glycine leach only, achieved approximately 70% dissolution of the total available gold as determined by cyanidation.

Dissolution of gold for the Fairview sample was significantly improved by both calcining and adding glycine to the leaches. While the calcining would have allowed refractory gold grains to be exposed to leaching, this did not proportionally improve the leaching without the addition of a stabilizer. When the stabilizer (glycine) was used for the uncalcined ores, gold dissolutions of 44% were achieved, which represents just less than 65% of the exposed gold. If it is assumed that on calcining much of the gold was made available, then the thiosulfate glycine leach recovered approximately 70% of the total available gold.

Cyanide dissolutions cannot be used as a proxy for total exposed gold in the Barbrook sample due to preg-robbing. The dissolution of gold for thiosulfate glycine-bearing leaches was high (59.9%) relative to cyanide (24.13%), highlighting the amount of gold lost to preg-robbing. The calcined glycine leaches reached 75% dissolution. Once the leach was completed, the remaining Thiosulfate was determined using an iodimetric titration (Zipperian et al., 1988; Deutsch, 2012). The remaining thiosulfate concentrations for the above leaches were all above 20% of the original amount.

Table 2 Summary of thiosulfate leaching results, where ticks represent processes that were used on the material either prior or during the leach. Crosses represent where this process did not occur.

Thiosulfate Leaching Data			
Ore	Calcined	Glycine	Dissolution %
VCR	✗	✗	42.1
	✓	✗	26.9
	✗	✓	65.8
	✓	✓	60.2
Fairview	✗	✗	0
	✓	✗	45.7
	✗	✓	43.7
	✓	✓	73.8
Barbrook	✗	✗	5.2
	✓	✗	40.2
	✗	✓	59.9
	✓	✓	75.8

5. Residue Mineralogy

To better understand In order to gain a better understanding as to why the leaches were consistently achieving incomplete dissolutions, which was generally in the order of 70% of the total available gold, the mineralogy of the residue was studied. To facilitate locating a significant number of gold grains, the residue of the glycine, thiosulfate VCR leach was selected for study. The residue was upgraded using heavy liquid separation (HLS). The HLS was prepared using tetrabromoethane (TBE) at a specific gravity of 2.96 g/cc in a centrifuge to ensure fine gold was able to settle out. The floats were assayed to ensure the HLS was successful while the sinks were used to prepare polished sections for Mineral Liberation Analyser (MLA) study. From the mineralogical data (Figure 1), it is evident that there was extensive exposed gold, as well as gold associated with sulfides (pyrrhotite, arsenopyrite and pyrite) and to a lesser extent quartz, chlorite, muscovite and siderite. The gold in the residues was significantly finer than in the feed, with the liberated gold grains consistently finer than 30 µm, while the average grain size of the feed was greater than 220 µm.

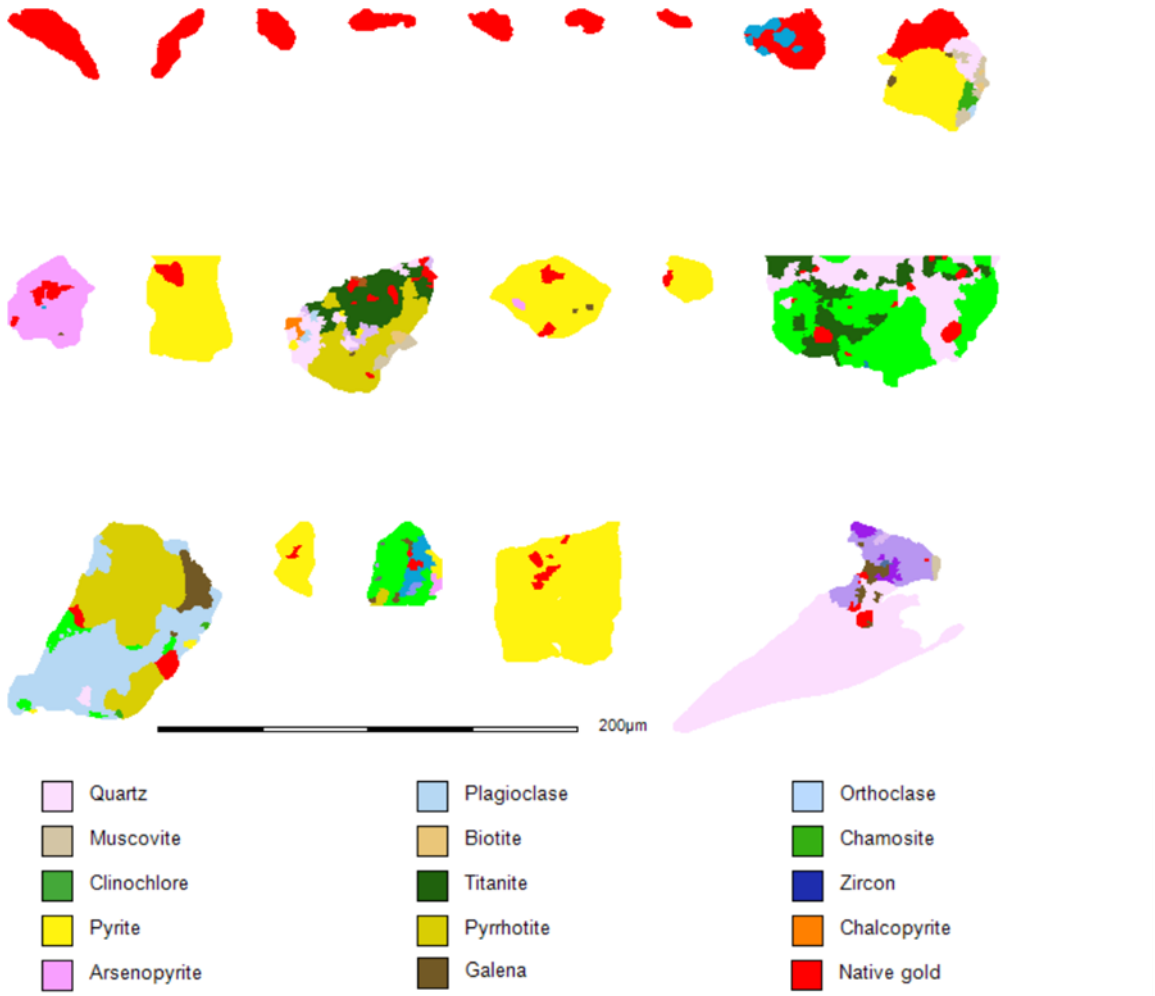


Figure 1 False color MLA images of gold grains (red) and their associated minerals found in the VCR thiosulfate glycine residue

The remaining liberated gold grains could either be due to gold precipitation or coarse gold grains present in the feed that did not undergo complete dissolution. The BSE images of the remaining gold (Figure 2) show compact gold with embayments that suggest incomplete dissolution, rather than more fine grained open-structured gold one might predict with precipitation.

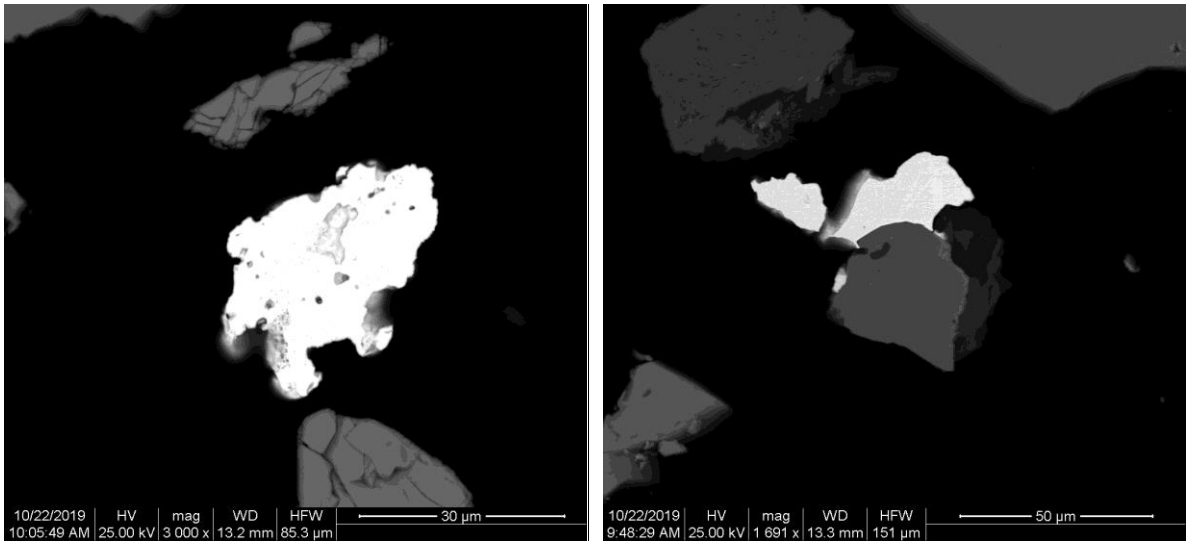


Figure 2 BSE images of gold remaining in the residue of the calcined, thiosulfate glycine leached material with embayments suggesting incomplete dissolution.

6. Discussion

The limited dissolution achieved by the thiosulfate leaches originally occurred due to the instability of the gold Thiosulfate complex. The presence of both sulfides and catalytic copper competing for thiosulfate would reduce the amount of thiosulfate available for gold leaching and complexation. It was found that this stability was significantly improved by the addition of glycine, with up to 50% increase in dissolution by the addition of a single reagent.

Unfortunately, even with the addition of a stabilizing agent, the thiosulfate consistently achieved only 70% of the dissolution achieved by cyanide. To understand this, the results from the mineralogy and post-leach titrations must be considered. The titrations suggest that there was sufficient (not excess) reagent in the leach solution for the dissolution to still be active. However, the mineralogy shows a number of fine-grained, well-liberated gold grains. The average grain size is significantly smaller than was present in the feed, and the texture of the residual gold grains suggest that these particles have undergone a degree of dissolution. As it seems likely that the glycine sufficiently addressed the reagent instability issue, we cannot assume the incomplete dissolutions are due to inadequate thiosulfate or precipitation.

The thiosulfate leaching system can, under ideal conditions, achieve similar kinetics to cyanide. However should conditions be non-ideal, notably slower dissolution will occur (Langhans et al., 1992; Breuer and Jeffrey, 2000; Jeffrey et al., 2001; Muir and Aylmore, 2004b; Aylmore, 2005). After 24 hours of leaching the pH and redox conditions were within the optimal range, and thiosulfate was still present in solution, however the lower concentration in the solution may have further reduced the already disadvantaged leach kinetics.

Results from these leaches are encouraging as reasonable dissolutions were achieved for all ore types, in the presence of a low cost stabilising agent. The Barbrook ore, when leached with thiosulfate and glycine far surpassed the results attained with cyanide.

7. Conclusion

Thiosulfate leaches are hindered by their reagent instability, however the introduction of a stabilizing agent can significantly improve the total dissolution. To further improve recovery, it appears that extended leaching times could allow for dissolutions approaching levels achieved by cyanide. Despite this, dissolutions achieved on a variety of ore types were encouraging and further research may well allow for increased dissolution and further establish thiosulfate as a dominant competitor to cyanide.

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