AN INVESTIGATION OF THE FLOTATION BEHAVIOUR OF THE SULPHIDE GOLD ORES

SÜLFÜRLÜ ALTIN CEVHERLERİNİN FLOTASYON DAVRANIMLARININ İNCELENMESİ

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ETHICS

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- all cited studies have been fully referenced
- I did not do any distortion in the data set
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25/04/2014

TUĞBA ÖZGEN

ABSTRACT

AN INVESTIGATION OF THE FLOTATION BEHAVIOUR OF THE SULPHIDE GOLD ORES

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In this thesis, flotation behavior of a refractory gold ore with high sulphide content (S%>5) and production of a flotation concentrate with high gold content were investigated. Pre-concentration of sulphidic gold ores to produce gold with high recovery and high grade before leaching (bioleaching, cyanide leaching) is a common method in mineral industry.

During experimental study, the relationship between the gold recovery and sulphur recovery was analyzed. The effects of different types of collectors, pH, fineness of grinding, activation with $CuSO_4$, sulphidization and pulp density on flotation performance were examined. The results showed that there is a linear relationship between gold and sulphur recovery.

At the end of the flotation experiments, a preconcentrate with high gold and sulphur content was produced and the concentrate was leached. According to the results of the leaching tests, high cyanide and lime consumption with low gold recovery was observed due to the high sulphur content in the concentrate.

The reason for low gold recovery could be due to complex mineralogy, i.e. locked gold grains in sulphide matrix and not gained by conventional leaching methods. However, gold can be recovered by roasting or pressure oxidative leaching methods.

Keywords: Flotation, Refractory gold ores, Leaching
Advisor: Prof. Dr. Zafir EKMEKÇİ, Hacettepe University, Mining Engineering Department, Mineral Processing Division

ÖZET

SÜLFÜRLÜ ALTIN CEVHERLERİNİN FLOTASYON DAVRANIMLARININ İNCELENMESİ

Tuğba ÖZGEN
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Bu çalışmada, sülfür içeriği yüksek (S%>5), refrakter karakterdeki bir altın cevherinin flotasyon davranımı ve altın içeriği yüksek bir konsantrenin üretilebilirliği araştırılmıştır. Sülfür içeren altın cevherlerinin liç öncesi (biyoliç, siyanür liçi) zenginleştirme yapılarak tenörü ve verimi yüksek konsantre elde edilmesi mineral endüstrisinde uygulanan bir metottur.

Deneylerde, flotasyon koşulları değiştirilerek ,altın ve sülfür verimi arasındaki ilişki incelenmiştir. Farklı toplayıcıların, pH'ın, öğütme inceliğinin, bakır sülfat ile aktivasyonun, sülfidizasyonun, palp yoğunluğunun, refrakter bir altın cevherinin flotasyon performansına olan etkileri araştırılmıştır. Sonuçlara göre, altın ve sülfür verimi arasında doğrusal bir ilişki olduğu gözlenmiştir.

Çalışmalar sonucunda altın içeriği yüksek bir önkonsantre üretilmiş ve üretilen konsantreye liç işlemi uygulanmıştır. Liç sonuçlarına göre ise sülfür içeriğinin yüksek olmasından dolayı yüksek siyanür, kireç tüketimi ve düşük altın verimi gözlenmiştir.

Buna sebep olarak ise mineralojik olarak altının sülfür matrisi içerisinde kapanım halinde olduğu ve konvansiyonel liç yöntemleri ile altının kazanılamayacağı, sülfür ve altın içeriği yüksek konsatreyi öncesinde kavurarak ya da yüksek basınç altında liç ederek altının kazanılabileceği düşünülmektedir.

Anahtar Kelimeler: Flotasyon, Refrakter Altın Cevherleri, Liç

Danışman: Prof.Dr. Zafir EKMEKÇİ , Hacettepe Üniversitesi Maden Mühendisliği Bölümü Cevher Hazırlama Anabilim Dalı

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TABLE OF CONTENTS

Page

LIST OF TABLES

Page

LIST OF FIGURES

1. INTRODUCTION

The concentration of minerals by froth flotation depends primarily upon the differences in the hydrophobicity (or wettability) of mineral particles. Mineral particles to be floated must selectively attach to air bubbles; the particle bubble aggregates then rise through the pulp phase up to the froth phase. A few minerals like talc, sulphur, graphite, stibnite, coal and molybdenite are naturally hydrophobic as a result of preferential cleavage along planes that have all chemical bonds satisfied (Woods, 1984); therefore, they can be floated without collectors.

The application of flotation on a reasonable scale within the gold-mining industry commenced in the early 1930s following the introduction of watersoluble flotation collectors (specifically xanthates and dithiophosphate collectors) that allowed differential flotation of sulfide minerals (Weinig and Carpenter, 1937; Rabone, 1939; Richards and Locke, 1940; Taggart, 1945).

Prior to that time, a few gold mines in Canada, Australia and Korea built flotation plants as the first step in the treatment of complex and refractory gold ores (Richart, 1912, Taggart, 1927). Flotation collectors on these plants were oils (Hoover, 1916; Taggart, 1927) that generated bulk low-grade gold concentrates, which were difficult to filter and dry. Pre- and post-Second World War and up until late 1960s, most of the flotation activity in the gold industry took place in Canada (Carter, 1957). During this period, Canada was recognized as the second largest gold producer and a sizeable amount of the gold production came from the treatment by flotation of copper–gold ores, refractory gold ores and complex gold ores.

The demand for sulfuric acid initiated by the booming uranium industry during the late 1960s provided the catalyst for the installation of pyrite flotation plants on numerous gold mines in South Africa (Bushell, 1970). After roasting the pyrite flotation concentrate to generate sulfur dioxide for the sulfuric acid plant, the remaining calcine was cyanide leached to remove additional liberated gold.

The worldwide gold boom in the 1980s and 1990s created new opportunities in Australasia (Woodcock and Hamilton, 1993, Africa (Broekman et al., 1987) and the Americas for the exploitation of medium-sized refractory gold deposits by flotation and further treatment of the concentrates by bacterial and pressure leaching. Many copper flotation plants around the world, and particularly those in the Americas, have enough gold in the ore to ensure that special attention is given to maximize the recovery of gold into the copper concentrate. A comprehensive list of these operations, with details of the flotation reagent regimes and circuit configurations that were in existence during the 1980s, is provided by Bassarear (1985). Since that time there has been a significant increase in the availability of selective flotation collectors for gold (Nagaraj, 1997) and these are now widely in use on many large and new copper flotation plants around the world (Damjanovic and Goode, 2000; Winckers, 2002).

In recent years, studies on gold ores with complex mineralogy is also popular in Turkey such as Efemçukuru (Izmir) and Mollakara (Ağrı) gold deposits. Efemçukuru is a high grade, underground operation located in Izmir Province in western Turkey. It uses gravity circuits followed by flotation to produce a flotation concentrate and a gravity concentrate. Mollakara Project is at Ağrı Diyadin region, 100 km north of Van Province. Mineralization is Carlin type gold deposit. Mineralization is separated in two zones oxidation and sulphidation. Preliminary works were completed by the end of 2012 to calculate reserve for oxide ore. Flotation and pressure oxidation leaching can be applied to Mollakara gold deposit sulfide zone.

The main objectives of this thesis was to investigate the flotation behavior of a gold ore from one of Koza Gold’s mine operations. The ore has a complex mineralogy and is not amenable for direct cyanide leaching. Sulphur content is high and it passivates the gold surfaces for penetration of cyanide. For these reasons floatability of the ore is investigated at different conditions. The aim is to produce a concentrate with high gold recovery and grade for the following operation such as roasting or pressure oxidation leaching.

1.1. Application of Flotation

Free metallic gold can generally be recovered very effectively by flotation although more commonly it is recovered together with sulfide minerals, where gold is intimately associated with the sulfides as fine unliberated grains (in solid solution or as discrete inclusions), or occurs with barren, hydrophobic sulfides. The most common gold-bearing sulfides are pyrite, arsenopyrite, and to a lesser extent pyrrhotite.

The flotation of gold from sulfide-free ores containing very low concentrations of free gold is difficult because of the low mass of material reporting to the concentrate and the high density of gold (19.3 g/ cm3). For example, 0.005% (50 g/t Au) would be very high grade ore compared with grades of > 0.5 % for most copper, lead, or zinc ores treated by flotation. This results in very poor froth stability, decreased recovery, and/ or concentrate grade. Despite these factors, close to 100% recovery of free gold has been achieved by flotation under optimized conditions in some applications, with concentration ratios of 30:1 and 300:1 achieved (Leaver, E.S., and J.A. Woolf. 1934. Pages 9-17 in Flotation of Metallic gold; Relation of Particle size to Floatability Report of Investigation 3226. Washington D.C: U.S. Bureau of Mines).

Gold flotation is inherently a slow rate process compared to the flotation of other naturally floating minerals, such as chalcopyrite, chalcocite, and sphalerite. Ore mineralogy has a profound effect on the flotation conditions employed. The selection of operating pH, type and amount of frother, collector system and, where necessary, activators and/ or depressants, are all critical to achieve effective recovery (and concentrate grade) for a given ore type.

1.1.1. Free Gold Flotation with an Oxide or Silicate Gangue

For the treatment of placer deposits of gravity concentrates, most of the gangue minerals are oxides or silicates, which are hydrophilic, and strong collectors may be used to maximize gold recovery with little concern for corecovery of sulfides. This type of flotation is rare but has been proposed for low-grade ores where the gold is too fine to be recovered effectively by gravity concentration. Thus, conditions can be selected solely for gold recovery and not to optimize selectivity against sulfide minerals (Wengian, W., and G.W. Poling. 1983. Methods for recovering fine placer gold. Canadian Institute of Mining and Metallurgy Bulletin 76 (860):47).

1.1.2. Free Gold with a Sulfide Gangue

When the gangue mineral has no economic value, the objective is to selectively recover gold from the barren sulfide (and silicate) gangue. Optimum reagent selection is required and high gold recoveries are difficult to achieve without corecovery of some sulfide minerals. Commonly, free gold can be floated at neutral pH with little or no collector addition. If sulfide components in the gangue are of value, then conditions should be selected to give the best economic return of all economic minerals, not necessarily the maximum recovery of gold or any other single material.

1.1.3. Unliberated Gold in a Sulfide Gangue

When gold is very fine (< 10 µm) and intimately associated with sulfide minerals (either in solid solution or as fine inclusions), cyanidation performance is typically poor. Flotation may be used as a preconcentration step to allow more expensive refractory ore treatment to be performed on a smaller fraction of the material. Under these conditions, the flotation circuit is usually operated to maximize the recovery of all the minerals containing gold (bulk flotation).

1.1.4. Flotation of Gangue Minerals

Gangue minerals may be removed by flotation prior to the gold recovery stage to improve performance or reduce reagent consumption, for example, carbonates that affect pH control during oxidative pretreatment or carbonaceous material, which can adsorb gold during cyanidation. The use of flotation in this way also acts as a preconcentration step.

1.2. Chemistry of Gold Flotation

Free Gold is naturally floatable in most industrial systems, which means that it can be recovered without collector addition. This is due to the adsorption of hydrocarbons (and other surface reactions), which are dependent on gold's metallic properties, particularly its high electrical conductivity, which allows surface electrochemical reactions to occur catalytically and selectively. In noncomplexing aqueous media, gold is unreactive because gold ions and oxides are thermodynamically unstable, requiring strongly oxidizing conditions for their formation.

Gold hydrophobicity is enhanced by the addition of flotation collectors such as xanthates, DTP (Dithiophosphate), and MTB (Mercaptobenzothiozole), as used in sulfide mineral flotation. Usually collector concentrations in the range of 25 to 75 g/t are used. The mechanism by which gold hydrophobicity is enhanced is similar to that of certain sulfides such as pyrite. Xanthate ions are oxidized at the gold surface to form the neutral dimer diaxanthogen.

1.3. Collectors in Gold Flotation

Flotation with xanthate collectors involves the anodic oxidation of the collector that may involve sub-processes such as metal xanthate formation, chemisorption of the xanthate ion and oxidation of the xanthate to form dixanthogen (Groot, 1987; Monte et al., 1997). These adsorb onto mineral surfaces, rendering the mineral hydrophobic. It is generally accepted that the xanthate species responsible for the flotation of free gold is dixanthogen (Miller et al., 1986).

This is a neutral oil that will adsorb onto the surface of any naturally hydrophobic solid, rendering it floatable (Gardner and Woods, 1974). Dixanthogen may form on gold by either the application of an applied potential or by a mixed potential mechanism in a pulp that involves the reduction of oxygen. Studies have shown that the development of a finite contact angle and the onset of flotation of gold.

particles occur at a potential close to that of dixanthogen formation (Woods et al., 1994). Dixanthogen is the active hydrophobic species in the flotation of gold, pyrite and certain other sulfides commonly encountered in gold extraction. The potentials required to oxidize the xanthate series are given in Table 1.1.

Table 1.1. Reduction potentials for homologous series of dithiolate- thiol couples (Finkelstein, N.P and G.W. Poling. The role of dithiolates in the flotation of sulfide minerals)

Homologue	Xanthate	Monothio-carbonate	Dithiophosphate
Methy	-0.004	0.020	0.315
Ethyl	-0.060	0.002	0.255
n-Butyl	-0.091	-0.022	0.019
Isopropyl	-0.096	ND	0.196
n-Butyl	-0.127	-0.038	0.122
Isobutyl	-0.127	ND	0.158
n-Amyl	-0.159	-0.080	0.050
Isoamyl	ND	ND	0.086
Hexyl	ND	-0.12	-0.015

ND= not determined

The rapid decomposition reactions in the acidic region are much better understood than alkali decomposition, which is slower and has a complex mechanism. The general ranges of stability of thiol collectors are given in Table 1.2.

Table 1.2. pH stability ranges for various thiol collectors (King R.P.,editor.1982. Principles of Flotation. Johannesburg: South African Institute of Mining and Metallugy)

Collector	pH Range
Dithocarbamate	5-12
Dithiophosphate	4-12
Dixanthogen	1-11
Mercaptobenzothiazole	4-9
Thionocarbamate	4-9
Xanthate	8-13

The longer-chain xanthates are more readily oxidized, generating dixanthogen at lower potentials (Gardner and Woods, 1974). An increase in thiol chain length increases the maximum contact angle, thereby increasing the hydrophobicity of the surface species. Both these attributes favour the use of longer chain xanthates, such as potassium amyl xanthate (PAX) for the flotation of free gold. It is quite common to encounter silver and other precious metals forming alloys with native gold. The positive effect that silver has on gold floatability was first recognized in experiments using plates of pure gold, silver and gold–silver alloys. The adsorption of ethyl xanthate on silver is generally thought to take place through an electrochemical mechanism of metal xanthate formation on the surfaces (Talonen et al., 1991; Woods et al., 1992; Yoon and Basilio, 1993). For ethyl xanthate, the presence of silver in gold leads to silver xanthate formation at a potential proportionately lower than for dixanthogen formation on pure gold (Leppinen et al., 1991). As a consequence, the flotation of gold–silver alloys can be achieved at potentials considerably lower than that for gold. Xanthate ions chemisorb on silver at potentials below the region at which silver xanthate deposits (Woods et al., 1992). Chemisorbed ethyl xanthate results in finite contact angles on silver surfaces and the initiation of flotation appears to result from the chemisorption process.

For more rapid flotation dixanthogen may play a supporting role. The chemisorbed sub-monolayer (Woods et.al., 1995) could be important in retaining the dixanthogen at the gold surface through hydrophobic interactions between the adsorbate and the bulk phase (Woods et al., 1994). The xanthogen formates are produced by reacting alkyl chloroformate with xanthate salts.

Dithiophosphates (Nagaraj, 1997; Allan and Woodcock, 2001) are useful secondary collectors (sometimes referred to as promotors) to xanthates in gold flotation. It has been known for a long while that Aeropromotor 208 is an effective promoter in gold flotation (Leaver and Woolf, 1934a; Rabone, 1939). Dithiophosphate (DTP) collectors are stable under more acidic conditions than xanthates and hence are used when flotation at $pH < 8$ is desirable.

These collectors will not float pyrite effectively under alkaline conditions and therefore have good selectivity for recovery of other nonferrous sulfides. DTPs generally give a lower recovery (although possibly with higher selectivity) than the corresponding xanthate and are less readily oxidized, forming the neutral dimer dithiophosphatogen as follows;

$$[Et(DTP)]_2 + 2e \rightleftharpoons 2Et\,(DTP)_2^{\,-} \quad E^0 = 0.225\ (V)$$

Dithiophosphorous acids are known also to adsorb on gold under certain conditions but they are usually considered not to be selective for gold.

The monothiophosphates provide a good selectivity for gold values with a high silver content (Nagaraj et al., 1991) and are able to recover gold selectively from some sulfide ores. Silver has been shown to assist adsorption of discresyl monothiophosphate onto gold (Nagaraj et al., 1991; Basilio et al., 1992a). The monothiophosphates are more stable and stronger than xanthates, dithiophosphates and xanthogen formates. They have also found application for selective gold flotation from primary gold ores or for improving gold recovery in basemetal sulfide flotation in alkaline circuits. Monothiophosphates are now used widely on copper–gold flotation plants.

Mercaptobenzothiazole (MBT, Aeropromoter 404) is a fairly specialized collector (Finkelstein and Poling, 1977) and is the preferred collector for the flotation of gold and gold-carrying pyrite in acid circuits (O'Connor and Dunne, 1991). It is also recommended for oxidized and partially oxidized pyritic gold ores. MBT exists mainly in the non-ionized form in acid and alkaline solutions and both forms are more stable than the corresponding forms of xanthate. It is also a useful secondary collector in the treatment of copper–gold ores and the addition is characterized by a heavily mineralized froth and fast flotation kinetics. Mercaptobenzothiozole is despite being used in many industrial plants, including those for flotation of oxidized copper anda zinc minerals. It is a well-investigated collector. It is used widely for free gold and pyrite flotation under acidic conditions. MBT has a pK_a of about 7 and is ionized in alkaline solutions. It has a lower tendency to form the dithiolate, $(MBT)_2$, than other thiol collectors and instead forms metal- MBT species, which render the mineral surface hydrophobic.

The ethoxycarbonyl thiourea collectors were introduced in the late 1980s (Nagaraj and Avontins, 1988). These collectors were developed as selective collectors for operation at reduced pH values.

Amine-based collectors have been used to float gold and gold-bearing pyrite (Levin and Veitch, 1970). The application is limited as the amine collector is selective for pyrite at high pH values (>10) only.

Dithiocarbamates (DTCs) such as isopropyl ethyl thiocarbamate, are most commonly used for flotation of copper sulfide minerals. They are relatively stable collectors which oxidize to form a dithiolate, $(DTC)_2$, in a manner similar to xanthates as follows:

$$(DTC)_2 + 2e \rightleftharpoons 2DTC^-$$

Very few gold flotation plants use single collectors and the preference is either to add a blended collector, that is, a mixture of a number of collectors, or collectors added separately in smaller quantities than those of the primary collector. These collector combinations result in better overall flotation recoveries. The term synergism is often applied to this phenomenon. The order of addition of the different collectors can also be important in improving flotation performance (O'Connor et al., 1990). Dithiophosphates are probably the most widely used promoters in gold flotation (Pickett, 1978; O'Connor and Dunne, 1991).

1.4. Frothers in Gold Flotation

Frothers are usually nonionic surface active molecules which, in flotation, are used to produce a large, stable air-water interface to ensure that floated material remains in the froth to allow recovery to concentrate. The strength and stability of the froth is important when floating free gold. There appears to be a preference for polyglycol ether-based frothers on most gold plants in combination with one or other frothers. When selectivity is required or, in the case of copper–gold ores, where a copper concentrate is sold to a smelter, a weaker frother such as methyl isobutyl carbinol (MIBC) is preferred. The choice of a particle size-balanced frother is also an important consideration in gold flotation as this promotes composite particle recovery in the scavenger flotation circuit. As a rule, the glycol or polypropylene glycol methyl ether frothers are ideal for this application (Klimpel, 1997). The blended interfroth frothers have found wide acceptance on Australian gold plants and the base reagent is an alkyl aryl ester (Goold, 1990).

1.5. Activators in Gold Flotation

Activators are chemicals that enhance the flotation performance of a specific mineral or minerals. These are usually inorganic compounds which react with or are adsorbed such that a stronger reaction is possible with a collector. Activation implies improved floatability of a mineral after the addition of a soluble base metal salt or sulfidizer. It is generally thought that the metal or sulfide ion adsorbs onto the mineral surface thus changing its surface chemical properties. In this way, the flotation response can be improved and/or the pH range of flotation for the mineral can be extended, the rates of flotation can be increased and selectivity can be improved. In general, activators are added before collectors in order to establish mineral surface conditions most suitable for effective collector action.

1.5.1. Metal Salts

Early work on gold particles with copper sulfate showed no improvement in recovery but an increased rate of flotation of gold (Leaver and Woolf,1934). More recent laboratory testwork on a refractory gold ore has shown, however, that a 5%increase in free gold flotation recovery is achievable when adding copper sulfate (Teague et al., 1999b).

It is widely accepted that the main purpose of copper sulfate in the flotation of sulfide gold carriers is to enhance the flotation of the sulfides and, in particular, pyrrhotite (Mitrofanov and Kushnikova, 1959), arsenopyrite (Gegg, 1949; O'Connor et al., 1990) and pyrite (Bushell and Krauss, 1962).

The sequence of copper sulfate addition (before or after the collector) is important. For arsenopyrite and pyrrhotite, the preference is for copper sulfate to be added first (Teague et al., 1999b; Monte et al., 2002) and recovery differentials as large as 20% are apparent when this is done (Teague et al., 1999b). The addition of copper sulfate has been found to increase the rate of pyrite flotation, providing an overall increase in gold recovery because of the gold association with pyrite (Allison et al., 1982; Duchen and Carter, 1986). The adsorption of copper onto pyrite and pyrrhotite is pH-dependent, smaller quantities being adsorbed at alkaline conditions. For some ores, the addition of copper sulfate at intermediate pH values such as 7–10 may be harmful and may reduce pyrite recovery (Bushell, 1970).

The activation of the mineral surface by adsorption of copper ions to allow the enhanced adsorption of collector has been touted as one mechanism that provides the improved flotation performance (Leppinen, 1990; Leppinen et al., 1995; Nagaraj, 1997; Nagaraj and Brinen, 1995). The redox potential of the pulp will also increase with the addition of copper sulfate, thereby increasing the oxidizing environment for thiol collectors, thus favouring improved flotation performance (Nicol, 1984).

1.5.2. Sulfidization

The application of sulfidizers (sodium sulfide and sodium hydrosulfide) to enhance the flotation of oxidized ores is well known (Jones and Woodcock, 1984; Oudenne and de Cuyper, 1986; O'Connor and Dunne, 1991). The first detailed laboratory study of the influence of sodium sulfide on the flotation of gold-bearing ores was undertaken in the mid 1930s (Leaver and Woolf, 1935). The outcome from this study was that, in general, sodium sulfide retards the flotation of gold, although for some ores there was benefit in its addition. Similar comments are to be found in the literature since that time (Taggart, 1945; Aksoy and Yarar, 1989). Sulfide ions appear to act as flotation activators at low concentrations (less than 10^{-5} M) and as a strong depressant at concentrations above 10^{-5} M (Aksoy and Yarar, 1989). The addition of sulfide ions converts some coatings on mineral surfaces into sulfides

(Healy, 1984) and subsequent xanthate addition will promote flotation. For successful activation, the sulfide activator should be added slowly and at starvation quantities.

1.6. Depressants in Gold Flotation

Depressants for native gold that are usually introduced during the flotation process include compounds such as calcium ions, chloride ions, calcium carbonate, cyanide, sodium silicate, sodium sulfite, ferric and heavy metal ions, tannin and related compounds, starch and other organic depressants and many others (Taggart, 1945; Broekman et al., 1987; Marsden and House, 1992; Lins and Adamian, 1993; Allan and Woodcock, 2001; Chryssoulis, 2001).

All of these may competitively adsorb on the gold surface thus preventing the adsorption of the collector(s) added. It has also been suggested that the ferric ions, which would be in the form of hydrated oxides, may act as a physical barrier between the air bubble and gold surface but this effect is reversed simply by washing with water (Aksoy and Yarar, 1989). However, flotation of native gold often proceeds satisfactorily in the presence of many of these compounds.

1.7. pH Modifiers

An important consideration when selecting the reagent scheme for the flotation of a particular ore is the choice of pH value and pH modifier. Commonly used industrial modifiers and their characteristics in flotation is given in Table 1.3.

Table 1.3. pH modifiers used in flotation (American Cyanamid. 1986. Mining Chemicals Handbook. Parsippanny, NJ: American Cyanamid

Reagent	Usual addition g/t	Sulfides	Precious Metals
Lime	250- 2500	Depresses iron sulfides ; also lead, marmaritic zinc and certain copper minerals if excess used	Depresses gold. Little effect on silver sulfides.
Soda Ash	250- 1500	Assists separation of sulfides from each other by acting as gangue slime dispersant. Aids recovery of arsenopyrite when used with copper sulphate.	Assists flotation of precious metals and sulfides
Alkaline Silicates	250- 1500	Disperses gangue slimes, assists grade and recovery. Produces brittle- type froth. Depressed quartz and silicates.	In controlled amounts, aids selectivity and grade of concentrate.
Sodium hydroxide	250- 2000	Gangue slime regulator. With copper sulphate, activates arsenopyrite	Some assistance o recovery of free gold.
Alkaline phosphates	250- 1000	Improves grade for some sulfide ores by dispersing gangue slimes. Particularly effective on ores containing iron-oxide slimes.	Assists recovery of precious metals from slimy ores.
Sulphuric acid	250- 2500	Assists recovery of iron sulfides, especially after depression by lime or cyanide.	Assists recovery of gold in dilute pulp. Less benefit in thick pulp.

1.8. Influence of Conditions on Gold Flotation

1.8.1 Eh of the Flotation of Pulp

Eh has been shown to be a significant parameter in the flotation of precious metals and sulfide minerals (Woods, 1984; Jones and Woodcock, 1984; Hintikka and Leppinen, 1995). The Eh provides a measure of the oxidizing or reducing capacity of the solution phase in a flotation pulp. When a mixture of two or more sulfide minerals is ground in a steel mill, there are galvanic interactions between the minerals and the steel grinding media. Redox reactions that occur cause current flow leading to surface oxidation of the metals and minerals. Minerals and metals with high rest potential (i.e., noble) act as a cathode, while minerals and metals with lower rest potential (active component) act as the anode.

The cathodic reaction will consume oxygen and produce hydroxyl ions while the anodic reaction for a sulfide mineral will provide metal cations and elemental sulfur. The generic reactions are shown below:

Cathodic Reaction: $\frac{1}{2} O_2 + H_2O + 2e^- \longrightarrow 2OH^-$

Anodic Reaction : $MeS \longrightarrow Me_2 + S + 2e^-$

Overall Reaction : $MeS + 1/2O_2 + H_2O \longrightarrow Me^{+2} + S + 2OH^-$

The hydroxyl ions can adsorb onto mineral surfaces or react with metal ions in solution to form hydroxides that precipitate on mineral surfaces. Predicting the Eh in a flotation pulp is complicated by the many galvanic interactions that occur in the system (Rao et al., 1976; Martin et al., 1992). In practice, sulfide flotation plants operate at pulp potentials that are in the range of +50 to +200mV vs. SHE (Grano, 2004) and the value depends on the pulp pH, type and quantity of sulfide minerals, the type and quantity of grinding media (Rao et al., 1976), and the amount of air–pulp contact in and after the grinding circuit (Teague et al., 1999). The pulp potential in many aerated systems is deemed to be too oxidizing to achieve maximum mineral selectivity by flotation. This is based on the fact that each mineral has a characteristic potential where flotation starts. At too anodic potentials, the potential ranges for individual minerals overlap, causing simultaneous flotation of several minerals and consequently poor selectivity (Ralston, 1991; Chander, 2003).

Furthermore, surface oxidation and precipitation of metal hydroxides may be severe enough in certain high-Eh pulps to cause a decrease in flotation performance (Fornasiero and Ralston, 1992).

Eh–pH diagrams are available, and these indicate that the chemisorption of xanthate occurs over a wide range of Eh and pH values (Woods et al., 1992) implying that the flotation of gold can be successfully accomplished over a wide range of conditions.

It has been shown under laboratory conditions that the optimum Eh range for native gold flotation (Hintikka and Leppinen, 1995) is between +10 and +60mV (vs. SCE). Ethyl xanthate has been shown to form dixanthogen on pure gold at potentials above approximately +200mV (Leppinen et al., 1991).

Laboratory testwork on mixed-sulfide gold-bearing ores showed that the flotation recovery of iron sulfides was highest at +50mV and the recovery began to decrease at higher potentials (Hintikka and Leppinen, 1995). For copper-activated pyrite, the optimum Eh at a pH value of 9 was +35 mV (Shuhua et al., 2004). Copper minerals, on the other hand, reached a maximum recovery at values of between +150 and +200mV (Hintikka and Leppinen, 1995). The recovery of arsenic minerals was found to progressively increase up to +150mV followed by a slight drop at +200mV and rapidly decreased at higher potentials.

1.8.2. Flotation Gases and The Impact of Oxidation on Flotation

The presence of oxygen is a prerequisite for the flotation of sulfide minerals with thiol collectors (Gardner and Woods, 1974). Electrochemical investigations have shown that the role of oxygen is to provide a cathodic reduction process that allows anodic oxidation processes, involving the flotation collector, to occur on the mineral surface. For example, the collector PAX, which is used on many gold flotation plants, is readily soluble in water and dissociates into positively charged potassium ions and negatively charged polar amyl xanthate ions in a pulp.

Chemisorption of the xanthate ion onto the mineral surface then occurs and can render some minerals hydrophobic. For some mineral systems, the chemisorption step is followed by electrochemical oxidation of the adsorbed xanthate to give oxidized species (dixanthogen) that render the sulfide surface hydrophobic. Oxygen for the chemical reactions is introduced into flotation pulps by air entrainment in the grinding and classification circuits and during pumping of the pulp. More oxygen is provided during flotation as air is added in the flotation cells to generate bubbles.

Reduced flotation performance may also result from oxidation products of easily oxidized minerals, such as pyrrhotite and arsenopyrite. These oxidation products can attach unselectively on minerals in the pulp, thus enhancing their floatability (Hintikka and Leppinen, 1995). Reduced selectivity can also arise from the formation of elemental sulfur, thiosulfate, metal hydroxides and other surface layers that are frequently detected in aerated mineral slurries (Smart, 1991). In flotation processes where excessive oxidation causes problems, the application of a reducing agent, such as sodium sulfide, is useful to restore the pulp potential to an appropriate level.

1.8.3. Modification of pH for flotation

An important consideration when selecting the reagent scheme for the flotation of a particular ore is the choice of pH value and pH modifier (Bulatovic, 1997). Lime and sulfuric acid are presently the most common pH modifiers. In the past, soda ash (sodium carbonate) was extensively used in preference to lime for gold flotation (Taggart, 1945). Sodium carbonate is a common additive to precipitate heavy-metal ions and calcium ions while buffering the solution in the pH range 8–9; all of these conditions are favourable for the flotation of free gold (Allan and Woodcock, 2001). The pH value chosen for gold flotation is dependent on a number of factors (Broekman et al., 1987) and the selection usually takes account of the type and quantity of gangue components (both sulfide and silicate) in the ore. Certain clay minerals are very floatable in the pH range 5–9 and if these are present in the ore, then pH values outside this range are chosen for flotation (Bushell, 1970).

The recent application of surface analytical techniques has provided compelling evidence of the loss of free gold due to depression induced by surface-bound calcium and hydroxyl ions (Chryssoulis, 2001). The benefit to gold recovery by reducing the pH value from 10.5 to 9 has been demonstrated at the Candelaria Copper Mine in Chile, where a 10% increase in gold recovery was realized at the lower slurry pH (Kendrick et al., 2003).

Pyrite and arsenopyrite float well in the pH range 3–10. This excludes the addition of an activator such as copper sulfate. The depression of pyrite at high pH values can be overcome by the addition of more collector, stronger collector and copper sulfate. Pyrrhotite floats best in acid circuits. The pH range for flotation for gold sulfide ores containing pyrite and arsenopyrite is generally between 7 and 9. Flotation may be carried out at higher pH values of 9 to 10, since this favours the depression of talc minerals and avoids the use of expensive gangue depressants.

Copper minerals are floated in alkaline circuits. There is a preference to float at high pH values to depress pyrite if a reasonable amount of pyrite is present or at natural pH value if there is only a small amount of pyrite present.

1.8.4. Particle Size and Shape in Flotation

It is well known that particle size is an important parameter in flotation and that size limits exist at which minerals will and will not float. The high particle-density of gold and its malleable and ductile properties that favour the propagation of platy particles, further compound this effect. Platy/flaky particles are formed in the treatment process, particularly in grinding, or during transportation events in nature (Rickard, 1917; Aksoy and Yarar, 1989). During these events, some gold particles are impregnated with nonfloatable particles (Taggart, 1945; Pevzner et al., 1966), inhibiting flotation. Passivation of a gold-particle surface may also occur after considerable hammering by steel grinding-media (Pevzner et al., 1966). On the other hand, it is postulated that the surface of the gold could become more active and therefore more floatable due to work hardening (Allan and Woodcock, 2001).

It has been suggested that the practical particle size limits for gold flotation are around 5–200 µm (Allan and Woodcock, 2001). Particles as small as 3 µm have been floated at laboratory scale (Allison and Dunne, 1985), while actual measurements indicate that the flotation performance on many gold plants decrease rapidly below 10 µm (Chryssoulis, 2004). At the coarse end, gold particles as large as 300 µm (Leaver and Woolf, 1934b) and 700 µm have been floated in laboratory flotation cells under specific operating conditions and high collector additions (Lins and Adamian, 1993). Flotation of 590 µm gold particles has been reported on an industrial scale with ‘unit’ flotation cells (Leaver and Woolf, 1934).

Pulp density and aeration rates influence flotation-cell pulp hydrodynamics and are important parameters in extending the particle-size limits of gold flotation. There is conflicting commentary on the best pulp density for gold particle flotation, both a high pulp-density (Leaver and Woolf, 1934a) and a low pulp-density being recommended (Fahrenwald et al., 1936; Lins and Adamian, 1993).

1.8.5. Slime Coatings and Floatable Non-Sulfide Gangue

The deleterious impact of clay slimes on gold flotation is well known (Rickard, 1917; Leaver and Woolf, 1932, 1934a; Richards and Locke, 1940; Orel et al., 1986). The failure of free gold and sulfide minerals to float has at times been shown to be related to the presence of coatings of colloidal or near-colloidal gangue or silicate material adhering to the mineral surface.

These coatings are formed under pulp conditions in which the sulfide particles and silicate particles are oppositely charged. Gangue minerals that are known to cause problems include talcose and carbonaceous minerals, bentonite clay, goethite [FeO(OH)] (Leaver and Woolf, 1934), iron oxide and manganese slimes, pyrophyllite [$AlSi_2O_5OH$] (O’Connor and Dunne, 1991) and carbonates (Da Silva et al., 1989). Slime coatings are controlled by the use of gangue-dispersing agents. Sodium silicate is widely used for this purpose and is most effective when the alkalinity is carefully controlled. Sodium sulfide has also been found to be an effective dispersing agent. In addition to coating the mineral surface, the gangue particles may coat the bubble surface, affecting the ability of any gold and sulfide particles to attach to the air bubbles (Allison et al., 1982; Aksoy and Yarar, 1989).

Other more recent remedies to overcome the problem of slime coatings have included physical methods such as removal of the slimes by cycloning (Bushell, 1970; Broekman et al., 1987), high-intensity attritioning (Duchen and Carter, 1986; Valderrama and Rubio, 1998; Allan and Woodcock, 2001) and a lower pulp density (Bulatovic, 1997).

Organic compounds of high molecular weight that maintain a state of dispersion of deleterious slime components by forming wettable coatings on the gangue particles are used for much the same purpose as the inorganic dispersing agents. These organic compounds are referred to as organic gangue depressants. Typical examples are glue, starch, dextrin, gum arabic, carboxymethylcellulose and the more recent modified-guar gums.

Selection of the correct depressant type and dosage is critical, as an overdose results in both loss of free gold (Leaver and Woolf, 1934) and sulfides that contain gold (Steenberg, 1984). The anionic polymers (guar gums, cellulose gums, modified lignin sulfonates) generally have a negligible depressant capability on sulfide minerals while the cationic polymers (starches, dextrin, tannin derivatives, oxycellulose) are capable of acting as sulfide mineral depressants (Bulatovic, 1999). The combination of a collector and depressant is also important since in the flotation of pyrite, for example, guar gum will have a more adverse effect when used with MBT than with xanthate (Steenberg, 1984).

Porphyry copper–gold ores usually contain some gangue components that are highly floatable and contaminate the copper concentrate. Maintenance of a high copper-concentrate grade requires that gangue depressants be used. Silicates, guars and carboxymethyl cellulose are the common depressants applied in the copper industry. Carbonaceous and graphitic minerals are soft and flaky, and easily broken down during grinding. During flotation, the carbon floats readily owing to its fine grain size, natural hydrophobicity, platy nature and low density. Graphitic carbon and clays can be the cause of poor gold recovery on many refractory gold or flotation plants.

1.8.6. Natural Metal and Organic Coatings on Gold

Most coatings on mineral surfaces are detrimental to flotation, but in some cases, the effects can be overcome (Allan and Woodcock, 2001). Many types of surface coatings have been reported to occur on native gold particles. Perhaps the most difficult coatings to cope with are hydrated iron oxides (Richart, 1912; Richards and Locke, 1940; Brooke et al., 2003).

The surfaces of gold particles can become coated naturally with precipitates of iron, from oxidized sulfides in an orebody or from rusting iron, such as iron grinding media, as first reported by Head (1936). Gold from placer deposits heavily stained or coated by iron oxides or impregnated by hydrophilic minerals is not easily floated (Wang and Poling, 1983). Tarnished gold has been found also to have a markedly higher mercury content compared to ‘shiny’ gold (Chryssoulis et al., 2003a).

Some gold flotation pulps may contain humic and tannin substances (organic decay products from wood and vegetation) and sulfide ions from sulfide mineral (Aksoy and Yarar, 1989) that are reported to impact on gold flotation. Humic acid has been found to be only marginally deleterious to gold flotation (Aksoy and Yarar, 1989) and some naturally occurring organic coatings can be removed by conditioning with sodium hydroxide or acid solution (Johns, 1935). Methods to improve the flotation of gold-coated particles include more collector, acid treatment (Brooke et al., 2003), chelating agents, water wash and organic acids (Bulatovic, 1997). Flotation of some coated particles, even when a high dosage of collector is employed, is reported to be weak and erratic.

1.8.7. Liberation

The primary mineralogical requirement for the effective flotation of gold is that it is liberated (i.e., as free gold grains), or that it occurs in composite particles which are floatable. This latter condition usually requires gold to be a component of a particle with a predominantly gold and/or sulfide surface. If oxides or silicates form a composite with gold, then floatability would be lower and dependent on their respective surface areas and hydrophobicities.

1.9. Flotation Circuits

Flotation is one of the methods in gold processing. Other mineral processing methods for gold containing ores were summarized in Figure 1.1. and effects of selected variables on flotation were summarized in Figure 1.2. Flotation circuit configuration on most gold mines can be divided into a number of categories.

Open circuits with no cleaning at all, and open and closed circuits with single stage and two stages of cleaning. Open circuits have the advantage of no feedback from the effects of non-steady-state operation and therefore are inherently more stable than the closed-circuit configuration. Closed and open-circuit flotation cleaning is used on gold mines where high-grade concentrates are required for roasting and smelting. Under these conditions, it is difficult to maintain very high gold and sulfide flotation recoveries, while also producing an acceptable grade of concentrate. Where there is no constraint on concentrate quality, high gold and sulfide flotation recoveries are achievable to the extent that a discardable gold flotation tail is possible.

Cleaning-circuit configuration, either single or two stages of cleaning, and cleaner residence time are related to the particle size of the sulfides in the flotation feed and also the presence or absence of floatable gangue components.

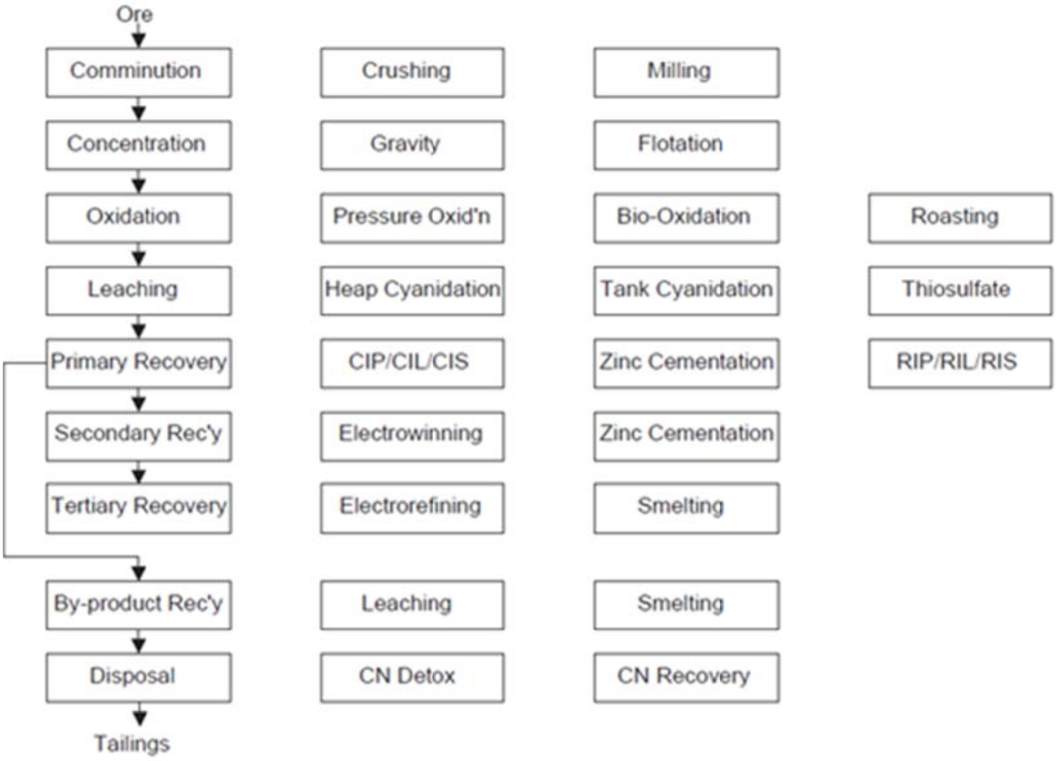
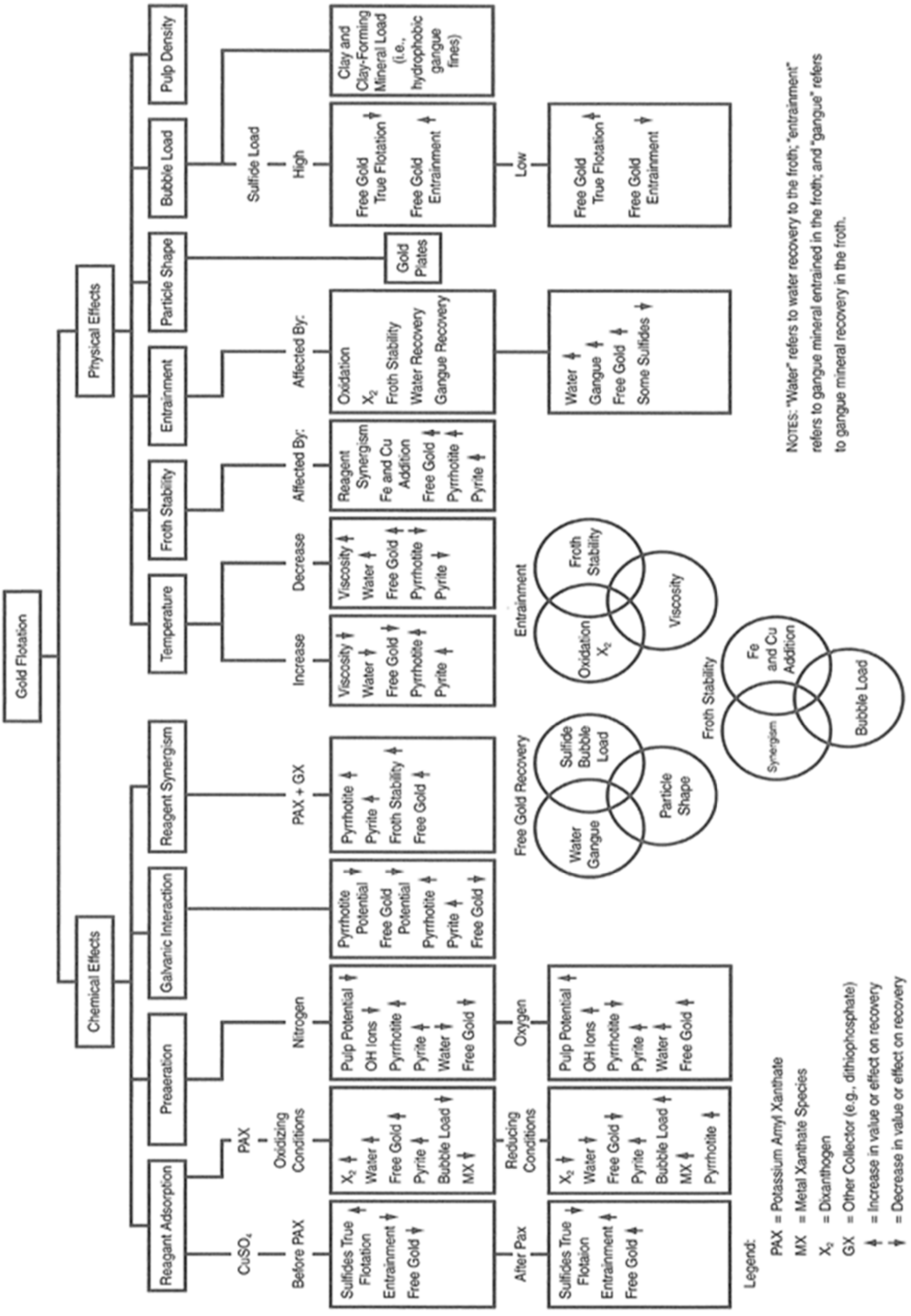


Figure 1.1. Generic flowsheet for processing gold-containing ores (Mike D. Adams, 995, Advances in gold Ore Processing).

Unit flotation cells (Hasting, 1937; Taggart, 1945) and the more recent. Flash flotation cells (Kalloinen and Tarainen, 1984) are installed in grinding circuits with the purpose of improving the overall flotation recovery of free gold (Taggart, 1945; Suttill, 1990; Laplante and Dunne, 2002). The aim is to remove as much of the free gold contained in the circulating load of the grinding mill before it is overground or is covered with coatings of iron, sulfide or other coatings that will lower flotation recoveries. Improved overall gold flotation recoveries of 2–10% have been quoted. Furthermore, the inclusion of Unit and Flash flotation cells will generally provide better flotation stability and performance. Improved overall gold flotation recoveries from 3% to 10% have been quoted. Column flotation cells are used in roughing and cleaning duties on a number of mines treating gold ores (Lane and Dunne, 1987). A column cell typically provides higher concentrate grades compared to a mechanically agitated cell; however, losses of coarse gold may be higher in the column cell.



Legend:
PAX = Potassium Amyl Xanthate
MX = Metal Xanthate Species
X_2 = Dixanthogen
GX = Other Collector (e.g., dithiophosphate)
↑ = Increase in value or effect on recovery
↓ = Decrease in value or effect on recovery

NOTES: "Water" refers to water recovery to the froth; "entrainment" refers to gangue mineral entrained in the froth; and "gangue" refers to gangue mineral recovery in the froth.

Figure 1.2. Conceptual Conceptual model for gold flotation, showing effect of selected variables (Teague, A.J., J.S.J. Van Deventer, Minerals Engineering).

1.10. Flotation Practice in Gold Mines

It is not unusual for gold to be present in several forms in any given ore deposit. A typical example of a mixed gold ore is one having some level of free or native gold occurring with the remainder being associated with various types of sulfide minerals (Teague et al., 1999a). The selection of an optimal flotation reagent scheme is very dependent on the specific mineralogy of the gold ore to be processed.

1.10.1. Refractory Gold Ores

Successful concentration of gold in refractory sulfide ores is almost exclusively dependent on the association of the gold with the sulfides. Refractory gold ores commonly contain free gold, sub-microscopic gold, carbonaceous material, base metals, pyrite, marcasite, arsenopyrite and pyrrhotite. Clays and graphitic carbon are the most troublesome accessory components in some of these ores, as far as gold concentration is concerned.

Arsenopyrite has very similar properties to pyrite and the flotation conditions for its recovery are similar to pyrite. Arsenopyrite is marginally less hard and more brittle than pyrite and pyrrhotite. During milling, the arsenopyrite is therefore subject to more overgrinding than pyrite and pyrrhotite. The difference in the recovery of these minerals is due not only to the difference in the surface chemical properties of the particle but also to the difference in their overall size distribution.

1.10.2. Arsenopyrite, Pyrrhotite and Pyrite Ores

In Brazil, the application of flotation for gold-bearing ores started in the 1980s. There are three important Brazilian plants using flotation to process gold ores. At the Sa˜o Bento plant, gold and pyrite are floated into a bulk concentrate for further processing by bacterial and pressure leaching. The Morro Velho mine located at Raposos treats graphitic gold ores that contain free gold and gold associated with pyrite. The first stage of flotation to remove free gold and graphite is accomplished with MBT at neutral pH. The remaining gold containing pyrite is floated in a second stage with the addition of dithiophosphate. The two flotation concentrates are combined ahead of roasting.

In North America, four gold mines treat refractory gold ores. At the Lone Tree and Twin Creeks mines in Nevada, USA, the fine-grained arsenian pyrite that contains most of the gold is floated using the nitrogen-based N_2TEC technology (Simmons, 1997). Flotation takes place at neutral or slightly acid pH values with the addition of lead nitrate as activator and xanthate as collector. The Campbell Mine in Ontario (Chang et al., 1983) treats an arsenopyrite–pyrite ore and the flotation concentrate is pressure leached. Flotation reagents consist of copper sulfate as activator, PAX and AF208 as collector and the frother is Dowfroth 250. Flotation is conducted at natural pH (O'Rourke et al., 2000). At the Snip Mine in British Columbia, Canada, the gold ore is floated at natural pH value (8.5) using Aerophine 3418A and a dithiophosphate, the frother is Dowfroth 250. The flotation concentrate is filtered, bagged and sent to a smelter.

There are six gold mines in the Australasian region that treat refractory gold ores. The largest of these is the Porgera Gold Mine in Papua New Guinea, where a pyrite–arsenopyrite flotation concentrate is pressure leached. Flotation takes place at natural pH values and PAX is the only collector. The frother is a mixture of MIBC and Dowfroth 400. The Macraes Gold Mine in New Zealand floats a bulk arsenopyrite–pyrite–pyrrhotite sulfide flotation concentrate at natural pH value. Flotation reagents include copper sulfate, sodium isobutyl xanthate (SIBX) as the primary collector, a dithiophoshate promoter and two frothers, MIBC and Interfroth 60. There are two gold mines in Australia, the Wiluna Gold Mine in Western Australia and the Beaconsfield Mine in Tasmania that use bacterial oxidation to process bulk arsenopyrite–pyrite flotation concentrate prior to cyanide leaching. At the Wiluna mine, the gold ore is floated at a slightly elevated pH of 8.2 using copper sulfate, PAX and Interfroth 50. A Unit flotation cell is included in the milling circuit at the Beaconsfield mine. Copper sulfate is added to the mill while SIBX is distributed to three different locations in the flotation circuit. At the Kanowna Belle Gold Mine in Western Australia, the combined Flash flotation concentrate, first rougher concentrate and rougher–cleaner concentrate that contains pyrite and arsenopyrite is roasted. Flotation is undertaken at natural pH values with the addition of copper sulfate, PAX and a guar gum depressant.

There are three gold mines on the African continent that have flotation plants treating refractory gold ores. The Ashanti Mine in Ghana employs a combination of column and mechanical flotation cells to produce a bulk concentrate for bacterial leaching. The other two mines, the Fairview Gold Mine and the Sheba Gold Mine, both in South Africa, carry out flotation at natural pH values and add SIBX or thiocarbamates as collector. Copper sulfate is also used at the Sheba flotation plant.

In Australia, the trend is to use the stronger and longer-chain xanthate PAX as collector as this appears to give slightly higher recoveries and better concentrate grades. Most flotation plants treating refractory gold ores add only a single collector and this is also the case on most Australian flotation plants.

1.10.3. Gold Ores Containing Telluride Minerals

The Emperor Mine in Fiji that selectively floated telluride minerals in the 1970s and 1980s provides the best example of an integrated telluride-flotation circuit. Lime was added to the grinding circuit to depress the sulfide minerals.

The telluride minerals were removed with the addition of a small amount of polyoxypropylene glycol ether frother at a pH value of 9. The telluride concentrate was cleaned after the addition of a small amount of sodium silicate as a gangue slime depressant and froth modifier. Soda ash addition to the flotation tailings provided a pH value of 9.5 prior to conditioning the slurry with xanthate and Terric 402 frother and then floating a sulfide concentrate. The sulfide concentrate was reground before cleaning and the cleaned concentrate was roasted ahead of the cyanide-leaching circuit. The telluride concentrate was treated in a separate circuit to recover the gold and tellurium.

The Emperor Mine in Fiji and the Kalgoorlie Consolidated Gold Mine in Kalgoorlie, presently float a bulk sulfide–telluride concentrate for roasting. The flotation reagent suites at these two mines are a combination of copper sulfate, xanthate and frother.

1.10.4. Pyritic Gold Ores

South Africa still has three reasonably sized operating pyrite flotation plants, compared to the flurry of activity in the 1990s when some 14 pyrite flotation plants were in existence. The three flotation plants that are currently in operation are to be found at the Vaal Reefs Gold Mine and at the Chemwes and Ergo residue re-treatment operations. Copper sulfate is used at two plants (Vaal Reefs and Chemwes), while single collectors are added at the Vaal Reefs Mine (Aeropromoter 407) and Chemwes operation (SIBX). A blended thiol collector is used at the Ergo flotation plant. The pH of flotation is near neutral at all the three plants (pH range 7- 8.5).

In North America, the Sonora Mine in the Mother Lode district of California found that a combination of xanthate and Aero 5688 promoter with small additions of depressant was the best option for floating the pyrite– gold ores that occasionally contained large amounts of hydrophobic talc. The Rio Paracatu Minerac-a˜o (RPM) Mine, in the Paracatu district in Brazil produces a pyrite concentrate with minor amounts of arsenopyrite, which is reground before cyanide leaching.

At the Royal Oak Mine in Ontario, Canada, the predominantly pyritic gold ore is treated by flotation to produce a concentrate that is reground to 38 μm prior to pre-aeration and cyanide leaching. Flotation reagents are PAX, dithiophosphate and Dowfroth 250. Flotation reagents at the Stawell Gold Mine in Victoria, Australia, where carbonaceous pyrite–pyrrhotite gold ores with minor quantities of arsenopyrite are floated at a slightly elevated pH value, include copper sulfate, PAX and Terric 407 frother. The concentrate is reground before cyanide leaching.

1.10.5. Copper- gold Ores

Copper–gold ores are significant contributors to the overall world gold production. Flotation is the principal process for pre-concentration of the copper–gold minerals for subsequent smelting treatment. Gold recovery, although important, is not always considered when optimizing the copper circuit. The need to reject gangue and iron sulfides during cleaning of the copper concentrate, invariably leads to losses of gold. The detrimental effects of depressants, which are introduced in the cleaner circuit, can often be overcome by changing the flowsheet and by adding supplementary collectors. In the treatment of copper–gold ores, mixed collector systems appear to be the norm and selective gold promoters are widely in use. The processing characteristics of copper–gold ores vary from ore to ore and are closely related to the mineralogical composition of the ore. Based on composition, the ores can be classified into porphyry copper–gold ores containing pyrite, porphyry copper–gold ores with negligible pyrite content and altered supergene copper–gold ores. Supergene alteration leading to the formation of oxide and secondary copper minerals is common. The amount of free gold varies among deposits and is typically higher in the supergene ores. The porphyry-type ores frequently includes free-gold recovery circuits, including Flash flotation and gravity concentration, to improve the overall gold recovery.

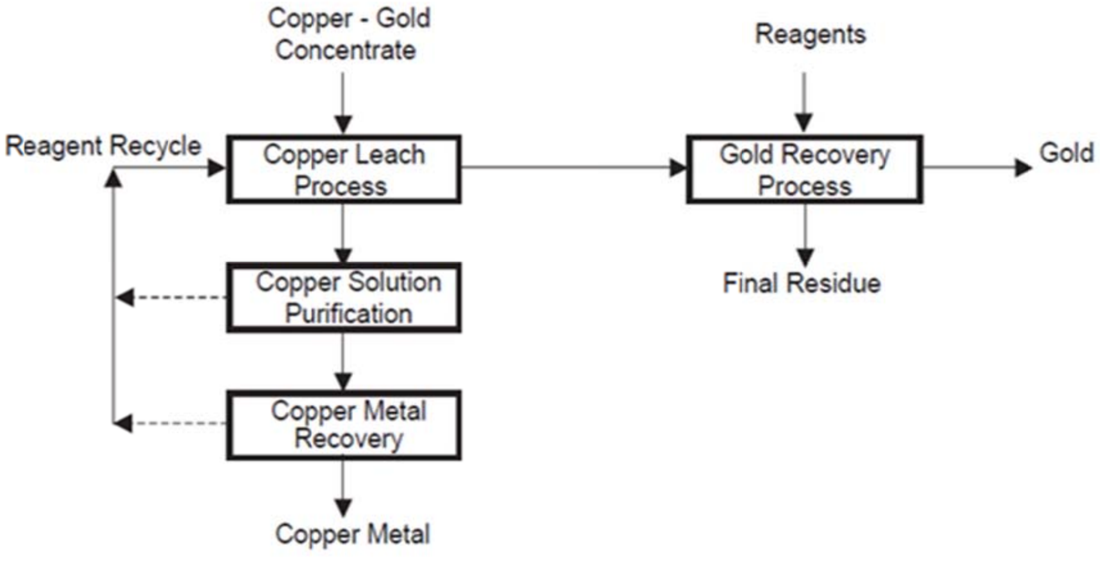


Figure 1.3. Block flowsheet for hydrometallurgical treatment of copper–gold concentrates.

2. EXPERIMENTAL PROCEDURE

2.1. Overview of Test Work Program

The purpose of this research is to investigate flotation behaviour of a sulphidic gold ore from Ovacık (Turkey). According to the flotation performance of the ore a prefeasibility study will be done if the ore is economically recovered by flotation.

First of all, various flotation tests were done. The floatability of the ore was examined by changing the flotation conditions. In the experiments, the effects of collector type and dosage, pH, pulp density, sulfidization and particle size were investigated. At the best flotation conditions which means high gold recovery and grade obtained, a preconcentrate with high gold content was produced for leaching operations. The analyses were done by Koza Gold Operations chemistry lab, accredited by TURKAK and leaching tests were performed at Koza Gold's metallurgical lab.

Mineralogical investigation is a critical and integral aspect of any prefeasibility study as well as one of the quality control measures in the process optimization step. The ore used for the experiments contains around 4-6 g/t Au and % 4-6 S. Moreover, the relationship between the gold and sulphur was also established with the help of flotation results. The mineralogy of the gold is the most important parameter that affects the flotation and also cyanide leaching performance. The refractory gold ores are difficult to process by conventional methods.

2.2. Ore Characterization and Preparation of Samples

The gold ore used in this study was taken from one of Koza Gold's mine (Z vein). The deposit was discovered at 2010. According to the preliminary stage of the explorations, the total reserve is 14296 ounce Au (Table 2.1). This reserve is refractory and not recovered by conventional cyanide leaching processes. The major mineral in the ore is quartz. Major sulphide mineral is pyrite. The ore also contains kaolinite, sphalerite, chalcopyrite, galena and tetrahedrite.

Table 2.1. The distribution of S% (>2.5 %) and Au ppm of Z vein in the whole deposit.

TONNES	AUCUTOK*	OUNCES	%	Zones S/ All %
1439	0,73	34	0,24	0,05
106175.05	1,56	5312	37,16	8,24
66328.67	4,20	8950	62,61	13,89
Total		14296	100	22,18

* Au grade that is calculated by ordinary Kriging method.

The samples were taken from the stocked ore with the help of mine geologists. Totally 200 kg of sample was used. Samples were split into 2 kg of packages for flotation tests.

A grindability (Bond test) was done in Koza Metallurgy Lab to determine the work index. For this study 15 kg crushed samples was used and closing size is 106 µm. The result is shown in Table 2.2.

Table 2.2. Z vein Grindability

Sample name	Z Sample
Feed f80 (µm):	1471
Product p80 (µm):	81
Grindability (grams/rev):	0.9505
Ball Mill Work Index (kwh/t):	20.63
Test sieve (µm):	106

2.3. Reagents Used in Flotation Experiments

2.3.1. Collectors

The reagents used for these tests were KAX (potassium amyl xanthate), Aero 7249, Aero 407, Aero 6697, Maxgold as collectors. Very few gold flotation plants use single collectors and the preference is either to add a blended collector, that is, a mixture of a number of collectors, or collectors added separately in smaller quantities than those of the primary collector. These collector combinations result in better overall flotation recoveries. The term synergism is often applied to this phenomenon. The order of addition of the different collectors can also be important in improving flotation performance.

Properties of Aero 7249 : AERO 7249 promoter is a formulated product that is used extensively in many Cu-Au plants, where it provides optimum recovery of both Cu and Au by combining the advantages of dithiophosphates and monothiophosphates, and provides excellent selectivity against iron sulfides.

Properties of Aero 407 : A stronger collector than AERO 404 promoter. May substantially replace xanthates in many applications, while being more selective against iron sulfides in alkaline circuit. Useful for treating a wide range of precious and base-metal ores, particularly those of Cu, Ni and Zn. Excellent for bulk flotation of poly-metallic ores and pyritic gold ores in acid circuits.

Properties of Aero 6697: AERO 6697 promoter is a novel collector based on monothiophosphate chemistry. AERO 6697 promoter is in commercial use at a number of operating locations around the world.

Properties of MAXGOLD: Collector for bulk sulfide flotation and precious metal values including free gold, electrum, and tellurides. It can be used at low dosages (5g/t-10g/t) in flash float cells improving recovery of precious metals and gold-bearing iron sulfides (FeS_2, $Fe_{(1-x)}S$)

2.3.2. Frothers

Dow 250 was used as frother. The Dowfroth 250 is one of the most common used ether frothers, which can be made from the reaction of methanol and propylene oxide.

2.4. Bulk Flotation Testing Procedure

In the first stage, bulk flotation tests were performed to roughly understand the behavior of the ore. The mineral samples were splitted for tests and a head assay analysis was performed for one of the samples. Fractional distribution of Au % and S % was shown in Table 2.3.

Table 2.3. Fractional Distribution of Gold Grade in Feed

Feed Assay (μm)	Weight (g)	% Wt	Grade		Distribution (%)	
			Au g/t	S(%)	Au	S
+106	203.6	10.18	3,06	2,36	7,43	5,37
+75	241.3	12.06	4,32	4,19	12,43	11,30
+53	226.1	11.30	4,65	5,25	12,54	13,26
+38	243.1	12.16	5,13	5,79	14,88	15,73
-38	1085.9	54.30	4,07	4,48	52,72	54,35
Head Assay (calculated)	2000	100	4,19	4,48	100,00	100,00
HeadAssay (analysed)			4,26	4,27		

Three grinding tests were performed to derive the grinding calibration curve. For this reason, the ore was ground for 20 min, 30 min and 60 min in lab scale ball mill and p80 values of 154 μm, 65 μm, 35 μm were obtained respectively.

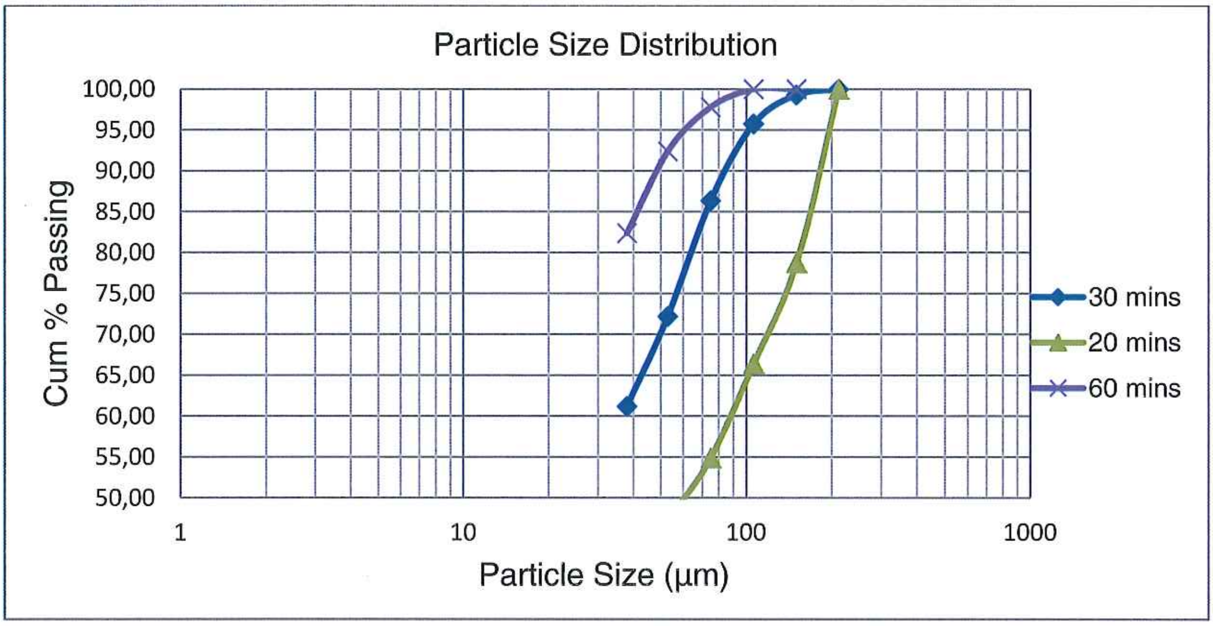


Figure 2.1. Particle Size Distribution of the Ore

According to grinding curve 65 µm (30 mins) was selected for flotation experiments. After determining the target size, the flotation test were started to find out the most effective collector for gold recovery. KAX, Maxgold, Aero 7249 (Cytec 7249), Aero 407 (Cytec 407) and Aero 6697 (Cytec 6697) was used.

2.5. Flotation Experiments

2.5.1. Determining the Effect of Collector Type

The flotation procedure to determine the most effective collector type is as follows;

- The flotation tests were performed at natural pH (changing between 7- 8)
- Collector was added in grinding stage and the ore was mixed with 1333 ml water.
- The ground ore (30 min; %60 solid in ball mill) was taken to the 4.5 lt flotation cell. The rotation speed was around 1800 rpm. The pulp density was adjusted to 30% solid.
- Collector was added again to the cell and started to conditioning for 2 minutes. DOW 250 was added as frother and conditioned 1 min.
- Air is turned on and concentrates were collected at 1.min, 5.min and 10. min as C1, C2, C3. For the last concentrate, C4, extra collector was added and flotation was continued for extra 10 minutes.

Table 2.4. Conditions of Flotation Experiments to Determine Effects of Collector Type

Test No	Reagents	Collector Dosage (g/t)	Frother; (g/t)
Z11	KAX+ MAXGOLD	KAX+ MAXGOLD; 125+100	DOW250; 10
Z12	KAX+ Aero 7249	KAX+ Aero 7249; 125+100	DOW250; 10
Z13	KAX+Aero 407	KAX+Aero 407; 125+100	DOW250; 10
Z14	KAX+ Aero 6697	KAX+ Aero 6697; 125+100	DOW250; 10
Z16	KAX	KAX; 100+100	DOW250; 10

During experiments half of the collectors were added in grinding stage and remained amount added in conditioning stage. Total flotation time was 20 minutes. KAX (%1 solution) was used as main collector.

Figure 2.2. Flotation Test Overview

Figure 2.3. Overview of Bubbles

The flotation tests were performed to determine the most effective collector on gold recovery. Other parameters (sulfidization, activation, pH, particle size, pulp density) effects were investigated separately at the best collector type. The concentrates and tails were filtered, dried and analyzed in aqua regia to determine the Au, Ag. For aqua regia analysis 50 grams sample is used to get accurate results. Cu, Fe, As analysis were performed with ICP and Leco instrument was used to determine sulfur content.

2.5.2. Determining The Effects of pH, NaHS and $CuSO_4$ on Gold Recovery

Effects of chemical conditions on gold recovery were investigated. First of all, pH was changed. Most minerals exhibit an optimum pH range for a given collector. While some minerals can often be floated at the natural pH of the ores, in most cases the pH has to be adjusted for maximum recovery and selectivity. The most commonly used reagents for acid circuit flotation, is sulfuric acid. The pH was adjusted by H_2SO_4. The procedure was the same except pH that was decreased to 6.0 by sulfuric acid.

Secondly, effect of sulfidisation and activation was investigated. The addition of sulfide ions converts some coatings on mineral surfaces in sulfides (Healy, 1984) and subsequent xanthate addition will promote flotation. NaHS was used as sulfidiser. Effect of two different dosage of the NaHS was investigated which is 500 g/t (10 ml;%10 by solution) and 2000 g/t (40ml;%10 by solution). NaHS was added in to the mill before collector in order to establish mineral surface conditions most suitable for effective collector addition.

Activators are chemicals that enhance the flotation performance of a specific mineral/minerals. Addition of Cu(II) ions, which activate minerals such as sphalerite ZnS, pyrite (FeS_2), pyrrhotite (FeS), stibnite ($Sb_2 S_3$), and arsenopyrite (FeAsS). $CuSO_4$ was used as activator. The dosage was 500 g/t (10 ml; %10 by solution).

In general activators are added before collectors in order to establish mineral surface conditions most suitable for effective collector addition. $CuSO_4$ was added in grinding stage during experiments. The chemical conditions of the experiments was summarized at Table 2.5.

Table 2.5. Conditions of Flotation Experiments to Determine The Effects of pH, NaHS, $CuSO_4$.

Test Number	Reagents	Collector Dosage (g/t)	Frother; (g/t)
Z15	KAX+ Aero 7249+ H_2SO_4	KAX+ Aero 7249; 125+100	DOW250; 10
Z17	KAX+ Aero 7249+ NaHS (500 g/t)	KAX+ Aero 7249; 125+100	DOW250; 10
Z18	KAX+ Aero 7249+ NaHS (2000 g/t)	KAX+ Aero 7249; 125+100	DOW250; 10
Z22	KAX+Aero 7249+ $CuSO_4$ (500 g/t)	KAX+ Aero 7249; 125+100	DOW250; 10

2.5.3. Determining the Effects of Particle Size and Pulp Density

The ore was ground 20 minutes (155 µm) and 60 minutes (35 µm). The conditions were same for both sizes. Standard flotation procedure was applied. The natural pH was 8.0. The standard conditions as summarized at Table 2.6. were also applied to determine the effect of pulp density. The natural pH was 8.1 After grinding the pulp was transferred into a 8 lt cell and the solid % was adjusted to 20. The speed was around 1800 rpm.

Table 2.6. Conditions of Flotation Experiments to Determine The Effects of Particle Size and Pulp Density

Test Number	Reagents	Collector Dosage (g/t)	Frother; (g/t)
Z20	KAX+ Aero 7249 20 mins	KAX+ Aero 7249; 125+100	DOW250; 10
Z21	KAX+ Aero 7249 60 mins	KAX+ Aero 7249; 125+100	DOW250; 10
Z23	KAX+ Aero 7249 %20 solid	KAX+ Aero 7249; 125+100	DOW250; 10

2.5.4. Experiments on Stocked Ore

The ore which is stocked in an open atmospheric conditions for months was used to investigate effect of surface oxidation. The experiments were performed in the presence of NaHS (500 g/t) and absence of NaHS. 10 ml of NaHS (%10 by solution) was added to the mill before grinding. The natural pH was 7.6. The same procedure was applied for the test without NaHS. The only difference was the collector addition, which was in grinding stage (standard conditions).

2.6. Flotation Tests with Cleaning and Scavenging Stages

Flotation details was summarized as follows and flowsheet was shown in Figure 2.4 and Figure 2.5.

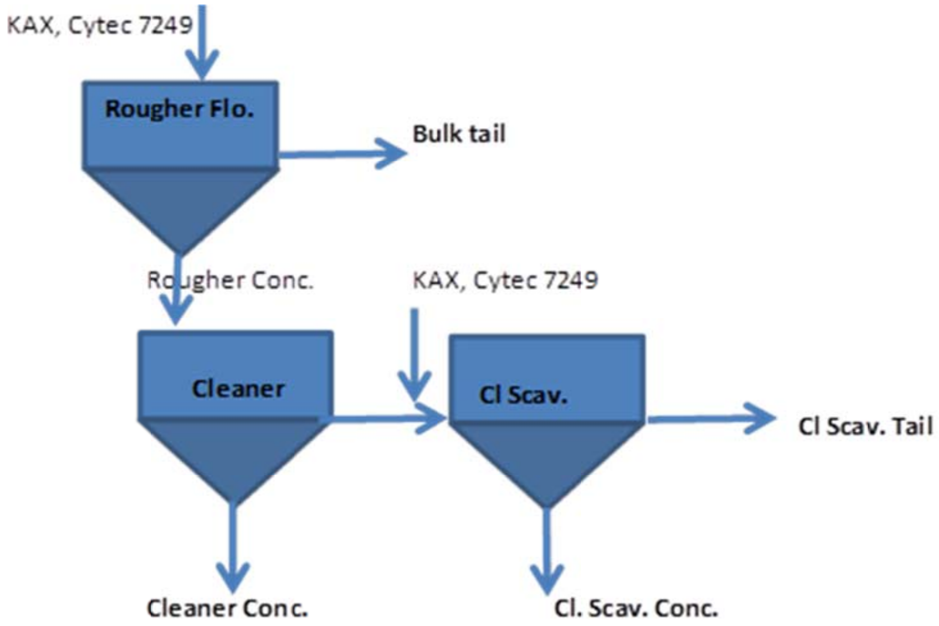


Figure 2.4. Flowsheet of Cleaner Test 1

- Ore was ground 30 mins and was transferred into 4.5 lt flotation cell. Natural pH was 9.15. 25 g/t KAX (5ml; %1 solution) and 25 g/t Aero 7249 were conditioned for 2 mins. 10 g/t (20 µl) Dow 250 was added and conditioned for 1 min. Then rougher flotation was performed for 5 mins (C1).

- Extra 25 g/t KAX (5 ml) and 25 g/t Aero 7249 (50 µl) were added and conditioned for 2 mins and flotation was performed for 5 mins (C2). Same procedure above was performed for concentrate C3. Rougher flotation concentrates, C1+C2+C3 were combined and cleaned for 5 minutes.
- After, cleaner concentrate C1 was taken, 25 g/t Aero 7249 and 7.5 g/t KAX (1.5 ml) were added and conditioned for 2 mins and cleaner scavenger flotation was performed for 2 mins.

In the second test, collector addition point at the circuit was changed and the flowsheet below was performed. Ore was ground for 30 mins and transferred into 4.5 lt flotation cell. Natural pH was 9.20. 25 g/t KAX (5ml; %1 solution) and 25 g/t (50µl) Aero 7249 were conditioned for 2 mins. 10 g/t Dow 250 was added and conditioned for 1 min. Then, rougher flotation was conducted for 5 minutes to get C1. Rougher concentrate C1 was cleaned 5 minutes in a 1L cell at 1300 rpm without collector. Rougher scavenging was performed by the addition of 25 g/t KAX and 25 g/t Aero 7249 for 5 mins and for final stage rougher scavenging concentrate and cleaner tail was combined and cleaned without collector.

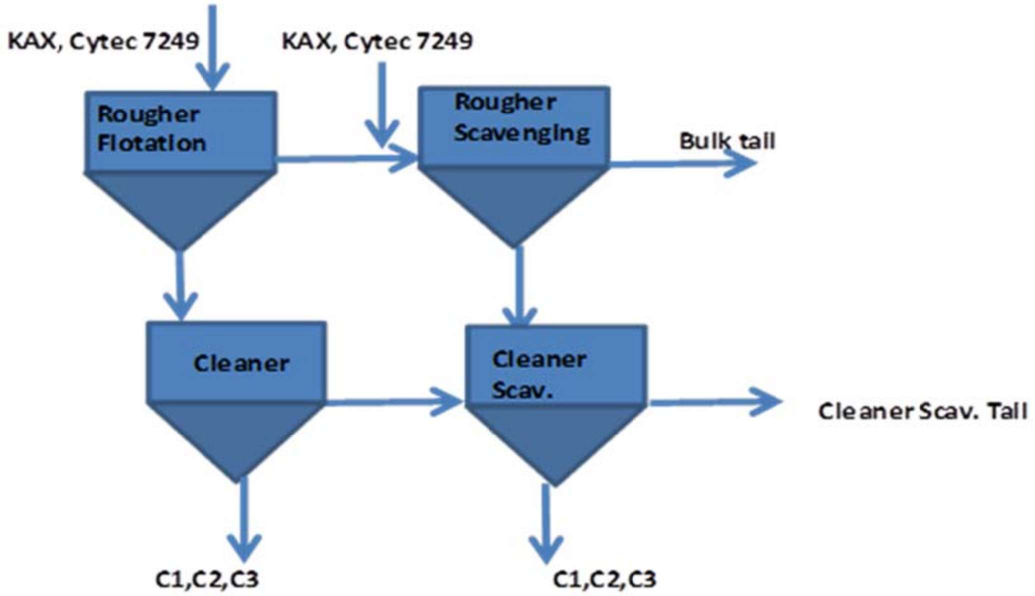


Figure 2.5. Flowsheet of Cleaner Test 2

The effect of conditions and chemicals on recovery was determined by bulk flotation tests. Upgrading of the concentrate was investigated by the cleaner tests. The aim of these tests was to produce a concentrate with high gold content and recovery for the following (leaching) circuit. In this case, the cyanidation tests were done to investigate leachability of gold, effects of gold mineralogy. The leaching tests were performed in Koza gold metallurgy laboratories.

2.7. Cyanidation Tests (Bottle Roll Tests)

The aim of bottle roll tests is to determine the recovery of valuable metals by direct intensive cyanide leaching such as gold, silver or copper. Cyanide leaching test can be done directly to the ore, pre-concentrate or middlings as well. To perform bottle roll tests, the following installation, machine was used (Figure 2.6.).

Figure 2.6. Bottle Roll Test Machine

Equipments used for test: 2.5 lt bottles, funnel, bottle roll test machine, beakers, $AgNO_3$ solution for free cyanide determination, burettes, oxygen meter, pH meter and pipettes.

Reagents used for test were NaCN (sodium cyanide) and CaO (lime), rhodanine (is an indicator to determine free cyanide). Bottle roll test was explained step by step as follows in Table 2.7.

1. Grind the ore to proposed size with rod mill.
2. Number the bottles and note on the experiment table.
3. Weigh the bottles and note on the experiment table.
4. Pour the 450 gram weight of grind sample to the bottles with placing the cone on it.
5. Pour 550 gram weight of water to the bottle (to adjust the solid ratio to %45)
6. Shake the bottle kindly in order to mix solid and water in the bottle properly.
7. Submerge the oxygen probe which the distance is 2 cm from the tap of the bottle in order to measure the dissolved oxygen. Shake the bottle with circular motion. Note the value from the oxygen meter on the experiment table as the value is steady state. Drain the slurry of probe with rinsing with a small amount of distilled water to the bottle.
8. Measure the first pH value of the slurry with the procedure written in the 7th step and note this value on the experiment table. Increase the pH value to proposed pH value with adding lime. Note the final pH value and amount of added lime. Shake the bottle about one minute with circular motion in every addition of lime to the bottle and then note the pH value.
9. Add the proposed amount of NaCN with weighing and note this value

Proposed NaCN, ppm * 0,55 = amount of NaCN to be weighed, mg NaCN

10. Shake the bottles in a circular motion for the last time and place them carefully on the roller. The rotational speed of the bottle must be 30 rev/min. Open the rotameter to 15 – 40 L/h level of oxygen, then hang down the thin tubes into the bottle. Confirm by observation that the oxygen form foam/boils inside the pulp.

11. Turn the bottle roll on.
12. Put the bottles down after 1-2 hours. Measure the oxygen; note them down. Measure the pH; note them down and increase the pH to the required level with lime addition. Weigh the bottles; note them down and wait around 45 minutes for the pulp to be settled down. After the pulp settles down enough, draw 30 ml of solution from the uppersurface of the pulp using pipet. (12)
13. Use 10 ml of the drawn solution for the silver nitrate titration of NaCN analyses. (look at free cyanide analyses procedure). Keep the remaining of the solution for Au-Ag analyses. Add used amount of NaCN to the solution.

a- If the solution is less than 550 g, the amount of NaCN that should be added;

NaCN, mg = (required NaCN, ppm) x 550 g – measured NaCN, ppm) x solution, g

After NaCN addition, fill the solution up to 550 g.

b- If the solution is more than 550 g the amount of NaCN that should be added;

NaCN, mg = (required NaCN, ppm) x solution, g – (measured NaCN, ppm) x solution, g

14. Weigh NaCN on a dry basis; note it down. Shake the bottles gently in a circular motion and then place them on the roller again.
15. Repeat the 12. - 14. steps at 2., 6., 24., 30. hours.
16. After the completion of leach, look at the pH and oxygen levels of 48. hour and note them down; weigh the bottles. Draw the solution.
17. Separate solid and solution using press filter (Filtration procedure). Perform solid and solution analysis separately.

Table 2.7. Conditions of Bottle Roll test for Flotation Concentrate and Tail

	Concentrate	Tail
Feed Au ppm.	23.07	0.92
Feed Ag ppm.	9.17	0.82
Initial NaCN, ppm.	1592	959
Added NaCN, g.	0.9	0.5
Leach time, hours	48	48
Initial pH	6.0	7.2

3. RESULTS AND DISCUSSION

3.1. Effects of Collector Type

The collector type is very important in flotation. There are many of collectors used in gold flotation. Cytec special gold collectors and xanthate were used to find out the most effective collector in terms of gold grade and recovery.

Table 3.1. Effect of KAX on gold and sulphur recovery

Product	Z16 (KAX)					
	Mass Pull		Grade		Cum. Recovery %	
	gr	%	Au g/t	S %	Au	S
C1	32.6	1.73	37.51	40.51	9.59	15.47
C1+C2	134.2	7.12	38.36	27.08	40.37	42.57
C1+C2+C3	234.9	12.46	31.85	22.11	58.67	60.83
C1+C2+C3+C4	342.5	18.17	26.60	15.55	71.44	62.41
Tail	1542.8	81.83	2.36	2.08	100	100
Feed	1885.3	100.00	6.76	4.53		

The assays were done at least with 50 g of samples. By the help of assay results the overall recovery of Au, Cu, Fe, As, Ag, S% were determined. In this section gold and sulphur results are given. The tests conditions and the full results are given in Appendixes.

According to the results only KAX was not enough to recover gold with sulphide. The gold and sulphur recoveries were around % 71 and % 62 respectively as shown on Table 3.1. For this reason Cytec reagents were used with KAX.

The selection of the collectors was done based on collector chemistry and their usage in flotation experiments. Reagents and effects on recovery were summarized on Table 3.2;3.3;3.4;3.5 and on Figure 3.1.

Table 3.2. Effect of KAX+ MAXGOLD on Gold and Sulphur Recovery

Product	Z11 (KAX+ MAXGOLD)					
	Mass Pull		Grade		Cum. Recovery %	
	gr	%	Au g/t	S %	Au	S
C1	36.6	1.96	43.17	32.73	11.45	11.22
C1+C2	143.3	7.66	47.47	25.50	49.29	34.24
C1+C2+C3	204.1	10.91	39.42	23.01	58.30	44.00
C1+C2+C3+C4	340.4	18.19	28.98	23.88	71.49	76.19
Tail	1531	81.81	2.57	1.66		
Feed	1871.4	100.00	7.37	5.70		

Table 3.3. Effect of KAX+ Aero 7249 on Gold and Sulphur Recovery

Product	Z12 (KAX+ Aero 7249)					
	Mass Pull		Grade		Cum. Recovery %	
	gr	%	Au g/t	S %	Au	S
C1	144	7.68	30.76	18.49	32.50	23.15
C1+C2	272	14.51	31.81	19.93	63.49	47.13
C1+C2+C3	337.5	18.00	27.74	18.48	68.68	54.23
C1+C2+C3+C4	499.1	26.63	22.49	19.33	82.34	83.86
Tail	1375.4	73.37	1.75	1.35		
Feed	1874.5	100.00	7.27	6.14		

Table 3.4. Effect of KAX+ Aero 407 on Gold and Sulphur Recovery

Product	Z13 (KAX+ Aero 407)					
	Mass Pull		Grade		Cum. Recovery %	
	gr	%	Au g/t	S %	Au	S
C1	176.3	9.29	31.36	20.55	40.13	29.12
C1+C2	250.6	13.20	33.47	21.21	60.88	42.73
C1+C2+C3	283.8	14.95	31.08	20.34	64.02	46.40
C1+C2+C3+C4	355.6	18.73	27.16	24.70	70.11	70.61
Tail	1542.7	81.27	2.67	2.37		
Feed	1898.3	100.00	7.26	6.55		

Table 3.5. Effect of KAX+ Aero 6697 on Gold and Sulphur Recovery

Product	Z14 (KAX+ Aero 6697)					
	Mass Pull		Grade		Cum. Recovery %	
	gr	%	Au g/t	S %	Au	S
C1	130.8	7.11	25.71	16.45	26.68	20.59
C1+C2	207.8	11.30	31.73	17.51	52.31	34.82
C1+C2+C3	268.6	14.61	27.71	15.72	59.05	40.40
C1+C2+C3+C4	347.2	18.88	23.37	15.41	64.38	51.19
Tail	1491.4	81.12	3.01	3.42		
Feed	1838.6	100.00	6.86	5.68		

The most effective collector type was determined as KAX+ Aero 7249. The gold and sulphur recoveries were % 82 and % 84 respectively. Aero 7249 is a mixture of monothiophosphate and dithiophosphate and a strong Cu and other valuable metals collector. It can be used for free native gold flotation. Figure 3.1. shows that the maximum Au and S recoveries were obtained with KAX + Aero 7249 mixture and the lowest recoveries were obtained with KAX+ Aero 6697.

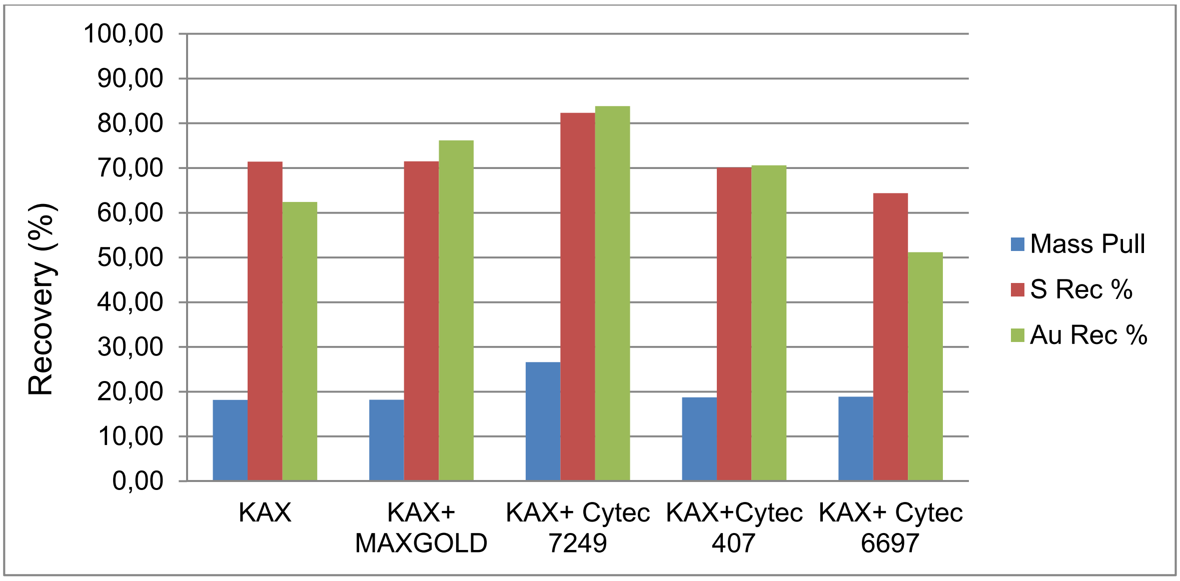


Figure 3.1. Mass Pull vs Au& S Recovery

According to the Figure 3.1. Z12 condition produces high gold recovery as a result of high mass pull. S recovery is higher than Au recovery with KAX only and KAX+

Aero 6697. However, with the other collectors Au recovery was equal or higher than S recovery. This means that Aero 7249, 407 and MAXGOLD are specific gold collectors.

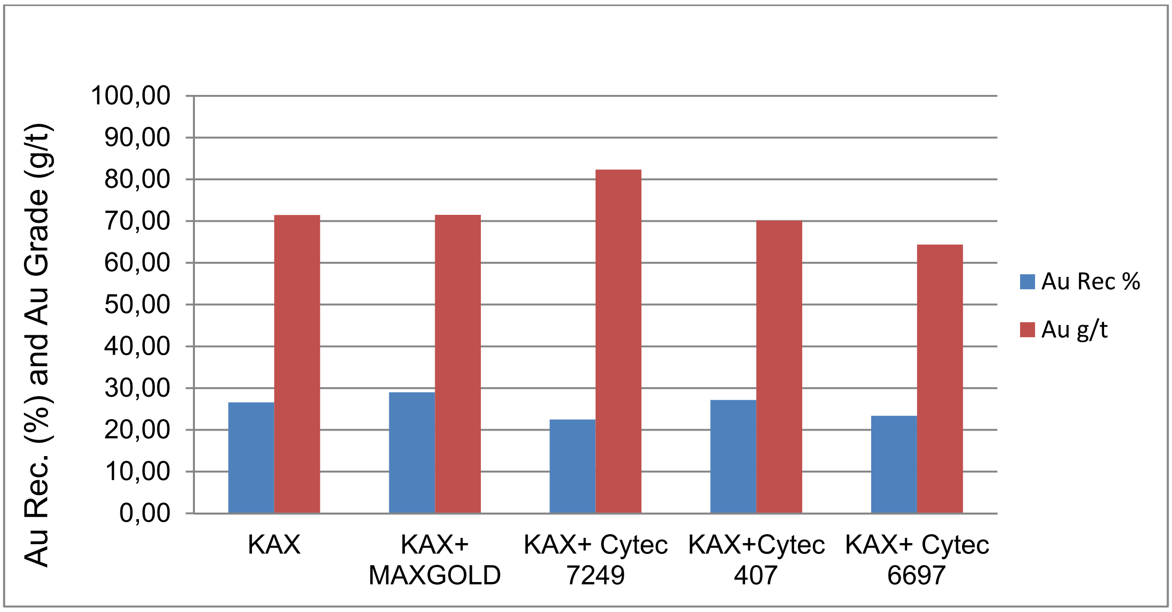


Figure 3.2. Au Recovery vs Au Grade

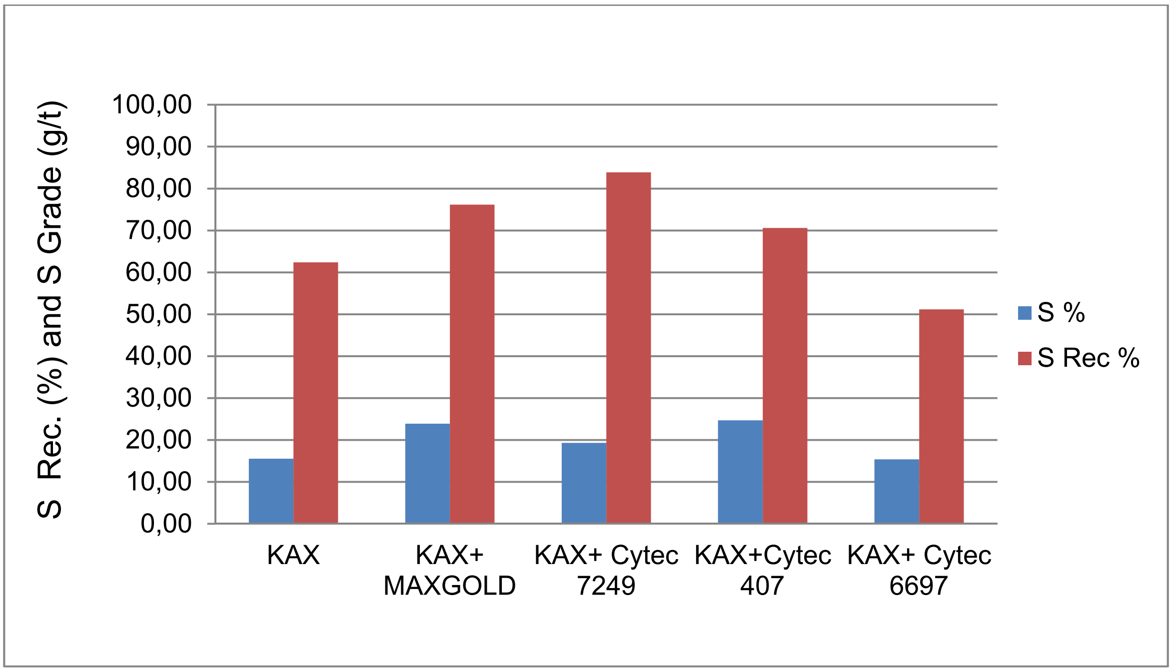


Figure 3.3. Sulphur Recovery vs Sulphur grade

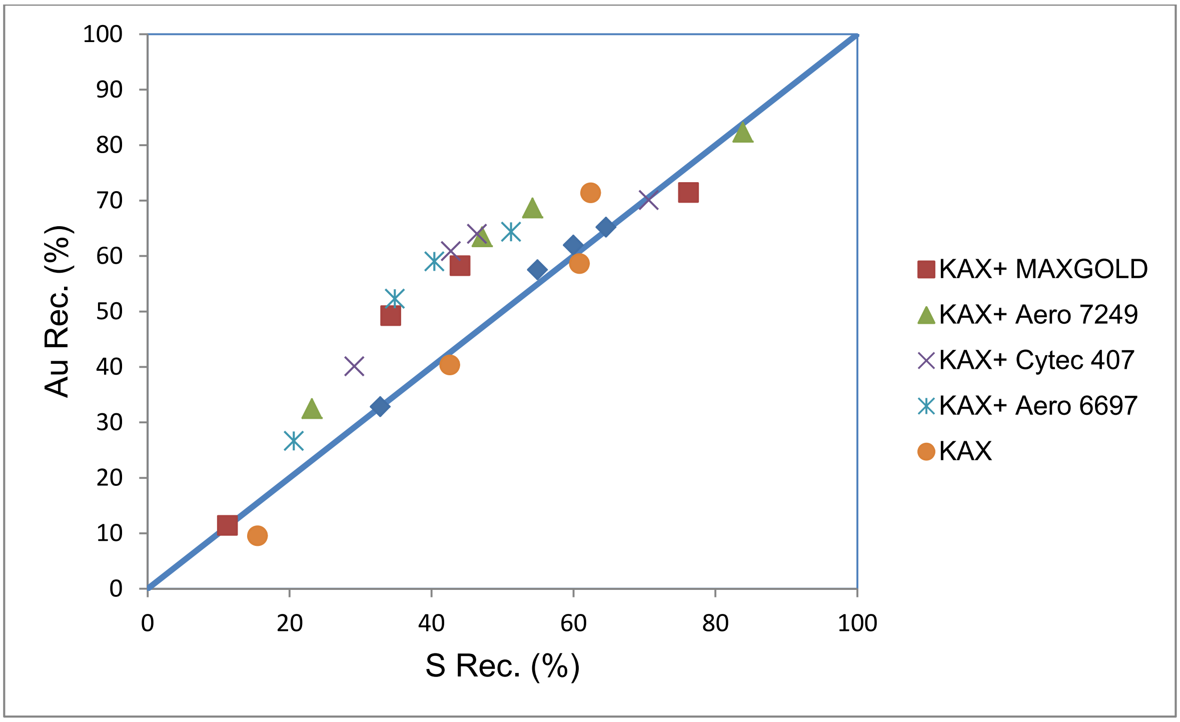


Figure 3.4. Effects of Collector Type on Gold and Sulphur Recovery

According to Figure 3.4. KAX+ Aero 7249 , KAX+ MAXGOLD and KAX+ Aero 407 collector mixtures are more selective than the others on gold recovery. Gold is reported not to float as readily or as rapidly as most of the mineral sulfides and thus, it is inherently a slow kinetic process (Leaver and Woolf, 1934a; Klimpel, 1997). Conditions that may further aggravate this are an excessive amount of collector; high clay slimes content (Taggart, 1945; Botelho de Sousa et al., 1986), surface coatings, and cold process-water conditions (More and Pawson, 1978; O'Connor et al., 1988).

Large free gold particles float more slowly than finer particles. Collector addition in stages has been shown to accelerate free gold flotation kinetics and more so for coarse flaky particles (Chryssoulis et al., 2003). Another way to achieve a similar outcome is to select a collector depending on the surface composition of the gold. The impact of higher collector addition is more obvious for the harder to float coated (tarnished) gold particles (Chryssoulis and Dimov, 2004). In terms of flotation kinetics, the results were compared as shown in Figure 3.5.

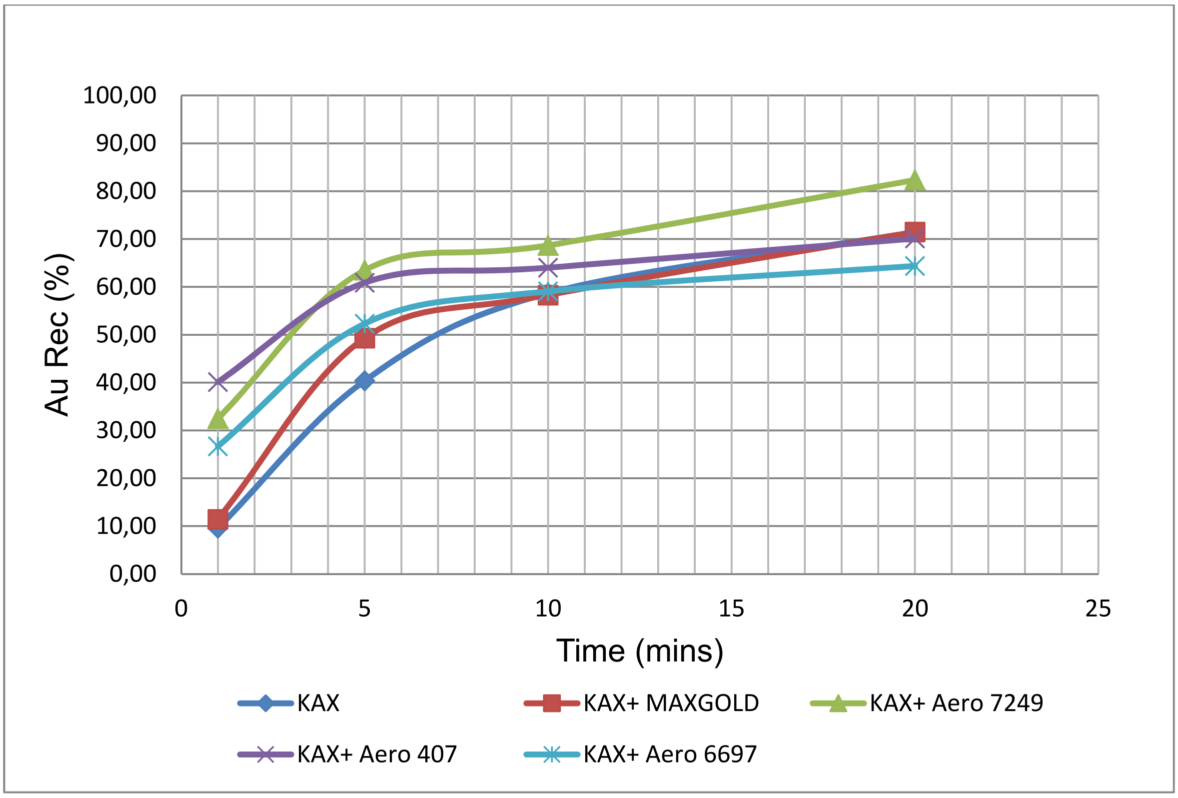


Figure 3.5. Effect of collector type on flotation kinetics of gold flotation

Flotation time is important due to the effect of collector type on flotation kinetics. Recovery can be increased as the time passes. The results in Figure 3.5 shows that Aero 7249 increases the flotation kinetics. At the beginning of the flotation KAX and MAXGOLD shows slower flotation. The type of the collector affects the kinetics and selectivity of the flotation. After 20 minutes, Aero 407 and MAXGOLD reaches the same Au and S recovery. Aero 7249 is the most selective collector type on gold recovery. Mass pull increased the recovery at Z12. The silver recovery was also related with gold recovery. The gold could be associated as electrum and native gold.

3.2. The Effect of pH on Gold Recovery

After determining the optimum collector mixture, effect of pH was tested by decreasing the pH to 6.0 with H_2SO_4. The gold recovery did not change so much as compared to flotation result at standard condition while sulphur recovery decreased obviously as seen in Table 3.6 and Figure 3.6. The gold grade was around 29 ppm.

Clay minerals are very floatable in the pH range 5–9 and if these are present in the ore, then pH values outside this range are chosen for flotation. Clay minerals were not observed in the ore during core logging by geology department , so that the flotation has been done at natural pH.

The adverse effect of high pH on the flotation of metallic gold when lime is added has been discussed widely in the literature (Leaver and Woolf, 1932; Taggart, 1945). The recent application of surface analytical techniques has provided compelling evidence of the loss of free gold due to depression induced by surface-bound calcium and hydroxyl ions (Chryssoulis, 2001).

Table 3.6. Flotation results at pH 6

Product	Z15 (KAX+ Aero 7249)					
	Mass Pull		Grade		Cum. Recovery (%)	
	gr	%	Au g/t	S %	Au	S
C1	94.3	5.05	42.81	33.89	31.02	24.60
C1+C2	159.4	8.54	52.89	32.63	64.79	40.05
C1+C2+C3	181.2	9.71	50.14	31.14	69.82	43.44
C1+C2+C3+C4	269	14.42	38.11	28.82	78.77	59.68
Tail	1596.8	85.58	1.73	3.28		
Feed	1865.8	100.00	6.97	6.96		

Sulphuric acid assists the recovery of iron sulfides. In this case, the results indirectly shows that sulphidic gold carrier can not be only the iron sulphides, because of the fact that sulphur recovery is low. Pyrite and arsenopyrite float well in the pH range 3–10 while pyrrhotite floats best in acid circuits (Allison and Dunne, 1985).

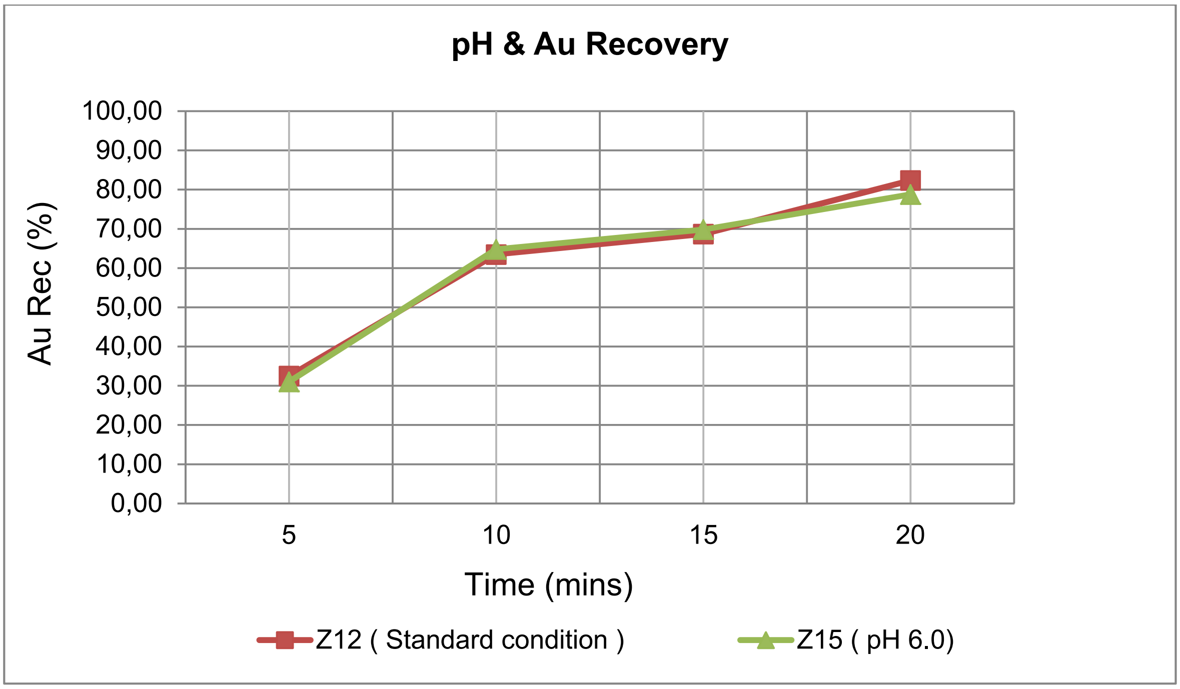


Figure 3.6. Effect of pH on Gold Recovery

3.3. Effect of NaHS on Gold Recovery

In these experiments with NaHS total collector dosage was 225 g/t , overall flotation time was 20 mins and the frother dosage was 10 g/t. For the experiments with NaHS the results are shown in Table 3.7 and Table 3.8.

Table 3.7. Addition of NaHS (500 g/t) on Gold Recovery

Product	Z17 (KAX+ Aero 7249)					
	Mass Pull		Grade		Cum. Recovery %	
	gr	%	Au g/t	S %	Au	S
C1	41.8	2.22	81.98	25.98	26.91	8.52
C1+C2	115.9	6.15	52.98	25.88	48.22	23.54
C1+C2+C3	180.2	9.56	41.08	25.31	58.13	35.79
C1+C2+C3+C4	323.5	17.16	28.41	23.47	72.17	59.57
Tail	1561.3	82.84	2.27	3.30		
Feed	1884.8	100.00	6.76	6.76		

Table 3.8. Addition of NaHS (2000 g/t) on Gold Recovery

Product	Z18 (KAX+ Aero 7249)					
	Mass Pull		Grade		Cum. Recovery %	
	gr	%	Au g/t	S %	Au	S
C1	63.1	3.38	63.89	28.22	34.63	14.76
C1+C2	156.8	8.39	41.65	26.58	56.10	34.54
C1+C2+C3	231.2	12.38	32.62	24.32	64.79	46.60
C1+C2+C3+C4	378.3	20.25	23.49	21.85	76.32	68.51
Tail	1489.9	79.75	1.85	2.55		
Feed	1868.2	100.00	6.23	6.46		

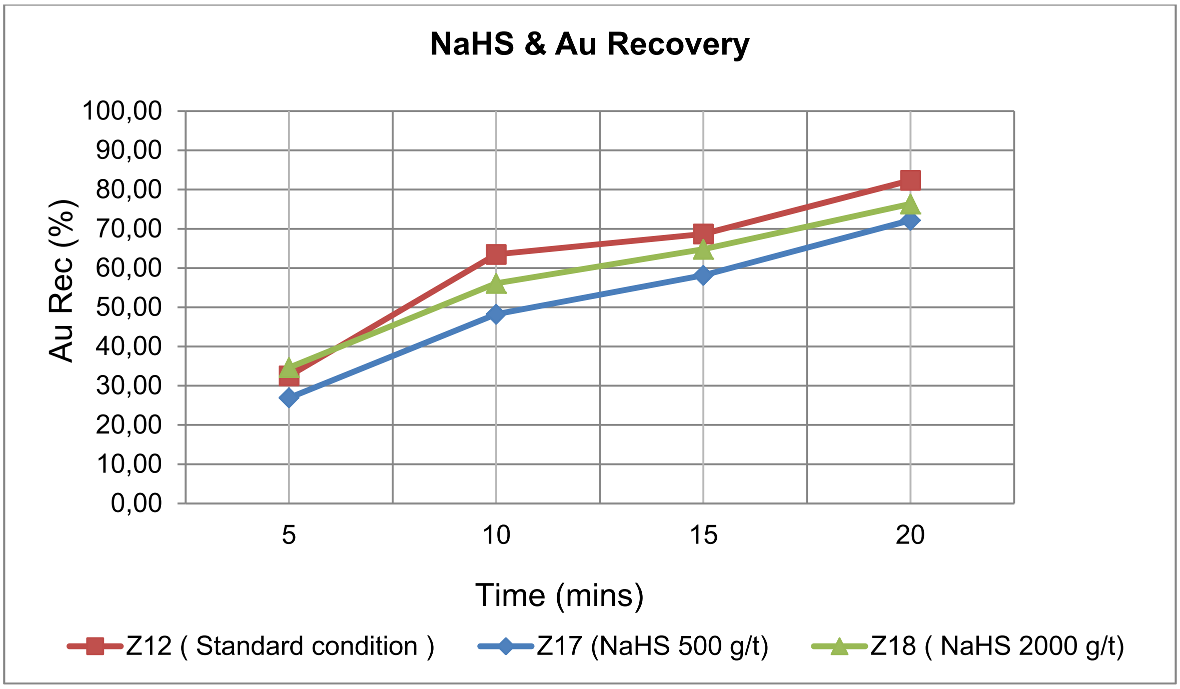


Figure 3.7. Effect of NaHS addition on Au Recovery

According to the Figure 3.7. increasing the NaHS dosage increases the gold and sulphur recovery. The reason for this situation can be that the mass pull increases the gold and sulphur recovery. The mineralogy of the gold particles are important in the addition of NaHS. If all the gold particles are associated with sulphur, NaHS addition promote the gold and sulphur recovery. However in this case, when compared to standard condition (Z12), NaHS does not increase recovery. This could be due to the free gold particles and also mass pull. In standard conditions mass pull was higher than Z17 and Z18 conditions. High mass pull decreases the gold grade in condition Z18.

3.4. Effect of $CuSO_4$ on Gold Recovery

It is widely accepted that the main purpose of copper sulfate in the flotation of sulfide gold carriers is to enhance the flotation of the sulfides and, in particular, pyrrhotite arsenopyrite and pyrite. The activation of the mineral surface by adsorption of copper ions to allow enhanced adsorption of collector has been thought as one mechanism that provides the improved flotation performance. The redox potential of the pulp will also increase with the addition of copper sulfate, thereby increasing the oxidizing environment for thiol collectors, thus favouring improved flotation performance (Nicol, 1984).

Table 3.9. Effect of $CuSO_4$ (500 g/t) on gold recovery

Product	Z22 (KAX+ Aero 7249)					
	Mass Pull		Grade		Cum. Recovery %	
	gr	%	Au g/t	S %	Au	S
C1	58.4	3.07	60.83	32.99	27.55	15.01
C1+C2	133.3	7.00	53.67	30.20	55.49	31.38
C1+C2+C3	188.4	9.90	43.34	27.93	63.33	41.01
C1+C2+C3+C4	295.3	15.51	32.39	28.91	74.18	66.53
Tail	1608.5	84.49	2.07	2.67		
Feed	1903.8	100.00	6.77	6.74		

In some situations, copper sulfate is classified as a froth modifier (O’Connor and Dunne, 1991). As an example, when floating pyrite, it was found that when copper sulfate was not added or added prior to or after the collector, then the froth contained a high proportion of slime and lower grades of concentrates were produced (Allison and Dunne, 1985). In recent laboratory testwork on a pyritic gold-ore, the effect of copper sulfate addition was to increase the concentrate mass and water recoveries (Bradshaw, 1997).

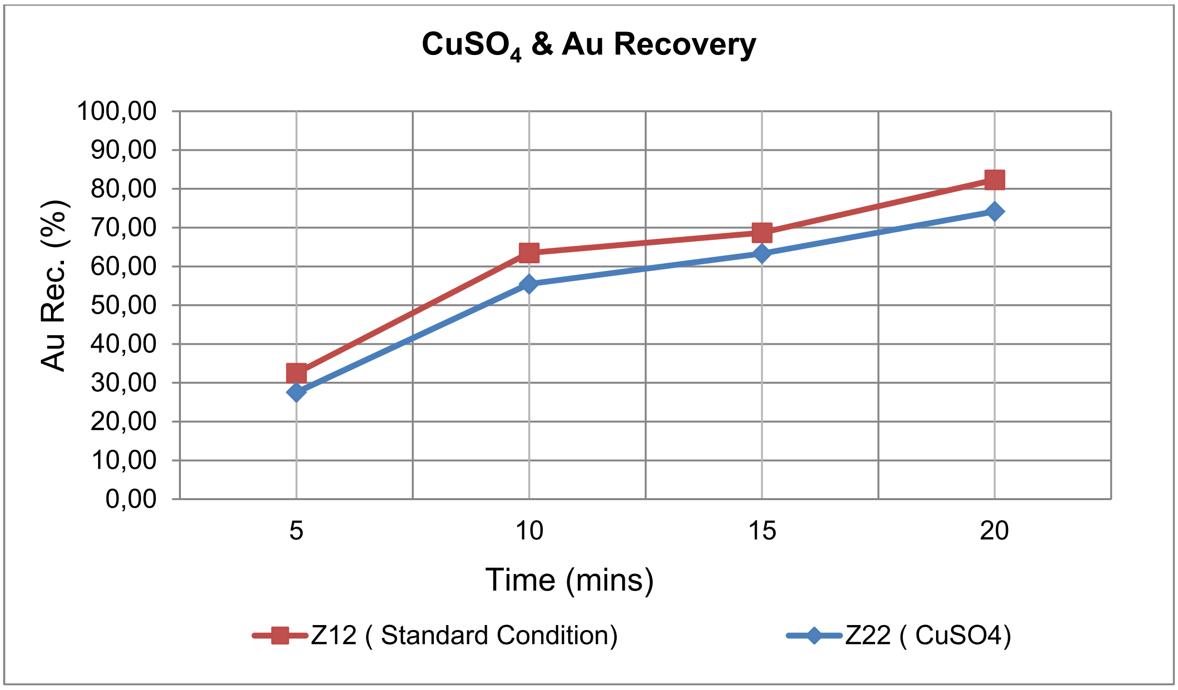


Figure 3.8. Effect of $CuSO_4$ on Au Recovery

According to the results shown in Figure 3.8. Addition of $CuSO_4$ did not increase the recovery. This could be due to the low mass and water recovery or the dosage of the copper sulphate could be high in the experiment.

Too high an addition of copper sulfate is known to have negative effect on flotation recovery (O'Connor et al., 1990; Bulatovic, 1997; Yan and Hariyasa, 1997; Monte et al., 2002). Copper is known to be an oxidant for xanthate, so when it is added to the pulp, it will oxidize greater amounts of xanthate to dixanthogen in solution, and this could be the reason for the sulfide and gold depression instead of being activated (Finkelstein and Poling, 1997; Teague et al., 2000).

It is claimed that copper activation is incompatible with simultaneous high free-gold and sulfide flotation recoveries. Addition of $CuSO_4$ increased the Au and S grade by decreasing mass pull. $CuSO_4$ addition did not increase the rate of flotation.

3.5. Effect of Particle Size on Gold Recovery

The standard test were done after 30 mins ground. To determine the effect of grind size on gold recovery the ore ground 20 mins and 60 mins. The ore grind size did not significantly affect gold recovery. The results has been shown in Table 3.10.

Table 3.10. Effect of Particle Size on Gold Recovery

Product	Z20 (KAX+ Aero 7249) 155 µm (p80)					
	Mass Pull		Grade		Cum. Recovery %	
	gr	%	Au g/t	S %	Au	S
C1	63.2	3.80	58.75	33.74	29.64	15.55
C1+C2	120.5	7.25	56.10	31.38	53.97	27.58
C1+C2+C3	158.1	9.51	48.53	29.36	61.25	33.85
C1+C2+C3+C4	244.9	14.73	36.90	31.15	72.15	55.63
Tail	1418.1	85.27	2.46	4.29		
Feed	1663	100.00	7.53	8.24		
Product	Z21 (KAX+ Aero 7249) 35 µm (p80)					
	Mass Pull		Grade		Cum. Recovery %	
	gr	%	Au g/t	S %	Au	S
C1	71.4	4.15	17.90	9.71	10.63	5.61
C1+C2	146.1	8.50	32.55	11.72	39.55	13.87
C1+C2+C3	230.7	13.42	30.73	12.10	58.95	22.60
C1+C2+C3+C4	379.3	22.06	24.14	19.36	76.15	59.43
Tail	1340	77.94	2.14	3.74		
Feed	1719.3	100.00	6.99	7.19		

According to the Figure 3.9., the rate of flotation is very low at fine sizes. However, as the flotation time rises the recovery and mass pull increases. At the beginning of the flotation, fine gold particles are not carried to the froth phase. This can be due to the low water recovery. At the coarser size (155 µm), free gold particles could be transferred to the concentrate rapidly and C1 concentrate Au recovery was the same as result at 65 µm which was around 28%. When the Z20 and Z21 condition are compared with Z12 condition, the reason of the lower gold recovery at Z20 and Z21 is the mass pull. Coarse particles floats faster than fine particles at the beginning but overall result shows that the optimum grinding size both Au and S is 65 micron.

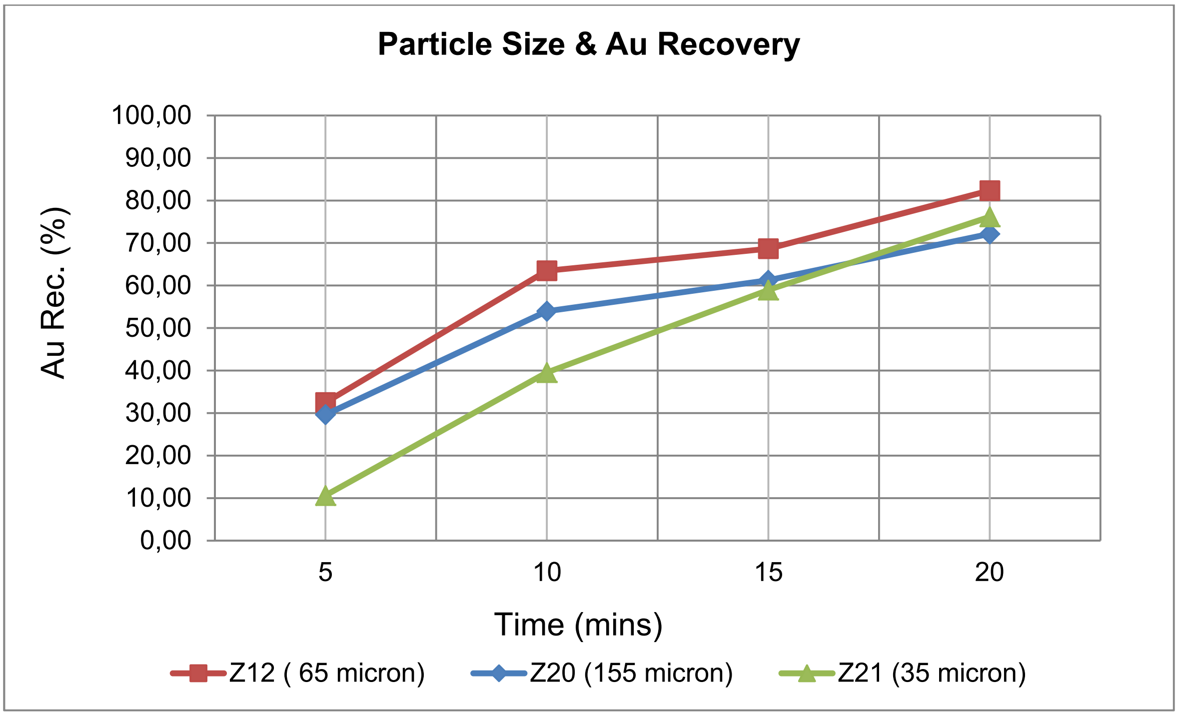


Figure 3.9. Effect of Particle Size on Au Recovery

3.6. Effect of Pulp Density on Gold Recovery

Pulp density and air flowrate influence flotation-cell pulp hydrodynamics and are important parameters in extending the particle-size limits of gold flotation. There is conflicting commentary on the best pulp density for gold particle flotation, both a high pulp-density (Leaver and Woolf, 1934a) and a low pulp-density being recommended. Flotation pulp density was around 30 % solid in standard tests. To see the effect of pulp density, flotation was performed at 20% solid by using 8 lt flotation cell. The test results with % 20 pulp density is given in Table 3.11 and comparison of the Au recovery with % 20 and % 30 w/w solid content is given in Figure 3.10. According to the results the more lower the density, less the gold and sulphur recovery. Little amount of frother was also added to increase bubbles during experiments, because of the fact that there was not seen a stable froth. Mass pull was very low.

Table 3.11. The Effect of Low Pulp Density on Gold Recovery

Product	Z23 (KAX+ Aero 7249)					
	Mass Pull		Grade		Cum. Recovery %	
	gr	%	Au g/t	S %	Au	S
C1	38.4	1.86	94.57	50.62	27.93	15.06
C1+C2	76.2	3.69	74.61	38.69	43.73	22.85
C1+C2+C3	101.9	4.93	62.27	33.82	48.80	26.70
C1+C2+C3+C4	178.4	8.63	42.91	33.70	58.88	46.58
Tail	1889	91.37	2.83	3.65		
Feed	2067.4	100.00	6.29	6.24		

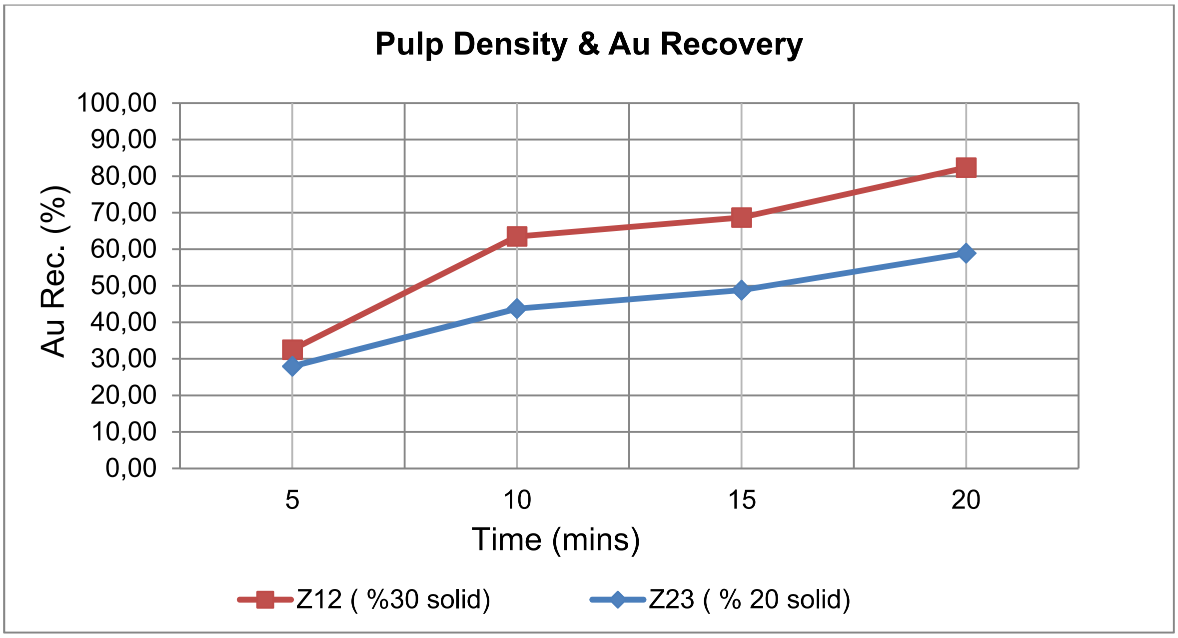


Figure 3.10. Effect of Pulp Density on Au Recovery

In terms of flotation kinetics at low pulp density gold and sulphur particles are not carried to the froth phase. The rate of flotation was very low in Z23 condition because of low density. Both mass pull and recovery were also low while Au and S grades high. During flotation the water recovery was very high and a stable froth was not observed. This situation can cause high gold and sulphur grade which was 42 g/t Au and 33.70 % S. Gold particles floats more than sulphur particles.

3.7. Results of Experiments on Stocked Ore

The oxidized ore was floated with and without NaHS to investigate if there was a negative effect on oxidation of the sulphur minerals in the ore. The results given in Table 3.12. and 3.13. show that, the ore is not oxidized fast in the atmospheric conditions. The final recovery and grades of gold, sulphur and mass pull on oxidized ore and the standard ore were approximately the same. However, surface oxidation decreases the flotation rate considerably in Figure 3.11.

Table 3.12. NaHS addition on oxidized ore

Product	Z25 (KAX+ Aero 7249)					
	Mass Pull		Grade		Cum. Recovery %	
	gr	%	Au g/t	S %	Au	S
C1	106.5	5.31	62.13	51.96	40.89	40.93
C1+C2	226.3	11.27	46.42	39.27	64.92	65.73
C1+C2+C3	282.1	14.05	40.77	34.79	71.08	72.58
C1+C2+C3+C4	349.4	17.41	35.63	31.20	76.94	80.62
Tail	1658	82.59	2.25	1.58		
Feed	2007.4	100.00	8.06	6.74		

Table 3.13. Absence of NaHS on oxidized ore

Product	Z26 (KAX+ Aero 7249)					
	Mass Pull		Grade		Cum. Recovery %	
	gr	%	Au g/t	S %	Au	S
C1	49.5	2.32	76.24	20.79	21.79	7.64
C1+C2	128	6.00	66.22	23.56	48.94	22.40
C1+C2+C3	178.6	8.37	52.65	22.27	54.29	29.54
C1+C2+C3+C4	375.6	17.61	34.83	28.55	75.54	79.64
Tail	1757.8	82.39	2.41	1.56		
Feed	2133.4	100.00	8.12	6.31		

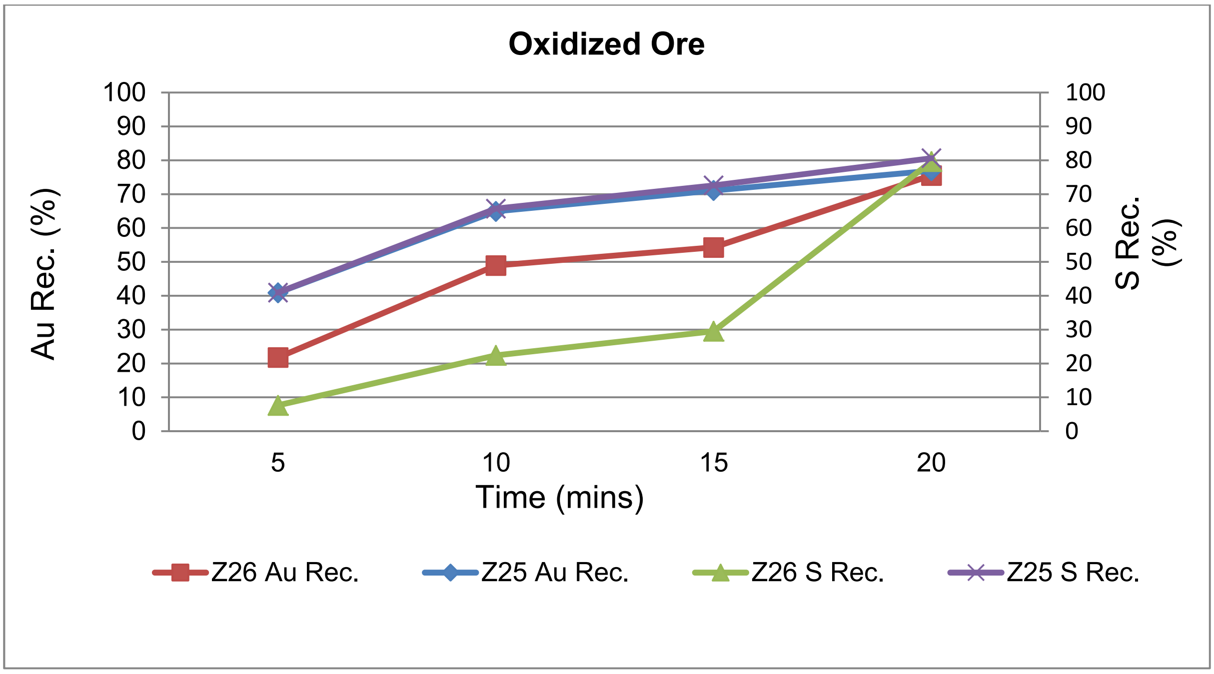


Figure 3.11. Oxidized ore vs Standard Ore

3.8. Cleaning and Scavenging Test Results

To produce a pre-concentrate with maximum recovery and grade, open cleaning tests were performed. The Au recovery and grade were around 80-84% and 10-12 g/t respectively. The results were shown on Figure 3.12.

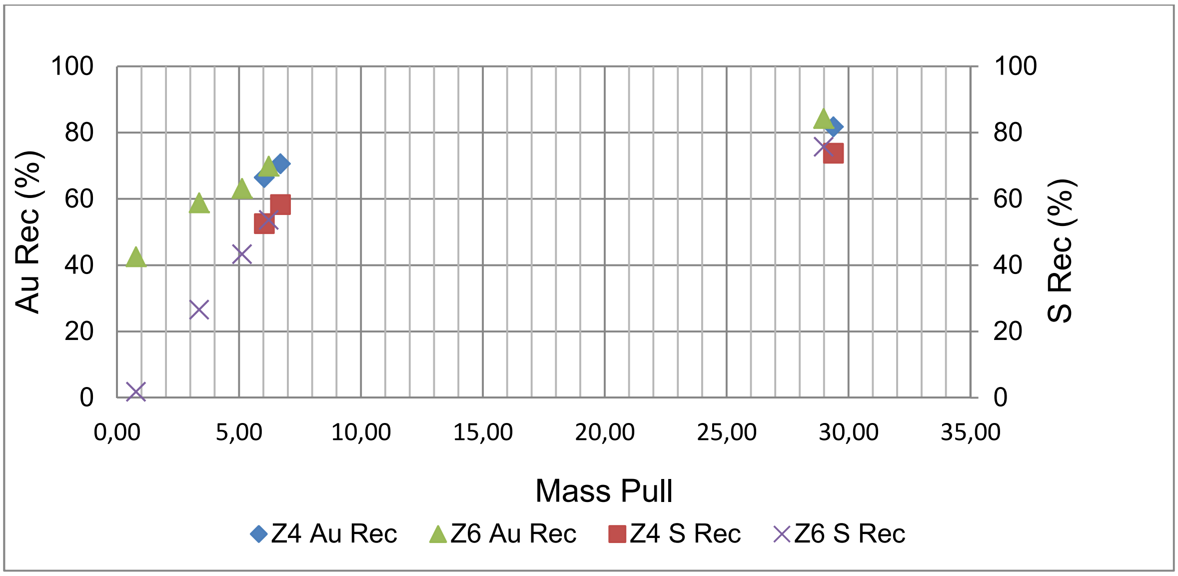


Figure 3.12. Mass pull vs Au& S recovery at cleaning and scavenging stages

The mass balance according to the elemental assays shown in Table 3.14 and Table 3.15. were done by JKSimfloat and compared with experimental results.

Table 3.14. Metallurgical balance assay results (Z4)

	%	Fe	As, g/t	Au, g/t	S %
Cleaner 1 Concentrate	6.03	37.79	14673.00	40.09	30.29
Cleaner Scavenger Conc.	0.67	33.11	13480.00	22.83	29.97
Cleaner Scavenger Tail	22.68	3.53	1126.60	1.79	2.38
Bulk Tail	70.62	2.46	581.30	0.94	1.29
Feed (Calc)	100.00	5.04	1641.25	3.64	3.48

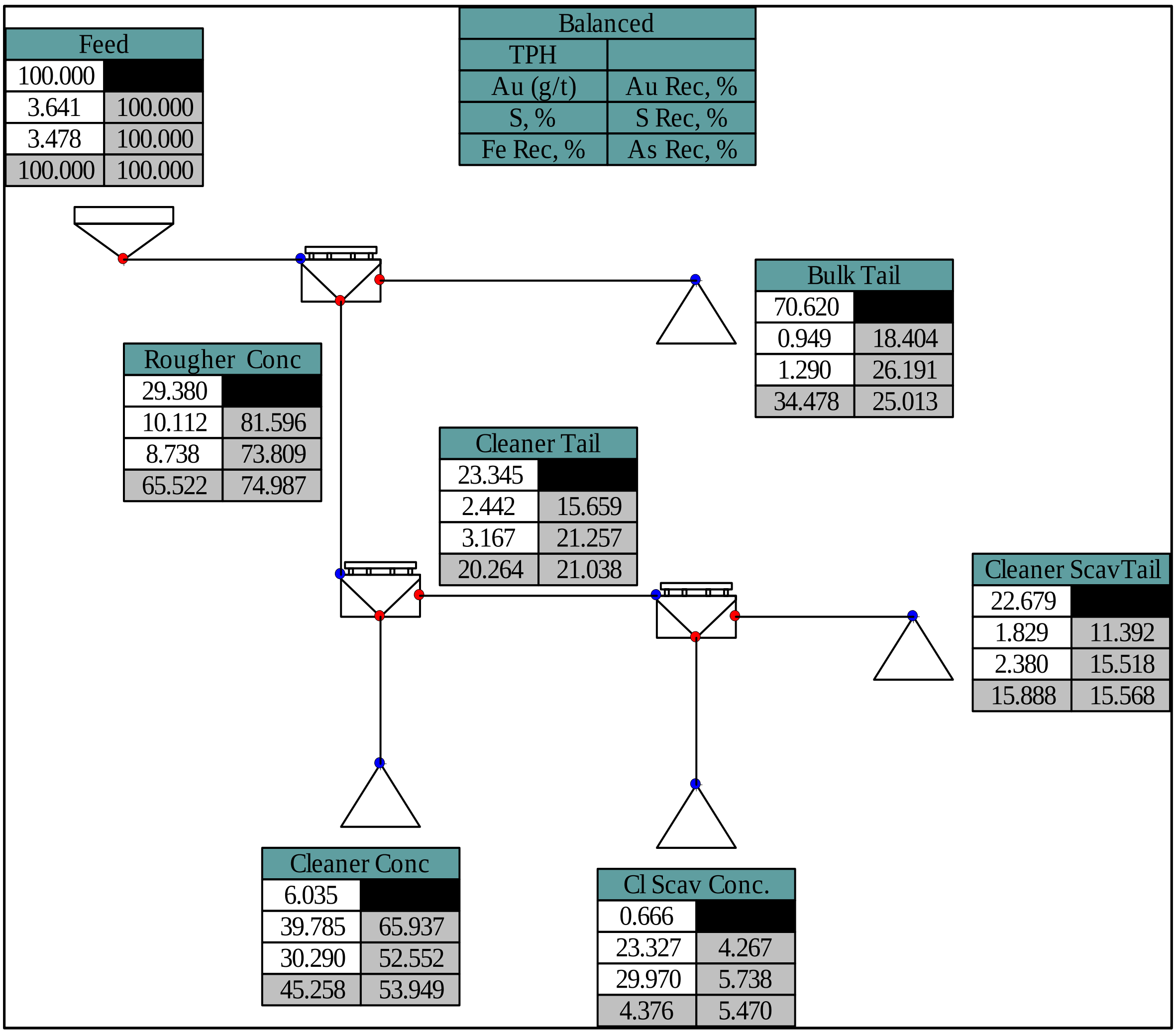


Figure 3.13. Mass balance of cleaning test 1 (Z4)

Table 3.15. Metallurgical balance assay results (Z6)

	%	Fe	As, g/t	Au, g/t	S %
Cleaner Conc	0.77	9.05	2699.30	225.00	8.82
Cleaner Scavenger C1	2.59	42.89	17017.00	25.54	35.48
Cleaner Scavenger C2	1.76	49.11	19618.00	10.00	35.21
Cleaner Scavenger C3	1.09	39.47	16270.00	24.95	35.08
Cleaner Scavenger Tail	22.76	4.86	1620.10	2.57	3.60
Bulk Tail	71.02	2.17	606.29	0.90	1.27
Feed (Calc)	100.00	5.13	1785.34	4.07	3.71

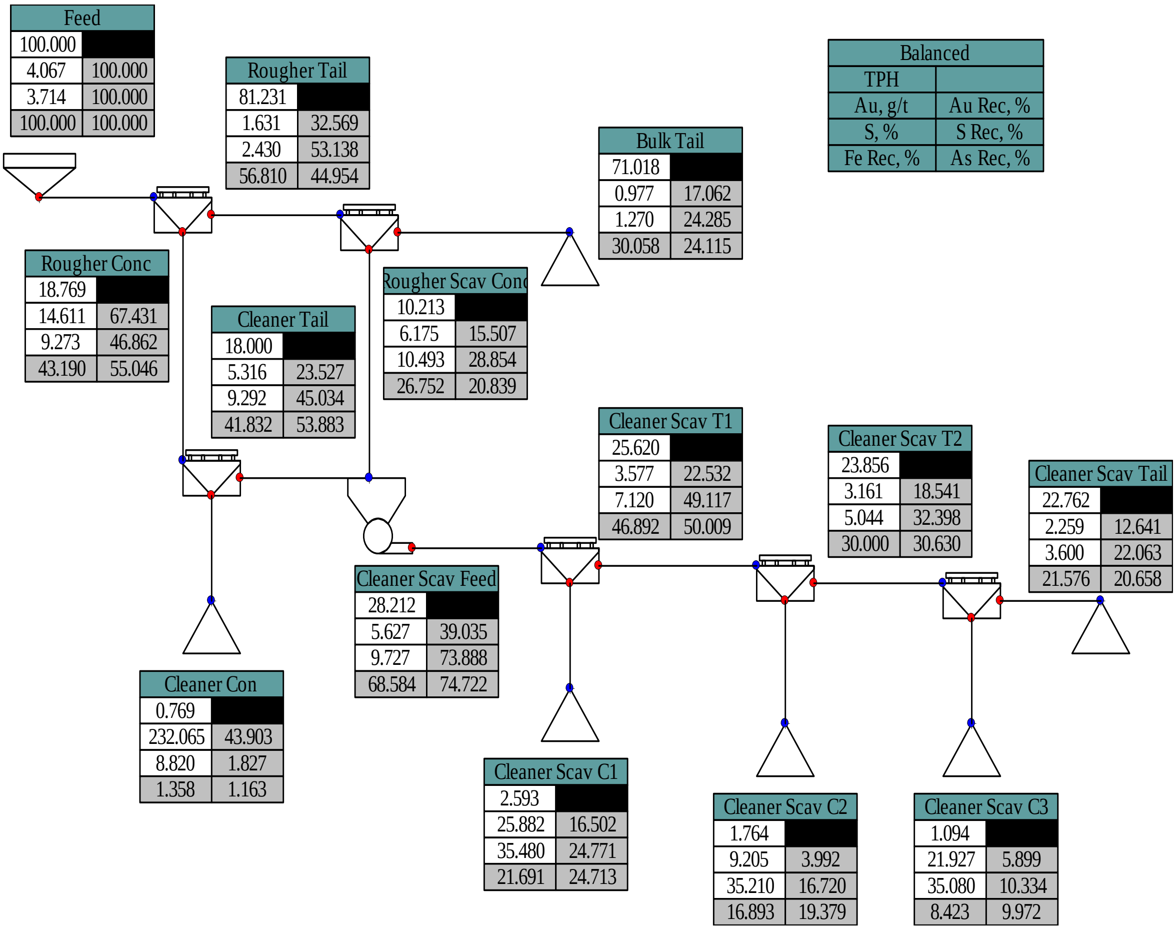


Figure 3.14. Mass balance of cleaning test 2 (Z6)

Experimental results and metallurgical balance according to elemental assays were nearly the same in Z4 and Z6 conditions. The difference was that experimental overall gold recovery in Z6 condition is 84.28 % while calculated gold recovery from JKSimfloat is 82.94%. This can be come from elemental assays.

3.9. The Relationship Between Gold & Sulphur

The relationship between Au and S and also As and Fe were determined according to the results to have an idea about the mineralogy of the ore without a gold deportment study.

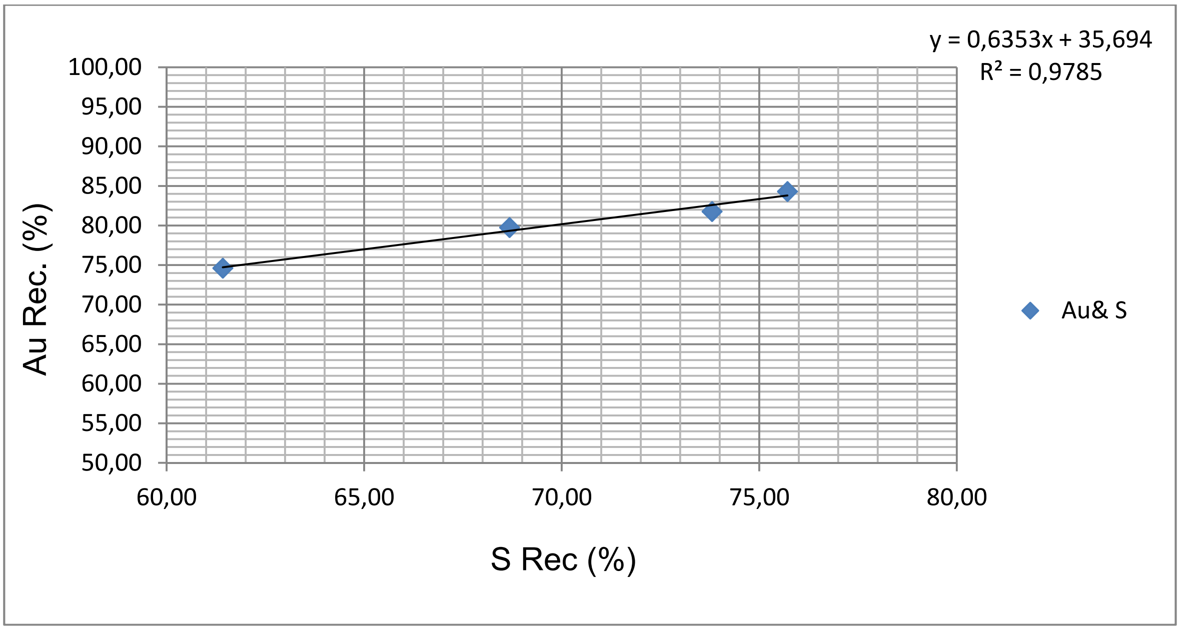


Figure 3.15. The relationship between Au and S from flotation test results

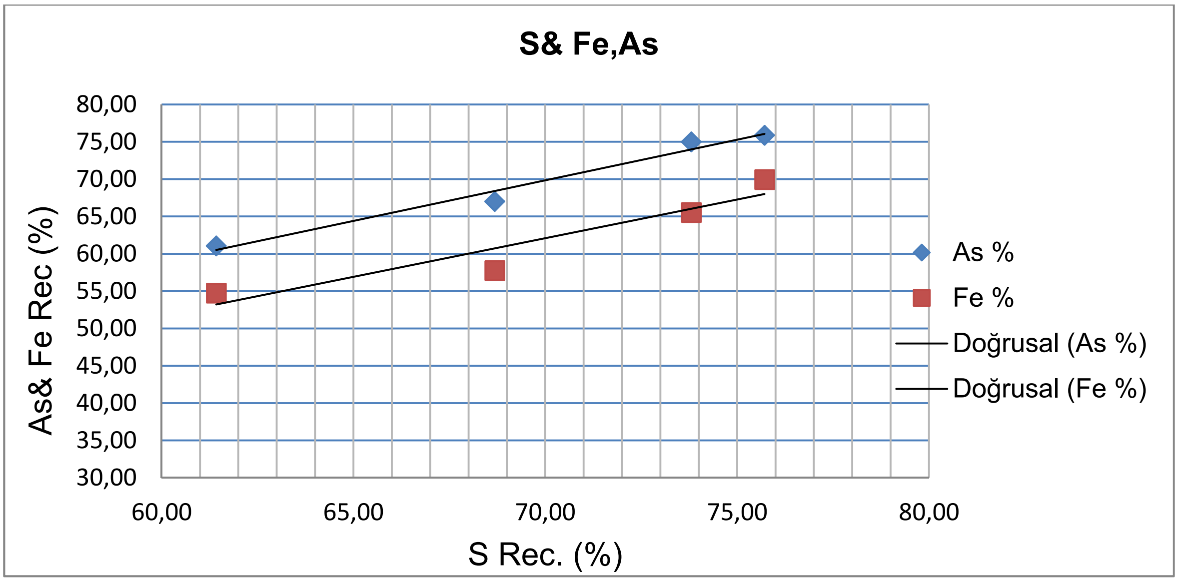


Figure 3.16. The relationship between S & Fe,As from flotation test results

A linear relationship between S and Fe,As were obtained from the results as shown in Figure 3.16. This relationship can explain that iron and arsenic sulphides could be the main gold carrier sulphides. However it does not mean that gold carriers is only sulphides. Because of the fact that gold is inside silicates as well from cyanide and flotation test results. Although chemical conditions changed gold recovery was stabled around 80-85 %. This result demonstrates that some of the gold ore is still with silicates and not be recovered by flotation.

3.10. Cyanide Leaching Results (Bottle Roll Test Results)

According to the flotation test results, Z12 conditions were selected to produce a bulk concentrate for cyanide leaching. The results as shown in Table 3.17. that the gold was not only associated with sulphide but also disseminated in sulphide matrix which has a refractory behavior. The gold recovery from concentrate was around % 55. Although it was leached with intensive cyanide, the gold could not be recovered completely. The reason for this situation is that cyanide could not able to penetrate the gold grains to dissolve them. 1500 ppm of cyanide is very intensive as compared to the conventional cyanide leaching plants. Conventional leaching circuits cyanide consumption changes between 0.20 kg/t and 0.80 kg/ t. However, if the ore containes cyanicides like pyrite pyrhotite, chalcocite, covellite, orpiment or realgar, reagent consumptions are very high.

If the ore is refractory like Z vein flotation concentrate is not directly leached due to the high sulphide content and gold in sulphide. Sulphur in leaching operations consumes high cyanide, lime and oxygen as also seen from the results. Sulphur dissolves in cyanide and forms cyanide complexes. These complexes retards the gold dissolution due to the lack of reagents and surface passivization of gold by sulphur. For these reasons that as mentioned above, to recover the gold inside the sulphur, pressure oxidation leaching techniques and ultrafine grinding, mechanical activation by breaking sulphide matrix, can be used by mineral industry.

Table 3.16. Intensive cyanide leaching results

	Concentrate	Tail
% Au Rec	55.31	47.83
% Ag Rec	46.13	34.15
Tail Au ppm.	10.31	0.48
Feed Au ppm.	23.07	0.92
Tail Ag ppm.	4.94	0.54
Feed Ag ppm.	9.17	0.82
Final NaCN, ppm.	280	200
Final oxygen, ppm.	27	28
Final pH	10.3	10.44
NaCN consumption kg/t	1.672	0.890
Feed anlayzed, % S	19.84	0.95
Solution, Fe ppm.	4.02	0.78
Solution, Cu ppm.	22.58	5
Solution, Ni ppm.	2.2	0.67
Solution, As ppm.	0.59	0.48
Solution, Sb ppm.	0.42	0.09

The concentrations of cyanide used to dissolve gold in ores are typically much higher than the stoichiometric amount required, owing to the solubility of other minerals. Previous literature has indicated that higher levels of cyanide are related to the competition of other ionic species for cyanide (Marsden and House, 1992). This is one explanation, but it is not the only one. Additional cyanide is needed to counteract the retarding effect of the coating formed on gold grains by the dissolution of metallic sulfides. In addition, the leaching kinetics targeted must match the retention time of the circuit.

The most commonly used equation for the dissolution of gold in a cyanide solution, known as Elsner's equation, is

$$4Au + 8NaCN + O_2 + 2H_2O \longrightarrow 4Na(AuCN)_2 + 4NaOH$$

Elsner's equation shows that oxygen is critical for the dissolution of gold. The maximum dissolved oxygen content of a dilute cyanide solution is 8.2 ppm with the injection of air at 25 ^{o}C. Conventional cyanidation is usually performed at a pH higher than 10 and an O_2 concentration greater than 6 ppm. If the concentration of dissolved oxygen drops below 4 ppm, gold dissolution rates are greatly reduced. On the other hand, the rate of gold dissolution will increase markedly as the concentration of dissolved oxygen rises above 10 ppm. An oxygen-enriched operation is one that is conducted at 12–18 ppm O_2 by sparging oxygen into the slurry.

Gold dissolution is an electrochemical reaction in which oxygen takes up electrons at one part of the metallic surface (the cathodic zone), while the metal gives them up at another (the anodic zone). According to this reaction, at low cyanide concentrations, the dissolution rate is a function of cyanide concentration. At high cyanide concentrations, the dissolution rate is a function of oxygen concentration. Gold dissolution is a heterogeneous reaction that is controlled by the diffusion of both reacting species (O_2 and CN^-) through the Nernst boundary layer. The rate of metal dissolution increases linearly with increasing cyanide concentration until a maximum dissolution is reached, beyond which there is a slight retarding effect. In the case of cyanidation plants processing gold ores with metallic sulfides, oxygen is not only used to optimize plant throughput but also to enhance gold extraction. Because the dissolution of sulfide minerals consumes oxygen , it is important to avoid a reduction in the gold-leaching rate associated with low dissolved oxygen in the pulp. Below equation is related to cyanide consumption and passivation of gold with sulfide:

$2MS + 2\,(x+1)CN^- + O_2 + 2H_2O \longrightarrow 2M\,(CN)_x^{(2-x)-} + 2CNS^- + 4OH^-$
$MS + 2OH^- \longrightarrow M(OH)_2 + S^{-2}$
$2M(OH)_2 + \frac{1}{2}\,O_2 + H_2O \longrightarrow 2M(OH)_3$
$2S^{-2} + 2O_2 + H2O \longrightarrow S_2O_3^- + 2OH^-$

3.11. Cyanide Leaching Tests with Lead Nitrate

Reactions kinetics are extremely important because most cyanidation plants operate at maximum throughput. Other factors affecting gold-leaching kinetics are grain size, galvanic effects, agitation speed, temperature, foreign ions, cyanicides and preg-robbing. Coarse gold is usually recovered by gravity prior to leaching. Lead, mercury salts, bismuth and thallium ions (Guzman et al., 2000) accelerate the rate of dissolution of gold. The addition of lead salts to cyanide solutions has already been a common practice (used for more than 60 years) aimed at counteracting the harmful effect of sulfides, while mercury, bismuth and thallium are not used because of their extremely high toxicity. Recent electrochemical studies have quantified and explained the accelerating effect of these ions. The most common method used to control lead nitrate addition is based on the relationship between the gold content of the leach residue, the dissolved oxygen concentration and the free cyanide concentration of the leach solution. Because the lead concentration is always very low (0.1 mg/L), it is consequently not possible to use the analysis of the dissolved lead to control its addition. At longer leaching times or higher mineral concentrations, arsenopyrite and pyrite were found to be detrimental to gold-leaching. Lead nitrate was used for leaching tests, but the recovery did not changed appreciably. It is directly applied to the ground ore not to the flotation concentrate.

4. CONCLUSIONS

During the experiments flotation characteristics of the Z vein was determined for the further cyanide process. Before the flotation tests there was not any gold deportment and mineralogy study. According to geological studies the ore is coming from high sulphidization porphyric system overprint. This type of an ore did not be recovered by conventional agitation leaching process due to high cyanide, lime and oxygen consumption. Because of these reasons bulk flotation tests were performed to investigate the flotation behavior of the ore. Effect of collector type was investigated first and Aero 7249 gave the best result in terms of gold recovery. Furthermore, the effects of other chemical conditions were also determined to obtain the maximum gold recovery.

As a result of flotation tests, a preconcentrate with % 82 Au recovery and approximately 23 g/t Au grade was produced for cyanide leaching. Cyanide leaching tests showed that silicates inside the ore contain gold. Because of that the recovery did not increase more than 84%. Main sulphur gold carriers can be pyrite and arsenopyrite. However results does not show directly this relation. Sulphur recovery during tests has a linear relationship with gold recovery which means the ore is not a free- milling type ore , it is refractory.

Successful concentration of gold in refractory sulfide ores is almost exclusively dependent on the association of the gold with the sulfides. Refractory gold ores commonly contain free gold, sub-microscopic gold, carbonaceous material, base metals, pyrite, marcasite, arsenopyrite and pyrrhotite Clays and graphitic carbon are the most troublesome accessory components in some of these ores, as far as gold concentration is concerned. Arsenopyrite has very similar properties to pyrite and the flotation conditions for its recovery are similar to pyrite.

Certain gold ores, mainly those containing sulfides, arsenides or tellurides, are best treated by producing a flotation concentrate. This is almost always the case in treating refractory gold ores, where the refractory nature of the ore is due to sulfide mineral species. It is also the case where it is desirable to reduce the mass of material that needs to be treated to recover the gold, or where differential flotation is used to produce a separate concentrate of a valuable metal which co-exists with the gold, and which would otherwise hinder the recovery of gold. A separate gold concentrate is then produced for further processing.

Cyanide tests performed with lead nitrate to investigate the effect on gold recovery, but lead nitrate did not change the cyanidation results. Preaeration was also performed before flotation tests to oxidize the sulphide minerals in the ore but consumption was too high because of cyanicides.

Conventional cyanide leaching technics (CIP process) without flotation, gravity separation v.s could not able to recover these kind of refractory gold ores. The main idea is the mineralogy must be known well before the ore processing circuits designed.

5. FURTHER STUDIES

In a very broad sense the definition of hydrometallurgy is that branch of metallurgy concerned with the science and art of extraction of metals from their ores by processes involving solution in water.

As a further study pressure oxidative pretreatment can be investigated for the flotation concentrate to recover the gold from sulphides. Although pressure oxidation of gold ores and concentrates does not directly recover gold in a strict sense of the above definition, it is an essential step to enable the gold to be recovered and accordingly is part of hydrometallurgy oxidative pretreatment is a requirement for gold ores with a significant fraction of submicroscopic gold in sulfide minerals. Ore mineralogy in general and more specifically the process mineralogy of gold is one of the criteria used for selecting the most appropriate pretreatment option. Carbonate minerals need to be removed prior to pressure oxidation or bioleaching, and this is often achieved in a sulfuric acid pre-oxidation step. Finely disseminated submicron gold-bearing pyrite and arsenopyrite inclusions become liberated in the acid leach step, and hence rendered amenable to cyanidation, unlike inclusions in microcrystalline quartz (chert), which remain enclosed and therefore are not accessible for dissolution.

Mineralogy is an aspect often overlooked, but can be very important if dealing with a metallurgically difficult ore or a known refractory ore. By understanding the mineralogical makeup of the gold bearing and associated species, better-informed decisions can be made with respect to treatment options.

For these reasons a detailed mineralogical study on flotation tail can be done to see the minerals that reject the gold to the tailings. These are sulphides, tellurides or carbonecous material that contains gold and not recovered during flotation.

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APPENDIXES

Metallurgical Balance Z4

Product	Mass		Assay Results of Z4				
	gr	%	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Cleaner 1 Concentrate	118.7	6.03	37.79	14673	16.74	40.09	30.29
Cleaner Scavenger Concentrate	13.1	0.67	33.11	13480	8.42	22.83	29.97
Cleaner Scavenger Tail	446.1	22.68	3.53	1126.6	1.17	1.79	2.38
Bulk Tail	1389.1	70.62	2.46	581	0.61	0.94	1.29
Feed (Calc)	1967	100.00	5.04	1641.25	1.76	3.64	3.48

Product	Recovery of Z4				
	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Cleaner 1 Concentrate	45.26	53.95	57.32	66.44	52.55
Cleaner Scavenger Concentrate	4.38	5.47	3.18	4.18	5.74
Cleaner Scavenger Tail	15.89	15.57	15.06	11.15	15.52
Bulk Tail	34.48	25.01	24.44	18.23	26.19
Feed (Calc)	100.00	100.00	100.00	100.00	100.00

Metallurgical Balance Z6

Product	Mass		Assay Results of Z6				
	gr	%	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Cleaner C1	8.6	0.43	9.05	2699.3	79.19	225	8.82
Cleaner C2	3.9	0.19					
Cleaner C3	2.9	0.14					
Cleaner Scavenger C1	51.9	2.59	42.89	17017	8.76	25.54	35.48
Cleaner Scavenger C2	35.3	1.76	49.11	19618	8.43	10	35.21
Cleaner Scavenger C3	21.9	1.09	39.47	16270	8.04	24.95	35.08
Cleaner Scavenger Tail	455.6	22.76	4.86	1620.1	1.29	2.57	3.6
Bulk Tail	1421.5	71.02	2.17	606	0.70	0.9	1.27
Feed (Calc)	2001.6	100.00	5.13	1785.34	1.86	4.07	3.71

Product	Recovery Z6				
	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Cleaner C1	1.36	1.16	32.69	42.57	1.83
Cleaner C2					
Cleaner C3					
Cleaner Scavenger C1	21.69	24.71	12.19	16.28	24.77
Cleaner Scavenger C2	16.89	19.38	7.98	4.34	16.72
Cleaner Scavenger C3	8.42	9.97	4.72	6.71	10.33
Cleaner Scavenger Tail	21.58	20.66	15.75	14.38	22.06
Bulk Tail	30.06	24.12	26.67	15.72	24.28
Feed (Calc)	100.00	100.00	100.00	100.00	100.00

Metallurgical Balance Z11

Product	Mass		Assay Results of Z11				
	gr	%	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	36.6	1.96	27.74	39774	37.49	43.17	32.73
Rougher C2	106.7	5.70	21.63	29260	34.17	48.94	23.02
Rougher C3	60.8	3.25	17.69	24849	17.18	20.45	17.13
Rougher C4	136.3	7.28	21.66	41748	9.4	13.35	25.2
Bulk Tail	1531	81.81	3.81	3690	2.11	2.57	1.66
Feed (Calc)	1871.4	100.00	7.05	9312.94	5.65	7.37	5.70

Product	Recovery of Z11				
	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	7.70	8.35	12.98	11.45	11.22
Rougher C2	17.51	17.91	34.48	37.84	23.02
Rougher C3	8.16	8.67	9.88	9.01	9.76
Rougher C4	22.39	32.65	12.12	13.19	32.19
Bulk Tail	44.24	32.42	30.55	28.51	23.81
Feed (Calc)	100.00	100.00	100.00	100.00	100.00

Metallurgical Balance Z12

Product	Mass		Assay Results of Z12				
	gr	%	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	144	7.68	18.98	19341	21.57	30.76	18.49
Rougher C2	128	6.83	19.81	21463	25.16	33	21.55
Rougher C3	65.5	3.49	13.77	13540	8.95	10.8	12.48
Rougher C4	161.6	8.62	22.73	29127	6.7	11.52	21.09
Bulk Tail	1375.4	73.37	2.43	1887	1.70	1.75	1.35
Feed (Calc)	1874.5	100.00	7.03	7320.25	5.51	7.27	6.14

Product	Recovery of Z12				
	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	20.73	20.30	30.06	32.50	23.15
Rougher C2	19.23	20.02	31.16	30.99	23.98
Rougher C3	6.84	6.46	5.67	5.19	7.11
Rougher C4	27.86	34.30	10.48	13.66	29.63
Bulk Tail	25.35	18.92	22.63	17.66	16.14
Feed (Calc)	100.00	100.00	100.00	100.00	100.00

Metallurgical Balance Z13

Product	Mass		Assay Results of Z13				
	gr	%	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	176.3	9.29	20.04	21510	23.18	31.36	20.55
Rougher C2	74.3	3.91	22.83	23119	29.2	38.48	22.77
Rougher C3	33.2	1.75	14.06	12117	12.6	13.04	13.77
Rougher C4	71.8	3.78	21.57	29996	7.6	11.68	41.94
Bulk Tail	1542.7	81.27	4.28	3715	2.17	2.67	2.37
Feed (Calc)	1898.3	100.00	7.29	7268.13	5.57	7.26	6.55

Product	Recovery of Z13				
	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	25.51	27.49	38.67	40.13	29.12
Rougher C2	12.25	12.45	20.53	20.75	13.60
Rougher C3	3.37	2.92	3.96	3.14	3.68
Rougher C4	11.18	15.61	5.16	6.09	24.21
Bulk Tail	47.68	41.54	31.68	29.89	29.39
Feed (Calc)	100.00	100.00	100.00	100.00	100.00

Metallurgical Balance Z14

Product	Mass		Assay Results of Z14				
	gr	%	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	130.8	7.11	17.25	18007	18.59	25.71	16.45
Rougher C2	77	4.19	18.2	17214	29.18	41.95	19.32
Rougher C3	60.8	3.31	11.79	11163	11.47	13.98	9.58
Rougher C4	78.6	4.27	16.51	18500	6.28	8.55	14.35
Bulk Tail	1491.4	81.12	5.53	5108	2.29	3.01	3.42
Feed (Calc)	1838.6	100.00	7.57	7305.22	5.05	6.86	5.68

Product	Recovery of Z14				
	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	16.21	17.54	26.19	26.68	20.59
Rougher C2	10.07	9.87	24.20	25.63	14.24
Rougher C3	5.15	5.05	7.51	6.74	5.57
Rougher C4	9.32	10.83	5.32	5.33	10.79
Bulk Tail	59.25	56.72	36.78	35.62	48.81
Feed (Calc)	100.00	100.00	100.00	100.00	100.00

Metallurgical Balance Z16

Product	Mass		Assay Results of Z16				
	gr	%	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	32.6	1.73	32.57	33061	29.05	37.51	40.51
Rougher C2	101.6	5.39	24.46	25053	22.99	38.63	22.77
Rougher C3	100.7	5.34	17.71	19702	13.35	23.17	15.48
Rougher C4	107.6	5.71	20.43	26476	8.73	15.13	1.25
Bulk Tail	1542.8	81.83	4.08	2676	2.67	2.36	2.08
Feed (Calc)	1885.3	100.00	7.33	6675.07	5.14	6.76	4.53

Product	Recovery of Z16				
	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	7.68	8.56	9.78	9.59	15.47
Rougher C2	17.98	20.23	24.12	30.78	27.10
Rougher C3	12.90	15.77	13.88	18.30	18.26
Rougher C4	15.90	22.64	9.70	12.77	1.58
Bulk Tail	45.54	32.81	42.53	28.56	37.59
Feed (Calc)	100.00	100.00	100.00	100.00	100.00

Metallurgical Balance Z17

Product	Mass		Assay Results of Z17				
	gr	%	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	41.8	2.22	16.95	19396.12	39.51	81.98	25.98
Rougher C2	74.1	3.93	19.17	22166.83	21.21	36.62	25.82
Rougher C3	64.3	3.41	9.64	12420.56	13.32	19.63	24.28
Rougher C4	143.3	7.60	10.24	13783.21	8.78	12.47	21.15
Bulk Tail	1561.3	82.84	1.86	2521	1.65	2.27	3.30
Feed (Calc)	1884.8	100.00	3.78	4861.64	4.20	6.76	6.76

Product	Recovery of Z17				
	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	9.95	8.85	20.87	26.91	8.52
Rougher C2	19.95	17.93	19.86	21.31	15.01
Rougher C3	8.71	8.72	10.82	9.91	12.25
Rougher C4	20.61	21.56	15.90	14.03	23.78
Bulk Tail	40.79	42.96	32.55	27.83	40.43
Feed (Calc)	100.00	100.00	100.00	100.00	100.00

Metallurgical Balance Z18

Product	Mass		Assay Results of Z18				
	gr	%	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	63.1	3.38	21.92	25980.09	32.28	63.89	28.22
Rougher C2	93.7	5.02	8.49	11416.11	19.12	26.67	25.47
Rougher C3	74.4	3.98	6.24	9073.09	10.92	13.6	19.55
Rougher C4	147.1	7.87	10.59	15329.97	6.75	9.13	17.97
Bulk Tail	1489.9	79.75	1.04	1445	1.22	1.85	2.55
Feed (Calc)	1868.2	100.00	3.08	4171.00	3.99	6.23	6.46

Product	Recovery of Z18				
	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	24.05	21.04	27.34	34.63	14.76
Rougher C2	13.83	13.73	24.04	21.47	19.78
Rougher C3	8.07	8.66	10.90	8.69	12.06
Rougher C4	27.09	28.94	13.33	11.54	21.91
Bulk Tail	26.95	27.63	24.39	23.68	31.49
Feed (Calc)	100.00	100.00	100.00	100.00	100.00

Metallurgical Balance Z20

Product	Mass		Assay Results of Z20				
	gr	%	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	63.2	3.80	26.97	27278.34	35.43	58.75	33.74
Rougher C2	57.3	3.45	15.64	20540	31.97	53.18	28.77
Rougher C3	37.6	2.26	8.94	11887.35	16.36	24.25	22.89
Rougher C4	86.8	5.22	15.06	23118.54	10.05	15.72	34.4
Bulk Tail	1418.1	85.27	1.62	2730	1.49	2.46	4.29
Feed (Calc)	1663	100.00	3.93	5548.20	4.61	7.53	8.24

Product	Recovery of Z20				
	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	26.06	18.68	29.19	29.64	15.55
Rougher C2	13.70	12.76	23.88	24.33	12.02
Rougher C3	5.14	4.84	8.02	7.28	6.28
Rougher C4	19.98	21.75	11.37	10.89	21.78
Bulk Tail	35.12	41.97	27.54	27.85	44.37
Feed (Calc)	100.00	100.00	100.00	100.00	100.00

Metallurgical Balance Z21

Product	Mass		Assay Results of Z21				
	gr	%	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	71.4	4.15	7.69	10178.76	11.53	17.9	9.71
Rougher C2	74.7	4.34	9.4	12278.41	25.47	46.55	13.65
Rougher C3	84.6	4.92	9.19	10859.85	14.37	27.58	12.75
Rougher C4	148.6	8.64	17.05	21886.09	9.78	13.92	30.62
Bulk Tail	1340	77.94	3.05	4085	1.88	2.14	3.74
Feed (Calc)	1719.3	100.00	5.03	6565.76	4.60	6.99	7.19

Product	Recovery of Z21				
	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	6.35	6.44	10.40	10.63	5.61
Rougher C2	8.12	8.13	24.04	28.92	8.25
Rougher C3	8.99	8.14	15.36	19.40	8.73
Rougher C4	29.29	28.81	18.36	17.20	36.83
Bulk Tail	47.25	48.49	31.83	23.85	40.57
Feed (Calc)	100.00	100.00	100.00	100.00	100.00

Metallurgical Balance Z22

Product	Mass		Assay Results of Z22				
	gr	%	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	58.4	3.07	26.09	29474.26	33.72	60.83	32.99
Rougher C2	74.9	3.93	21.8	26244.98	28.08	48.09	28.03
Rougher C3	55.1	2.89	16.39	21773.4	13.68	18.36	22.44
Rougher C4	106.9	5.62	11.7	17715.68	8.46	13.08	30.63
Bulk Tail	1608.5	84.49	1.83	2964	1.47	2.07	2.67
Feed (Calc)	1903.8	100.00	4.34	6065.69	4.25	6.77	6.74

Product	Recovery of Z22				
	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	18.46	14.91	24.33	27.55	15.01
Rougher C2	19.78	17.02	25.98	27.94	16.36
Rougher C3	10.94	10.39	9.31	7.85	9.64
Rougher C4	15.15	16.40	11.17	10.84	25.52
Bulk Tail	35.66	41.28	29.21	25.82	33.47
Feed (Calc)	100.00	100.00	100.00	100.00	100.00

Metallurgical Balance Z23

Product	Mass		Assay Results of Z23				
	gr	%	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	38.4	1.86	13.31	14225.58	48.77	94.57	50.62
Rougher C2	37.8	1.83	10.79	13269.87	32.75	54.34	26.58
Rougher C3	25.7	1.24	13.49	17402.98	16.73	25.68	19.38
Rougher C4	76.5	3.70	21.32	28552.48	11.92	17.13	33.53
Bulk Tail	1889	91.37	3.00	4237	2.36	2.83	3.65
Feed (Calc)	2067.4	100.00	4.14	5651.22	4.31	6.29	6.24

Product	Recovery of Z23				
	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	5.97	4.68	21.02	27.93	15.06
Rougher C2	4.76	4.29	13.89	15.80	7.78
Rougher C3	4.05	3.83	4.83	5.08	3.86
Rougher C4	19.05	18.70	10.23	10.08	19.87
Bulk Tail	66.18	68.51	50.03	41.12	53.42
Feed (Calc)	100.00	100.00	100.00	100.00	100.00

Metallurgical Balance Z25

Product	Mass		Assay Results of Z25				
	gr	%	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	106.5	5.31	26.15	33280.59	23.3	62.13	51.96
Rougher C2	119.8	5.97	20.74	27424.46	15.16	32.45	27.99
Rougher C3	55.8	2.78	13.56	17987.42	9.1	17.88	16.6
Rougher C4	67.3	3.35	12.91	16820.47	6.52	14.09	16.16
Bulk Tail	1658	82.59	1.38	1563	1.24	2.25	1.58
Feed (Calc)	2007.4	100.00	4.57	5757.18	3.64	8.06	6.74

Product	Recovery of Z25				
	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	30.33	30.67	33.99	40.89	40.93
Rougher C2	27.06	28.43	24.88	24.03	24.80
Rougher C3	8.24	8.68	6.96	6.17	6.85
Rougher C4	9.46	9.80	6.01	5.86	8.04
Bulk Tail	24.92	22.42	28.16	23.06	19.38
Feed (Calc)	100.00	100.00	100.00	100.00	100.00

Metallurgical Balance Z26

Product	Mass		Assay Results of Z26				
	gr	%	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	49.5	2.32	16.53	21276.51	28.53	76.24	20.79
Rougher C2	78.5	3.68	24.8	27832.42	24.97	59.9	25.31
Rougher C3	50.6	2.37	17.64	21037.43	10.83	18.31	19.01
Rougher C4	197	9.23	21.59	29597.47	8.82	18.68	34.24
Bulk Tail	1757.8	82.39	1.54	2087	1.25	2.41	1.56
Feed (Calc)	2133.4	100.00	4.98	6469.05	3.68	8.12	6.31

Product	Recovery of Z26				
	Fe	As, g/t	Ag, g/t	Au, g/t	S %
Rougher C1	7.71	7.63	17.98	21.79	7.64
Rougher C2	18.34	15.83	24.95	27.15	14.76
Rougher C3	8.41	7.71	6.98	5.35	7.14
Rougher C4	40.06	42.25	22.12	21.25	50.09
Bulk Tail	25.49	26.58	27.97	24.46	20.36
Feed (Calc)	100.00	100.00	100.00	100.00	100.00

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