

**DOKUZ EYLUL UNIVERSITY
GRADUATE SCHOOL OF NATURAL AND
APPLIED SCIENCES**

**RECOVERING EFEMÇUKURU GOLD ORES
BY FLOTATION**

**by
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**June, 2010
İZMİR**

RECOVERING EFEMÇUKURU GOLD ORES BY FLOTATION

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Graduate School of Natural and Applied Sciences of Dokuz Eylül University
In Partial Fulfillment of the Requirements for the Degree of Master of Science
in Mining Engineering, Mineral Processing Program**

**by
M. Baran TUFAN**

June, 2010

IZMIR

M.Sc THESIS EXAMINATION RESULT FORM

We have read the thesis entitled “**RECOVERING EFEMÇUKURU GOLD ORES BY FLOTATION**” completed by **M.BARAN TUFAN** under the supervision of **ASSOC. PROF. DR. EROL KAYA** and we certify that in our opinion it is fully adequate, in scope and in quality, as a thesis for the degree of Master of Science.

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RECOVERING EFEMÇUKURU GOLD ORES BY FLOTATION

ABSTRACT

The aim of this study is to investigate the beneficiation of Efemçukuru gold ore by flotation. The flotation procedure is decided due to the mineralogy and characteristics of the gold ore in the region.

The different variables were studied and analyzed to achieve the highest gold recovery with a reasonable gold grade. The high recovery in gold beneficiation is becoming more important since the gold prices continue to increase.

Knelson concentrator was also studied in detail to observe the recoverability of gold ore by gravity methods. The combination of both methods suggested an alternative processing procedure to cyanide leaching.

In conclusion, the optimum flotation parameters obtained are given and the related mechanisms are discussed in detail.

Keywords : Gold, froth flotation, bulk flotation, gravity techniques, Knelson concentrator

EFEMÇUKURU ALTIN CEVHERİNİN FLOTASYON YÖNTEMİ İLE ZENGİNLEŞTİRİLMESİNİN ARAŞTIRILMASI

ÖZ

Bu tez çalışmasının amacı, Efemçukuru altın cevherinin flotasyon yöntemiyle zenginleştirilmesinin araştırılmasıdır. Flotasyon çalışmalarında uygulanan prosedür, bölgedeki altın cevherinin yataklanması ve karakteristik özelliklerine bağlı olarak belirlenmiştir.

Altın cevherinin en yüksek altın verimi ve makul bir altın tenörü ile kazanılması için farklı değişkenler araştırılmıştır. Altın fiyatlarının hızla yükselmesi, altın veriminin arttırılmasını daha önemli hale getirmektedir.

Flotasyon yönteminin yanında, cevherin gravite yöntemiyle zenginleştirilmesi de Knelson konsantratörü kullanılarak araştırılmıştır. Bu iki yöntemin birleştirilmesi, siyanür liçine alternative bir yöntem sunmaktadır.

Elde edilen en uygun flotasyon koşulları ve bu koşulları oluşturan mekanizmalar tartışılmıştır.

Anahtar sözcükler : Altın, köpük flotasyonu, toplu flotasyon, gravite yöntemleri, Knelson konsantratörü

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CHAPTER ONE

INTRODUCTION

1.1 About Gold

Gold is a metallic element. Its atomic number is 79. It is soft, shiny, yellow, dense, malleable and ductile. It does not react with most chemicals but is attacked by chlorine and fluorine. This metal occurs in the form of nuggets or grains in rocks. It is also found in alluvial deposits. Gold is generally measured by grams. If alloyed with other metals, a term 'carat' or 'karat' is used to indicate the amount of gold present. Pure gold is 24 carats (Minerals zone, 2005).

1.1.1 History of Gold

Gold has a long and complex history. From gold's first discovery, it has symbolized wealth and guaranteed power. Gold has caused obsession in men and nations, destroyed some cultures and gave power to others.

Early civilizations equated gold with gods and rulers, and gold was sought in their name and dedicated to their glorification. Humans almost intuitively place a high value on gold, equating it with power, beauty, and the cultural elite. And since gold is widely distributed all over the globe, we find this same thinking about gold throughout ancient and modern civilizations everywhere (Only Gold, 2009).

Archaeological digs suggest the use of Gold began in the Middle East where the first known civilizations began. The oldest pieces of gold jewelry Egyptian jewelry were found in the tomb of Queen Zer and that of Queen Pu-abu of Ur in Sumeria and are the oldest examples found of any kind of jewelry in a find from the third millennium BC. Over the centuries, most of the Egyptian tombs were raided, but the tomb of Tutankhamen was discovered undisturbed by modern archaeologists. Inside

the largest collection of gold and jewelry in the world was found and included a gold coffin whose quality showed the advanced state of Egyptian craftsmanship and gold working (second millennium BC).

The Persian Empire, in what is now Iran, made frequent use of Gold in artwork as part of the religion of Zoroastrianism. Persian goldwork is most famous for its animal art, which was modified after the Arabs conquered the area in the 7th century.

When Rome began to flourish, the city attracted talented Gold artisans who created gold jewelry of wide variety. The use of gold in Rome later expanded into household items and furniture in the homes of the higher classes. By the third century AD, the citizens of Rome wore necklaces that contained coins with the image of the emperor. As Christianity spread through the European continent, Europeans ceased burying their dead with their jewelry. As a result, few gold items survive from the middle Ages, except those of royalty and from church hordes (Northwest Territorial Mint, 2005).

Herodotus (484–425 BC) refers to several great gold-mining centers in Asia Minor, and Strabo (63 BC) mentions gold mining in many different places. Pliny (23–79 AD) gives many details of ancient placer mining, which was extensive. The Romans had little of the metal in their own regions, but their military expeditions brought them major amounts in the form of booty. They also exploited the mineral wealth of the countries they had conquered, especially Spain, where up to 40,000 slaves were employed in mining. The state's accumulation of gold bars and coins was immense, but during the barbarian invasions and the collapse of the empire this gold was scattered, and gold mining languished in the middle Ages (Habashi, 2005).

Following the discovery of America at the end of the fifteenth century, the Spaniards transferred considerable amounts of gold from the New World to Europe. Although the conquistadors found a highly developed mining industry in Central America, their efforts to increase gold production were largely unsuccessful because most of the finds consisted of silver. It was not until the discovery of deposits in

Brazil, in 1691, that there was a noticeable increase in world gold production. Since about 1750 gold has been mined on a major scale on the eastern slopes of the Ural Mountains. In 1840, alluvial gold was discovered in Siberia then at Coloma, California in January 1848, a few days before the signing of a treaty between Mexico and the United States to end their hostilities. California was thus ceded by Mexico after a discovery that was apparently not known to either government. Coloma is about 50km southeast of Sacramento on the slopes of the Sierra Nevada. The discovery of gold in British Columbia was an epoch-making event. In the late 1850s, alluvial gold was found along the Thompson River, and in 1858 the famous Fraser River rush took place. Extraordinarily rich deposits were discovered in 1860 on Williams and Lightning creeks. For many years, British Columbia was the leading gold producer among the Canadian provinces and territories, but with the discovery of the Kirkland lake deposits in 1911, and the opening up of the Porcupine district in 1912, Ontario held first place ever since (Habashi, 2005).

Gold deposits were also found in Eastern Australia (1851), Nevada (1859), Colorado (1875), Alaska (1886), New Zealand and Western Australia (1892), and Western Canada (1896). However, these deposits soon lost much of their importance. The strongest impetus was given to gold production through the discovery of the goldfields of the Witwatersrand in South Africa in 1885. South African gold soon occupied a commanding position in the world market. Production grew continuously except for a short interruption by the Boer War (1899–1902). Gold mining in Ghana (Gold Coast) began to play a modest role in the twentieth century, although the deposits were known in the Middle Ages (Habashi, 2005).

1.1.2 Physical and Chemical Characteristics of Gold

Gold is yellow in color but can also occur in black or ruby when it is finely divided. The colloidal solutions are intensely colored and are often purple. Gold's plasmon frequency, lying in visible range, results in colors. It absorbs blue light and causes the red and yellow light to be reflected (Minerals zone, 2005).

Gold is most malleable and ductile. One gram can be beaten into a sheet of one square meter. It readily forms alloy with many other metals. With copper it yields redder metal, blue with iron, silver produces green, aluminum-purple and platinum-white. Gold is a good conductor of both heat and electricity. It is not affected by air and most reagents. Heat, moisture, oxygen, and most of the corrosive agents have very little chemical effect on gold. Halogens chemically alter gold and aqua regia dissolves it (Minerals zone, 2005).

Table 1.1 Physical and atomic properties of gold (Minerals zone, 2005).

Phase	Solid
Density	19.3 g/cm ³
Melting point	1064.18° C
Boiling point	2856° C
Heat of fusion	12.55 kJ/mol
Heat of vaporization	324 kJ/mol
Crystal structure	Cubic face centered

1.1.3 Utilization of Gold

The utilization of gold can be summarized as follows (Minerals zone, 2005);

- Gold and its alloy are often used in jewelry, coinage and a standard for monetary exchange in many countries.
- Due to its high electrical conductivity and resistance to corrosion, it is used as industrial metal.
- It is made into thread and used in embroidery.
- It performs a critical function in computers, communications equipment, spacecraft, jet aircraft engines and host of other products.

- The resistance to oxidation property, it is used as a thin layers electroplated on the surface of electrical connectors to make sure of good connection.
- It is used in restorative dentistry.
- Colloidal gold (a gold nanoparticle) is an intensely colored solution and is used as gold paint on ceramics prior to firing.
- Chlorauric acid is used in photography to tone silver image.
- Gold (III) chloride is utilized as catalyst in organic chemistry.
- It is used in awards.
- It is used for protective coatings on many artificial satellites as it is a good reflector of infrared and visible light.
- The isotope of gold, Au-198 is used in some cancer treatments and for other diseases.
- Gold flake is used in sweets and drinks.
- White gold serves as the substitute for platinum.
- Green gold is used in specialized jewelry.

1.1.4 Gold Production and Prices

Studies of the price and world market for gold in past academic, financial and semi popular literature have neglected production technology and have concentrated primarily on how the price of gold is determined by demand. This lack of treatment of gold supply was justified mainly on the grounds that annual gold output historically has represented a small percentage of the stock of gold in the world market. As a result, the price of gold was dominated overwhelmingly by decisions of investors and speculators who changed their holdings of gold in response to economic and political conditions and by demand for industrial purposes. Forecasts of the price of gold focused loosely on the factors determining the demand for gold from these varied sources (Duane, 1999).

Many economists and market analysts believe that gold spot prices are influenced by expected inflation. Analysts in the financial press routinely attribute substantial changes in the price of gold to changes in expected inflation. When unexpected changes in the Consumer Price Index (CPI) occur on the same day as large changes in the price of gold, analysts attribute the change in gold price to the changes in the inflation indicators (Laurence, 2009).

The gold prices are significantly increased for the past few years and that affected the mining industry positively. The trend in the gold prices is listed in table 1.2.

Table 1.2 The world gold price (\$/oz) between 1990 and 2007 (Gold Miners Association, 2006)

Years	Gold Price (\$/oz)
1990	383.51
1991	362.11
1992	343.82
1993	359.77
1994	384
1995	383.79
1996	387.81
1997	331.02
1998	294.24
1999	278.98
2000	279.11
2001	271.04
2002	309.73
2003	363.38
2004	409.72
2005	444.74
2006	635
2007	695.39

The world gold production between the years 1997 and 2007 is shown in figure 1.1, and the gold production of Turkey for the past 10 years is stated in table 1.3.

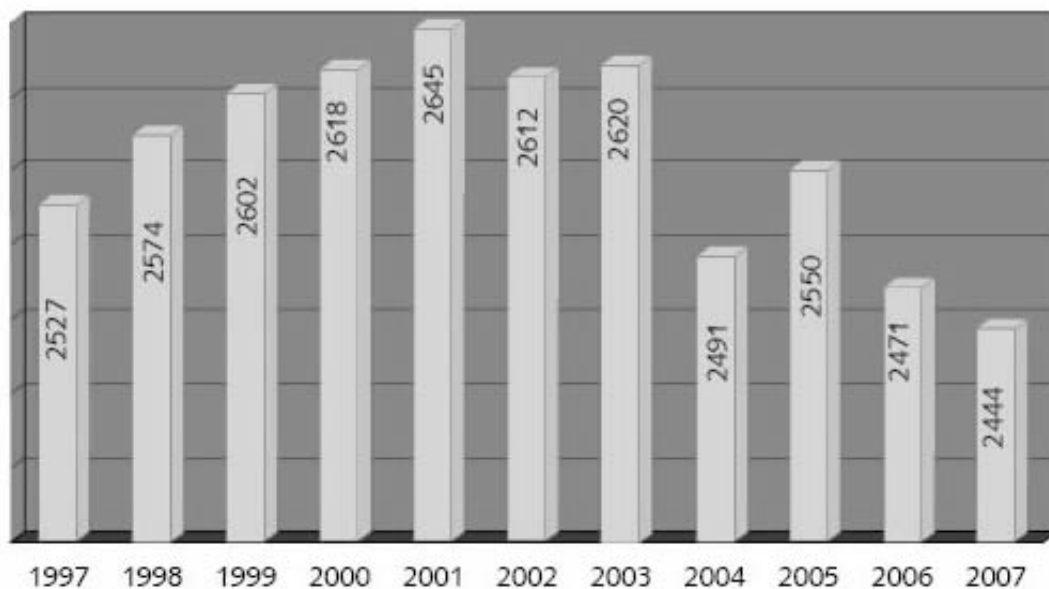


Figure 1.1 World gold productions (tons) between 1997 and 2007 (Gold Miners Association, 2006).

Table 1.3 The gold production of Turkey for the past 10 years (Gold Miners Association, 2006)

Years	Production (tons)
2000	0.0
2001	1.4
2002	4.3
2003	5.4
2004	5.0
2005	5.0
2006	8.0
2007	9.8
2008	11.0
2009	16.0

1.2 Gold Minerals and Alloys

Gold is the most inert metal; consequently there are not many naturally occurring gold compounds. The predominant occurrence is as native metal frequently alloyed with silver (Boyle, 1979). When the silver content exceeds 20%, the alloy is called electrum, an unofficial but universally accepted term. Other gold alloys are rare and generally confined to specific ores; for example, the two copper gold alloys: auricupride [Cu₃Au] and tetra-auricupride [AuCu] are found in higher gold-grade porphyry copper ores. Gold alloyed with platinum group elements (PGE) is encountered in PGE ore deposits and maldonite [Au₂Bi] is more common in the higher temperature mesothermal gold deposits. After native gold and electrum, tellurides are the most common gold minerals followed by aurostibite [AuSb₂]. Calaverite [AuTe₂] and sylvanite [(Au, Au) Te₂] are the most common tellurides comprising a significant fraction of the gold assay in a number of gold deposits. Somewhat unique characteristics of gold minerals that separate them from the other minerals include their high specific gravity, brightness (high reflectance) and hardness. In addition to the gold compounds, there are three important submicroscopic gold occurrences that cannot be ignored when considering processing options and recovery optimization. These are solid-solution gold, colloidal size particulate gold and surface-bound gold (Chryssoulis & McMullen, 2005). The most common gold minerals are listed in table 1.4.

Solid-solution gold refers to gold that is atomically distributed in the crystal structure of sulfide minerals like pyrite and arsenopyrite. The first indirect reference to solid-solution gold was made by Bürg (1930), who used the term invisible gold to describe submicroscopic gold in pyrite from the Bradisor mine in Romania. Under this term falls both solid-solution and colloidal gold.

Pyrite is the most common of the sulfide minerals and may also incorporate significant amounts of gold in its crystal structure, to the point where solid solution gold becomes the principal form of gold in the ore and pyrite its chief carrier (Thomas, 1997).

The term colloidal gold was introduced to describe discrete submicron gold inclusions in sulfide minerals, invisible by optical or conventional scanning electron microscopy (SEM), but detectable by SIMS in-depth concentration profiling (Chryssoulis, 1987).

Table 1.4 The most common gold minerals (Kucha, H., & Plimer, I.R, 2001)

Gold minerals

Native elements, alloys and metallic compounds

Native gold (<20 mol% Ag)	Au
Electrum (20–80 mol% Ag)	(Au,Ag)
Palladian gold (porpezite)	(Au,Pd)
Rhodian gold (rhodite)	(Au,Rh)
Iridic gold	(Au,Ir)
Platinum gold	(Au,Pt)
Goldamalgam	(Au,Ag)Hg
Weishanite	(Au,Ag) ₃ Hg ₂
Maldonite	Au ₂ Bi
Auricupride	Cu ₃ Au
Tetra-auricupride	AuCu
Hunchinite	Au ₂ Pb
Bogdanovite	Au ₅ (Cu,Fe) ₃ (Te,Pb) ₂
Bezsmertnovite	Au ₄ Cu(Te,Pb)
<i>Sulfide/selenite</i>	
Uytenbogaardite	Ag ₃ AuS ₂
Fischesserite	Ag ₃ AuSe ₂
Petrovskaita	AuAg(S,Se)
<i>Tellurides</i>	
Calaverite	AuTe ₂
Krennerite	(Au,Ag)Te ₂
Muthmannite	(Au,Ag)Te
Petzite	Ag ₃ AuTe ₂
Sylvanite	(Au,Ag) ₂ Te ₄
Kostovite	CuAuTe ₄
Montbrayite	(AuSb) ₂ Te ₃
Nagyagite	[Pb(Pb,Sb)S ₂] [Au,Te]
<i>Silicates/Other</i>	
As chlorite ^a	(Mg,Al,Fe) ₁₂ [(Si,Al) ₈ O ₂₀](OH) ₁₆
Auroantimonate	AuSbO ₃

1.3 Types and Characteristics of Gold Deposits

1.3.1 Gold in Shear Zones

Representatives of gold in shear zones are widespread. The deposits occur as veins, lodes, stockworks, pipes, and irregular masses in extensive fracture and shear zone systems in volcanic, intrusive and interbanded sedimentary rocks of all ages. Quartz is the most important gangue mineral. Other gangue minerals include calcite, dolomite, ankerite, barite, fluorite, rhodochrosite, rhodonite, adularia, microcline, albite, tourmaline, scheelite, chlorite, sericite and fuchsite. Pyrite is the most common metallic mineral, although arsenopyrite is abundant in many deposits. Other metallic minerals include loellingite, galena, sphalerite, chalcopyrite, pyrrhotite, pentlandite, acanthite, tetrahedrite-tennantite, pyrargyrite, proustite, polybasite, stephanite, miargyrite, stibnite, molybdenite, gold and silver tellurides, other tellurides, gold and silver selenides, native gold, native silver and gold alloys (Boyle, 1979). Oxidized and weathered zones contain marcasite, hematite, rutile, goethite, lepidocrocite, jarosite, and cryptomelane. Examples of gold deposits in fracture and shear zones are the Ku-kland Lake, Porcupine, Red Lake and Hemlo deposits in Ontario; Giant Yellowknife in Northwest Territories; Noranda - Rouyn, Val d'Or, Chibougamau and Belleterre deposits in Quebec; Kolar Goldfields in India; Barberton deposits in South Africa; Mother Lode in Sierra Nevada; Kalgoorlie deposits in Western Australia; Jinqingding gold deposits in China (Xu et al., 1994).

Much of the gold in these deposits occurs as platelets of native gold along shear zones, fractures and micro fractures in quartz and carbonate veins and in the walkock (Boyle, 1979). A smaller amount occurs as platelets in pyrite, arsenopyrite, and chalcopyrite, and some is present in veinlets in pyrite, arsenopyrite, chalcopyrite and gangue. The veinlets are commonly chalcopyrite, pyrite, calcite, quartz and tourmaline veinlets with small amounts of gold in them (Kojonen et al., 1993). A few are complex veinlets composed of sulphosalts, galena and sphalerite with minute inclusions of native gold. Some gold also occurs along mineral grain boundaries and as interstitial fillings, and some encapsulated gold in arsenopyrite, pyrite, gangue and

chalcopyrite is also present. Minor to significant amounts of “invisible” gold, gold tellurides, gold selenides and gold alloy may be present in these ores.

The gold ores in the Red Lake area in Ontario contain considerable amounts of gold in arsenopyrite as encapsulated gold and as “invisible” gold. Some of the arsenopyrite crystals are layered and some of the layers are enriched in gold. Some of the layered arsenopyrite crystals contain numerous inclusions of encapsulated gold, whereas others do not (Petruk, 2000).

1.3.2 Gold in Carlin-type Deposits

Carlin-type deposits occur in a metamorphosed and faulted sequence of calcareous siltstone, sandstone, silty limestone, chert, and siliceous mudstone (Branham & Arkell, 1995; Livermore 1996). Some ore horizons are rich in clay minerals such as kaolinite and illite, whereas other horizons are enriched in calcite and dolomite. The ores contain small amounts of pyrite, marcasite, arsenopyrite, and sometimes carbonaceous matter, jarosite and barite. The gold occurs as micron-size grains disseminated within argillized and silicified silty hornfels, marble and siltstone, as small pods of gold along faults, and as fine-grained gold associated with pyrite.

1.3.3 Gold in Volcanogenic Massive Sulfide Base Metal Deposits

Volcanogenic massive sulfide deposits contain from 1 g/t to 7 g/t gold. The gold occurs as discrete grains, veinlets, fissure fillings and “invisible” gold. It is associated with pyrite, arsenopyrite, chalcopyrite, galena, tetrahedrite and sphalerite (Petruk & Wilson, 1993; Healy & Petruk, 1990).

1.3.4 Gold Associated with Porphyry Copper Deposits

Many porphyry copper deposits contain gold in the main part of the deposit, and in the nearby surrounding rocks. Porphyry copper deposits consist of stockworks of mineralized quartz veins in intrusive host rocks (commonly granitic) and in associated volcanic rocks. The quartz veins contain inclusions and large grains of chalcopyrite, bornite, pyrite, molybdenite, and magnetite, and trace amounts of a variety of other minerals including galena, sphalerite and gold. The gold is commonly present as micro-veinlets in the quartz, chalcopyrite and pyrite, and possibly as “invisible” gold in pyrite and arsenopyrite (Petruk, 2000).

1.3.5 Gold in Conglomerate (Witwatersrand-type) Deposits

The Witwatersrand deposits in South Africa provide the best examples of gold in conglomerates and reefs. The gold commonly occupies interstitial spaces between the matrix minerals in the conglomerate (Hofineyer & Potgeiter, 1983), and occurs as irregular, jagged, flaky, plate-like and wire-like grains that are up to 1 cm long and a few micrometers thick. Some of the gold occurs as fracture fillings and as discrete grains in veinlets in fractured pyrite, chalcopyrite, arsenopyrite, cobaltite, uraninite, and quartz. Some also occurs in complex veinlets. The gold in the complex veinlets is often intergrown with authigenic minerals such as chalcopyrite, galena, tennantite, bismuthinite, sphalerite, and porous pyrite. Some of the gold also occurs as interstitial fillings between grains, particularly between pyrite grains. The least common occurrence is inclusions of gold in large pyrite and quartz crystals. Nevertheless, pyrite is the most important host for the gold, although chalcopyrite is the dominant ore mineral in all reefs (Oberthür & Frey, 1991; Harley & Charlesworth, 1994).

1.3.6 Invisible gold

Some gold deposits consist essentially of “invisible” gold in pyrite and arsenopyrite and are, therefore, classified as refractory gold deposits. The Suurikuusikko Au deposit in central Lapland, which was studied by Kojonen & Johanson (1999), is an example of such a deposit. The average “invisible” gold contents in pyrite and arsenopyrite are about 46 g/t and 279 g/t, respectively. The distribution of gold in the ore is 4.1 % free gold, 22.7 % “invisible” gold in pyrite and 73.2 % “invisible” gold in arsenopyrite. The gold content in pyrite varies from 22 to 585 g/t, and in arsenopyrite from 22 to 964 g/t. The pyrite grains are zoned and there is a general correlation between the Au and As contents in pyrite. In contrast there is a negative correlation between the Au and Sb contents in arsenopyrite (Kojonen & Johanson, 1999).

1.4 Gold Ore Processing Methods

The most common gold ore processing methods are leaching, gravity techniques, roasting, amalgamation and flotation. These methods are summarized below. The two main techniques used in this research are gravity method by Knelson concentrator and froth flotation. These two techniques are investigated and studied in chapter 3 in detail.

1.4.1 Leaching (Cyanidation)

Cyanidation is the most common technique for recovering gold. It is performed by leaching the ore in an alkaline solution that has a low concentration of alkaline cyanide (cyanidation). Other solutions such as thiourea, nitric acid, halides, etc., have been tested and used in special cases, but alkaline cyanide (Na-cyanide) is generally used because it is chemically robust and usually is very forgiving of non-optimum operating conditions. Furthermore, despite its toxicity, the cyanide ion is easy to oxidize and rendered harmless in gold plant tailings (Petruk, 2000).

The alkaline cyanide solution dissolves gold and silver at a pH of 10 to 11, and dissolves some gold tellurides at a higher pH (~12). Cyanidation is, however, usually performed at a pH of 10 to 11 hence, only native gold; electrum, gold alloy, secondary gold and native silver are dissolved.

Until recently gold was precipitated from the leach liquor by zinc cementation. In the last twenty years carbon adsorption processes, which include carbon in pulp (CIP), carbon in leach (CIL) and carbon in columns (CIC), have replaced many zinc cementation plants (Fleming, 1998). It was found to be considerably more economical to install and operate than the zinc cementation process, had a higher gold recovery, and was less vulnerable to impurities such as sulfides, arsenates and antimony in the leach liquor. The CIP process recovers gold directly from a pulp or slurry that contains 50 to 60 % solids. The gold is leached in one tank, and is adsorbed onto activated carbon in another tank. The loaded carbon is recovered by a screening device which has a screen mesh size that allows the gold depleted pulp to pass through while retaining the carbon granules. The CIL process operates in a similar manner to the CIP process but the activated carbon is added to the leaching tanks, and adsorption occurs simultaneously with the leaching. The CIC process operates by pumping the pregnant leach liquor up flow through a series of columns that are packed with activated carbon (Flemming, 1998).

1.4.2 Pressure Leaching

There has been a shift in recent years from the traditional roasting method of treating refractory gold ores to pressure leaching. A higher gold recovery is obtained by cyaniding the pressure leach residue than the roasted calcine, and the process produces a very stable ferric arsenate complex (FeAsO_4) (Fleming, 1998). Both pyrite and arsenopyrite are decomposed by pressure leaching and occluded gold is exposed. The invisible gold is released from the arsenopyrite and pyrite and it precipitates as secondary gold which is soluble in the cyanide solutions. Cyanidation of the residue will recover the gold that has been exposed, as well as the gold has been released

from the mineral structure. Some of the iron and sulfur that were released by the decomposition of the sulfides precipitate as hematite, ferric sulfate and jarosite.

1.4.3 Bioleaching

Bacterial leaching is a relatively new technique of processing refractory gold ores, and is being investigated by many companies. The technique has the same advantages as pressure leaching in that the pyrite, arsenopyrite and other sulfides are decomposed and the occluded gold is exposed. Similarly the invisible gold is released and precipitates as secondary gold that is soluble in the cyanide solution. The iron, arsenic and sulfur precipitate as relatively stable compounds. A major concern of bioleaching has been the residence time. Early pilot plant work at the Gencor Laboratory in South Africa required a 10 day residence time to achieve sufficient oxidation of an arsenopyrite concentrate for a 97 % gold recovery by cyanidation of the bioleach residue. After two years of operation the bacteria had adapted and mutated to the extent that the retention time had decreased to 4 days (Van Aswegen et al., 1988). In contrast, a typical pressure leaching operation requires a residence time of 1 to 2 hours. It is noteworthy that processing tests on the refractory gold ore from the large Suurikuusikko gold deposit in Finland gave recoveries of 10 % without bioleaching and 96 % with bioleaching (Harkönen et al., 1999).

1.4.4 Heap Leaching

Heap leaching of gold ores became a widely used technology in the gold mining industry in the 1990's. It is a low-cost method of recovering gold from low grade materials with recoveries of about 50 to 90 % of the contained gold. The technique involves percolating an alkaline cyanide solution through a heap and collecting the pregnant solution. The basic requirement is that the heap be porous enough for the cyanide solution to flow through, and that the ore pieces be permeable so that the solution can come in contact with the gold. This criteria is met by material from

oxidized ore zones above ore bodies, and by some primary ores. Bioleaching is used increasingly to release the gold contained in sulfides, and in some instance fine-grained ores are cast into briquettes to make them permeable. Investigations have shown that heap leaching can be performed in winter in permafrost conditions (Lakshmanan & McCool, 1987), and year round heap leaching operations have been developed in areas of severe winter conditions (Micheletti & Weitz, 1997)

1.4.5 Roasting

Roasting has been used for many years to recover the gold from refractory gold ores and is still widely used. When arsenopyrite is present in the ore a two-stage process is usually applied. A non-oxidizing first stage roast at 400-450°C is performed to remove the arsenic as volatile arsenic trioxide, followed by an oxidizing roast at 650 to 750°C, to produce a permeable hematite and SO₂ (Fleming, 1998). The invisible gold and insoluble gold minerals are converted to gold that is soluble in a cyanide solution, and the native gold and electrum remain in the residue as soluble gold minerals. The gold that was encapsulated in the sulfides and arsenopyrite is now in the permeable hematite and can come in contact with the cyanide solutions and be dissolved. The roasting may burn off the graphite and carbonaceous material and eliminate the preg-robbing characteristics of the ore. In some instances, however, residual carbon might be left and sometimes it may be a more active variety than the original one in the ore. A major drawback of roasting is that it is difficult to condense the volatile arsenic trioxide and to filter it from the off-gas (Petruk, 2000).

1.4.6 Flotation

Gold is generally recovered by flotation when it is a by-product in sulfide ores, as in porphyry copper, base metal and copper-gold ores. The gold is generally recovered in the copper concentrate, and is recovered from the copper concentrate by smelting and electrolysis. Gold tends to float readily in sulfide flotation cells, particularly in copper flotation cells. Hence all the liberated gold and much of the

unliberated but exposed gold is recovered in copper concentrates. Much of the attached gold is recovered, particularly in the rougher concentrate, even if it is attached to other minerals, such as pyrite. Unfortunately some of the attached gold that is recovered in the rougher cells may subsequently be lost in the cleaning stages, because the attached gold grains may be too small to maintain flotation of the particles. However, a significant amount of the gold commonly remains in the copper concentrate, hence much of the gold lost to flotation tailings is encapsulated gold grains (Petruk, 2000).

At some operations, as at the Kutema gold mine in Southern Finland, gold is recovered by flotation to produce a sulfide concentrate that can be smelted to recover the gold. The Kutema gold ore consists of disseminated banded to massive pyrite with various tellurides and minor base metal sulfides, arsenides and sulphosalts in a sericite-quartz schist. The gold occurs as inclusions in the pyrite, arsenopyrite and quartz, intergrown with tellurides and as free grains. About 82 to 87 % of the gold is recovered (Kojonen et al., 1999).

In some cases flotation is used as a scavenger to recover the gold that was not recovered by cyanidation because the gold was encapsulated in the sulfides (pyrite and chalcopyrite) and did not dissolve during cyanidation. In one case, which cannot be identified because of company confidentiality, the gold occurred in quartz veins and was associated with pyrite and chalcopyrite. The ore was crushed, ground and cyanided. About 85 % of the gold was recovered by cyanidation. A flotation circuit was installed to process the cyanidation tailings to recover more gold. The cyanidation tailings were reground and a copper concentrate, grading around 22% Cu, recovered around 30 % of the gold in the tailings. A study of the products in the flotation circuit showed that some relatively large gold grains, attached to pyrite, were lost to the tailings.

The flotation practice was changed to recover particles containing exposed gold attached to pyrite. The grade of the copper dropped to 17% Cu, but the gold recovery was increased by 8%. The smelter accepted the copper concentrate. In other cases

flotation is used prior to cyanidation to either pre-concentrate the ore, or to remove minerals such as secondary copper minerals from the cyanidation circuit (Petruk, 2000).

Flotation is used to pre-concentrate the ore only when there is a high recovery of gold in the pre-concentrate. If the pre-concentrate does not recover most of the gold, all the ore is processed by cyanidation.

1.4.6.1 Collectors in Gold Flotation

Gold hydrophobicity is enhanced by the addition of flotation collectors and no flotation plant relies solely on the natural floatability of gold for its recovery. Naturally occurring or free (liberated) gold is optimally recovered in a flotation circuit at natural or near-natural pulp pH values and with the addition of small amounts of collector. Inherently, naturally floating minerals float fast kinetically (Klimpel, 1999). Flotation tests on placer gold showed that fine placer gold typically floated readily with common sulfhydryl collectors and common frothers at natural pH without the addition of any special regulating reagents. Gold flotation recoveries ranged from 78 to 99%.

Flotation with xanthate collectors involves the anodic oxidation of the collector that may involve sub-processes such as metal xanthate formation, chemisorptions 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. This is a neutral oil that will adsorb onto the surface of any naturally hydrophobic solid, rendering it floatable (Gardner & 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. The longer-chain xanthates are more readily oxidized, generating

dixanthogen at lower potentials (Gardner & Woods, 1974). An increase in thiol chain length increases the maximum contact angle, thereby increasing the hydrophobicity of the surface species. Both these attributes favor 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. 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. As a consequence, the flotation of gold–silver alloys can be achieved at potentials considerably lower than that for gold. Xanthate ions chemisorbs on silver at potentials below the region at which silver xanthate deposits. 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 could be important in retaining the dixanthogen at the gold surface through hydrophobic interactions between the adsorbate and the bulk phase.

The xanthogen formates are produced by reacting alkyl chloroformate with xanthate salts. They are stable in acidic conditions unlike the xanthates from which they are formed and are stable in the pH range of 5–10.5. The formates appear to have superior pyrite rejection properties compared to xanthates and dithiophosphate (Ackerman et al., 2000).

Dithiophosphates are useful secondary collectors (sometimes referred to as promoters) to xanthates in gold flotation. It has been known for a long while that Aeropromotor 208 is an effective promoter in gold flotation. 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 and are able to recover gold selectively from some sulfide ores. Silver has been shown to assist adsorption of discredyl monothiophosphate onto gold. 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 (Dunne, 2005).

Mercaptobenzothiazole (MBT, Aeropromoter 404) is a fairly specialized collector and is the preferred collector for the flotation of gold and gold-carrying pyrite in acid circuits. 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.

The phosphine-based collector (Aerophine 3418A) has found application in flotation of silver and silver sulfides. 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. The dicresyl monothiophosphinate and the diisobutyl monothiophosphinate have been found to increase gold recovery significantly from either primary gold ores or gold-containing tailings when used in combination with standard thiol collectors. In the presence of silver, adsorption of the di- and monothiophosphinate was demonstrated by the formation of the corresponding silver complexes (Dunne, 2005).

Amine-based collectors have been used to float gold and gold-bearing pyrite. The application is limited as the amine collector is selective for pyrite at high pH values (>10) only. Industrial-scale application of amine collectors are recorded at Venterpost Gold mine in South Africa and at the Kerr Addison Mine in Canada where the sand fraction from the deslimed cyanide-leach tailing was floated and the concentrate was roasted (Ramsay, 1978).

Table 1.5 List of specific and blended collectors used for gold flotation (Cytex Industries Inc., 2002)

Collector composition – individual collectors	Cytex reagent details
Sodium <i>isobutyl</i> xanthate	AERO 317
Potassium amyl xanthate	AERO 343
Xanthogen formate formulation	AERO 3758
<i>Diisobutyl dithiophosphate</i>	AERO 3477
Mercaptobenzothiazole	AERO 404
<i>Monothiophosphate</i>	AERO 6697
<i>Dithiophosphate</i> formulation	Reagent S-9810
Collector composition – blended collectors	
<i>Dithiophosphate/monothiophosphate</i> formulation	AERO 7249
<i>Monothiophosphate/dithiophosphate</i> formulation	AERO 8761
<i>Dithiophosphate/monothiophosphate/dithiophosphate</i> formulation	Reagent S-9913
<i>Dithiophosphate/mercaptobenzothiazole</i> formulation	AERO 405
<i>Dithiophosphate/mercaptobenzothiazole</i> formulation	AERO 7156
Thionocarbamate/ <i>dithiophosphate</i> formulation	AERO 3926
Thionocarbamate/ <i>dithiophosphate</i> formulation	AERO 473
Modified thionocarbamate/ <i>dithiophosphate</i> formulations	AERO 5744/5
<i>Dicresyl dithiophosphate</i> formulation	Reagent S-8985
Modified thionocarbamate/ <i>dithiophosphate/monothiophosphate</i> formulation	Reagent S-9889
Ethyl octyl sulfide formulation	Reagent S-701
<i>Dithiocarbamate/sodium hydrosulfide</i> formulation	Reagent S-3730

1.4.6.2 Frothers in Gold Flotation

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.

1.4.6.3 Activators in Gold Flotation

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 increased and selectivity improved (Dunne, 2005).

Early work on gold particles with copper sulfate showed no improvement in recovery but an increased rate of flotation of gold. 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. The reason for improved flotation recovery and rate is not understood as the mechanism of surface activation, if it exists, is different from that for sulfide minerals (Allan & Woodcock, 2001).

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 & Kushnikova, 1959), arsenopyrite (Gegg, 1949; O'Connor et al., 1990) and pyrite (Bushell & Krauss, 1962).

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. 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).

Lead nitrate or acetate is often used for the activation of stibnite in preference to copper sulfate (Oberbilling, 1964). The reason appears to be price related, lead salts being cheaper, as copper sulfate has been shown to be the superior activator for many stibnite ores. At the Three Mile Hill Gold Mine in Western Australia, lead nitrate was added as an activator to assist in the preferential flotation of arsenopyrite from

pyrrhotite, while lead acetate was used at the Surcease Mine in California to produce a bulk sulfide concentrate for roasting. The recommended activator to float arsenopyrite in the nitrogen-based N2TEC process is lead nitrate (Simmons et al., 1999).

1.4.6.4 Sulfidization

The application of sulfidizers (sodium sulfide and sodium hydrosulfide) to enhance the flotation of oxidized ores is well known (Jones & Woodcock, 1984; Oudenne & de Cuyper, 1986; O'Connor & 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 & 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 & 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 & Yarar, 1989). The addition of sulfide ions converts some coatings on mineral surfaces in sulfides (Healy, 1984) and subsequent xanthate addition will promote flotation. For successful activation, the sulfide activator should be added slowly and at starvation quantities.

1.4.6.5 Depression of Gold in 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 & House, 1992; Lins & Adamian, 1993; Allan & 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 & Yasar, 1989). However, flotation of native gold often proceeds satisfactorily in the presence of many of these compounds. In general, the results reported by different authors are not in good agreement (Allan & Woodcock, 2001). It is likely that other components in solution or on the surface of the gold that were not measured provide the answer for the different outcomes.

Lime cannot be considered as just a pH modifier and studies have shown that calcium is strongly adsorbed on sulfide minerals and gold at pH values at and above 10 (Healy, 1984; Chryssoulis, 2001). This adsorption is enhanced if excess sulfate in the pulp promotes calcium-sulfate coatings on particles. Desorption of calcium from the surface by reducing the pH can be assisted by the use of specific calcium-complexing ions such as polyphosphate. Furthermore, if the calcium release is attempted while adding excess activator, then a hydrophilic hydroxide coating can result (Healy, 1984). Metals ions introduced from the circuit water, or from soluble metal ions in the ore, may adsorb and nucleate as hydroxide coatings on all particle surfaces, thus inhibiting collector adsorption. The recommended method of flotation treatment (Healy, 1984) is to operate at as low a pH value as practical, avoid rapid increases in pH, add activator slowly or condition separately and keep the tailings dam at a pH of minimum solubility.

1.4.6.6 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 & 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).

1.4.6.7 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; Askoy & Yasar, 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 & Woodcock, 2001).

1.4.6.8 Electrical Double Layer

The electrical double layer that forms at the mineral–solution interface is generated by the presence of potential-determining ions in the mineral–solution (pulp) system. The electrical double layer is important because collector, activator and depression adsorption depend on this, as does the attachment of some particles to bubbles. The sign and magnitude of the surface charges on the species in the system are important issues in the flotation system. A high surface charge on a mineral surface will inhibit the chemisorption of a collector. The dispersed and flocculated state of a mineral pulp is also controlled by the electrical double layer (Dunne, 2005).

1.4.7 Gravitational techniques

Gravity concentration is one of the oldest of all forms of mineral processing. Along with hand picking it remained the primary tool in the mineral processors arsenal for most of the last 2,000 years. However, with the advent of flotation and other processes the interest in gravity concentration declined. Little more than thirty years ago, young mineral processing engineers would not, willingly, have specialized in gravity concentration (Burt, 1999).

Gravity concentration techniques are used to supplement cyanidation and flotation techniques for recovering gold (Laplante et al., 1996), particularly when the ore contains gold nuggets that are too large to attach to bubbles in flotation cells, and too large to be dissolved completely during cyanidation. Evidence of gold nuggets in an ore is usually found by their presence on the crusher plates, grinding mill liners, grinding pump boxes and sumps. The gravity circuit in most gold concentrators is placed ahead of the cyanidation circuit, and a high grade gold concentrate is usually recovered. The Golden Giant Mine of Hemlo Mines Inc. provides an example of a gravity circuit ahead of the leaching circuit. The gravity circuit consists of Knelson concentrator and shaking tables (Honan & Luinstra, 1996), and recovers a gold concentrate that contains about 75% Au.

Jigging is a form of gravity concentration carried out by pulsing water through a screen on which lies a bed of crushed (and preferably sized) ore. The bed is alternately dilated and compacted by the pulsed flow of water through the screen so that the heavier, smaller particles penetrate the interstices of the bed and the larger high specific gravity particles fall under a condition probably similar to hindered settling. In the jig the pulsating water currents have a harmonic wave form each jig cycle being composed of pulsation (upward flow) and a suction (downward flow) stage (Hoşten, 2002).

Conventional jigs are often used to recover heavy minerals that are liberated at a coarse particle size from crushing/grinding circuits, thus avoiding subsequent over-grinding and loss. Centrifugal jigs use enhanced forces generated by their spinning motion to enable finer particle sizes and closer specific gravity (SG) minerals to be separated. The Kelsey jig is the most common example of this type of separator (Angove, 2005).

Spirals are one of the oldest gravity separators. There is a wide range of profiles available including low-grade, medium-grade, high-grade and fine mineral models, plus ones incorporating different wash water techniques. Careful monitoring and control of size distribution is important in achieving optimum results with spirals (Angove, 2005).

The MGS is a low-capacity high-performance gravity separator suitable for treating difficult fine particle feeds below 75 mm (Angove, 2005).

Falcon and Knelson concentrators are centrifugal type gravity separators also suited to fine particle-size feeds. These units come in batch and continuous configuration for both laboratory testing and operational application (Angove, 2005).

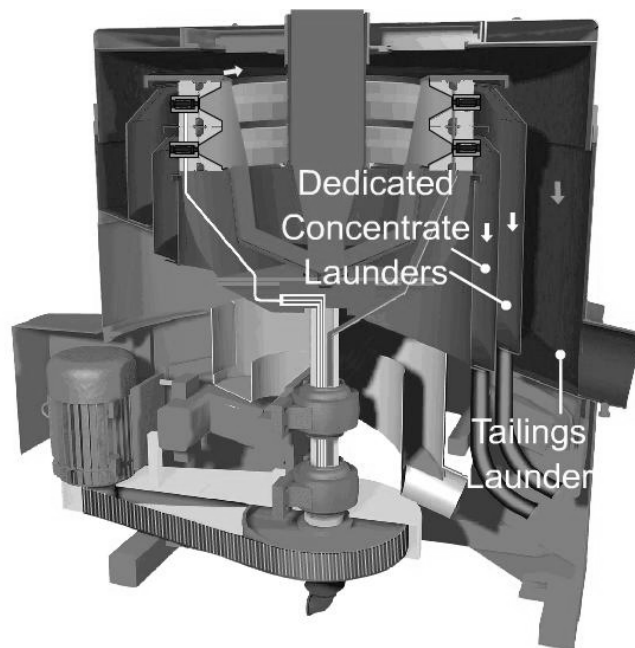


Figure 1.2 Schematic view of Knelson concentrator (Knelson, 2010).

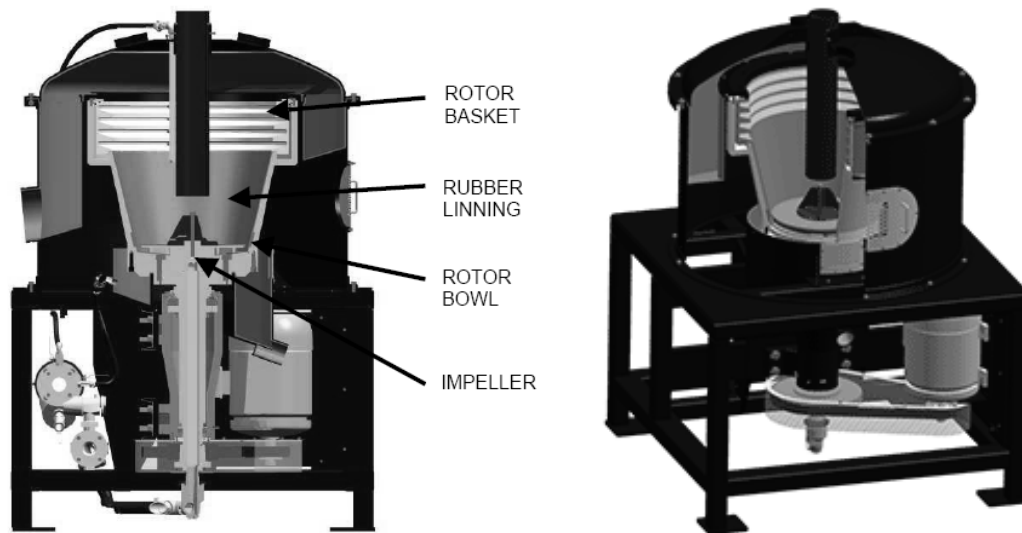


Figure 1.3 Schematic view of Falcon concentrator (Falcon, 2010).

1.4.8 Amalgamation

Although mercury was known to the ancient Chinese and Hindus, and has been found in Egyptian tombs dating back to 1600 BC, it was extracted and used only in the Roman times. Mercury ores occur in abundance in Italy and Spain. The Roman writer, Dioscorides mentioned its preparation from cinnabar. Pliny gave a method of purifying mercury by squeezing it through leather. He described the amalgamation process and introduced the term amalgam, from malagma (meaning I soften, because mercury softens gold).

Although the Romans were acquainted with the fact that mercury dissolves gold and silver, it does not appear that they applied this knowledge to the extraction of these metals from their ores. Vanoccio Biringuccio mentioned the amalgamation of ores in his book, *De la Pirotechnia*, published in 1540, as does Georgius Agricola in his book *De Re Metallica*, published in 1556. Mercury dissolves gold rapidly at ambient temperatures; an amalgam containing 10% gold is liquid, that containing 12.5% gold is pasty, and that containing 15% gold is solid (Habashi, 2005).

Stamp mills were closely associated with the amalgamation process. A stamp mill is a grinding machine where the grinding action takes place by falling weights. The ore from the crushers varying in size from 10–60mm is fed with water in the stamp mill. A stamp is a heavy iron pestle 3–5m long and about 7 cm in diameter carrying a weight 200–400 kg. The stamp is raised by a cam that is keyed on to a horizontal revolving shaft and allowed to fall by its own weight. Mortars are made of cast iron about 1.2m long, 1.2m high and 0.3m wide. The height of the drop of the stamp varies between 30 and 50 cm and the number of drops per minute varies from 30 to 100. Five stamps are usually present in one mortar. Fine powdered ore slurry containing native gold and silver is then passed over copper plates amalgamated with a thin layer of mercury onto which the noble metals adhere. Depending on the gold content of the ore, the amalgam is scraped once or twice a day and a fresh mercury surface is exposed. About 3–10 t of ore per square metre of mercury surface are usually employed, i.e. a mercury consumption of about 30–50 g/t ore. The amalgam is washed with water to remove any attached gangue particles, then pressed to remove excess mercury. Amalgam containing 40–50% gold can be obtained. It is charged on trays and heated in a horizontal retort to distil the mercury. A gold sponge is obtained, which is then melted with fluxes then cast as a bullion. Gold bullion contains some silver, copper and other metals, which are separated at mints or private refineries (Habashi, 2005).

The amalgamation process was used extensively for the recovery of gold and silver from their ores. In many countries it is now illegal to use this process because of the toxicity of mercury. In spite of this, the process is used on a large scale by numerous small illegal operators in several parts of Africa and by the so-called garimpos in the Amazon basin in Brazil. This practice has resulted in the mercury pollution of soil, rivers and also the atmosphere because the final step of gold recovery is usually done by heating with an oxygen flame in the open air to remove the mercury, a hazardous procedure (Habashi, 2005).

CHAPTER TWO

GEOLOGY, MINEROLOGY AND CHARACTERIZATION OF EFEMÇUKURU GOLD DEPOSIT

2.1 Geology of Efemçukuru District

The Efemçukuru project is located at the western end of the Izmir-Ankara Suture Zone, a major regional structure that extends northeast and then east from Izmir for almost 800 kilometers. The Izmir-Ankara Suture Zone marks the closure point of a subduction zone that separated the Sakarya and Anatolide-Tauride microplates plates during the late Cretaceous and early Paleocene Age. As the subduction zone closed, Neo-Tethyan sea floor between the two microplates was abducted onto the Anatolide-Tauride plate. Lenses of serpentine often associated with thrust faults and large olistoliths of recrystallized limestone were caught up in the melange-like complex that formed during the suturing process. Regionally extensive volcanism and intrusive activity were also associated with the subduction process. Subsequent mid-Tertiary dilation in western Turkey resulted in block faulting and the formation of the north-south orientated Seferihisar horst. The Efemçukuru project is situated in the central part of the Seferihisar horst. Younger Neogene sediments and volcanics fill the flanking graben structures (Eldorado Gold Corporation, 2006).

2.1.1 Local Geology

The immediate project area is comprised of a late Cretaceous to Paleocene-age volcano-sedimentary sequence, which has been regionally metamorphosed to greenschist facies. Intermediate to mafic submarine volcanics and interbedded mafic sediments (schist) in the northeast corner of the project area grade southward and westward into phyllites. Granitic intrusive reportedly outcrops in a restricted military radar station located approximately 3.5 kilometers north of the deposit area. The age of the granite is unknown; however, it is assumed to be early Tertiary in age and is probably subduction related granite (Eldorado Gold Corporation, 2006).

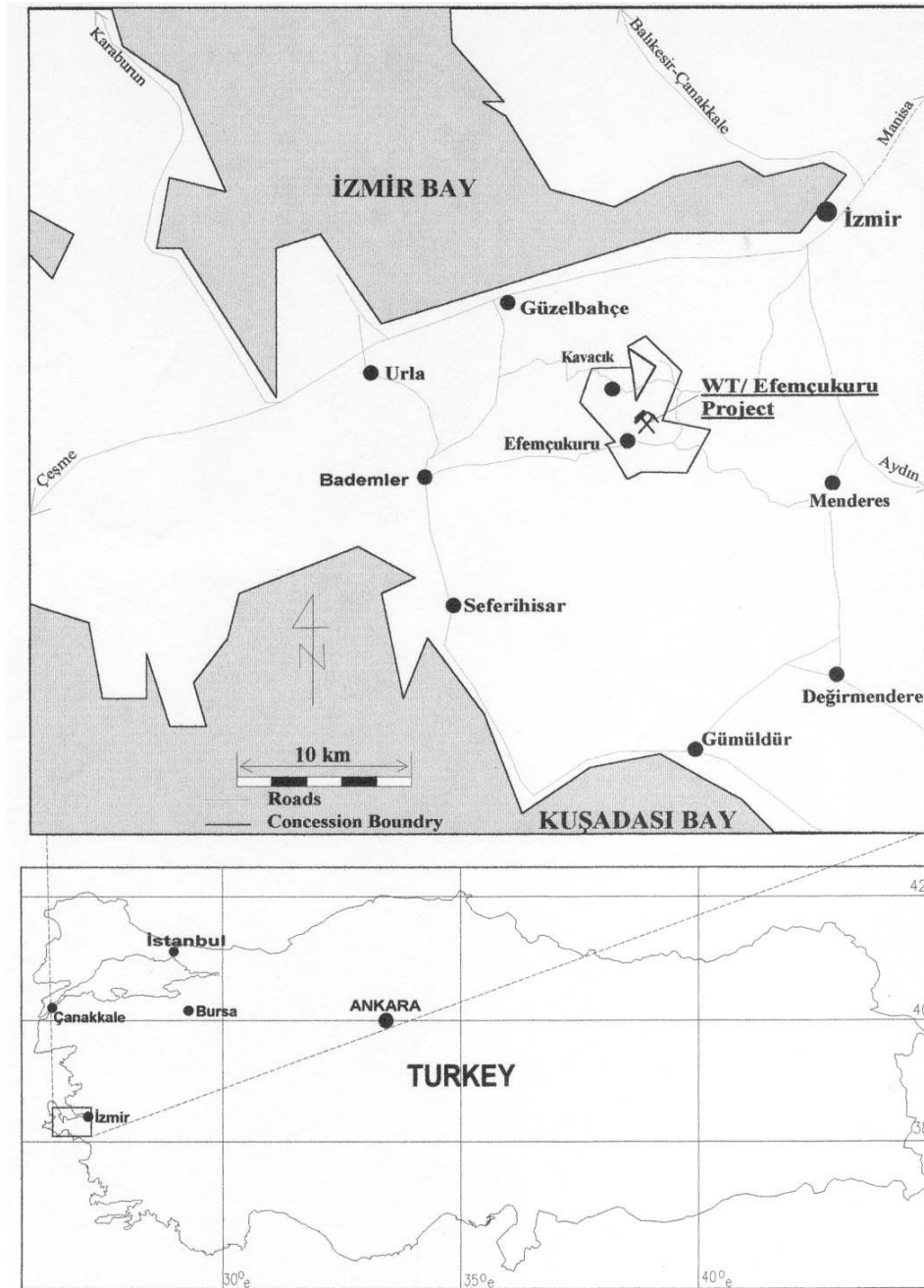


Figure 2.1 Location of Efemçukuru District (Eldorado Gold Corporation, 2006).

In the immediate deposit area, rhyolite dikes outcrop in and around a zone of hornfels altered phyllites. The rhyolite dikes are typically 1-2 meters wide and outcrop intermittently over distances up to 2,500 meters. The emplacement of the rhyolite dikes appears to have been controlled by west-northwest trending strike slip faults. The rhyolite dikes are thought to be the surface expression of a deeper intrusive body, which may be related to the granite beneath the radar station.

Phyllites and hornfels are the primary host rock for mineralization on the property. Where unaffected by hydrothermal alteration, the phyllites are typically soft, fissile, and have a well-developed foliation. Fractures in the phyllite are locally filled with thin metamorphic quartz-microcline veinlets. The phyllites were strongly deformed during regional tectonic events. Foliation strike and dip directions change quickly over short distances. The center of the deposit area is comprised of silica and calcsilicate altered phyllites referred to as hornfels. Thin-sections indicate that the hornfels consists of a mixture of silica, epidote, tremolite and actinolite, with traces of pyrite and pyrrhotite. The hornfels often forms bold rounded outcrops with only thin soil cover. Relict foliation textures are present in much of the unit, hence it is not a true hornfels, and however, the term has been retained because of its practical value as a field term (Eldorado Gold Corporation, 2006).

2.1.2 Infrastructure

Efemçukuru is located in hilly terrain in an area known as Teke Dagi. Relief extends from sea level to approximately 1000 meters; the project site is situated at between 575 and 700 meter elevations. Slopes in the area range from 2.5:1 to 1.5:1 with agriculture being carried out on slopes of 2.5:1 or 1 less.

The Teke Dagi area is populated by small, rural villages. The inhabitants rely primarily on agriculture with an annual cash crop of grapes for support. The project license centers on the Kokarpinar Valley covering predominantly forest lease with some private land holdings.

The climate of Efemçukuru is typically Mediterranean with hot, dry summers and cool, wet winters. Temperatures range from highs of 30°C in the summer to lows of 0°C in the winter months. The seasonal rains occur between November and April, with annual precipitation of approximately 740 mm (Eldorado Gold Corporation, 2006).

2.1.3 Vein System and Ore body

Efemçukuru, located in the Aegean Region, Turkey, is an example of vein-type epithermal gold deposit with related stockwork and replacement mineralisation. The veins are hosted by Late Cretaceous–Paleogene flysch facies rocks of the Izmir–Ankara zone, which were intruded by rhyolites from of Neogene volcanism. Gold mineralisation is associated with late pulses of magmatic hydrothermal activity and is present in zones of hydraulic brecciation adjacent to veins or stockwork zones around a dome-shaped small intrusion. The intrusion and later epithermal mineralisation are both controlled by NW–SE trending faults, and mineral deposits occur along strike with dips 60° to 80° to the northeast. Individual quartz veins associated with sulphide minerals and mineralized hornblende facies hornfels are other important ore-bearing formations. The alteration associated with mineralization is represented mainly by rhodonite, rhodochrosite, axinite, quartz, calcite and adularia in veins and stockwork zone and chlorite, sericite, illite and kaolinite in the wall rock. Homogenization temperatures of 200–300°C reflect emplacement of ore distant from inferred magmatic heat in epithermal environment. Due to wide range in salinity of the fluids, sulphide diversity in the deposit is rather rich than the low-sulphidation epithermal systems. The fluid inclusion data indicate that a complicated geothermal system existed. Both the gas data and the microthermometry data indicate that there was fluid mixing. Each type of mineralization has been studied with the aim of clarifying the paragenetic relationships among the different minerals (Oyman et al., 2003).

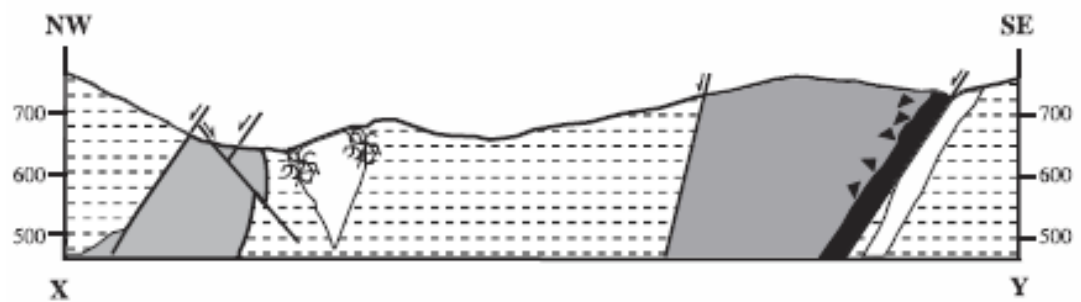
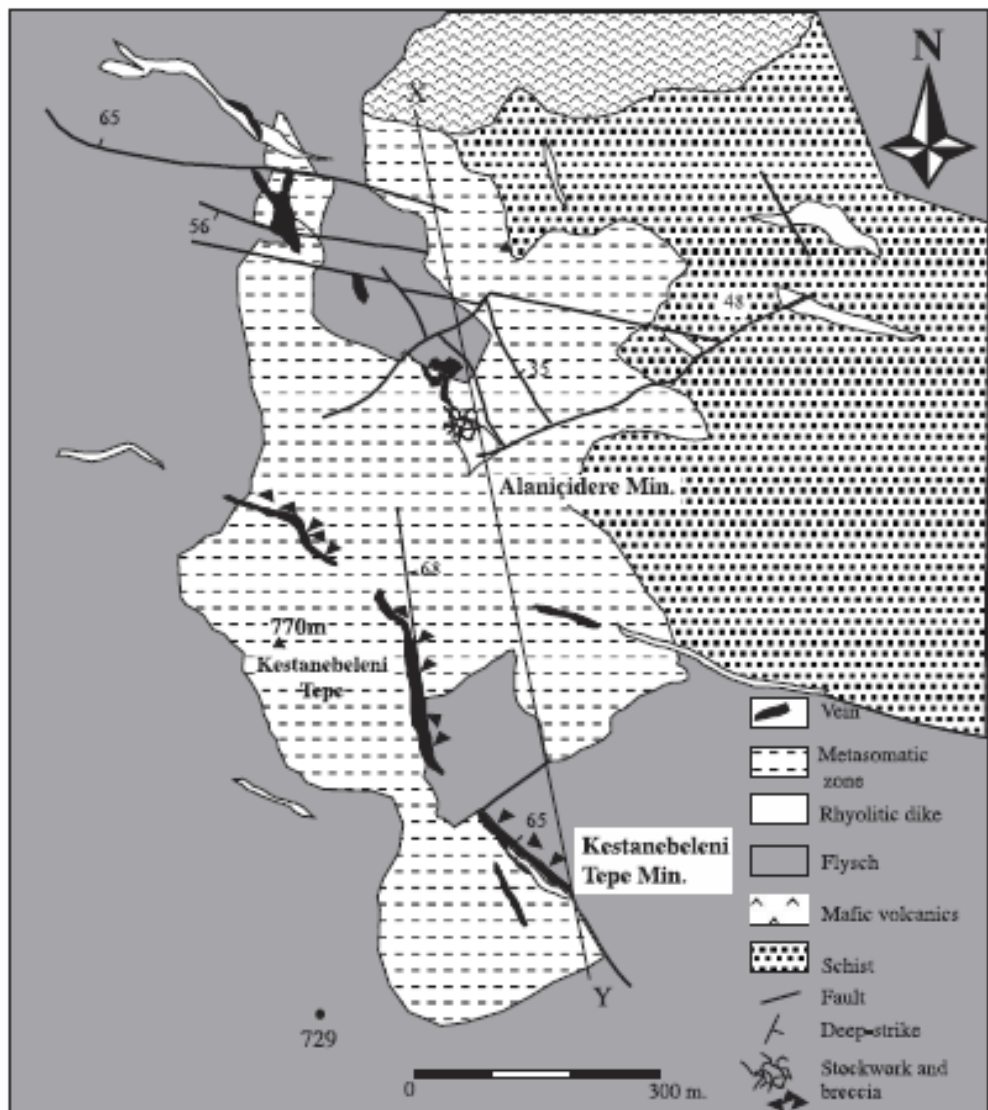


Figure 2.2 Geological map and cross section of ore body positions of Efemçukuru area (Oyman et al., 2003)

Mineralization due to intense hydrothermal activity represented by a vein system associated with multistage brecciation, stockwork zone, alteration and polymetallic precious metal deposition. The grade of the gold mineralization has been enhanced by secondary enrichment in the top 100 m of the vein. Results from the drill programs indicate that the ore body extends to a depth 300 m and remains open. The homogenization temperatures indicate deposition of ore in epithermal environment distant from a buried magmatic heat source (Oyman et al., 2003).

The vein system is emplaced along a NW–SE oriented main fault system that the rhyolitic intrusions crop out. Efemçukuru was discovered by Turkish Geological Survey (MTA) in the 1980s during the regional precious metal evolution of the base of old diggings and quarries in Western Anatolia. Following the drilling and sampling program, MTA turned over the rights of the area to “Tüprag Metal Madencilik San. ve Tic.”, a wholly owned subsidiary of Eldorado (Oyman et al., 2003).

2.1.4 Gold in Epithermal Systems

The Ovacik, Efemçukuru, Kucukdere, Agi Dagi and Sahinli deposits are associated with subaerial volcanic rocks in W and NW Turkey. The Ovacik Au–Ag deposit, located along the northern boundary of the ENE-trending Bergama Graben, is a typical example of a low-sulfidation vein deposit hosted in andesite porphyry of Early Miocene age. East-trending and NW-trending banded-quartz veins, known as M and S, respectively, dipping steeply north are the only economical veins. Higher Au grades with in excess of 100 g/t Au are associated with colloform to crustiform banded quartz–adularia veins and late stage breccias. Gold is mostly Au-rich electrum (Yigit, 2004).

The Efemçukuru, low-sulfidation epithermal Au deposit is hosted by rhyolite domes of Neogene and flysch facies rocks of Late Cretaceous to Paleogene age. This is a boron-rich epithermal system that consists of two major veins, Alanicidere and Kestanebeleni, and numerous late-stage small sulfide-rich veins. NW-trending faults

control not only vein-style mineralization with associated stockworks, breccias and replacement mineralization, but also act as loci for rhyolitic intrusions. Gold mineralization is associated with at least two distinct hypogene events: an early metasomatism and a later hydrothermal alteration and mineralization event consisting of least three stages. However, Au grades are enhanced as a result of supergene alteration in an oxidized zone up to 100 m depth (Oyman et al., 2003).

Fluid inclusions from quartz samples show a broad range of homogenization temperatures between 200 and 300°C, and salinities of 0 to 8 wt. % NaCl equiv., indicating the existence of a complicated geothermal system (Oyman et al., 2003). Alternating quartz, rhodonite and axinite bands with lesser amounts of calcite, rhodochrosite and adularia can be seen in some veins, which may reflect geochemistry of skarnified host rocks (Yigit, 2004).

CHAPTER THREE

EXPERIMENTAL STUDIES

3.1 Sampling and Size Reduction

The case of sampling is to prepare a sample that represents the bulk sample in all aspects, physical, chemical and mineralogical. This bulk sample should be subjected to size reduction for better mixing, proper sampling and reasonable size for treatment.

3.1.1 Sampling from the Efemçukuru Area

The sample gathering procedure from the Efemçukuru gold deposit is shown in figure 3.1 and 3.2. There were some difficulties in sampling since the mining company was not fully utilized and initiated working. The 850 kg sample was taken from the outcrops of old drilling sites.



Figure 3.1 Sampling from Efemçukuru gold deposit site



Figure 3.2 Outcrops of gold veins in one of the drilling sites in Efemçukuru

3.1.2 Size Reduction – Crushing

The size of the sample gathered was around 50 cm in diameter. Around 640 kg of the sample was crushed gradually below 3.35 mm (figure 3.3). The remaining sample was stored for achieves and further development of the processing

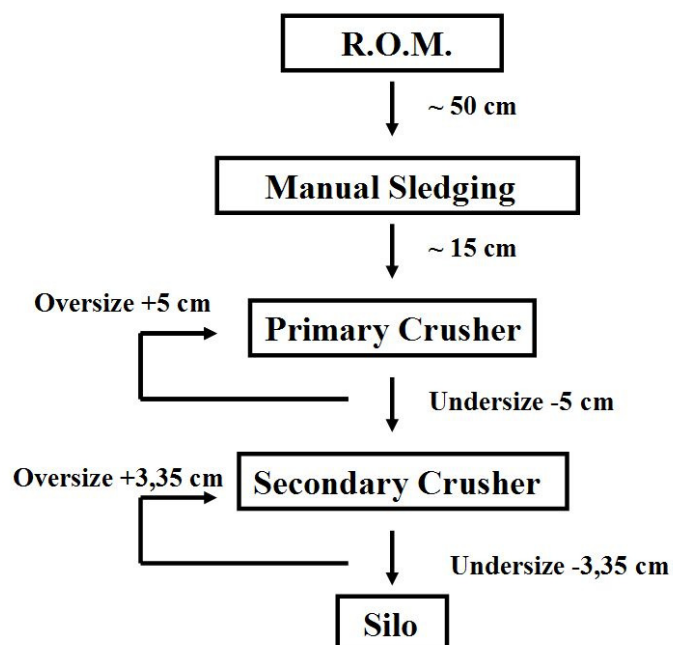


Figure 3.3 The flowsheet representing the crushing procedure of the sample

Different sizes (gap) of jaw crushers were used for primary and secondary crushers. The laboratory type crushers are shown in figure 3.4 and 3.5.

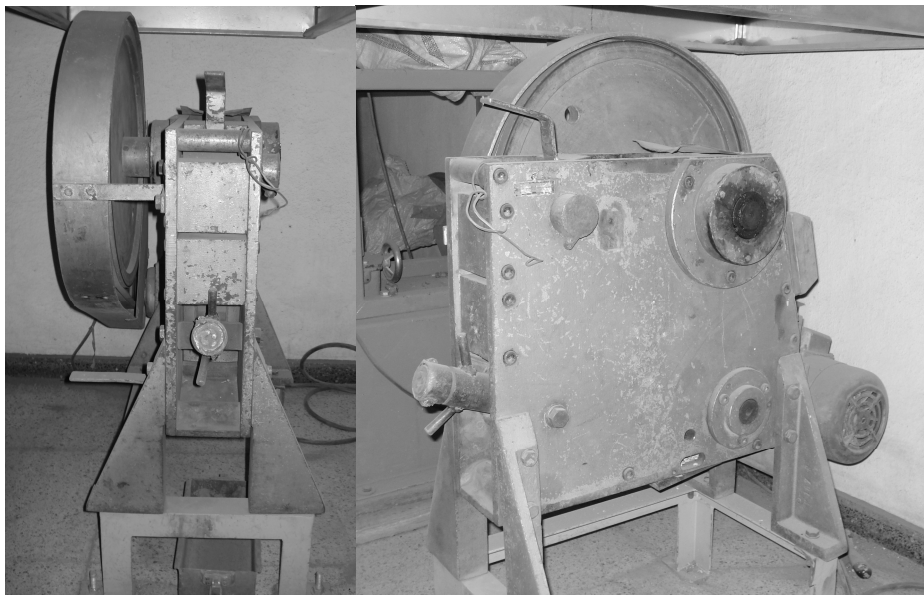


Figure 3.4 The primary jaw crusher used during size reduction and its gap

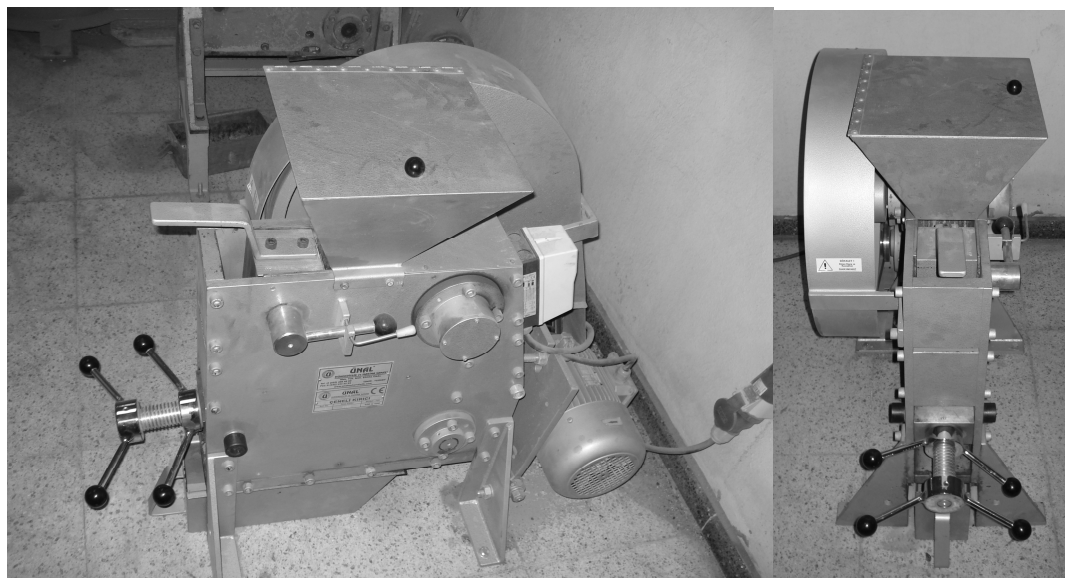
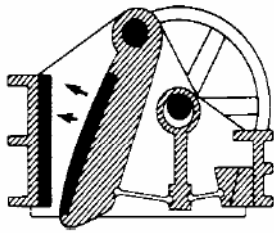


Figure 3.5 The secondary laboratory type jaw crusher used during size reduction



Jaw crusher (Blake-type, double-toggle):

Size: 125mm (gape*)x150mm to 1600x2100mm

Power: 2.25 to 225 kW

Preferred Reduction ratio: 7:1 (range 4:1 to 9:1)

Recommended product top size: 1600 to 100 mm

Figure 3.6 The schematic view and properties of a jaw crusher (Hoşten, 2002).

3.1.3 Subsample Preparation

The crushed pile of 640 kg's was divided by cone & quartering method (figure 3.7) and subsamples of 80 kg are formed. In this method, the material is flattened in the shape of a cone and we use a cross to separate the material into four parts. Two opposite quarters are taken and another cone and quartering is applied on these parts. The process continues till we have the desired amount and the size of the bulk material to consider as the whole sample. The efficiency may be poor if we apply this technique on small sized samples.

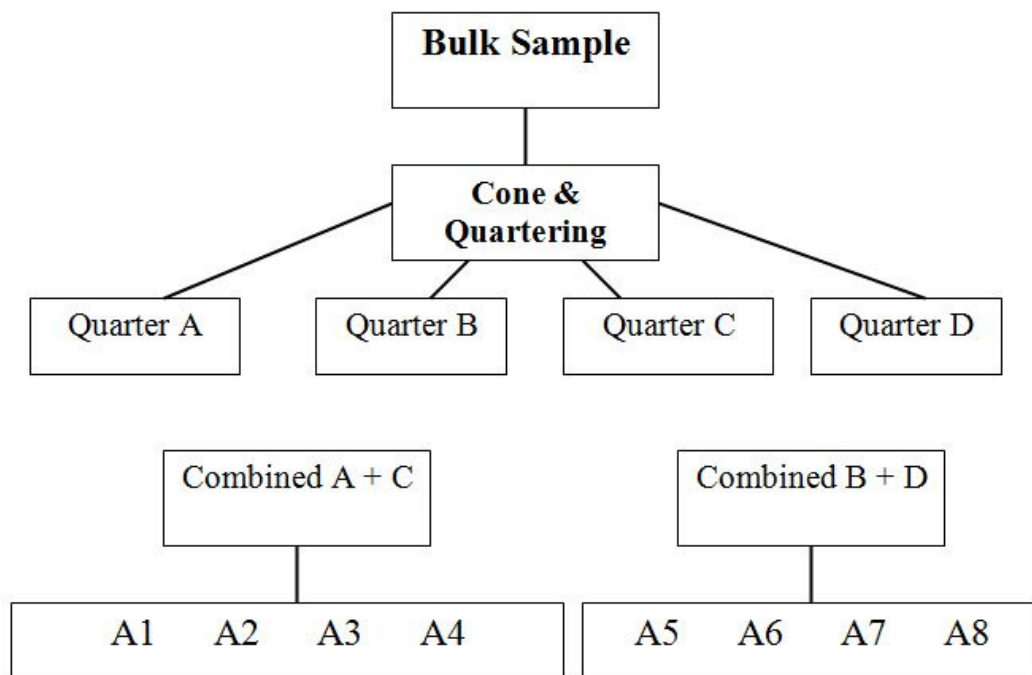


Figure 3.7 The flowsheet representing cone & quartering method.

The cone & quartering method was applied again to the subsamples of 80 kg to form subsamples of 10 kg. The subsamples of 10 kg's are divided into 8 by chute riffler method (figure 3.8) to form subsamples of 1250 g. The concept of using the chute riffler is separating the material into two nearly equal parts by make the material pass through the V shaped perspex of the riffler. This equipment is useful when it is used to split head samples of size down to less than one kilogram. The subsamples of 1250 grams are stored for the further experiments

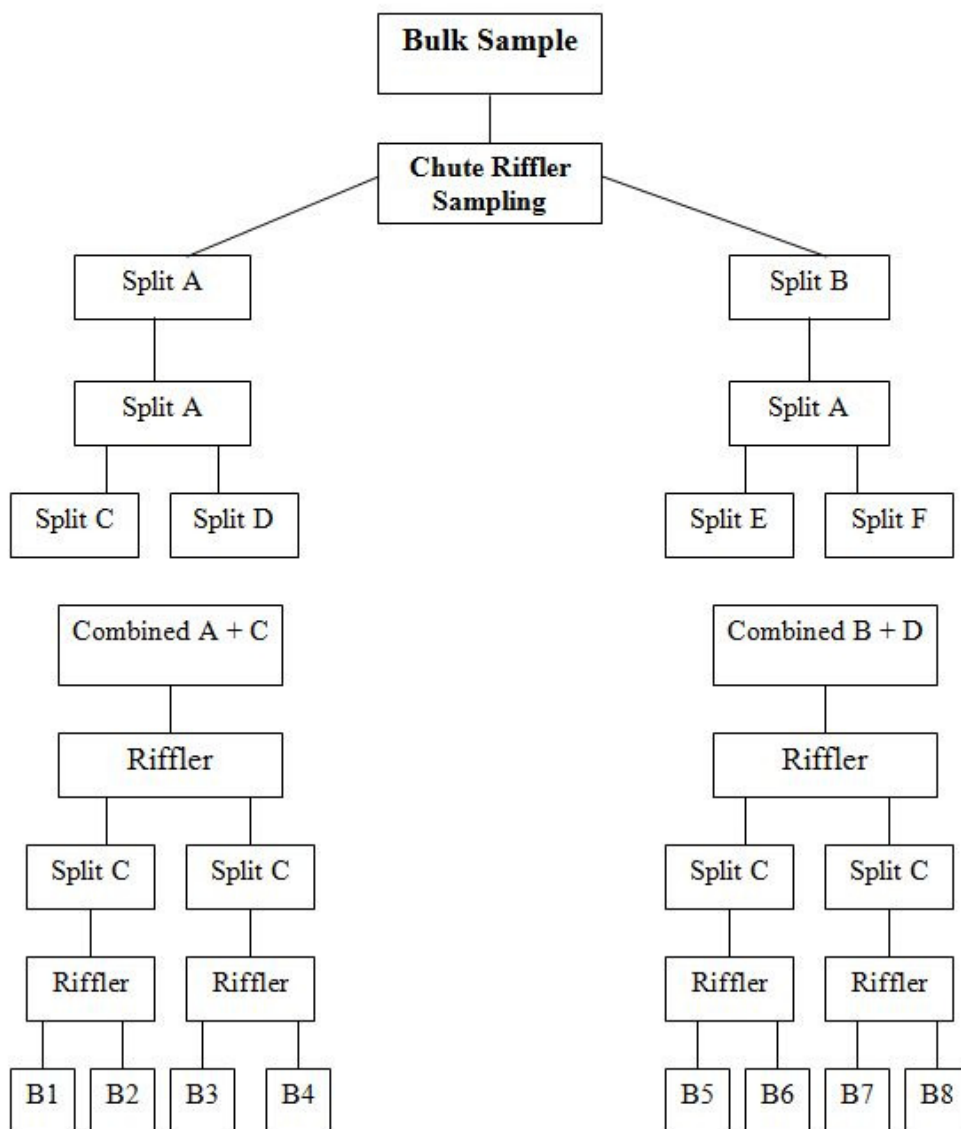


Figure 3.8 The flowsheet representing chute riffler method.

3.2 Characterization of the Sample

Efemçukuru, located in the Aegean Region, Turkey, is an example of vein type epithermal gold deposit related stockwork and replacement mineralization (Oyman et al., 2003). Arsenopyrite, pyrite (marcasite), sphalerite, galena, chalcopyrite are the main minerals with pyrrhotite, fahlore group, sphene, rutile and gold occurring as accessories. Pyrolusite, limonite (goethite, lepidocrocite) covellite-chalcocite, malachite, azurite are of secondary origin. The chemical characterization studies using atomic adsorption spectrum (AAS) had shown that the ore sample contained on the average of 6.7 g/t Au, 50 g/t Ag and 0.028% Cu, 0.26% Pb, 0.053% Zn. The results are shown in the table 3.1 below. In this study, characterization of ore minerals on polished sections has been done using the polarizing reflected light microscope which is the standard and widely used for the identification of the ore minerals.

Table 3.1 The chemical characterization of the Efemçukuru ore sample by AAS.

Element	Au	Ag	Cu	Pb	Zn	Ni	Fe ₂ O ₃	Mn
Grade (g/t), %	6.7	50.2	277.1	2592.3	534.0	94.7	5.67 %	2.8 %

Gold in the ore sample was observed in sulfides mainly as globular specks or tiny blebs (figure 3.9). The grain size of gold commonly varies between 5 to 30 μm . Pyrite is the most important sulfide phase hosting the gold. The grain size of pyrite is important in terms of hosting gold grains. Gold is preferentially hosted by cubic pyrite grains with a relatively coarse grains (<50 μm) (figure 3.9 A, B and C). The predominant occurrence of gold is electrum with lesser amounts of native gold. Electrum contains 61-69 wt% Au and 23-30 wt % Ag. Gold is also associated with tellurides.

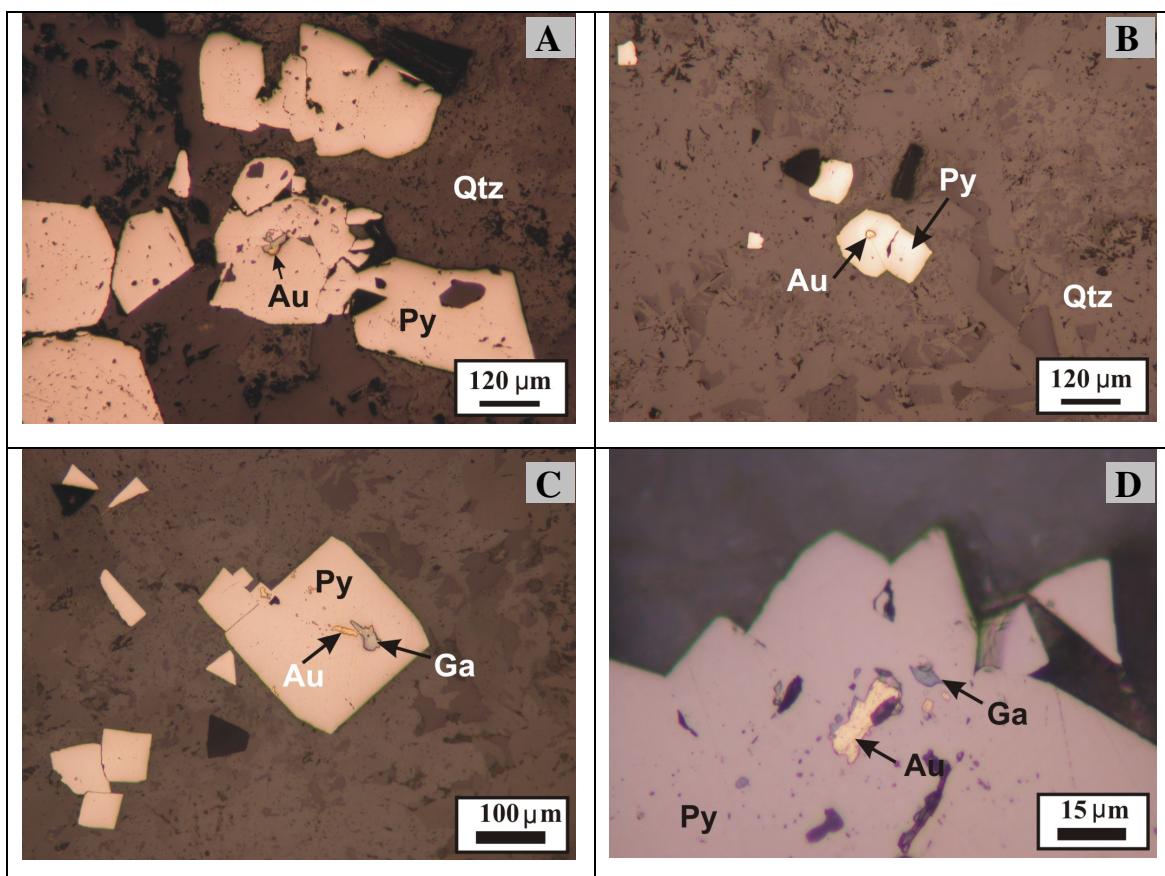


Figure 3.9 Reflected light photomicrographs of Efemçukuru ore (Plane polarized light, D oil immersion). A and B are electrum included in disseminated subhedral to euhedral pyrites in a quartz rich matrix of the vein. C and D are electrum and galena in pyrite crystals. The grain size of electrum is coarser than 10 micrometer in pyrites.

Sulfide rich parts of the veins generally comprise dominantly quartz, pyrite, sphalerite, galena, and lesser chalcopyrite, some other gangue minerals rhodonite, axinite, carbonates and a variety lesser mineral phases including minerals of precious metals such as gold, silver, and tellurides.

Pyrite is the most common sulfide in different parts of the vein (figure 3.9), vein breccia and stockwork enclosed to the vein and mineralized hornfels in Efemçukuru gold ore. The habit of the pyrites varied from cubic to pyritohedron and octahedron (figure 3.9 A, B and C). Whereas cubic is the most widespread habit in Efemçukuru pyrites. As one of the first minerals to crystallize it is replaced by other minerals such as sphalerite, galena, chalcopyrite, tetrahedrite-tenanthite.

Sphalerite is generally found as second earliest formed ore mineral accompanying to or nearly after pyrite. Sphalerite contains discontinuous veins and abundant oriented, fine grained inclusions of chalcopryite. The early sphalerite (figure 3.10 A) contains randomly oriented crystals of chalcopryite and pyrrhotite in different sizes and shape. Sphalerite II which precipitated later does not contain more than 5% inclusions.

Chalcopryite and galena generally appeared to have formed after both the pyrite and sphalerite (figure 3.10 B). Chalcopryite crystals contain bismuth sulphosalts and carbonate crystals. Fahlore group minerals are observed in minor amounts as late ore phase.

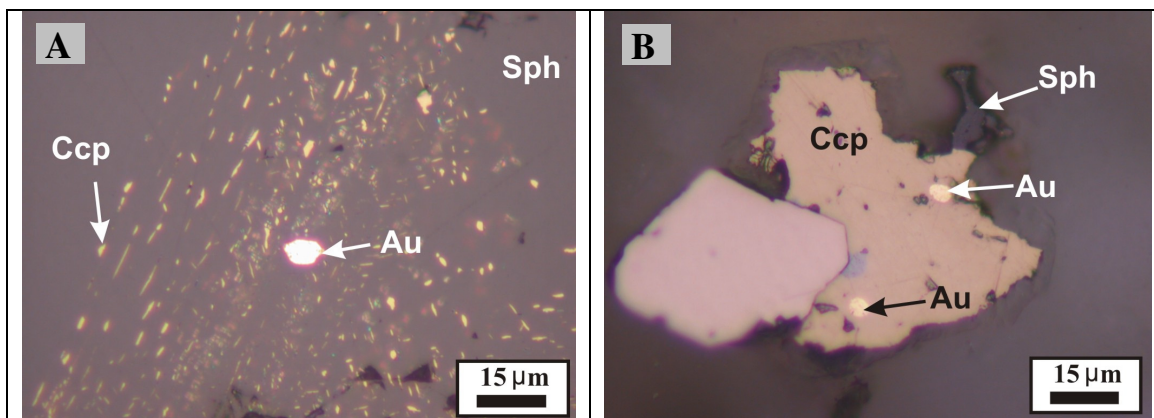


Figure 3.10 Reflected light photomicrographs of Efemçukuru ore (Plane polarized light, oil immersion). A is sphalerite that has chalcopryite inclusions aligned along crystallographic directions and about grain boundaries with gold. B is a gold bearing chalcopryite after euhedral pyrite

The oxide rich samples are mainly composed of quartz and iron oxides. The oxidation of iron (Fe) and sulfides of iron-copper (Fe-Cu) formed mainly goethite and secondly hematite. Some pyrite particles are fully or partially oxidized to goethite (figure 3.11 A). In those samples, some fine grained liberated gold formations can be observed (figure 3.11 B). All photomicrographs are taken by Tolga Oyman from Dokuz Eylül University.

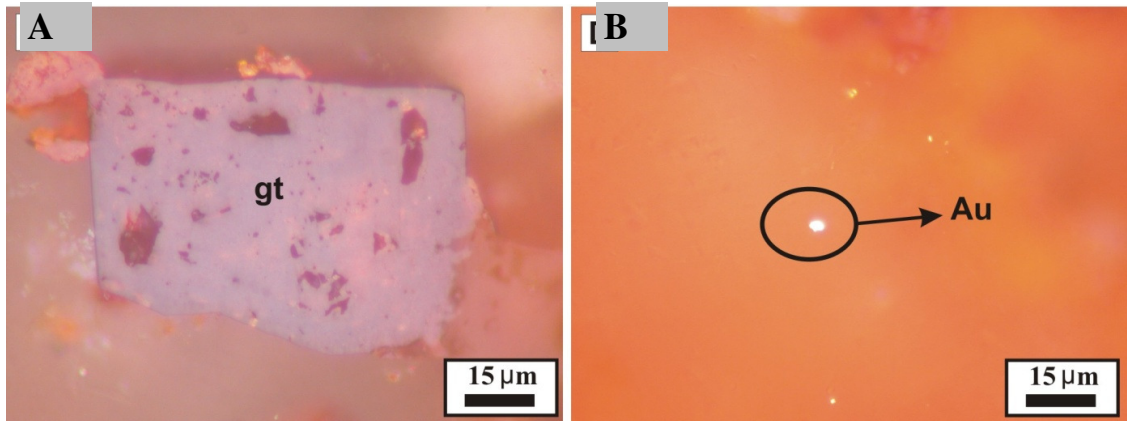


Figure 3.11 Reflected light photomicrographs of Efemçukuru ore (Plane polarized light). A is the formed goethite by oxidation of pyrites. B is the liberated fine gold particle.

Another characterization test to Efemçukuru gold sample was X-Ray diffraction (XRD) spectroscopy. The samples, sent to XRD were crushed to below 3.35 millimeters. The XRD results show the presence of quartz (SiO_2), orthoclase (KAlSi_3O_8), pyrite (FeS_2), silver (Ag) and goethite ($\text{FeO}(\text{OH})$). The diffractions of SiO_2 and trace elements are shown in figure 3.12.

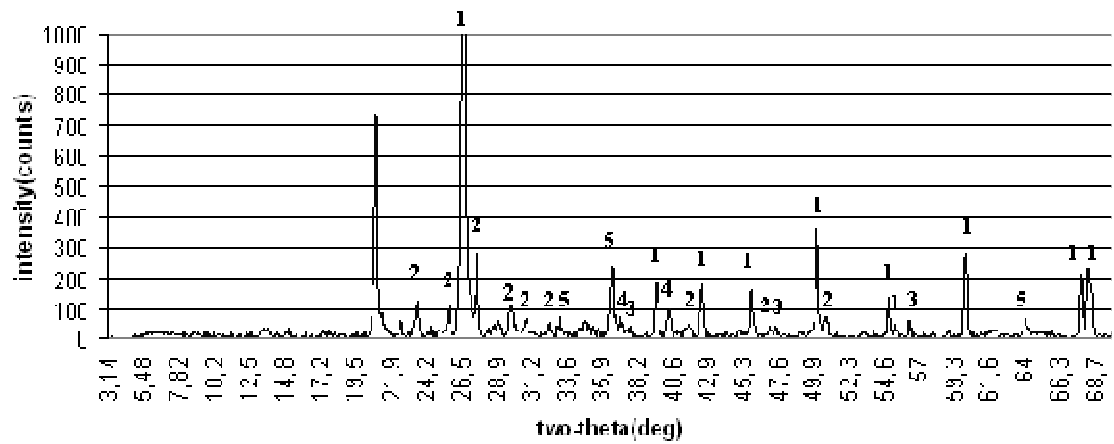


Figure 3.12 The XRD results of Efemçukuru gold ore crushed below 3.35 mm. (1- SiO_2 2- KAlSi_3O_8 3- FeS_2 4-Ag 5- $\text{FeO}(\text{OH})$)

3.3 Bond Work Index Estimation Studies and Grinding Tests

3.3.1 Bond Work Index Test

A Bond Ball Mill Work Index test is a standard test for determining the ball mill work index of a sample of ore. It was developed by Fred Bond in 1952 and modified in 1961 (JKMRC CO., 2006). This index is widely used in the mineral industry for comparing the resistance of different materials to ball milling, for estimating the energy required for grinding, and for ball mill scale-up. The test has existed for more than 40 years (Man, 2002). Because of the difficulty in determination of this index, alternatives to the standard method have been developed by many researchers (Vedat & Huseyin, 2003).

In the determination of ball mill work index 15 kg of representative ore at 100% + 3.35 mm is crushed to 100% - 3.35 mm (Amtech, 2006). The first grinding test is started with an arbitrarily chosen number of mill revolutions. At the end of each grinding cycle, the entire product is discharged from the mill and is screened on a test sieve. Fresh feed material is added to the oversize to bring the total weight back to that of the original charge. This charge is then returned to the mill. The number of revolutions in the second grinding cycle is calculated so as to gradually produce the 250% circulating load. After the second cycle, the same procedure of screening and grinding is continued until the test-sieve under size produced per mill revolution becomes constant for the last three grinding cycles. This will give the 250% circulating load (Bond, 1961).

The sieving set for the size analysis of the feed was, 3.350 mm, 2.360 mm, 1.700 mm, 1.18 mm, 0.850 mm, 0.600 mm, 0.425 mm, 0.300 mm, 0.212 mm and 0.150 mm (figure 3.13). The set for the product size analysis was, 0.150 mm, 0.125 mm, 0.106 mm, 0.075 mm, 0.053 mm and 0.045 mm. The ball charge weighted 20.14 kg in total with 285 balls (figure 3.14). The 700 cm³ feed material was prepared by

tapping and weighted. The first grinding test was for 150 revolutions at 70 rpm. Bond work index can be calculated by the equation below.

$$W_i = 44.5 / [P_i^{0.23} \times G_{bp}^{0.82} \times (10/\sqrt{P} - 10/\sqrt{F})]$$



Figure 3.13 The sieving set and the vibratory sieving machine used during size analysis



Figure 3.14 The Bond mill and the ball charge used during work index estimation

Size, US Std.		Feed				Product			
Feed	Prdct.	Weight	Percent Retained		% Passing	Weight	Percent Retained		% Passing
(μm)	(μm)	(grams)	Individual	Cum.	Cum.	(grams)	Individual	Cum.	Cum.
3.350		0,85	0,06	0,06	99,94				
2.360		242,40	16,88	16,93	83,07				
1.700		227,40	15,83	32,77	67,23				
1180		195,20	13,59	46,36	53,64				
850		140,80	9,80	56,16	43,84				
600		119,90	8,35	64,51	35,49				
425		97,90	6,82	71,32	28,68				
300	150	74,95	5,22	76,54	23,46	41,70	2,15	2,15	97,85
212	125	66,70	4,64	81,18	18,82	345,63	17,84	20,00	80,00
150	106	99,38	6,92	88,10	11,90	298,32	15,40	35,40	64,60
Minus 150	75	170,90	11,90	100,00	0,00	434,73	22,44	57,84	42,16
	53		0,00	100,00	0,00	357,96	18,48	76,33	23,67
	45		0,00	100,00	0,00	247,10	12,76	89,08	10,92
	Minus 45		0,00	100,00		211,44	10,92	100,00	
Totals		1436,38	100,00			1936,88	97,85		
Cir. Load Ratio	New Feed	Undersize in Feed	To be Produced	Mill Turns	Oversize In Product	Undersize In Product	Total Undersized Produced	Grams per Mill Rev.	
0,76	1442,94	171,68	240,59	150	1095,84	347,10	175,42	1,169	
3,15	347,10	41,30	370,97	317	1092,63	350,31	309,01	0,974	
3,00	350,31	41,68	370,59	380	1050,09	392,85	351,17	0,923	
2,57	392,85	46,74	365,53	396	1010,53	432,41	385,67	0,974	
2,28	432,41	51,45	360,82	370	986,87	456,07	404,62	1,092	

Figure 3.15 The Bond mill and the ball charge used during work index estimation

The results and the steps of a bond grindability test is given in figure 3.15. Work index of Efemçukuru gold sample was calculated as 20.73 kWh/t.

3.3.2 Grinding Tests

Grinding tests were performed by using a laboratory type rod mill (figure 3.16). The sample ground was 625 grams. The mill was operated at 83 rpm, with a pulp density of 65% solid by weight. Different grinding durations of 10, 20 and 30 minutes were applied and analyzed for size distribution and gold distribution.

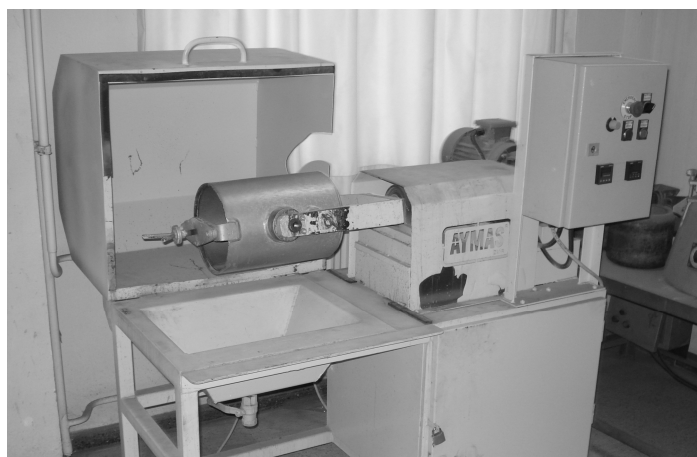


Figure 3.16 The laboratory type rod mill used for sample grinding

Table 3.2 Sieve - metal analysis of 625 g sample ground for 10 minutes

Size Fraction (microns)	Weight (g)	% Weight	Cum. % Weight Undersize	Au (g/t)	Cum. Au (g/t)	Cum. Au Dist. (%)
+ 106	10.63	1.73	100.00	_(*)	_(*)	_(*)
-106 +75	76.57	12.44	98.27	3.16	5.13	100.00
-75 +53	120.12	19.51	85.83	3.30	5.42	92.22
-53 +38	110.06	17.88	66.32	4.79	6.04	79.47
-38	298.31	48.44	48.44	6.51	6.51	62.48
Total	615.69	100,00		4.65		

(*) There is insufficient amount of sample in this size interval to perform the chemical analysis.

Table 3.3 Sieve - metal analysis of 625 g sample ground for 20 minutes

Size Fraction (microns)	Weight (g)	% Weight	Cum. % Weight Undersize	Au (g/t)	Cum. Au (g/t)	Cum. Au Dist. (%)
-75 +53	66.19	10.83	100.00	5.28	5.19	100.00
-53 +38	129.18	21.13	89.18	4.25	5.18	88.99
-38	416.09	68.05	68.05	5.47	5.47	71.69
Total	611.46	100.00		5.14		

Table 3.4 Sieve - metal analysis of 625 g sample ground for 30 minutes

Size Fraction (microns)	Weight (g)	% Weight	Cum. % Weight Undersize	Au (g/t)	Cum. Au (g/t)	Cum. Au Dist. (%)
-75 +53	13.04	2.13	100.00	16.76	4.89	100.00
-53 +38	51.93	8.47	97.87	5.61	4.63	92.71
-38	547.68	89.40	89.40	4.54	4.54	82.98
Total	612.65	100.00		4.89		

Table 3.5 Sieve – metal analysis of 625 g sample of feed sample before grinding

Size Fraction (mm)	Weight (g)	% Weight	Cum. % Weight Undersize
+ 3.35	1.00	0.16	100.00
-3.35 +2.26	83.98	13.72	99.84
-2.36 +1.7	65.25	10.66	86.12
-1.7 +1.18	61.70	10.08	75.46
-1.18 +0.850	50.07	8.18	65.38
-0.850 +0.600	49.15	8.03	57.20
-0.600 +0.425	45.22	7.39	49.18
-0.425 +0.300	46.20	7.55	41.79
-0.300 +0.212	28.65	4.68	34.24
-0.212 +0.150	23.03	3.76	29.56
-0.150 +0.125	14.97	2.45	25.80
-0.125 +0.106	9.90	1.62	23.36
-0.106 +0.075	22.19	3.62	21.74
-0.075 +0.053	22.25	3.63	18.12
-0.053 +0.038	10.81	1.77	14.48
-0.038	77.85	12.72	12.72
Total	612.22	100.00	

The cumulative size distributions of the grinding tests are shown in figure 3.17. In 10 minutes of grinding 48.44% of the sample passed under 38 microns. This value was 68.05% and 89.40% respectively for 20 and 30 minutes of grinding. The sieve analysis of the feed material crushed under 3.35 mm showed that 12.72% of the material was already below size 38 microns.

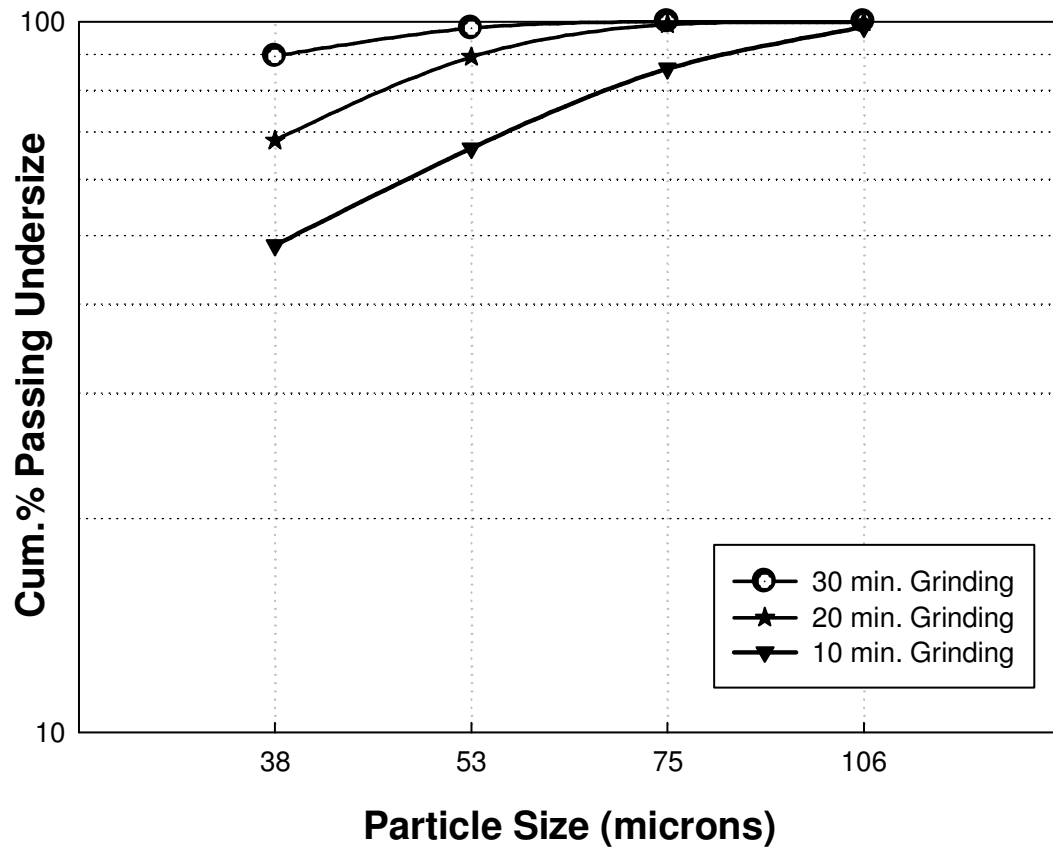


Figure 3.17 The cumulative size distribution of undersize after different periods of grinding

The gold (Au) contents and distributions of different grinding periods are shown in figure 3.18 and 3.19. The results support the characterization of the sample. Since, it is obvious that gold is mainly below 38 microns in size. The characterization studies stated that, the grain size of gold commonly varies between 5 to 30 μm . The distribution of gold for 30 minutes of grinding (89.40% of gold under 38 microns) shows that the liberation of gold would be in very fine sizes.

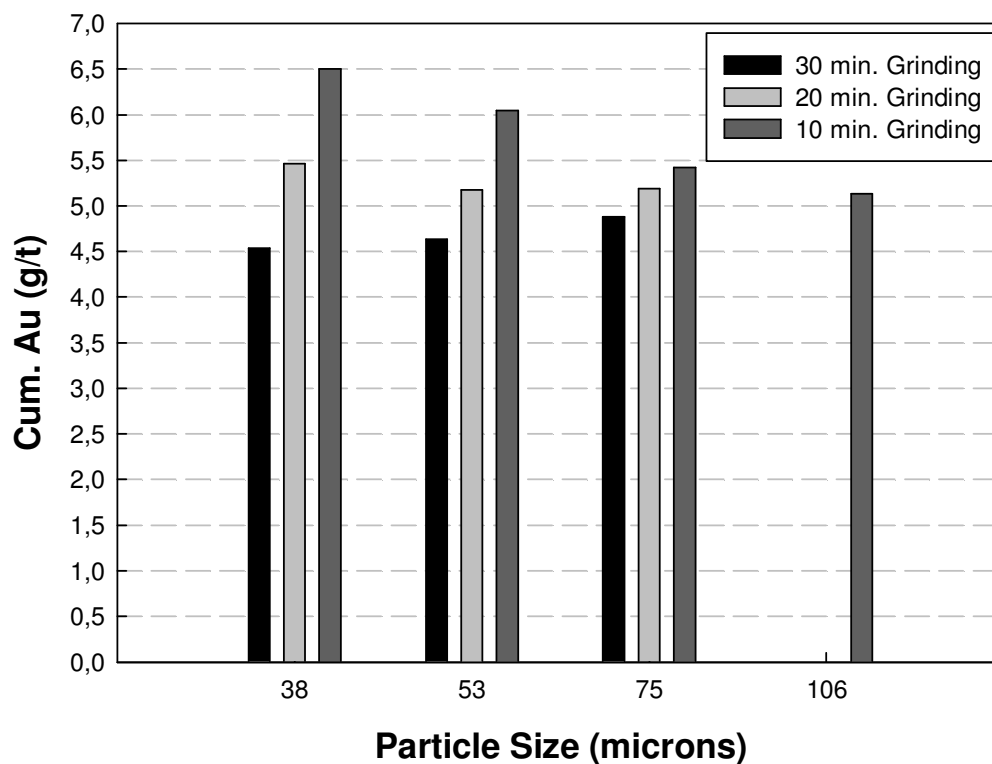


Figure 3.18 The cumulative gold (Au) contents after different periods of grinding

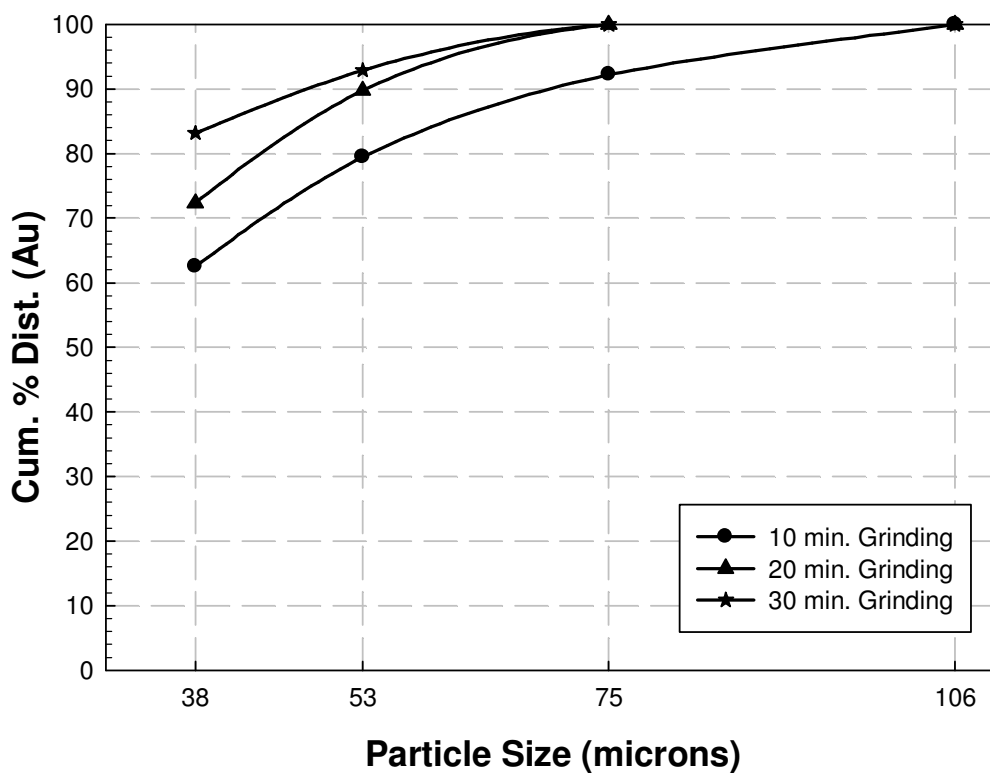


Figure 3.19 The cumulative gold (Au) distribution after different periods of grinding

3.4 Gravity Concentration Studies using the Knelson Concentrator

3.4.1 *The Knelson Concentrator*

The use of gravity concentration in the gold industry has grown significantly since the 1970s. This resurgence is in part due to the growing emphasis on sustainable development (Burt, 1999) where maximum recovery no longer holds precedence to improving the economics of the process. Demand for more efficient machines lead to the development of centrifugal devices such as the Knelson and Falcon concentrators. Silva (1987) demonstrated that Knelson concentrators (KC) are capable of achieving 96% recovery of free liberated gold coarser than $30\mu\text{m}$ with a limit of recovery down to $0.3\mu\text{m}$. The Knelson concentrators are commonly used in the gold industry as gravity concentration devices because of their ability to recover coarse and fine gold from both alluvial and primary deposits. The device is most commonly installed (Banisi et al., 1991) in the grinding circuit of a gold operation, where gold often accumulates due to its grinding and classification behavior.

Figure 3.20 provides an illustrated example of how concentration takes place within a KC. The feed material, most commonly cyclone underflow or ball mill discharge, enters through a central feed tube as a slurry. The slurry descends onto the base plate at the bottom of the rotating conical bowl and is thrown outwards as initially there is no centripetal force acting on the particles to maintain a circular path. As a result some of the particles get trapped within the grooves of the bowl forming a concentrate bed while the excess get carried upward into the tailings stream by the rising current of water. Injection of water through fluidization ports located in the ribs of the bowl prevents compaction of the concentrate bed. This creates a fluidized bed (Harris, 1984) which acts as a concentrating chamber for heavier particles such as coarse and fine gold, under enhanced gravitational force.

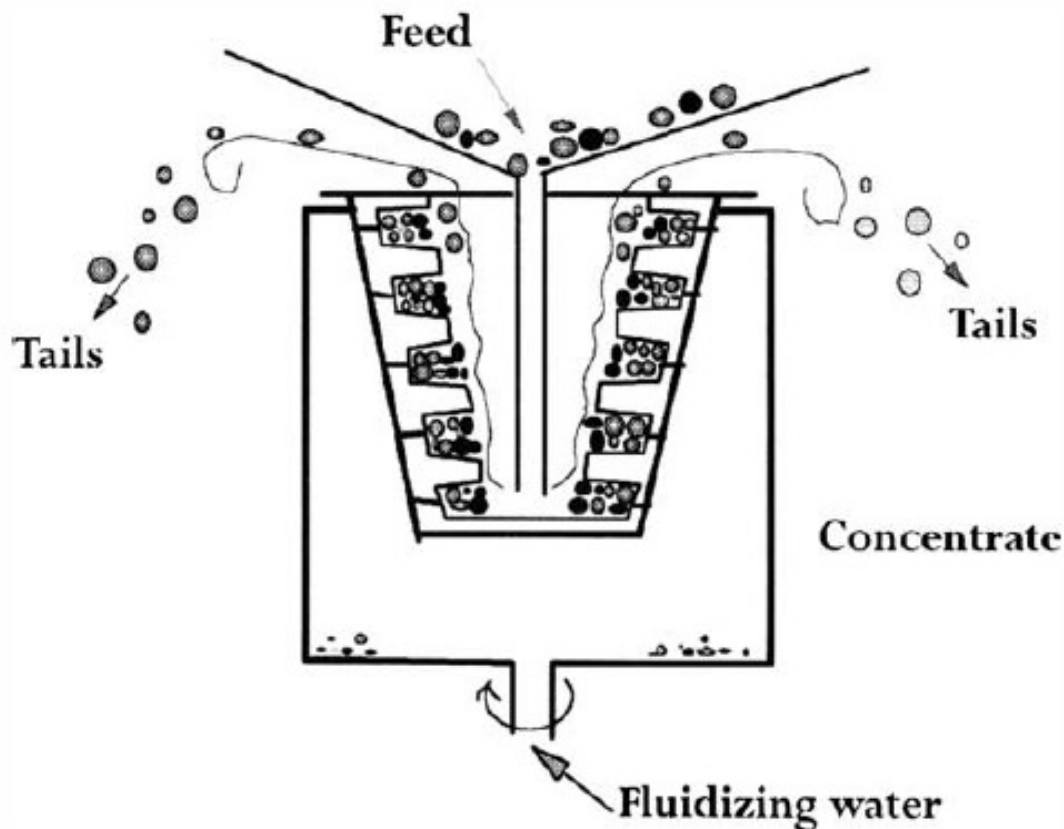


Figure 3.20 Schematic diagram of a Knelson concentrator.

Gold recovery using gravity concentration has advanced significantly over the past twenty years, largely because of the advent of high capacity centrifugal gravity concentrators such as Knelson concentrators (KC). Knelson concentrators are almost placed within the grinding circuit, either treating cyclone underflow or, less often the mill discharge to take advantage of gold's unusual behavior in grinding circuits. The fact that gold often accumulates in grinding circuits due to its grinding kinetics and classification behavior in cyclones is well documented (Laplante et al., 1994).

The Knelson concentrator is a centrifugal bowl-type concentrator which utilizes the principle of hindered settling classification in a centrifugal force field (Meza S, 1994). The Knelson gravity concentrator type used in these experiments is the laboratory type concentrator with 3 inches (7, 62 cm) cone diameter (figure 3.21). The standard operating conditions of the concentrator are listed in table 3.6.



Figure 3.21 Knelson MD3 type laboratory concentrator with 3 inches cone diameter

Table 3.6 The standard operating parameters of Knelson MD3 concentrator

	Model (D,inches)	Power (HP)	Centrifugal Speed of the cone (rpm)	Capacity (t/h)	Pulp Density (% Weight)	Max. Feed Size (mm)
Standards of the Concentrator	3	1/6	1200	0,065	0 – 75	2

3.4.2 The Experimental Data and Parameters of Knelson Concentrator Tests

The aim of the following experiments was determining the recovery and grade of Efemçukuru gold ore by using the Knelson gravity concentrator. The determination of the optimum conditions for the concentrator was observed by selecting different parameters. The variables and parameters studied are given in table 3.7.

Table 3.7 The variables and the parameters experimented

Variables	Parameters				
G Force (G)	60		90		120
Max. Feed Size (mm)	0.025	0.075	0.106	0.150	0.212
Water Pressure (psi)	2		4		8
Feed Rate (t/h)	0.024				
Feed Amount (g)	2000				
Pulp Density (%)	25-30				

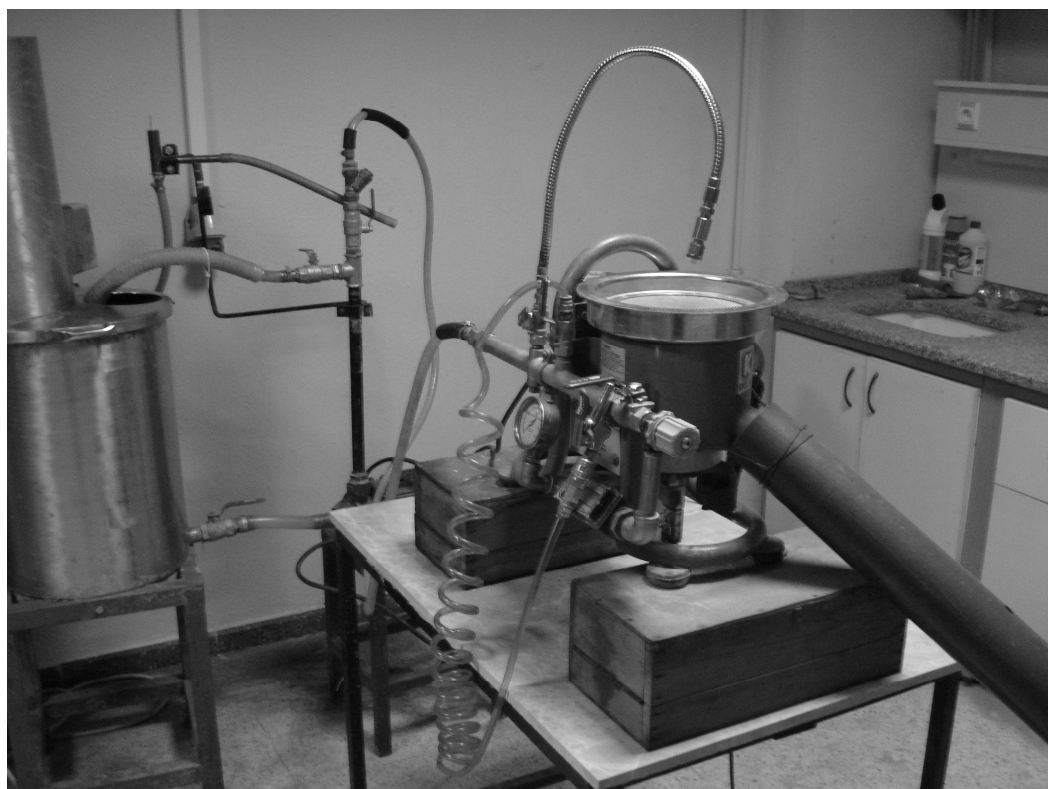


Figure 3.22 Experimental conditions of Knelson concentrator studies

3.4.3 The Effect of G Force and Particle Size in Knelson Concentrator Tests

The effect of G force tests on different sized samples were applied with 0.024 t/h feed rate, 4 psi water pressure, 2000 g feed amount and 25-30% pulp density.

3.4.3.1 Effect of G Force on the Sample with 0.025 mm top size

Table 3.8 The results of the chemical analysis of Knelson concentrate, gang and theoretical feed with maximum particle size of 0.025 mm

Size (mm)	G force	Type	Weight (g)	Au (g/t)	Recovery %
-0.025	60	Concentrate	72.08	60.50	26.59
		Gang	1927.92	6.24	73.41
		Feed	2000.00	8.20	
-0.025	90	Concentrate	64.20	108.13	42.33
		Gang	1935.80	4.89	57.67
		Feed	2000.00	8.20	
-0.025	120	Concentrate	90.30	59.50	32.76
		Gang	1909.70	5.77	67.24
		Feed	2000.00	8.20	

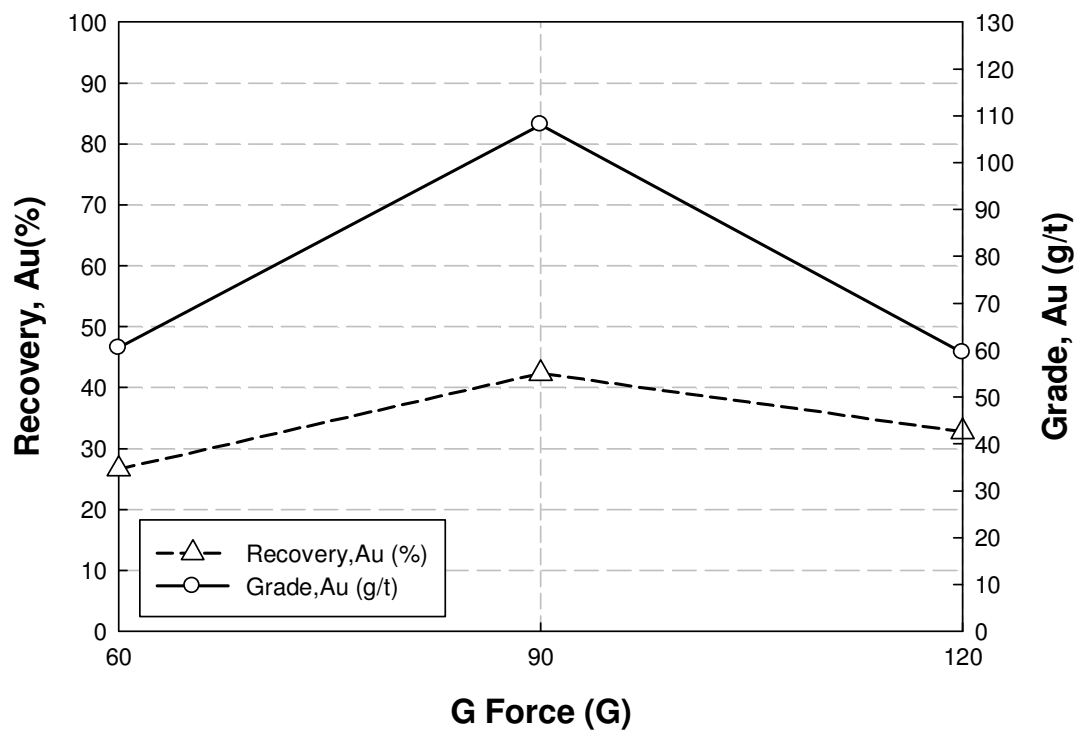


Figure 3.23 The Au recovery and grade of 3 different G forces on the sample with 0.025 mm top size

The Knelson concentrator running in 90 G achieved a total recovery 42.33% at the size of 25 microns with Au content 108.13 g/t. The results of 60 and 120 G experiments had low Au recovery and grades. The results will be discussed in chapter 4.

3.4.3.2 Effect of G Force on the Sample with 0,075 mm top size

Table 3.9 The results of the chemical analysis of Knelson concentrate, gang and theoretical feed with maximum particle size of 0.075 mm

Size (mm)	G force	Type	Weight (g)	Au (g/t)	Recovery %
-0.075	60	Concentrate	76.39	49.20	25.99
		Gang	1923.61	5.57	74.01
		Feed	2000.00	7.23	
-0.075	90	Concentrate	76.88	79.35	42.18
		Gang	1923.12	4.35	57.82
		Feed	2000.00	7.23	
-0.075	120	Concentrate	79.74	122.29	67.43
		Gang	1920.26	2.45	32.57
		Feed	2000.00	7.23	

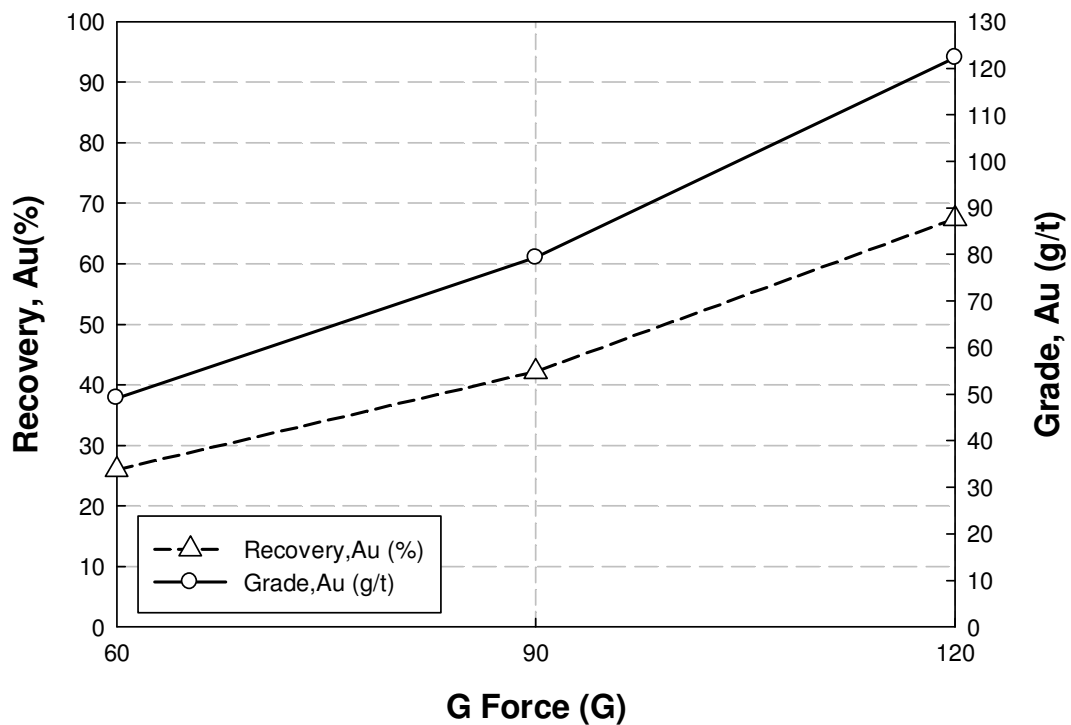


Figure 3.24 The Au recovery and grade of 3 different G forces on the sample with 0.075 mm top size

The Knelson concentrator running in 120 G achieved a total recovery 67.4 % at the size of 75 microns with Au content 122.3 g/t. The increase in the recovery and grade with increasing G force is shown in figure 3.24. Total recovery rises from 26% to 67.4 % by increasing the G force from 60 to 120 G.

3.4.3.3 Effect of G Force on the Sample with 0,106 mm top size

Table 3.10 The results of the chemical analysis of Knelson concentrate, gang and theoretical feed with maximum particle size of 0.106 mm

Size (mm)	G force	Type	Weight (g)	Au (g/t)	Recovery %
-0.106	60	Concentrate	65.86	50.43	28.62
		Gang	1934.14	4.28	71.38
		Feed	2000.00	5.80	
-0.106	90	Concentrate	74.66	72.59	46.70
		Gang	1925.34	3.21	53.30
		Feed	2000.00	5.80	
-0.106	120	Concentrate	87.95	92.74	70.28
		Gang	1912.05	1.80	29.72
		Feed	2000.00	5.80	

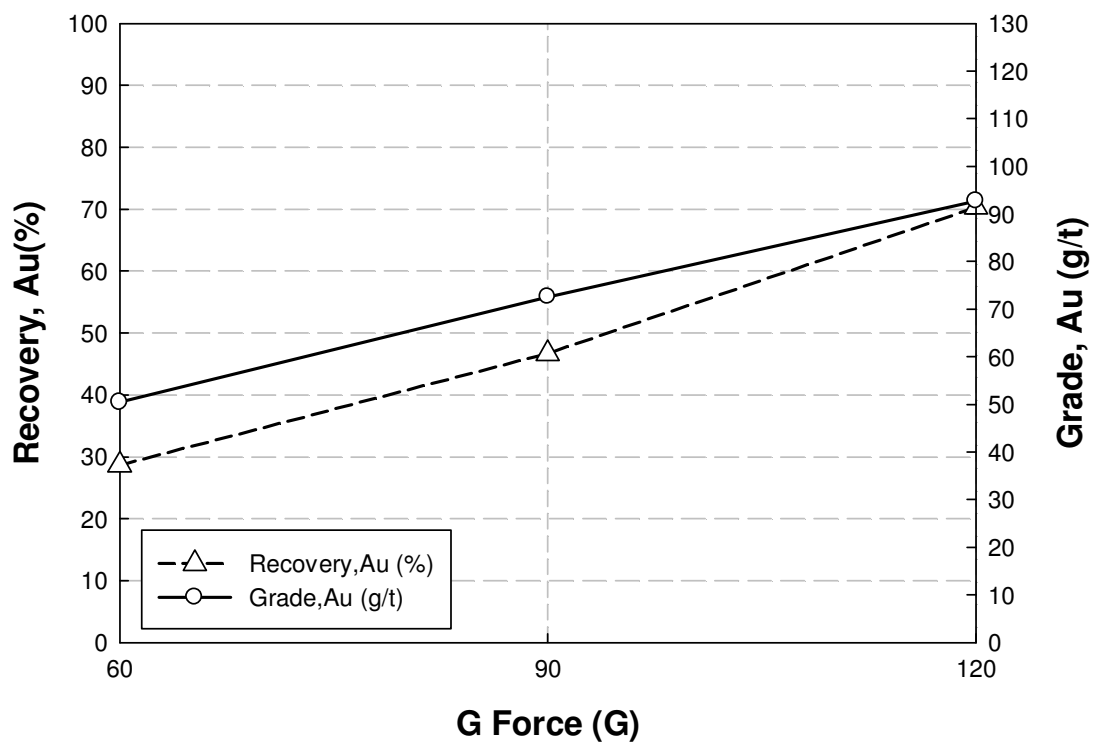


Figure 3.25 The Au recovery and grade of 3 different G forces on the sample with 0.106 mm top size

The results were obtained and it is observed that with the maximum feed size of 106 microns, increasing the G force had a positive effect on both grade and recovery of Au. Total recovery of 70.28% was achieved with Au content of 92.74 g/t. The gradual increase in recovery with increasing G force can be easily determined by examining figure 3.25 above.

3.4.3.4 Effect of G Force on the Sample with 0,150 mm top size

Table 3.11 The results of the chemical analysis of Knelson concentrate, gang and theoretical feed with maximum particle size of 0.150 mm

Size (mm)	G force	Type	Weight (g)	Au (g/t)	Recovery %
-0.150	60	Concentrate	71.30	56.42	36.97
		Gang	1928.70	3.56	63.03
		Feed	2000.00	5.44	
-0.150	90	Concentrate	79.86	72.05	52.89
		Gang	1920.14	2.67	47.11
		Feed	2000.00	5.44	
-0.150	120	Concentrate	71.95	89.20	58.99
		Gang	1928.05	2.31	41.01
		Feed	2000.00	5.44	

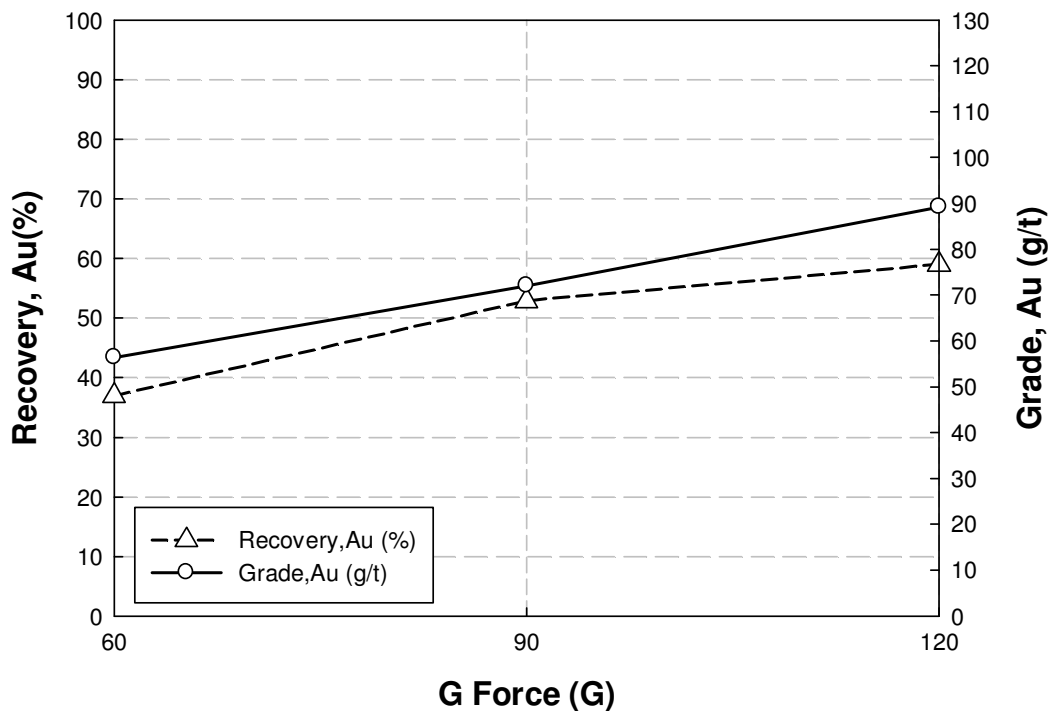


Figure 3.26 The Au recovery and grade of 3 different G forces on the sample with 0.150 mm top size

The Knelson concentrator running in 120 G achieved a total recovery 59% at the size of 150 microns with Au content 89.2 g/t. The increase in the recovery and grade with increasing G force is shown in figure 3.26. Total recovery rises from 37% to 59% by increasing the G force from 60 to 120 G.

3.4.3.5 Effect of G Force on the Sample with 0,212 mm top size

Table 3.12 The results of the chemical analysis of Knelson concentrate, gang and theoretical feed with maximum particle size of 0.212 mm

Size (mm)	G force	Type	Weight (g)	Au (g/t)	Recovery %
-0.212	60	Concentrate	70.04	86.94	38.50
		Gang	1929.96	5.04	61.50
		Feed	2000.00	7.91	
-0.212	90	Concentrate	69.20	81.35	35.60
		Gang	1930.80	5.27	64.40
		Feed	2000.00	7.91	
-0.212	120	Concentrate	72.28	78.32	35.80
		Gang	1927.72	5.27	64.20
		Feed	2000.00	7.91	

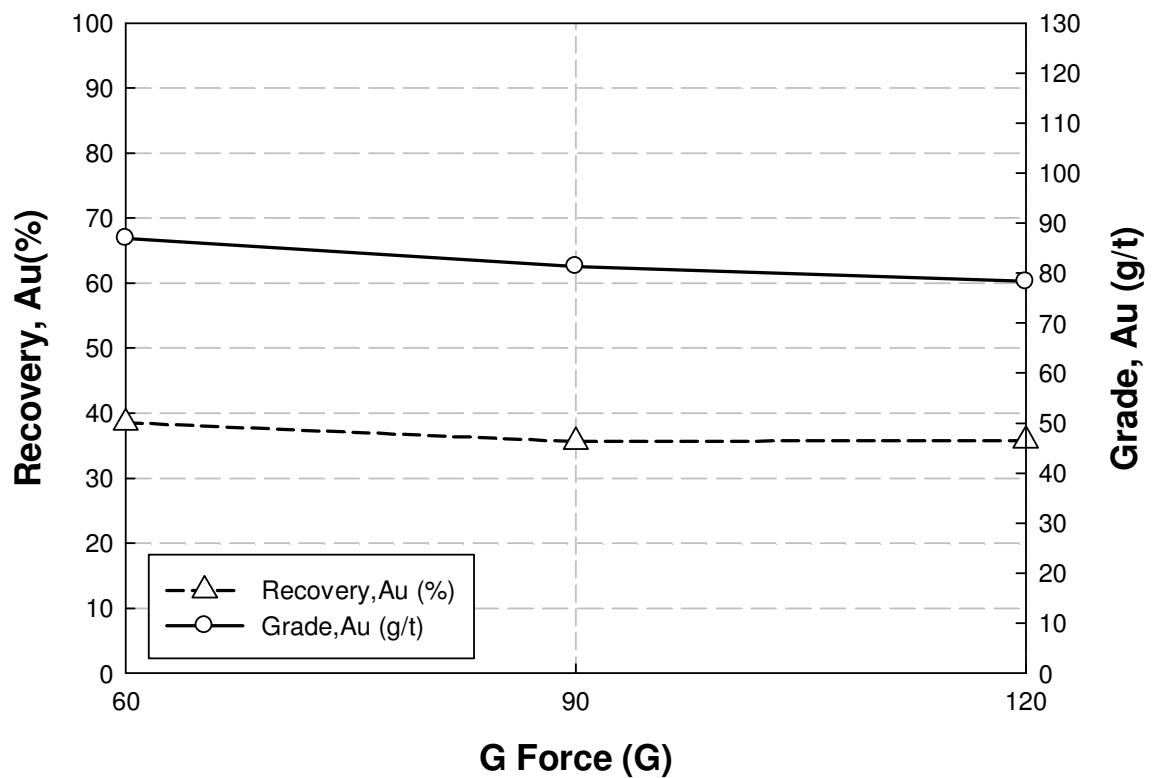


Figure 3.27 The Au recovery and grade of 3 different G forces on the sample with 0.212 mm top size

The experiments showed that, with the maximum feed size of 212 microns the increasing G force had a slightly negative effect on the recovery and grade of Au. The Au grade was between 78.3 and 87 g/t, while the recovery varied between 35.6% and 38.5%.

3.4.3.6 Effect of G Force on Samples with Different Particle Sizes

The effects of G forces and particle sizes on the gold recoveries and concentrate grades with a 7.6 cm laboratory type Knelson gravity concentrator operated at three different G forces and five different particle sizes (-25 μm , -75 μm , -106 μm , -150 μm , -212 μm) are summarized below.

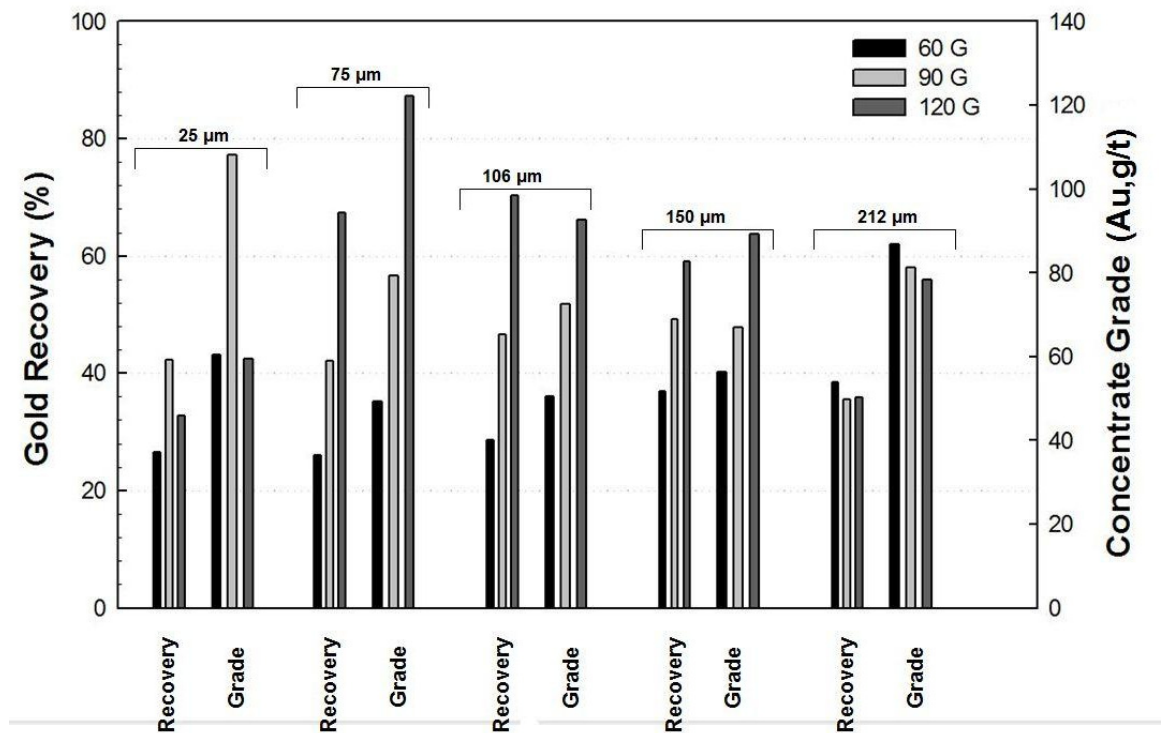


Figure 3.28 Effects of G forces and particle sizes on the gold recoveries and concentrate grades with a 7.6 cm laboratory type Knelson gravity concentrator operated at three different G forces and five different particle sizes (- 25 μm , -75 μm , -106 μm , -150 μm , -212 μm).

3.4.3.7 Effect of Particle Size on Knelson Concentrator Tests with 60 G

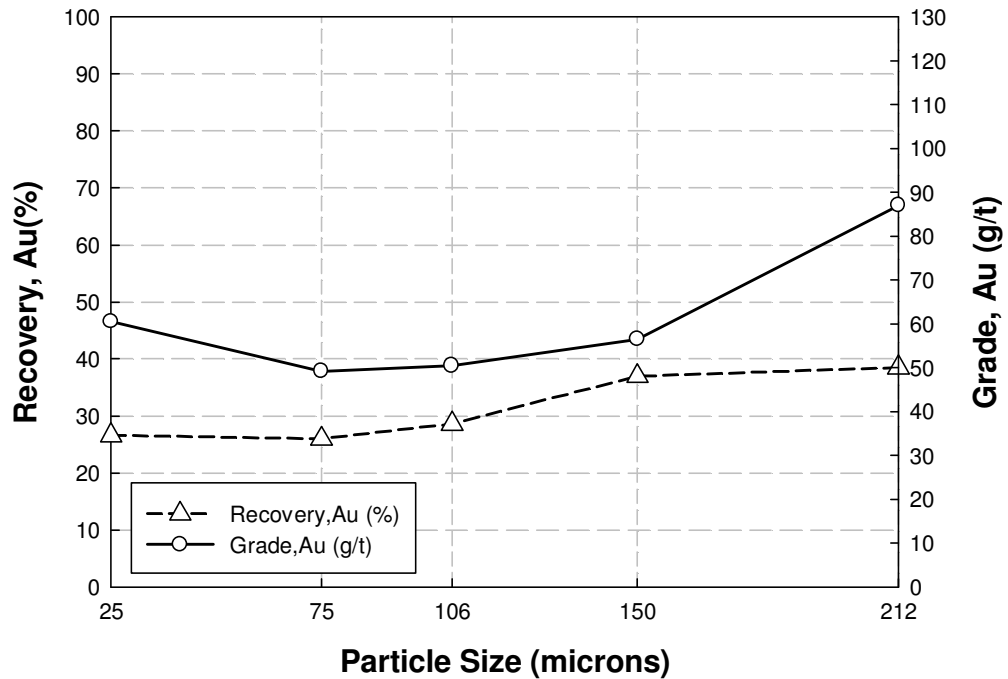


Figure 3.29 The Au recovery and grade of 5 different particle sizes on the tests with 60 G

The Knelson concentrator tests performed with 60 G showed that the coarser particle sizes are affected positively more than the fine sizes. The Au recovery has a slight increase with increasing particle size.

3.4.3.8 Effect of Particle Size on Knelson Concentrator Tests with 90 G

The Knelson concentrator tests performed with 90 G are shown in figure 3.29. The highest Au recovery was achieved with the particle size of -150 microns. However, the highest Au grade was achieved with the finest size, -25 microns. As the particle size gets coarser to -212 microns, a remarkable decrease in Au recovery can be observed.

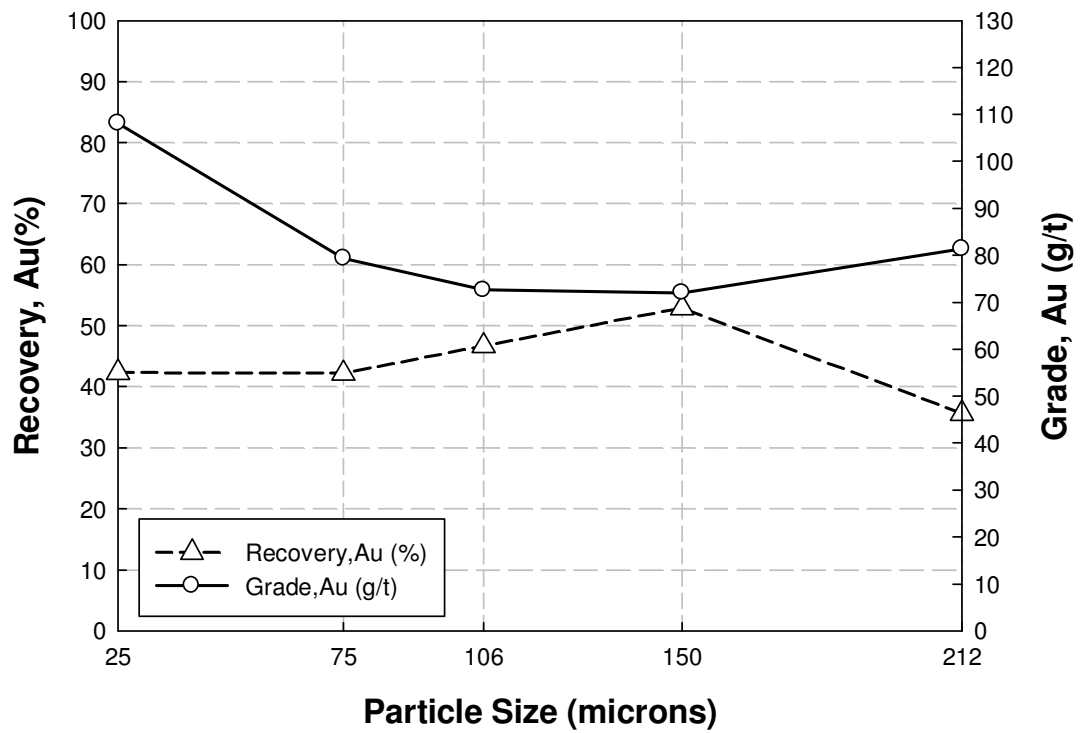


Figure 3.30 The Au recovery and grade of 5 different particle sizes on the tests with 90 G

3.4.3.9 Effect of Particle Size on Knelson Concentrator Tests with 120 G

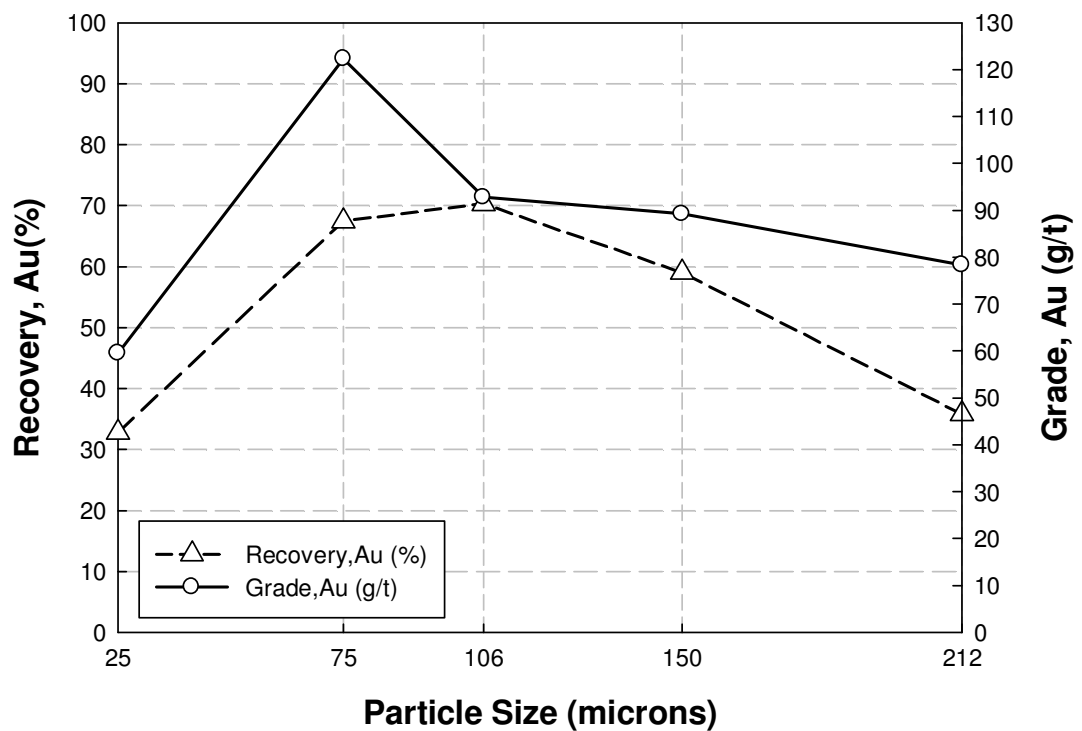


Figure 3.31 The Au recovery and grade of 5 different particle sizes on the tests with 120 G

Generally the highest Au recovery and grade results are achieved in the tests performed with 120 G. The highest Au recovery of 70.28% with 106 microns top size and highest Au grade of 122.29 g/t with 75 microns top size are achieved with 120 G.

3.4.4 The Effect of Water Pressure in Knelson Concentrator Tests

The highest Au recovery of 70.28 % with Au grade of 92.74 g/t was achieved in the experiment with 120 G, 4 psi water pressure and 0.106 mm top size. The effect of water pressure was studied by repeating the optimum result with water pressure of 2 and 8 psi.

Table 3.13 The results of the chemical analysis of Knelson concentrate and gang with 3 different water pressures applied on optimum test results

Size (mm)	G force	Water Pressure (psi)	Type	Weight (g)	Au (g/t)	Recovery %
-0.106	120	2	Concentrate	93.8	73.80	56.91
			Gang	1906.2	2.75	43.09
-0.106	120	4	Concentrate	87.95	92.74	70.28
			Gang	1912.05	1.80	29.72
-0.106	120	8	Concentrate	38.52	143.02	44.52
			Gang	1965.48	3.13	55.48

The results confirmed that the highest Au recovery and grade was achieved with the following parameters.

Table 3.14 The experiment parameters of the optimum Knelson concentrator tests

Variables	Parameters
G Force (G)	120
Max. Feed Size (mm)	0.106
Water Pressure (psi)	4
Feed Rate (t/h)	0.024
Feed Amount (g)	2000
Pulp Density (%)	25-30

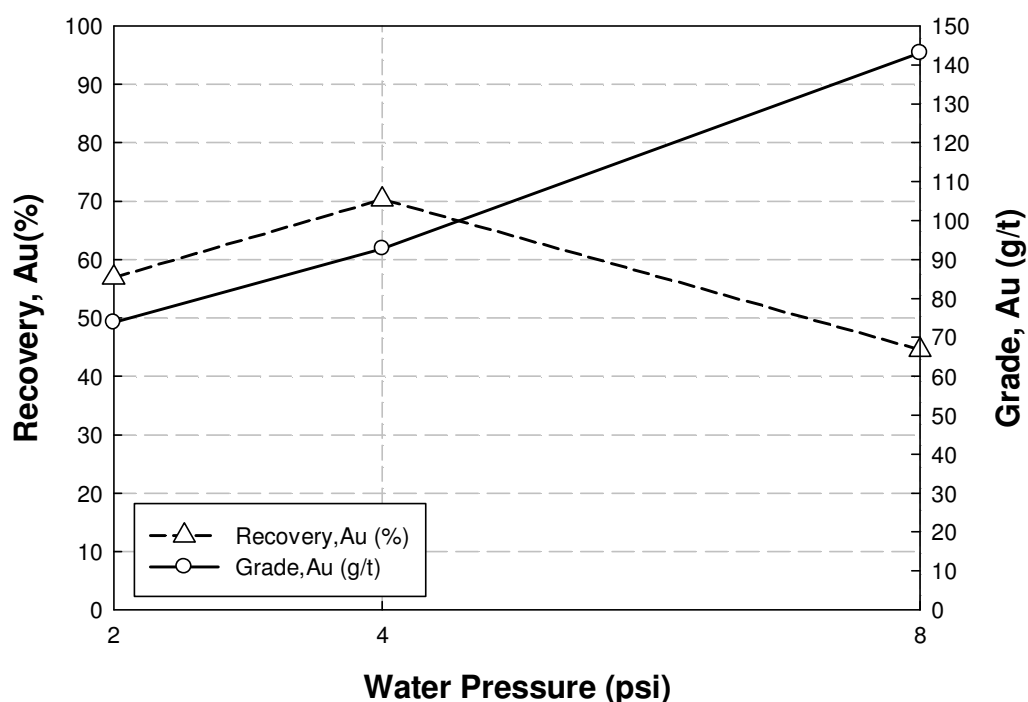


Figure 3.32 The Au recovery and grade of 3 different G forces on the sample with 0.212 mm top size

The water pressure increase has remarkable effect on the Au grade. The Au grade had increased from 73.80 g/t to 143.02 g/t while the water pressure had increased from 2 to 8 psi. The Au recoveries for both 2 and 8 psi water pressures were too low compared to 4 psi.

3.4.5 The Chemical Characterization of the Knelson Concentrate

The analysis of the Knelson concentrate gathered in optimum parameters are given in table 3.15. The results show that listed metals were approximately 1.6 times beneficiated except Au, Ni and Mn when compared with the feed analysis given in table 3.1. Au and Ni were the main beneficiated metals; on the other hand Mn content had not increased.

Table 3.15 The chemical characterization of the Knelson concentrate (optimum test) by AAS.

Element	Au	Ag	Cu	Pb	Zn	Ni	Fe ₂ O ₃	Mn
Grade (g/t), %	92.7	81.2	400.0	3990.0	850.0	2800.0	12.6%	1.9 %

3.5 Flotation Studies

The aim of the flotation investigation was to bulk float all the gold-containing minerals. Different variables were tested with different parameters to achieve the highest gold recovery with an acceptable gold grade.

3.5.1 Froth Flotation

Froth flotation utilizes the differences in physico-chemical surface properties of particles of the various minerals. The air-bubbles can only stick to the mineral particles if they can displace water from the mineral surface. It happens if the mineral is water repellent or hydrophobic. Most minerals are not water repellent in their natural state and flotation reagents must be added to the pulp.

The process of froth-flotation is used to separate a mineral or solid particulate matter from a mixture of finely grounded solids. Crude ore is ground to a fine powder and mixed with water, frothing reagents, and collecting reagents. When air is blown through the pulp, mineral particles stick to the bubbles, which both rise to form froth on the surface. The waste material (gangue) settles at the bottom. The froth with the concentrate is skimmed off and further dewatered.

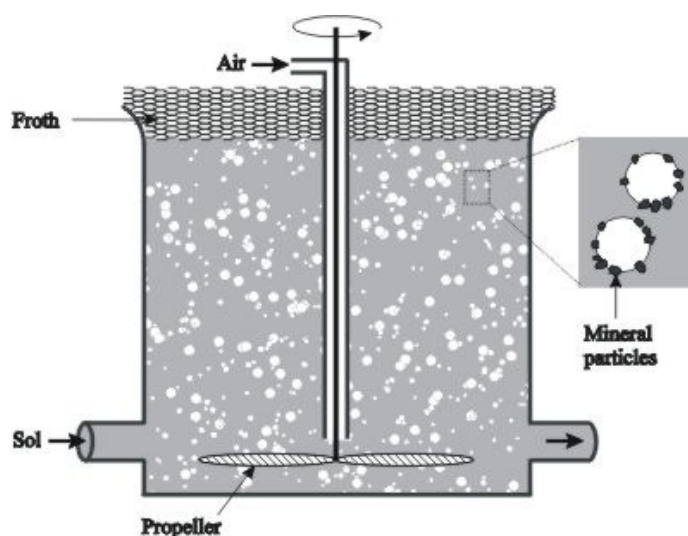


Figure 3.33 The illustration of froth flotation in Jameson cell.

3.5.2 The Equipment Used

Flotation can be performed in rectangular or cylindrical mechanically agitated cells or tanks, flotation columns or Jameson cells. Mechanical cells use a large mixer and diffuser mechanism at the bottom of the mixing tank to introduce air and provide mixing action. Flotation columns use air sparger. The Jameson cell uses neither impellers nor spargers, instead combining the slurry with air in a downcomer where high shear creates the turbulent conditions required for bubble particle contacting.

In the flotation experiments, a clone of Denver-12 mechanical flotation machine (figure 3.35), constructed in 1982 by “Baysan Makina Sanayi”, with a 2 liters stainless steel tank was used. The speed of impeller was adjusted to 1400-1500 rpm and the air pressure of 2.5 psi. The froths were manually skimmed with a plastic blade and collected in a zinc plate. The floated mineral was vacuum filtered and dried by a heating stove at 105°C.

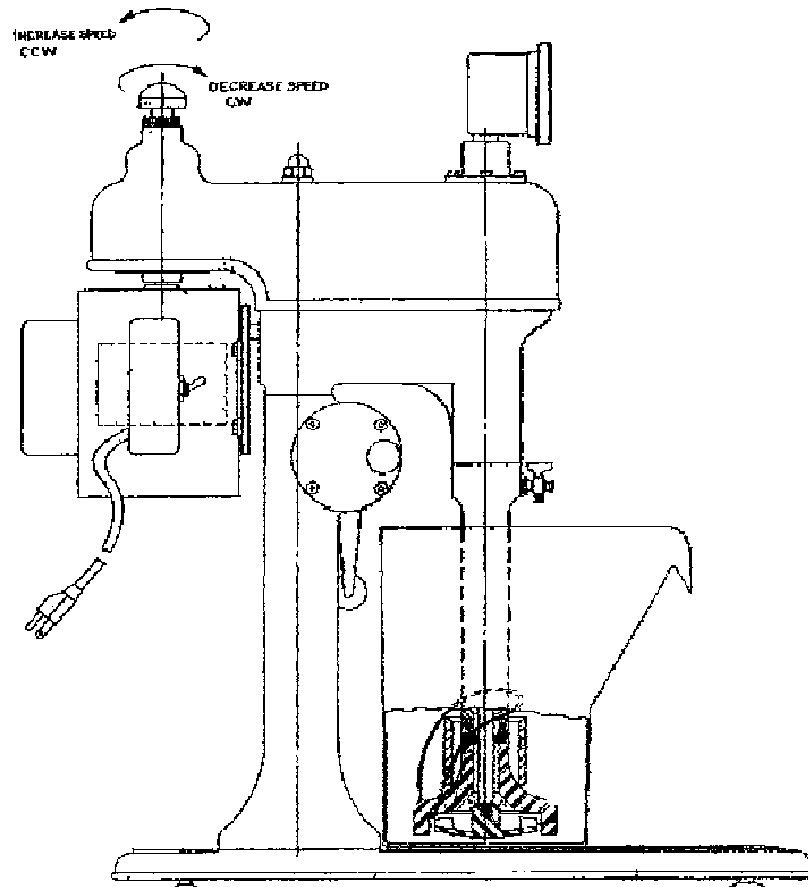


Figure 3.34 Schematic view of a Denver D12 Laboratory Flotation Machine (Lmine, 2009)



Figure 3.35 The Denver D12 flotation machine cloned by Baysan and its propeller.

The pH adjustments were done by using Denver Model 220 digital pH meter (figure 3.36).



Figure 3.36 The Denver Model 220 digital pH meter.

The chemical analyses of all samples (feed, concentrate and gang) were done by using “Analytik Jena – Nova 300” type atomic adsorption spectrum.



Figure 3.37 Analytik Jena – Nova 300 type atomic adsorption spectrum

3.5.3 The Flotation Agents

In general it was suggested that free gold floats in the presence of xanthate collectors and frothers but not if its particle size is too great or if reagents such as CaO or sodium sulfide are added to the pulp (Teague et al., 1999). Liberated gold, finer than about 150 μm floats readily with most collectors, in particular xanthates and dithiophosphates. The flotation response of free gold is better in pulps of high density (Dunne, 2005).

Flotation with xanthate collectors involves the anodic oxidation of the collector that may involve sub-processes such as metal xanthate formation, chemisorptions 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. This is a neutral oil that will adsorb onto the surface of any naturally hydrophobic solid, rendering it floatable (Gardner & 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. The longer-chain xanthates are more readily oxidized, generating dixanthogen at lower potentials (Gardner & Woods, 1974). An increase in thiol chain length increases the maximum contact angle, thereby increasing the hydrophobicity of the surface species. Both these attributes favor the use of longer chain xanthates, such as potassium amyl xanthate (PAX) for the flotation of free gold.

3.5.3.1 Collectors

The primary collector selected was Aero 350 (Potassium Amyl Xanthate, PAX). Aero 350 is the most powerful and least selective xanthate. Often used as a scavenger collector following a more selective rougher collector. Used widely in the flotation of Cu, Ni, Zn and Au containing iron sulfides. Aero 317 (Sodium Isobutyl Xanthate, SIBX) is a relatively strong collector used in the flotation of Cu, Pb, Ni, Zn and precious metals.

The effects of secondary collectors such as dithiophosphates and aerophine were also investigated. Aero 404 promoter is widely used for the flotation of tarnished and secondary Cu minerals, tarnished Pb and Zn minerals, and precious metals in alkaline circuits. Aero 404 (dithiophosphate/mercaptobenzothiazole) is an excellent collector for pyrite and auriferous pyrite in acid and neutral circuits. Aerophine 3418A promoter is a unique, P-based collector. Aero 3477 (Diisobutyl dithiophosphate) is a strong but selective collector for Cu, Ni and activated Zn minerals. It also improves recoveries of precious platinum group metals (Cytec, 2002).

3.5.3.2 pH Modifiers

The pH modifier selected for the initial experiments was lime. The depressing effect of lime on gold flotation was tested and investigated. Lime cannot be considered as just a pH modifier and studies have shown that calcium is strongly adsorbed on sulfide minerals and gold at pH values at and above 10 (Healy, 1984; Chryssoulis, 2001). Sodium hydroxide (NaOH) was used in further experiments as pH modifier.

3.5.3.3 Depressants

Sodium silicate (Na_2SiO_3) was used as depressant for non-sulfide gangue and dispersant to control slime coatings in flotation experiments. 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 & House, 1992; Lins & Adamian, 1993; Allan & Woodcock, 2001; Chryssoulis, 2001). All of these may competitively adsorb on the gold surface thus preventing the adsorption of the collector(s) added.

3.5.3.4 Activators

Copper sulfate (CuSO_4) was used as activator in flotation experiments. Early work on gold particles with copper sulfate showed no improvement in recovery but an increased rate of flotation of gold. 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. The reason for improved flotation recovery and rate is not understood as the mechanism of surface activation, if it exists, is different from that for sulfide minerals (Allan & Woodcock, 2001).

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 & Kushnikova, 1959), arsenopyrite (Gegg, 1949; O'Connor et al., 1990) and pyrite (Bushell & Krauss, 1962).

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. 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).

3.5.3.5 Sulfidizers

Sodium hydrosulfide (NaHS) was used as a sulfidizer in the flotation experiments. The application of sulfidizers (sodium sulfide and sodium hydrosulfide) to enhance the flotation of oxidized ores is well known (Jones & Woodcock, 1984; Oudenne & de Cuyper, 1986; O'Connor & 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 & Woolf, 1935).

3.5.3.6 Frothers

The frothers tested in flotation experiments are mainly product of Cytec Ind. Oreprep F-533, Oreprep X-133, Oreprep F-549, Oreprep F-597, Aerofroth 70 and pine oil were tested. Aerofroth 70 is an alcohol frother used when selectivity is important. The main frother used in flotation tests was pine oil. Pine oil is composed of various monohydric alcohols and other derivatives of terpene, with α -terpineol the major. It is a yellowish transparent oily liquid. Sparingly soluble in water. The pine oil is widely used in the flotation of various metallic and non-metallic minerals. It is mainly used in the flotation of various sulfide ores, such as lead, copper, zinc, and iron sulfide, and non-sulfide minerals. It exhibits some collecting properties, especially for readily floatable minerals, such as talc, graphite, sulphur, molybdenite

and coal etc. The froth produced by pine oil is more persistent than that produced by other frothers.

3.5.4 The Flotation Procedure

Flotation tests were accomplished with the Denver flotation cell at 1400-1500 rpm impeller speed and 2.5 psi air pressure. After grinding to the desired particle size, the pulp, with 625 g of sample weight and 30% solids, was transferred directly to the flotation cell with 2 liters volume. The conditioning time was 10 minutes for the collectors and 2 minutes for the frother. Flotation time was 4 minutes after testing periods of 2, 4, 8 and 16 minutes.

The flotation tests were accomplished first with primary collectors of potassium amyl xanthate (Aero 350) and xanthate sodium isobutyl (Aero 317) at different dosages of 50 to 200 g/t. The effect of secondary collectors [dithiophosphates/or modified dithiophosphates (Aero 404, Aero 3477 and Aerophine 3418A)] to float the gold were also investigated by adding them into the flotation cell after 2 minutes of flotation time with the primary collectors of Aero 350 and Aero 317. The pH was adjusted with lime at first and later with NaOH ranging from natural (around 5.6) to 9.5. The main frothers used were AeroFroth 70 (Cytec Industries Inc.) and pine oil. For sulfidization, different amounts of NaHS, for activation and depression, CuSO_4 and Na_2SiO_3 were used to increase gold recovery and grade. All the concentrate and tailings were collected, dried, weighed and chemically analyzed for Au.

The pulp in the cell was first stirred for 2 minutes to form a homogeneous pulp. After the pH adjustment, sulfidizer, depressant and activator were added in order. The primary collector was added and conditioned for 10 minutes. Aeration was applied during conditioning period of the collector. The frother was added in the last 2 minutes of conditioning. The pulp level was maintained constant by adding water as required. The concentrates were collected for 2 minutes and the froth was scrapped-off continuously.



Figure 3.38 Flotation in a 2 liter cell



Figure 3.39 The preparation of the samples by chemical decomposition for chemical analysis with AAS.

3.5.5 Primary and Secondary Collector Tests

Collector selection studies had the parameters of -106 microns particle size, pH 9.5 (adjusted with lime) and AF70 (35 g/t). The effects of secondary collectors were studied with 8 and 16 minutes of flotation. The secondary collectors were added after 8 minutes of flotation and compared with the flotation test with Aero 350 only. Also a second primary collector Aero 317 was studied. Another primary collector Aero 3477 was investigated at pH 12.5.

Table 3.16 The gold recoveries and grades for different primary and secondary collectors for 8 and 16 minutes of flotation.

Collector	Recovery at 8 min. (%)	Recovery at 16 min. (%)	Grade at 8 min. (g/t)	Grade at 16 min. (g/t)
Aero 350 + Aero 350	79.44	83.85	15.38	12.79
Aero 350 + Aerophine 3418A	77.73	84.06	13.64	9.88
Aero 350 + Aero 404	71.06	84.13	17.70	10.43
Aero 317 + Aero 317	80.44	88.23	17.68	12.93
Aero 3477 + Aero 3477	87.19	92.94	7.70	6.58

The effect of secondary collectors was negative since; adding the primary collector again in the second 8 minutes of flotation gave better results. The results showed that xanthate as primary collector (Aero 350 and Aero 317) reaches a reasonable gold grade with high recovery.

The flotation tests started with sulfidizer, depressant and activator at neutral pH. The comparison of Aero 350 with Aero 317 was studied at -106 microns particle size, neutral pH (5.5 - 5.8), collector (100 g/t), NaHS (0.1 kg/t), Na₂SiO₃ (1 kg/t) and CuSO₄ (200 g/t). Flotation duration was 4 minutes in total.

Table 3.17 The gold recoveries and grades for different primary and secondary collectors for 8 and 16 minutes of flotation.

Collector	Recovery at 2 min. (%)	Recovery at 4 min. (%)	Grade at 2 min. (g/t)	Grade at 4 min. (g/t)
Aero 350 + Aero 350	68.74	81.50	65.07	56.24
Aero 317 + Aero 317	68.90	86.21	38.42	37.14

The results show that, high gold recoveries and grades can be achieved with both collectors.

3.5.6 Frother Selection

Frother selection studies have two different procedures. The parameters of the first flotation studies were -106 microns particle size, pH 9.5 adjusted with lime, Aero 350 (100 g/t) and 35 g/t frother. The effects of frothers were studied with 8 and 16 minutes of flotation.

Table 3.18 The gold recoveries and grades for different frothers for 8 and 16 minutes of flotation.

Frother	Recovery at 8 min. (%)	Recovery at 16 min. (%)	Grade at 8 min. (g/t)	Grade at 16 min. (g/t)
Aero Froth 70	79.44	83.85	15.38	12.79
Oreprep X133	95.44	99.00	9.28	8.33
Oreprep 597	76.59	98.43	8.28	7.69
Oreprep 549	96.50	99.76	9.50	6.66

Aero Froth 70 reached a reasonable gold grade where other frothers did not work. The second flotation tests started with sulfidizer, depressant and activator at neutral pH. The comparison of AF 70 with pine oil was studied and higher gold recovery and grade was achieved by using pine oil. The constant variables were -106 microns particle size, neutral pH (5.5 - 5.8), Aero 350 (100 g/t), NaHS (0.1 kg/t), Na₂SiO₃ (1 kg/t) and CuSO₄ (200 g/t). The amount of frothers was 35 g/t for AF 70 and 100 g/t for pine oil. Flotation duration was 4 minutes in total.

Table 3.19 The gold recoveries and grades for AF70 and Pine oil for 2 and 4 minutes of flotation.

Frother	Recovery at 2 min. (%)	Recovery at 4 min. (%)	Grade at 2 min. (g/t)	Grade at 4 min. (g/t)
Aero Froth 70 (35 g/t)	56.56	68.93	88.21	49.07
Pine Oil (100 g/t)	68.74	81.50	65.07	38.42

The gold recovery in pine oil studies is significantly higher than AF 70. On the other hand, AF 70 reached 88.21 g/t gold grade in 2 minutes of flotation. The selection of frother is dependent on the aim of the study. Since, recovery is more important with reasonable gold grade, pine oil gives the optimum results for gold flotation.

3.5.7 Flotation Time

The parameters of the first flotation studies were -106 microns particle size, pH 9.5 adjusted with lime, Aero 350 (100 g/t) and AF 70 (35 g/t). Flotation duration was 24 minutes in total and samples were gathered and analyzed in 2, 4, 8, 16 and 24 minutes.

The gold grade decreases after 4 minutes of flotation remarkably however the gold recovery continues to increase at the end of 24 minutes. Flotation duration of 4 minutes is set as optimum to achieve a reasonable gold grade with high recovery.

Table 3.20 The gold recoveries and grades for different periods of flotation.

Time (min.)	Sample Weight (%)	Au (g/t)	Recovery (%)	ΣAu (g/t)	Σ Recovery (%)
2	13.3	26.40	56.09	26.40	56.09
4	9.2	11.45	16.77	20.30	72.86
8	9.9	4.18	6.58	15.38	79.44
16	8.7	3.18	4.41	12.79	83.85
24	6.7	2.48	2.67	11.33	86.52
Gang	52.3	1.61	13.48	6.25	-
Feed	100	6.25	-	6.25	-

3.5.8 Effect of Particle Size

The effect of particle size on flotation was investigated with three different grinding periods. The samples were ground for 10, 20 and 30 minutes. The maximum particle sizes were 106, 75 and 53 microns respectively. The flotation parameters were neutral pH (5.5 - 5.8), Aero 350 (100 g/t), pine oil (100 g/t), NaHS (0.1 kg/t), Na_2SiO_3 (1 kg/t) and CuSO_4 (200 g/t).

Table 3.21 The gold recoveries and grades for three different grinding periods for 2 and 4 minutes of flotation.

Grinding (min.)	Recovery at 2 min. (%)	Recovery at 4 min. (%)	Grade at 2 min. (g/t)	Grade at 4 min. (g/t)
10	68.74	81.50	65.07	38.42
20	61.15	73.63	57.65	30.84
30	49.96	67.78	49.85	28.88

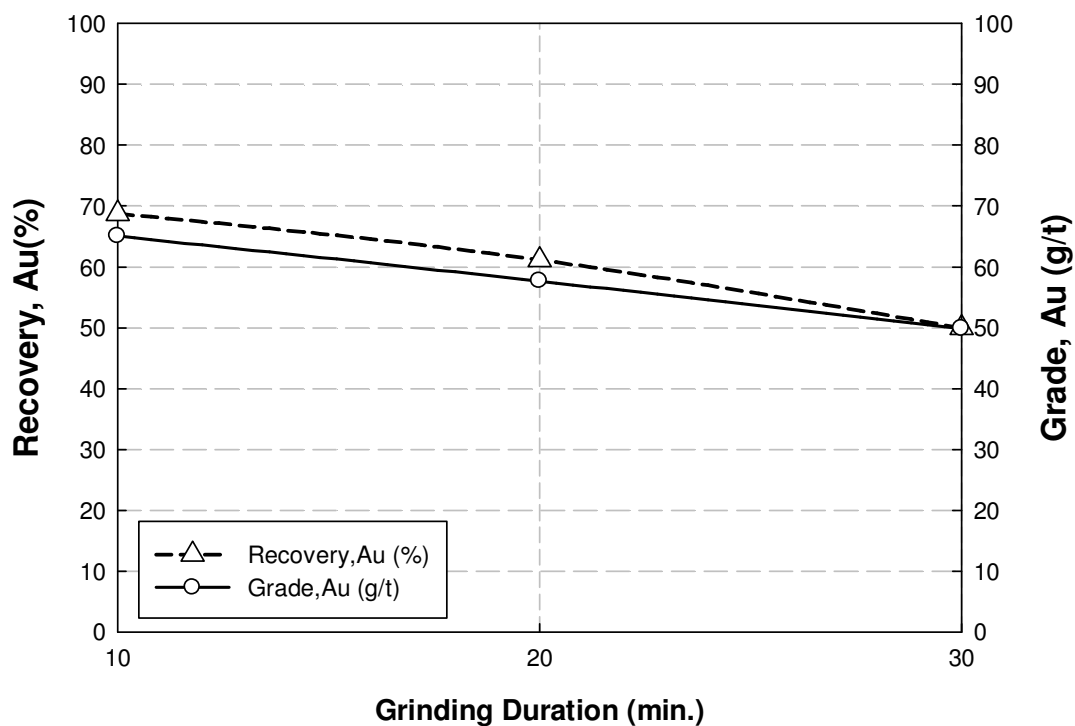


Figure 3.40 The gold recoveries and grades for different grinding periods for 2 minutes of flotation.

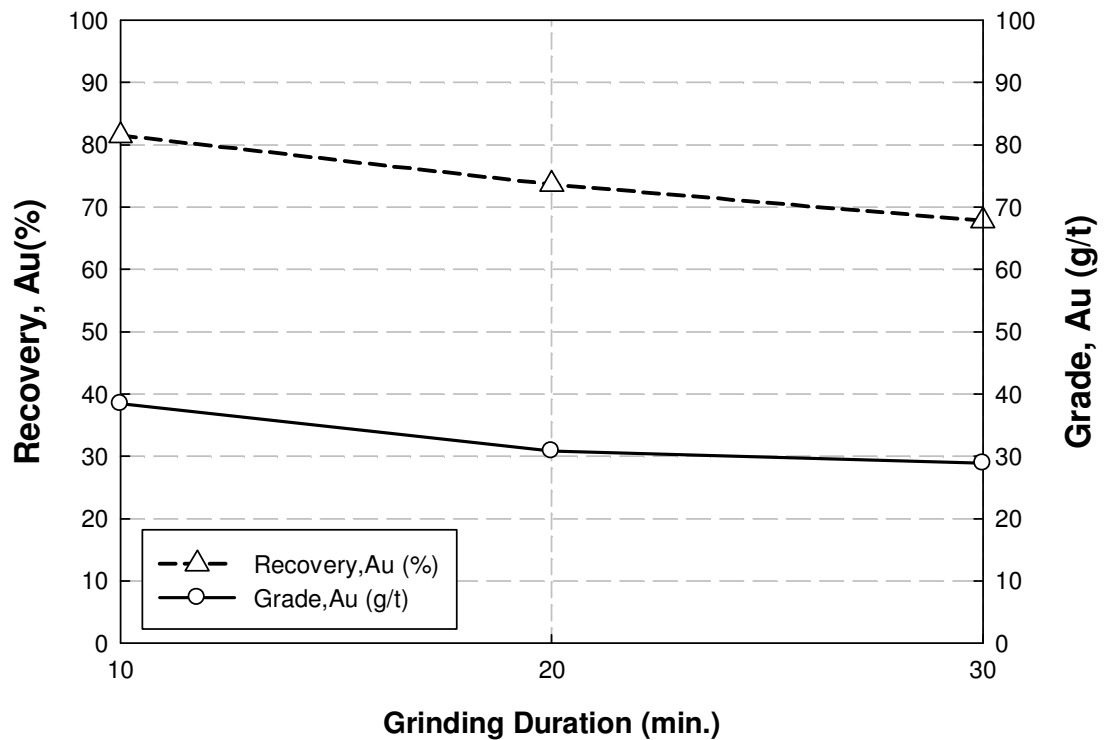


Figure 3.41 The gold recoveries and grades for different grinding periods for 4 minutes of flotation.

The decrease in gold recovery was unexpected since, it is well documented that the gold recovery increases with decreasing particle size. The result can be described by the insufficient dosages of reagents since the experiment parameters were constant. The finer the maximum particle size, the more liberated the gold as discussed in characterization of the sample. The more liberated the gold, the more reagent consumption is expected. The experiments are going to be repeated further with different dosages of collector, sulfidizer, depressant, activator and frother.

3.5.9 Effect of pH

There different pH levels were studied with constant parameters. The gold recovery and grades for 2 and 4 minutes of flotation were investigated with neutral pH (5.5 - 5.8), 7.5 and 9.5. The constant variables were -106 microns particle size, Aero 350 (100 g/t), pine oil (100 g/t), NaHS (0.1 kg/t), Na_2SiO_3 (1 kg/t) and CuSO_4 (200 g/t). Sodium hydroxide (NaOH) was used to adjust pH.

Table 3.22 The gold recoveries and grades for three different pH levels for 2 and 4 minutes of flotation.

pH	Recovery at 2 min. (%)	Recovery at 4 min. (%)	Grade at 2 min. (g/t)	Grade at 4 min. (g/t)
Neutral	68.74	81.50	65.07	38.42
7.5	71.12	77.85	45.90	31.63
9.5	64.53	73.24	52.31	27.55

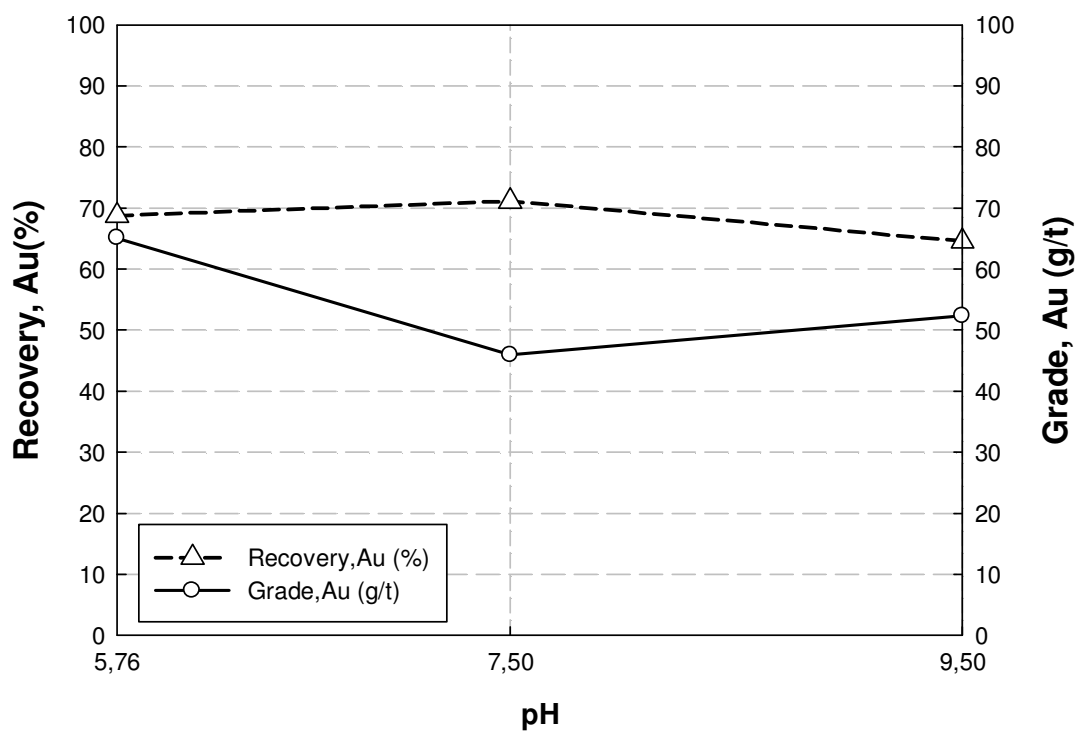


Figure 3.42 The gold recoveries and grades for different pH levels for 2 minutes of flotation.

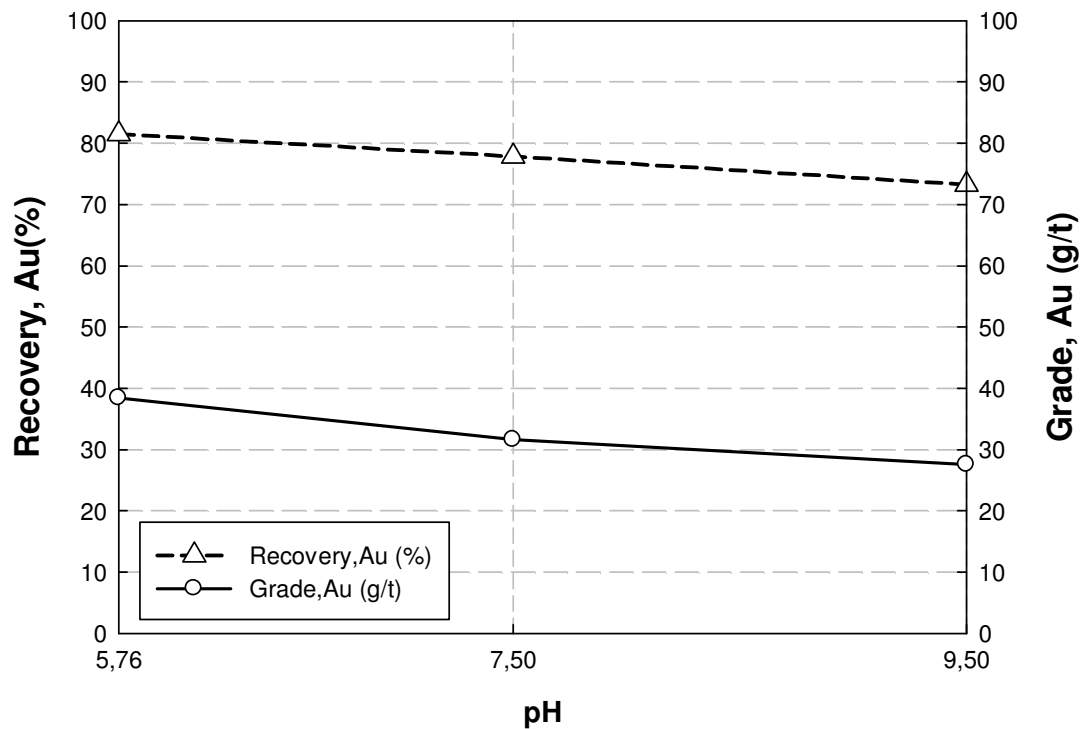


Figure 3.43 The gold recoveries and grades for different pH levels for 4 minutes of flotation.

The gold grade decreases with increasing pH. The highest grade of 65.07 g/t was achieved in 2 minutes flotation with neutral pH. The gold recovery was affected slightly by the increasing pH level. At pH 9.5, the gold recovery and grade suffers significantly at the end of 4 minutes. The optimum pH level is neutral pH with high recovery and acceptable gold grade.

3.5.10 Study on Collector Dosage

There different dosages of the primary collector Aero 350 were studied. The gold recovery and grades for 2 and 4 minutes of flotation were investigated with dosages of 50, 100 and 200 g/t Aero 350. The constant variables were –106 microns particle size, neutral pH (5.5 - 5.8), pine oil (100 g/t), NaHS (0.1 kg/t), Na₂SiO₃ (1 kg/t) and CuSO₄ (200 g/t).

Table 3.23 The gold recoveries and grades for different dosages of Aero 350 for 2 and 4 minutes of flotation.

Collector Dosage (g/t)	Recovery at 2 min. (%)	Recovery at 4 min. (%)	Grade at 2 min. (g/t)	Grade at 4 min. (g/t)
50	58.45	65.43	64.11	36.04
100	68.74	81.50	65.07	38.42
200	71.78	82.25	54.68	26.27

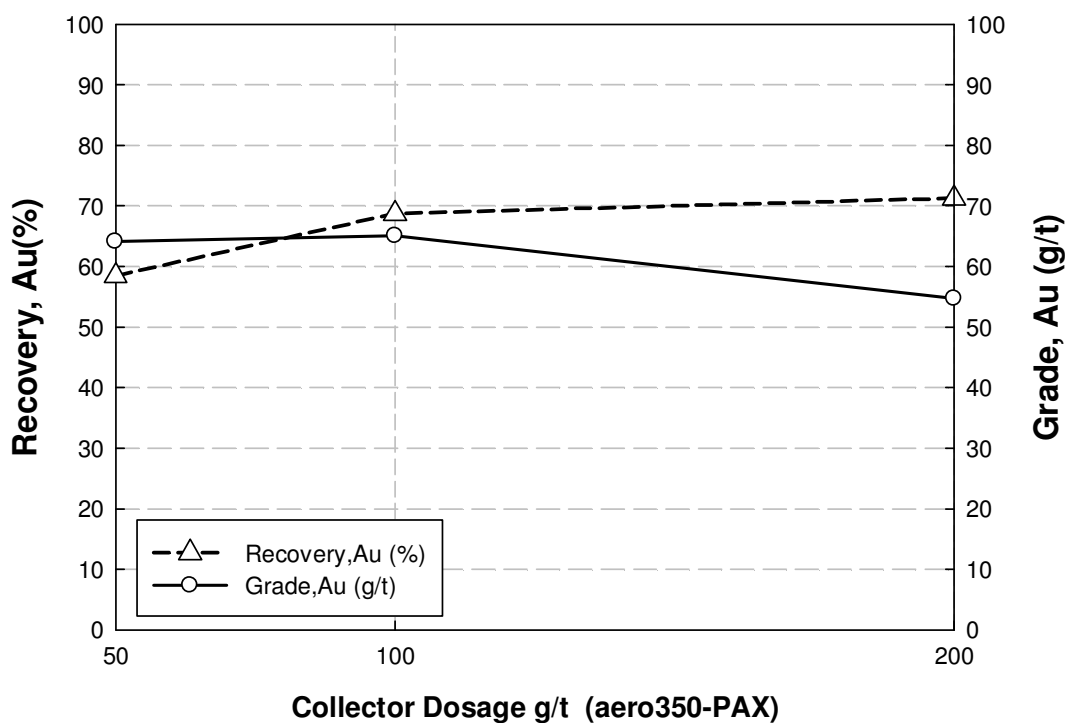


Figure 3.44 The gold recoveries and grades for different dosages of Aero 350 for 2 minutes of flotation.

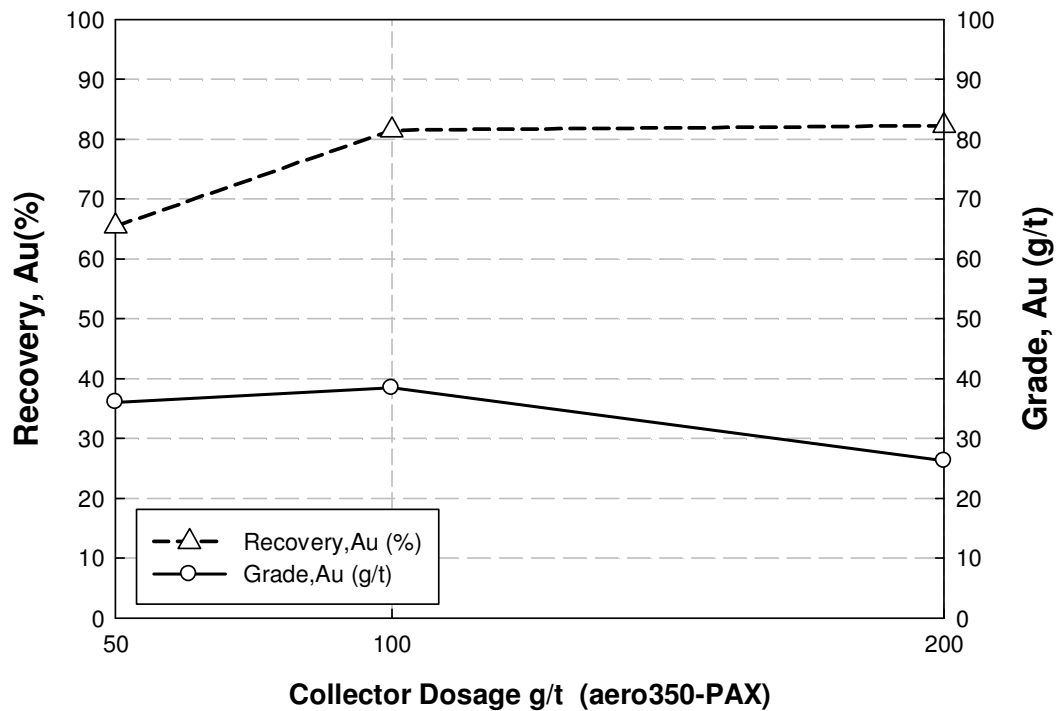


Figure 3.45 The gold recoveries and grades for different dosages of Aero 350 for 4 minutes of flotation.

The gold recovery increases significantly when the collector dosage increases from 50 to 100 g/t. The increase in recovery for 200 g/t dosage is not that remarkable, on the other hand the decrease in gold grade can be easily observed. The optimum primary collector dosage is 100 g/t of Aero 350. The change in the results is not sufficient for 200 g/t regarding the high collector consumption.

3.5.11 Effect of Sulfidizer (NaHS)

The effect of NaHS was investigated by adding sulfidizer, depressant and activator separately. The constant variables were -106 microns particle size, neutral pH (5.5 - 5.8), Aero 350 (100 g/t), pine oil (100 g/t), NaHS (0.1 kg/t), Na_2SiO_3 (1 kg/t) and CuSO_4 (200 g/t).

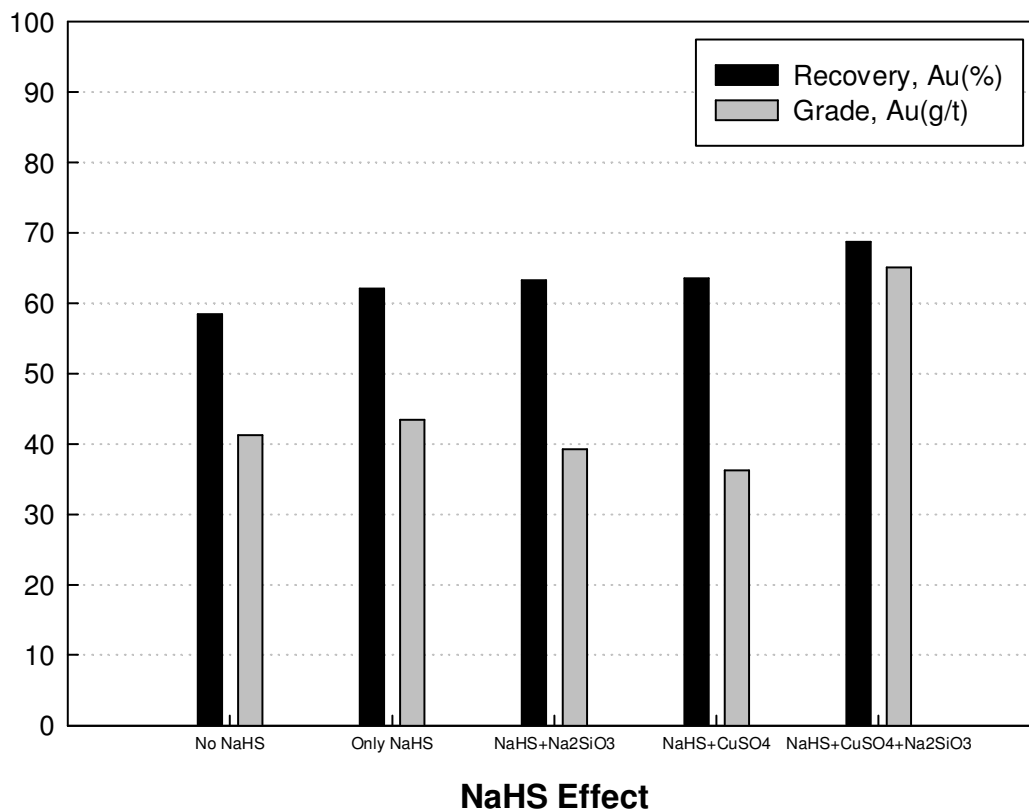


Figure 3.46 The effect of NaHS on flotation procedure for 2 minutes of flotation

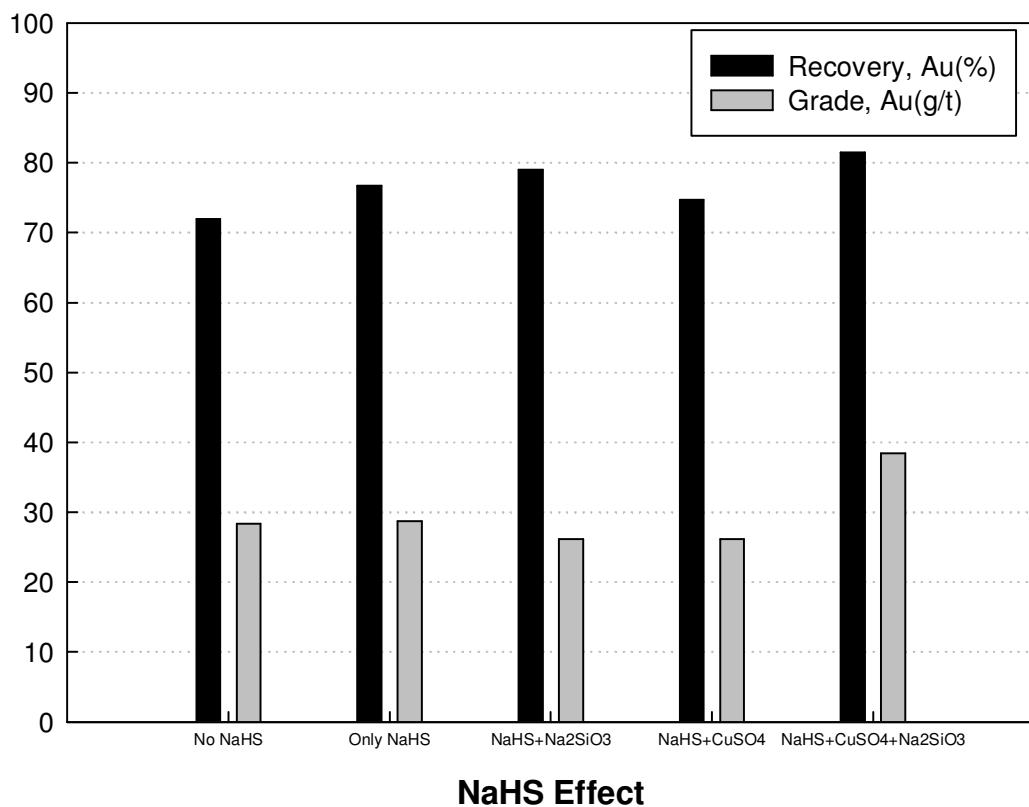


Figure 3.47 The effect of NaHS on flotation procedure for 4 minutes of flotation

The sulfidizer (NaHS) has positive effects on gold recovery if added single or with Na_2SiO_3 . The gold grade only increases when CuSO_4 is added. The effect of NaHS on gold grade for both 2 and 4 minutes of flotation is negligible.

3.5.12 Study on Sulfidizer (NaHS) Dosage

The effect of sulfidizer dosage was investigated with none, 0.05, 0.1 and 0.2 kg/t of NaHS. The constant variables were –106 microns particle size, neutral pH (5.5 - 5.8), Aero 350 (100 g/t), pine oil (100 g/t), Na_2SiO_3 (1 kg/t) and CuSO_4 (200 g/t).

Table 3.24 The gold recoveries and grades for different dosages of NaHS for 2 and 4 minutes of flotation.

NaHS Dosage (kg/t)	Recovery at 2 min. (%)	Recovery at 4 min. (%)	Grade at 2 min. (g/t)	Grade at 4 min. (g/t)
0	69.48	78.30	61.81	33.06
0.05	68.33	77.64	63.49	38.77
0.10	68.74	81.50	65.07	38.42
0.20	67.69	80.59	52.84	27.90

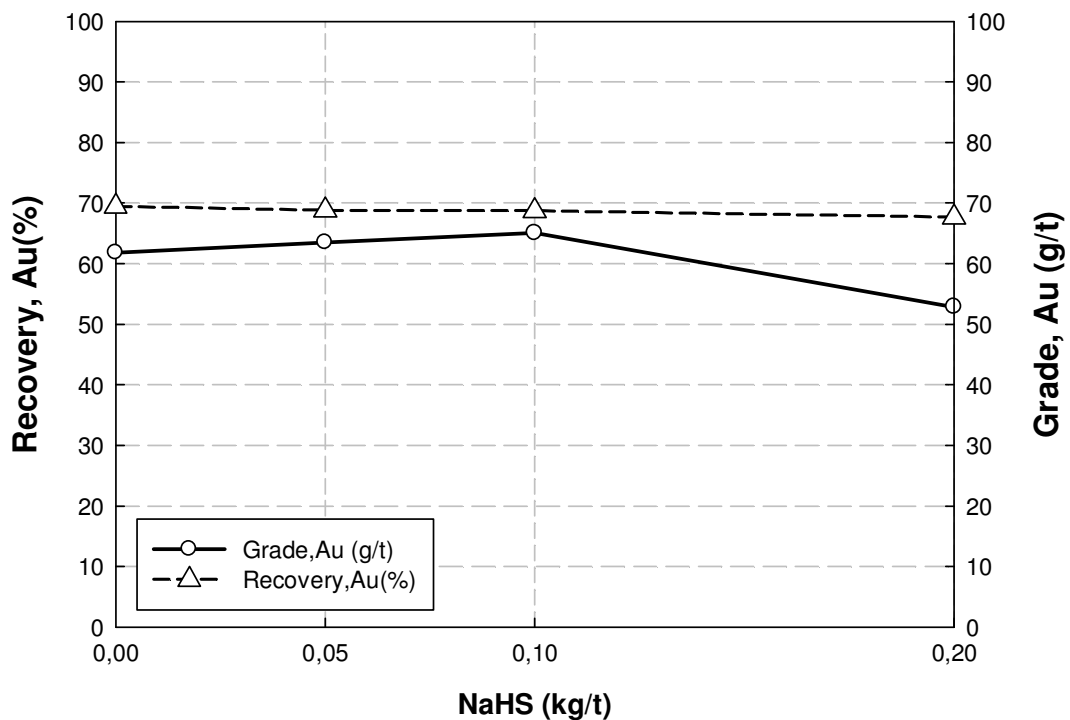


Figure 3.48 The gold recoveries and grades for different dosages of NaHS for 2 minutes of flotation.

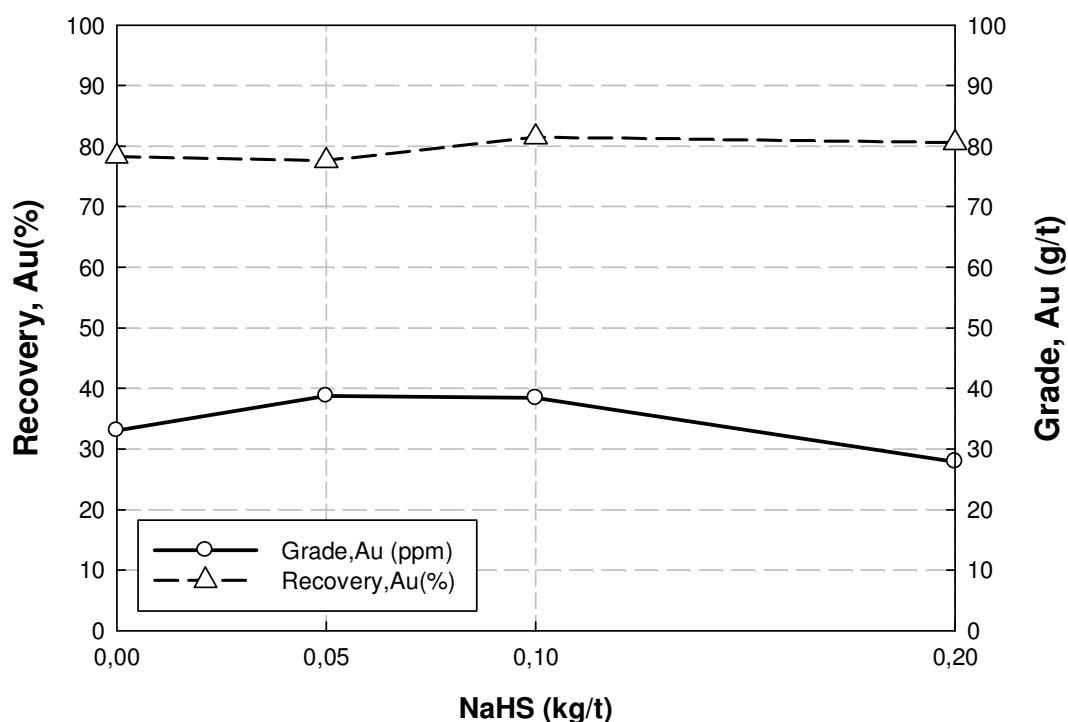


Figure 3.49 The gold recoveries and grades for different dosages of NaHS for 4 minutes of flotation.

The gold recovery and grades of 2 and 4 minutes of flotation are given above. It is observed that the dosage of NaHS has no effect on the gold recovery. On the other hand, the gold grade increases with increasing NaHS dosage till 0.1 kg/t. The depressing effect of NaHS is affecting the gold grade at higher dosages such as 0.2 kg/t and a significant decrease in gold grade is observed. The optimum dosage of NaHS for both 2 and 4 minutes of flotation is 0.1 kg/t with the highest gold recoveries and grades.

3.5.13 Effect of Depressant (Na_2SiO_3)

The effect of Na_2SiO_3 was investigated by adding sulfidizer, depressant and activator separately. The constant variables were –106 microns particle size, neutral pH (5.5 - 5.8), Aero 350 (100 g/t), pine oil (100 g/t), NaHS (0.1 kg/t), Na_2SiO_3 (1 kg/t) and CuSO_4 (200 g/t).

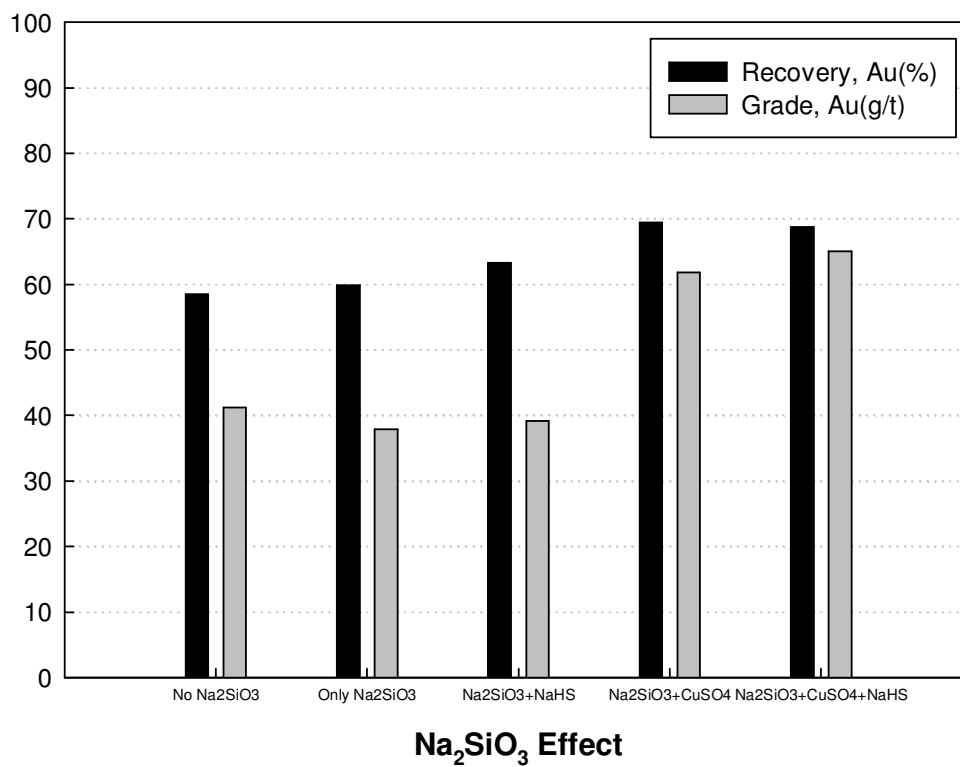


Figure 3.50 The effect of Na₂SiO₃ on flotation procedure for 2 minutes of flotation

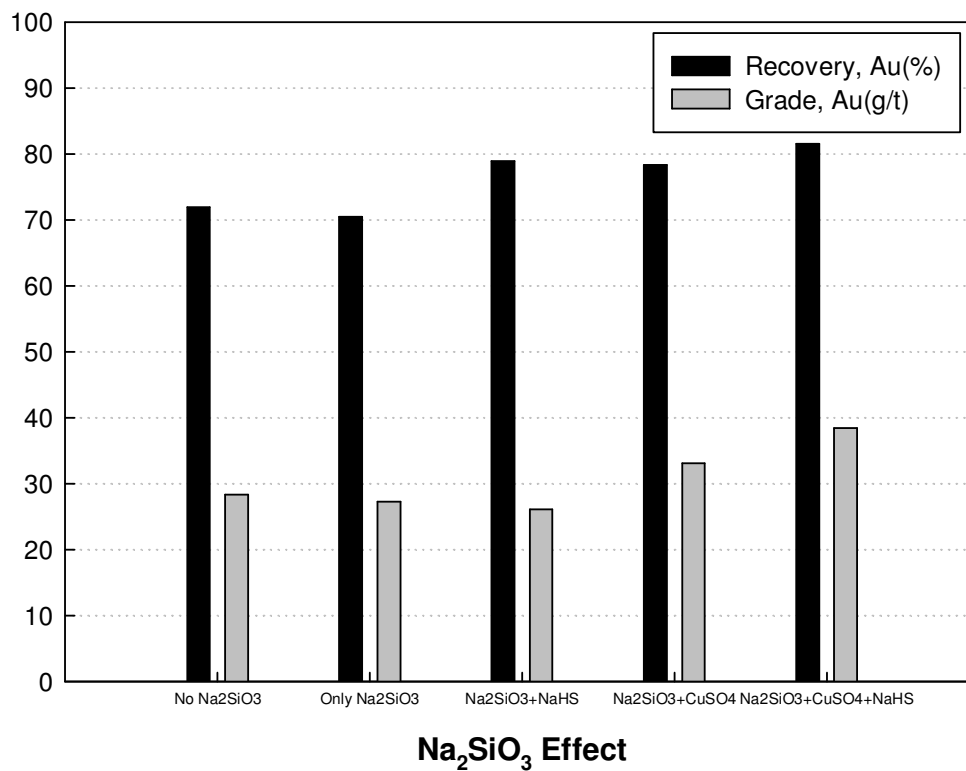


Figure 3.51 The effect of Na₂SiO₃ on flotation procedure for 4 minutes of flotation

The gold recovery was slightly increased with sodium silicate (Na_2SiO_3) addition. Sodium silicate with NaHS has positively affected the gold recovery however; the highest grades and recoveries were achieved after addition of copper sulfate.

3.5.14 Study on Depressant (Na_2SiO_3) Dosage

The effect of depressant dosage was investigated with none, 1 and 2 kg/t of Na_2SiO_3 . The constant variables were –106 microns particle size, neutral pH (5.5 - 5.8), Aero 350 (100 g/t), pine oil (100 g/t), NaHS (0.1 kg/t) and CuSO_4 (200 g/t).

Table 3.25 The gold recoveries and grades for different dosages of Na_2SiO_3 for 2 and 4 minutes of flotation.

Na_2SiO_3 Dosage (kg/t)	Recovery at 2 min. (%)	Recovery at 4 min. (%)	Grade at 2 min. (g/t)	Grade at 4 min. (g/t)
0	63.53	74.69	36.26	26.13
1	68.74	81.50	65.07	38.42
2	59.31	72.36	80.36	48.98

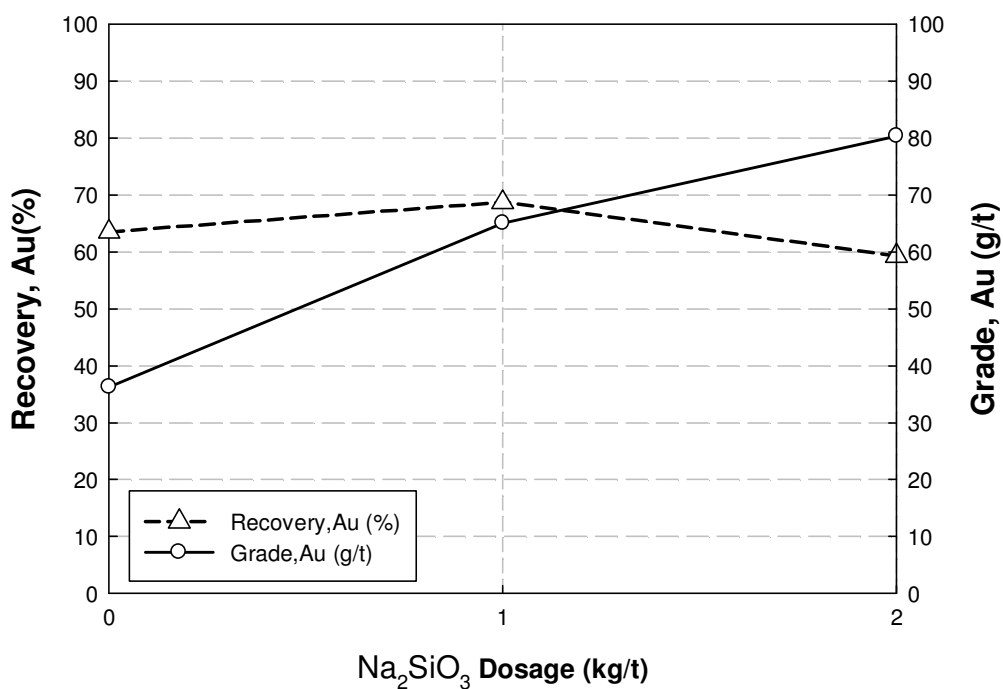


Figure 3.52 The gold recoveries and grades for different dosages of Na_2SiO_3 for 2 minutes of flotation.

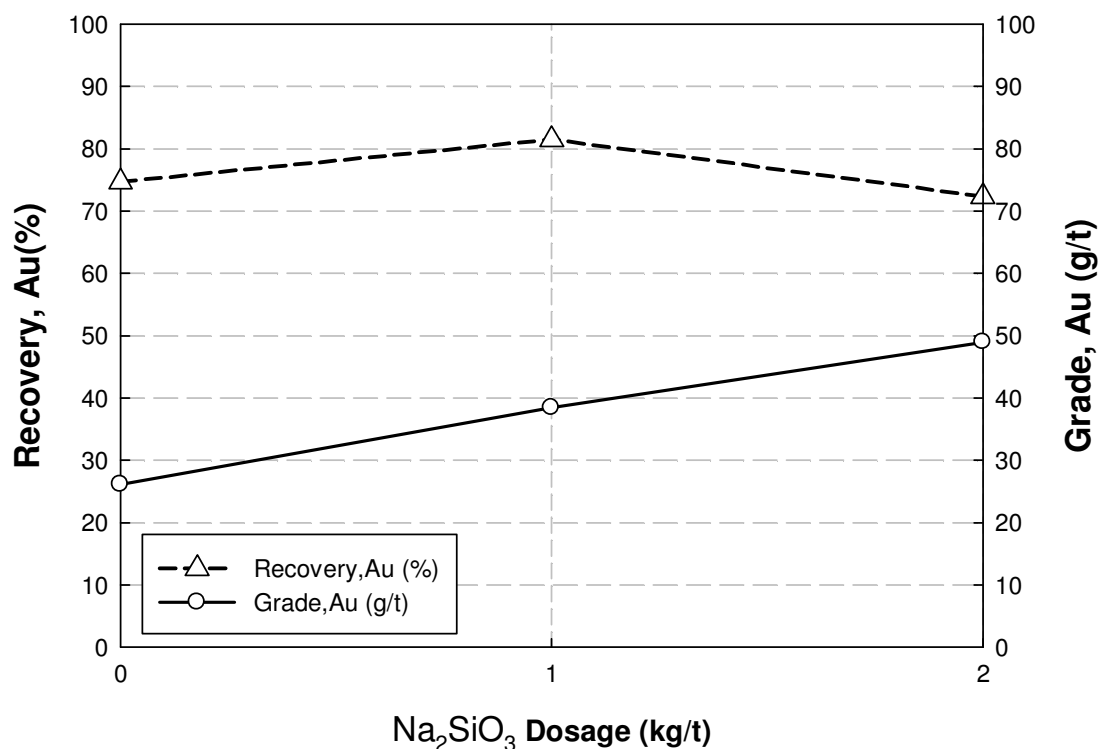


Figure 3.53 The gold recoveries and grades for different dosages of Na₂SiO₃ for 4 minutes of flotation.

The remarkable increase in gold grade with increasing sodium silicate amount is observed in flotation experiments. The grade increases from 36.26 to 80.36 g/t for 2 minutes of flotation with increasing dosage of sodium silicate. The highest gold recoveries were achieved by using 1 kg/t of sodium silicate for both 2 and 4 minutes of flotation.

3.5.15 Effect of Activator (CuSO₄)

The effect of copper sulfate was investigated by adding sulfidizer, depressant and activator separately. The constant variables were –106 microns particle size, neutral pH (5.5 - 5.8), Aero 350 (100 g/t), pine oil (100 g/t), NaHS (0.1 kg/t), Na₂SiO₃ (1 kg/t) and CuSO₄ (200 g/t).

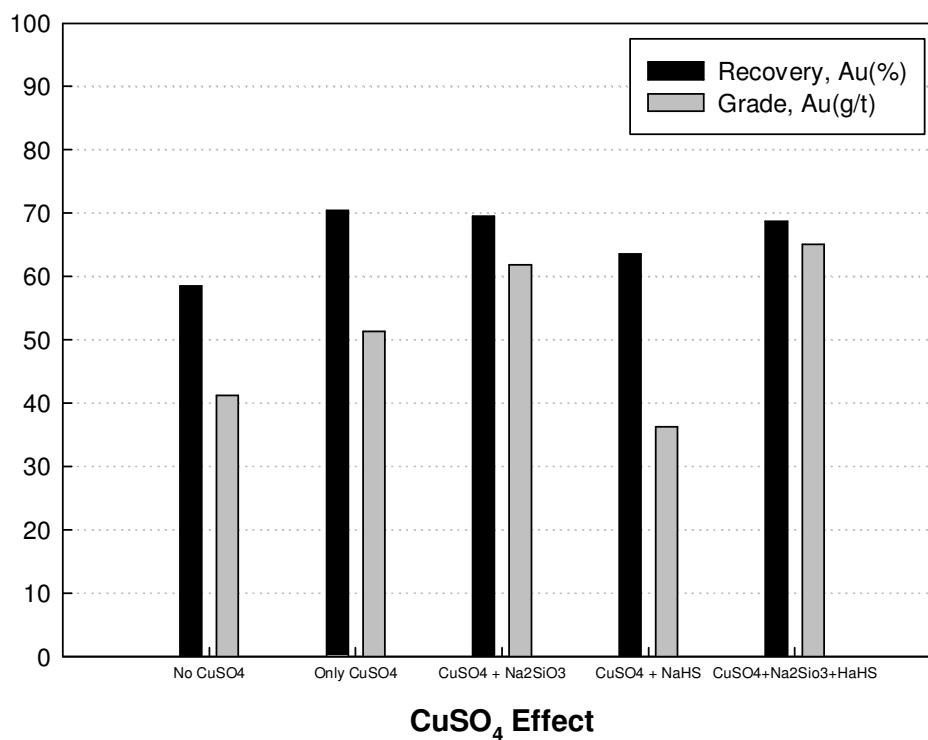


Figure 3.54 The effect of CuSO₄ on flotation procedure for 2 minutes of flotation

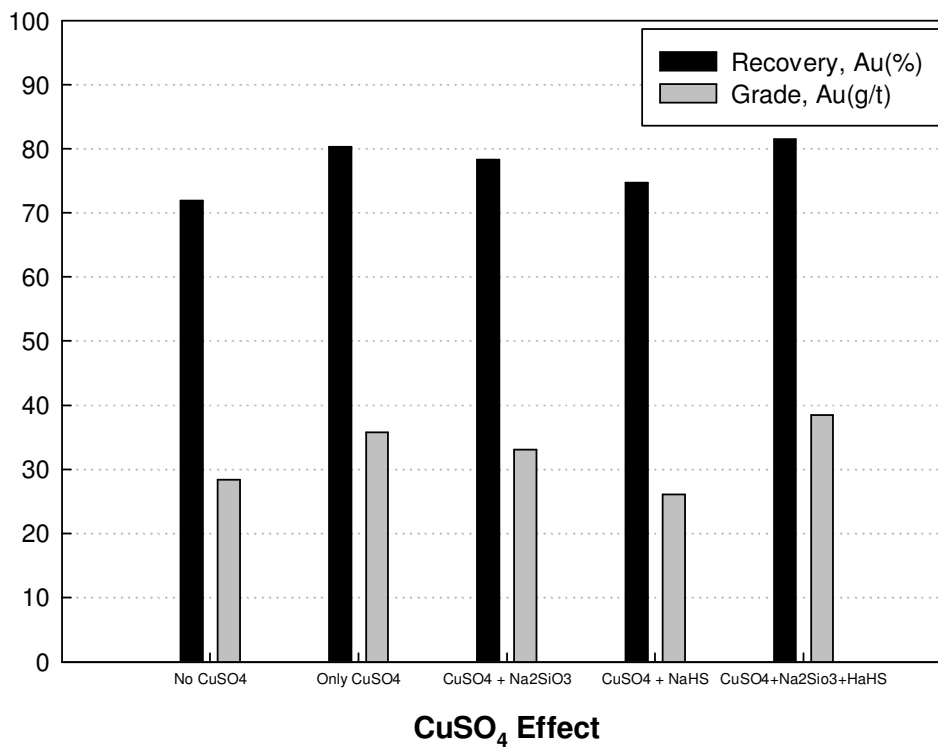


Figure 3.55 The effect of CuSO₄ on flotation procedure for 4 minutes of flotation

The effect of copper sulfate on both gold recovery and grade for 2 minutes of flotation is significant. The highest recovery and grade was achieved with the addition of the activator. Sodium silicate and copper sulfate additions affected the gold grade remarkably.

3.5.16 Study on Activator (CuSO_4) Dosage

The effect of activator dosage was investigated with none, 200 and 400 g/t of CuSO_4 . The constant variables were –106 microns particle size, neutral pH (5.5 - 5.8), Aero 350 (100 g/t), pine oil (100 g/t), NaHS (0.1 kg/t) and Na_2SiO_3 (1 kg/t).

Table 3.26 The gold recoveries and grades for different dosages of CuSO_4 for 2 and 4 minutes of flotation.

CuSO_4 Dosage (g/t)	Recovery at 2 min. (%)	Recovery at 4 min. (%)	Grade at 2 min. (g/t)	Grade at 4 min. (g/t)
0	62.87	82.11	37.61	21.66
200	68.74	81.50	65.07	38.42
400	76.84	85.07	76.07	44.06

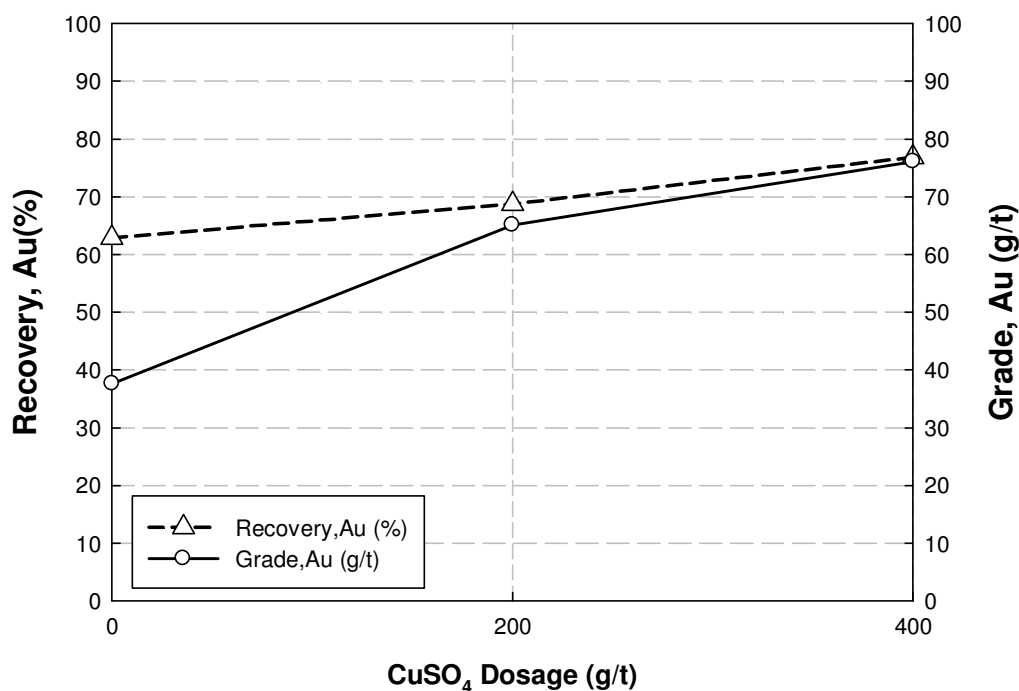


Figure 3.56 The gold recoveries and grades for different dosages of CuSO_4 for 2 minutes of flotation.

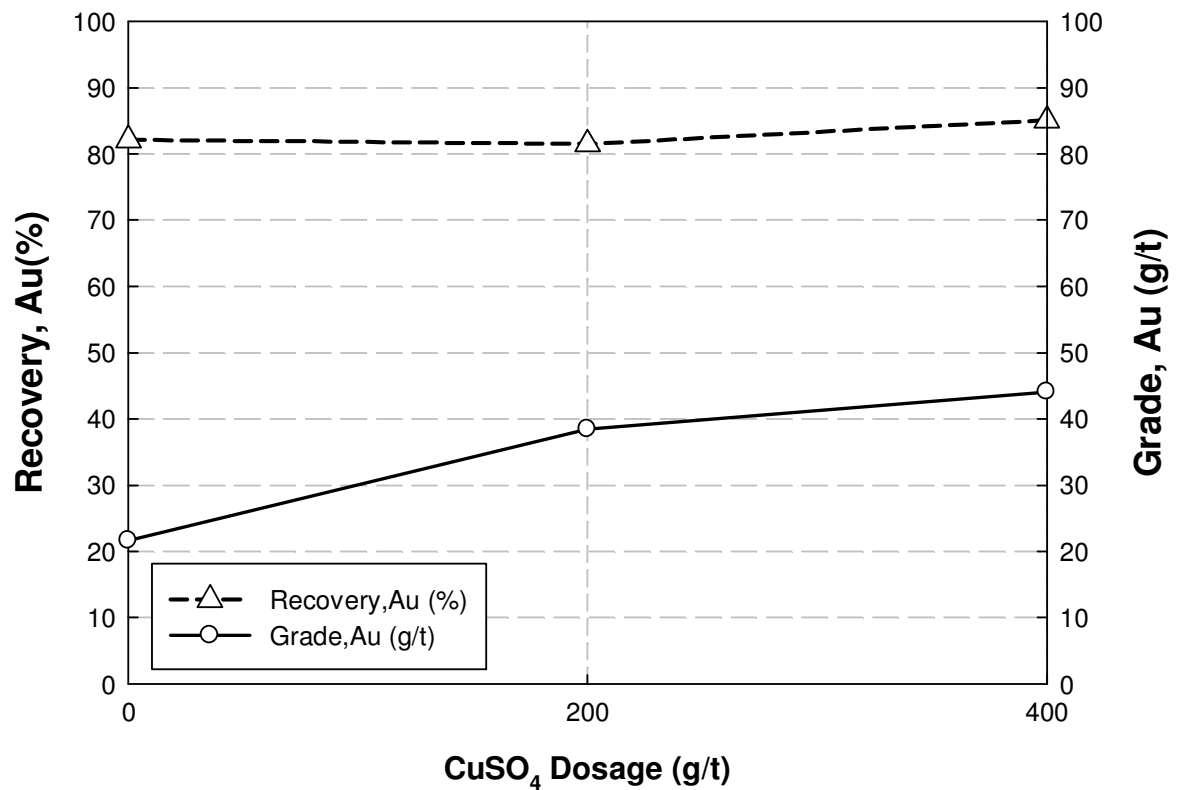


Figure 3.57 The gold recoveries and grades for different dosages of CuSO_4 for 4 minutes of flotation.

The gold recovery increased from 62.87 to 76.84 % with increasing copper sulfate amount in 2 minutes flotation. The increase in gold grade is more remarkable. In 2 minutes of flotation, the grade increased from 37.61 to 76.07 g/t. When flotation results at the end of 4 minutes are investigated, the effect of copper sulfate on recovery is negligible.

3.5.17 The Optimum Flotation Test Parameters

The experiment parameters of the optimum result achieved is listed below. The effect of particle size and higher dosages of copper sulfate will be studied further. Gold recovery of 85.1% with a grade of 44.1 g/t was achieved after 4 minutes of flotation with given parameters. The recovery was 76.84 % with a grade of 76.07 g/t at the end of 2 minutes.

Table 3.27 The variables and parameters of the optimum flotation test

Variables	Parameters
pH	Neutral (5.5 – 5.8)
Particle Size	- 106 microns
Flotation Time	2 + 2 minutes
Collector	Aero 350 (100 g/t)
Sulfidizer	NaHS (0.1 kg/t)
Depressant	Na ₂ SiO ₃ (1 kg/t)
Activator	CuSO ₄ (400 g/t)
Frother	Pine Oil (100 g/t)

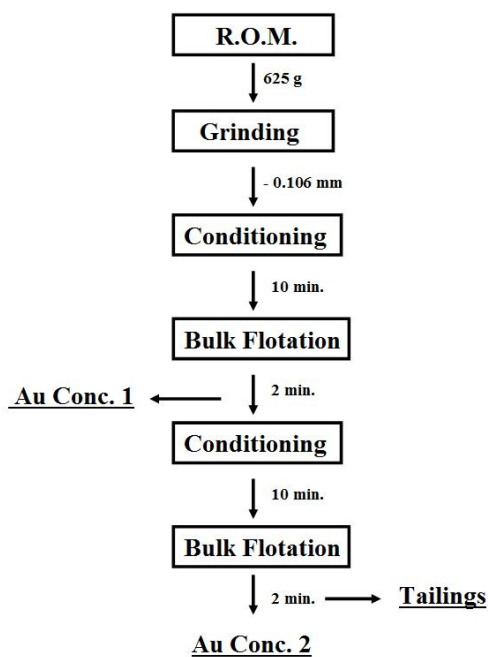


Figure 3.58 Flowsheet showing the bulk flotation circuit applied.

3.5.18 The Flowsheet Assessment combining Knelson Concentrator and Flotation

Gold recoveries of 71,44% with grades of 91,1 g/t are achievable with the Knelson gravity concentrator, however, these recoveries are considered very low when used alone in the circuit. On the other hand, higher gold recoveries (about 85.1%) but with relatively poor gold grades of 44.1 g/t were attained with flotation. Both the methods suffer to achieve an acceptable recovery and grade when used alone in the gold circuit.

A combination of Knelson gravity concentrator and flotation was considered to further increase the gold recoveries with acceptable gold grades. In this case, the tailings of the Knelson concentrator were scavenged to maximize the gold recovery. Knelson tailings contained average of 2 g/t gold grade. The generalized flowsheet arrangement for the combination of the Knelson gravity concentrator and the flotation is given in figure 3.59. Here, the final gold concentrate needs to be further processed to recover the gold. This can be accomplished by cementation, precipitation, cyanide leaching of the concentrate, biological or pressure oxidation, roasting processes.

When the Knelson concentrator tailings (middlings) were floated, a gold concentrate with 72.7% recovery and 14.1 g/t grades was achieved. The combination of the Knelson and flotation circuit brought the gold recovery from 71.4% to 92.2% with an average gold grade of 38.9 g/t.

Table 3.28. Gold recoveries and grades achieved with different circuit arrangements

Circuit arrangement	Recovery, %	Grade, Au g/t
Direct Knelson	71.4	91.1
Direct Leaching	82.2	-----
Direct Flotation	85.1	44.1
Knelson + Flotation	92.2	38.7

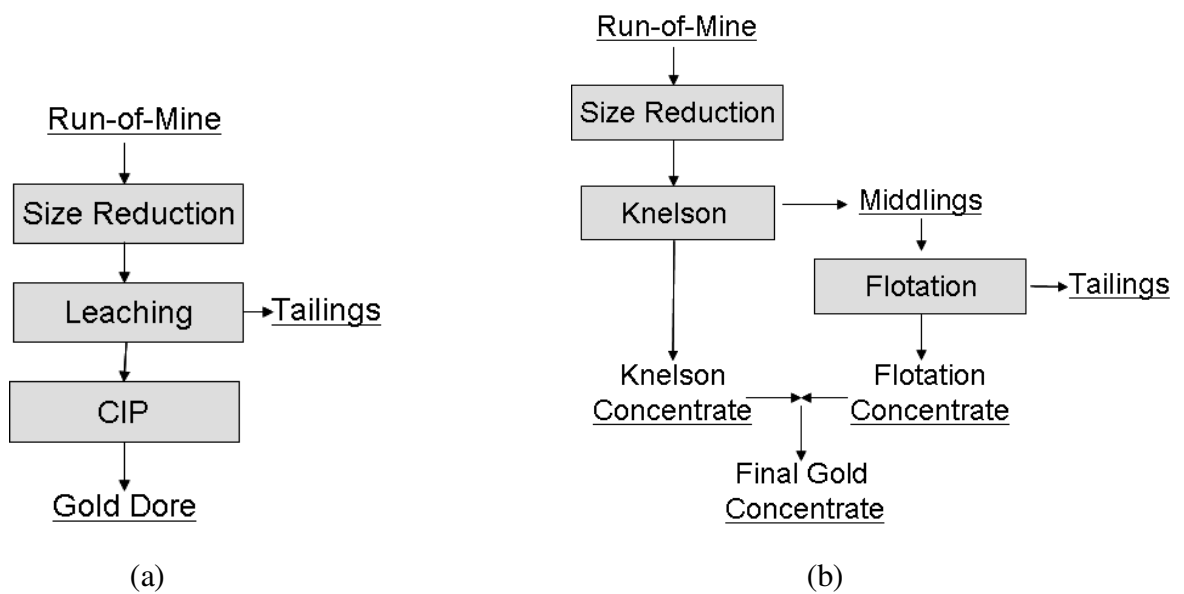


Figure 3.59 Generalized flowsheets showing (a) the conventional direct leaching and carbonization (CIP, carbon-in-pulp) method, b) the combination of the Knelson gravity concentrator and flotation arrangement.

CHAPTER FOUR

RESULTS AND DISCUSSION

4.1 Results and Discussion of Knelson Concentrator Studies

The aim of the Knelson concentrator studies was to determine the recoverability of Efemçukuru gold ore by gravity methods. The variables studied were the gravitational force, maximum particle size fed and water pressure.

The effect of G force on the gold recovery and grade was investigated. Three different G forces (60, 90 and 120 G) were applied with different experiment parameters. Experiments conducted in 120 G generally achieved the highest gold recovery and grade.

The effect of G force on samples with different particle top sizes (0.025, 0.075, 0.106, 0.150 and 0.212 mm) were investigated. Different G forces had different effects on each particle size. As the particle size got coarser, lower G forces became effective in recovery.

The sample with particle size of -0.025 mm did not give the expected results since; it was documented that as the gold is liberated, the recovery would increase. Where as, both gold recoveries and grades were insufficient regarding the whole study. The particle size of -0.025 mm was out of range for the recommended operating parameters of Knelson concentrator. So, it was tested that Knelson concentrator does not work well on very fine sizes. The highest recovery and grade achieved was at 90 G and 42.33 %, 108.13 g/t respectively.

The concentrator operating at 120 G was effective on both -0.075, -0.106 and -0.150 particle sizes. The highest gold recoveries and grades were achieved at 120 G. With increasing G force, a remarkable increase in recoveries was observed. The Knelson concentrator running in 120 G achieved a total recovery 67.4 % at the size of -0.075 mm with Au content 122.3 g/t. Total recovery rises from 26% to 67.4 % by

increasing the G force from 60 to 120 G. In the experiments with the particle size of -0.106 mm, recovery of 70.28% was achieved with Au content of 92.74 g/t. The gradual increase in recovery with increasing G force can be easily determined. The Knelson concentrator running in 120 G achieved a total recovery 59% at the size of 150 microns with Au content 89.2 g/t.

As the maximum particle size of the feed material got coarser, the effect of G force was undetermined. The experiments showed that, with the maximum feed size of 212 microns the increasing G force had a slightly negative effect on the recovery and grade of Au. The Au grade was between 78.3 and 87 g/t, while the recovery varied between 35.6% and 38.5%. The change in G force did not cause a significant change in both recovery and grade.

The water pressure increase had remarkable effect on the Au grade. The Au grade had increased from 73.80 g/t to 143.02 g/t while the water pressure had increased from 2 to 8 psi. The Au recoveries for both 2 and 8 psi water pressures were too low compared to 4 psi. The optimum test parameters of 120 G with the particle size of -0.106 mm were used for the studies of water pressure effect.

In general, the results show that gold recoveries increased with increasing G forces, except at the coarse size. This is expected as the liberation becomes problem at coarser sizes, giving poor recoveries. The effects of G forces on the recoveries are more pronounced at finer sizes, increasing substantially with increasing G forces.

Knelson gravity concentrator suffers to achieve acceptable gold recoveries when used alone. Thus, gravity cannot replace flotation and cyanidation (leaching) but it can reduce the circuit size, reagent use, and environmental impact.

4.2 Results and Discussion of Flotation Studies

The aim of the flotation investigation was to bulk float all the gold-containing minerals. Different variables were tested with different parameters to achieve the highest gold recovery with an acceptable gold grade. Primary collectors, secondary collectors, pH modifiers, activators, sulfidizers, depressants, frothers, particle sizes, flotation durations and pH levels were studied with different parameters.

In general it is well documented that the use of xanthate collectors in gold flotation is suggested. The aim of the bulk flotation is achieving high recoveries. One of the strongest xanthate type collectors Aero 350 was selected as the primary collector. The effects of secondary collectors were studied with 8 and 16 minutes of flotation. The secondary collectors were added after 8 minutes of flotation and compared with the flotation test with Aero 350 only. Also a second primary collector Aero 317 was studied. Another primary collector Aero 3477 was investigated at pH 12.5. The effect of secondary collectors was negative since; adding the primary collector again in the second 8 minutes of flotation gave better results. The results showed that xanthate as primary collector (Aero 350 and Aero 317) reaches a reasonable gold grade with high recovery.

The pH modifier selected for the initial experiments was lime. The depressing effect of lime on gold flotation was tested and investigated. Lime cannot be considered as just a pH modifier and studies have shown that calcium is strongly adsorbed on sulfide minerals and gold at pH values at and above 10 (Healy, 1984; Chryssoulis, 2001). Sodium hydroxide (NaOH) was used in further experiments as pH modifier.

The gold grade decreases after 4 minutes of flotation remarkably however the gold recovery continues to increase at the end of 24 minutes. Flotation duration of 4 minutes is set as optimum to achieve a reasonable gold grade with high recovery. The reactive consumption and feasibility is also taken account.

The effect of particle size on flotation was investigated with three different grinding periods. The samples were ground for 10, 20 and 30 minutes. The decrease in gold recovery was unexpected since, it is well documented that the gold recovery increases with decreasing particle size. The result can be described by the insufficient dosages of reagents since the experiment parameters were constant. The finer the maximum particle size, the more liberated the gold as discussed in characterization of the sample. The more liberated the gold, the more reagent consumption is expected. The experiments are going to be repeated further with different dosages of collector, sulfidizer, depressant, activator and frother.

The gold grade decreases with increasing pH. The highest grade of 65.07 g/t was achieved in 2 minutes flotation with neutral pH. The gold recovery was affected slightly by the increasing pH level. At pH 9.5, the gold recovery and grade suffers significantly at the end of 4 minutes. The optimum pH level is neutral pH with high recovery and acceptable gold grade.

The gold recovery increases significantly when the collector dosage increases from 50 to 100 g/t. The increase in recovery for 200 g/t dosage is not that remarkable, on the other hand the decrease in gold grade can be easily observed. The optimum primary collector dosage is 100 g/t of Aero 350. The change in the results is not sufficient for 200 g/t regarding the high collector consumption.

The application of sulfidizers (sodium sulfide and sodium hydrosulfide) to enhance the flotation of oxidized ores is well known (Jones & Woodcock, 1984; Oudenne & de Cuyper, 1986; O'Connor & Dunne, 1991). The sulfidizer (NaHS) has positive effects on gold recovery if added single or with Na_2SiO_3 . The gold grade only increases when CuSO_4 is added. The effect of NaHS on gold grade for both 2 and 4 minutes of flotation is negligible. It is observed that the dosage of NaHS has no effect on the gold recovery. On the other hand, the gold grade increases with increasing NaHS dosage till 0.1 kg/t. The depressing effect of NaHS is affecting the gold grade at higher dosages such as 0.2 kg/t and a significant decrease in gold grade

is observed. The optimum dosage of NaHS for both 2 and 4 minutes of flotation is 0.1 kg/t with the highest gold recoveries and grades.

The gold recovery was slightly increased with sodium silicate (Na_2SiO_3) addition. Sodium silicate with NaHS has positively affected the gold recovery however; the highest grades and recoveries were achieved after addition of copper sulfate. Sodium silicate (Na_2SiO_3) was used as depressant for non-sulfide gangue and dispersant to control slime coatings in flotation experiments. The remarkable increase in gold grade with increasing sodium silicate amount is observed in flotation experiments. The grade increases from 36.26 to 80.36 g/t for 2 minutes of flotation with increasing dosage of sodium silicate. The highest gold recoveries were achieved by using 1 kg/t of sodium silicate for both 2 and 4 minutes of flotation.

The effect of copper sulfate on both gold recovery and grade for 2 minutes of flotation is significant. The highest recovery and grade was achieved with the addition of the activator. Sodium silicate and copper sulfate additions affected the gold grade remarkably. The gold recovery increased from 62.87 to 76.84 % with increasing copper sulfate amount in 2 minutes flotation. The increase in gold grade is more remarkable. In 2 minutes of flotation, the grade increased from 37.61 to 76.07 g/t. When flotation results at the end of 4 minutes are investigated, the effect of copper sulfate on recovery is negligible.

The experiment parameters of the optimum result achieved can be listed as particle size -0.106 mm, pH neutral (5.5 - 5.8), 4 minutes of flotation time, primary collector Aero 350 (100 g/t), sulfidizer NaHS (0.1 kg/t), depressant Na_2SiO_3 (1 kg/t), activator CuSO_4 (400 g/t) and pine oil (100 g/t) as frother. Gold recovery of 85.1% with a grade of 44.1 g/t was achieved after 4 minutes of flotation with given parameters. The effect of particle size (with increasing reactive dosages) and higher dosages of copper sulfate will be studied further.

CHAPTER FIVE

CONCLUSION

In conclusion, recovering Efemçukuru gold ore was investigated with different aspects. Flotation and gravity techniques were both studied and optimum parameters were obtained.

Knelson gravity concentrator suffers to achieve acceptable gold recoveries when used alone. Thus, gravity cannot replace flotation and cyanidation (leaching) but it can reduce the circuit size, reagent use, and environmental impact.

Bulk Flotation studies with different variables were accomplished to achieve the highest gold recovery with an acceptable gold grade. Primary collectors, secondary collectors, pH modifiers, activators, sulfidizers, depressants, frothers, particle sizes, flotation durations and pH levels were studied with different parameters.

The experiment parameters of the optimum result achieved can be listed as particle size -0.106 mm, pH neutral (5.5 - 5.8), 4 minutes of flotation time, primary collector Aero 350 (100 g/t), sulfidizer NaHS (0.1 kg/t), depressant Na_2SiO_3 (1 kg/t), activator CuSO_4 (400 g/t) and pine oil (100 g/t) as frother. Gold recovery of 85.1% with a grade of 44.1 g/t was achieved after 4 minutes of flotation with given parameters.

The combination of the Knelson and flotation circuit brought the gold recovery from 71.4 % to 92.2 % with a gold grade of 38.9 g/t, thus, giving a better gold recovery than either the conventional cyanide leaching or flotation. The combined Knelson gravity concentrator - flotation circuit is a better alternative flowsheet than cyanidation at certain mines located near or next to agricultural and ecologically sensitive areas to reduce environmental concerns.

REFERENCES

- Ackerman, P. K., Harris, G. H., Klimpel, R. R. & Aplan, F. F. (2000). Use of xanthogen formats as collectors in the flotation of copper sulfides and pyrite. *Int. J. Min. Process* 58, 1–13.
- Ahmadi, R. & Shahsavari, R. (2008). Procedure for determination of ball Bond work index in the commercial operations.
- Aksoy, B. S. & Yazar, B. (1989). Natural hydrophobicity of native gold flakes and their flotation under different conditions. *In: Dobby, G. S., Rao, S. R. (Eds.), Processing of Complex Ores. Pergamon Press, New York*, 19–27.
- Allan, G. C. & Woodcock, J. T. (2001). A review of the flotation of native gold and electrum. *Miner. Eng.* 14 (9), 931–962
- Angove, J. (2005). Metallurgical testwork: Gold processing options, physical ore properties and cyanide management. *In: Mike D. Adams, (Ed.). Developments in Mineral Processing, Vol.15*, 97-107.
- Banisi, S., Laplante, A. R. & Marois, J. (1991). A Study of the Behaviour of Gold in Industrial and Laboratory Grinding, *CIM Bulletin, Nov 1991*, 72-78.
- Bond, F. C. (1961). Crushing and grinding calculations. *British Chemical Engineering* 6, 6-8.
- Boyle, R.W. (1979). The geochemistry of gold and its deposits. *Geol. Surv. Can. Bull.*, 280-554.
- Bradshaw, D. J. (1997). Synergistic effects between thiol collectors used in the flotation of pyrite. *Unpublished Ph.D. Thesis, University of Cape Town, South Africa*.

- Branham, A. & Arkell, B. (1995). The Mike Gold-Copper Deposit, Carlin Trend Nevada, in *Process Mineralogy XIII Qd, D. Hagni, TMS, Warrendale, PA (1995)*, 203-211.
- Broekman, B. R., Carter, L. A. E. & Dunne, R. C. (1987). Flotation. In: *The Extractive Metallurgy of Gold in South Africa, Monograph Series M7, vol. 1. South African Institute of Mining and Metallurgy, Johannesburg*, 235–275.
- Bulatovic, S. M. (1997). Flotation behaviour of gold during processing of porphyry copper-gold ores and refractory gold bearing sulfides. *Miner. Eng.* 10(9), 895–908.
- Burt, R. (1999). The Role of Gravity Concentration in Modern Processing Plants. *Minerals Engineering, Vol.12*, 1291-1300.
- Bushell, L. A. (1970). The flotation plants of the Anglo-Transvaal Group. *J. S. Afr. Inst. Min. Metall.* 70, 213–218.
- Bürg, G.H. (1930). Die Sichtbarmachung des feinverteilten goldes in goldhoffingen Erzen un ihre wirtschaftliche Bedeutung. *Metall. Erz.* 27 (13), 333–338.
- Chryssoulis, S. L. (2001). Using mineralogy to optimize gold recovery by flotation. *JOM* 53(12), 48–50.
- Chryssoulis, S.L., Cabri, L.J., Salter, R.S. (1987). Direct determination of invisible gold in refractory sulfide ores. In: Salter, R.S., Wyslouzil, D.M., McDonald, G.W. (Eds.). *Proc. of Intern. Symp. on Gold Metallurgy, Winnipeg, Canada. Pergamon Press, New York*, 235–244.

- Chryssoulis, S. L. & McMullen, J. (2005). Mineralogical investigation of gold ores. In: Mike D. Adams, (Ed.). *Developments in Mineral Processing, Vol.15*, 21-71.
- Cytec Ind. (2002). Mining Chemicals Handbook. Revised edition (106-125).
- Deniz, V. & Ozdag, H. (2003). A new approach to Bond grindability and work index: dynamic elastic parameters. *Minerals Engineering Journal* 16 (3), 211–217.
- Duane W. R. (1999). Gold prices and gold production. Evidence for South Africa. *Resources Policy*, 25, 69–76.
- Dunne, R. (2005). Flotation of gold and gold-bearing ores. In: Mike D. Adams, (Ed.). *Developments in Mineral Processing, Vol.15*, 309-343.
- Eldorado Gold Corporation (2006). Estimation of resources, Kestani Beleni structure, WT/Efemçukuru project, Turkey.
- Falcon, (2010). Falcon SB, Retrieved January 11, 2010 from <http://www.concentrators.net/falcon-sb-concentrators/falcon-sb-series-gravity-concentrators-separators.html>
- Fleming, C. A. (1998). Thirty Years of Turbulent Change in the Gold Industry, in *Proceedings, Canadian Mineral Processors, Mineral Processors Division, Ottawa, Canada*, 49-82.
- Gardner, J. R., Woods, R. (1974). An electrochemical investigation of contact angle and of flotation in the presence of alkyl xanthates, I, platinum and gold surfaces. *Aus. J. Chem.* 27, 2139–2148.
- Gegg, R. C. (1949). Milling and roasting at MacLeod-Cockshutt, *Can. Min. Metall. Bull* 42, 659–665.

Gold Miners Association (2006). Dünya Altın Üretimi, Retrieved December 12, 2009 from <http://www.altinmadencileri.org.tr/index.php?secim=9&mid=61>

Groot, D. R. (1987). The reactions of some thiol collectors at noble metals and pyrite electrodes. *Mintek Report No. M12*, 15.

Habashi, F. (2005). Gold – An historical introduction. In: *Mike D. Adams, (Ed.). Developments in Mineral Processing, Vol.15.*

Harkonen, I., Kojonen, K. K. & Johanson, B. (1999). The Early Proterozoic Suurikuusikko Refractory Gold Deposit, Kittila, Western Finnish Lapland, in *Proceedings 5th Biennial SGA meeting & 10th IAGOD meeting, London, UK*, 159-162.

Harley, M. & Charlesworth, E.G. (1994). Structural Development and Controls to Epigenetic, Mesothermal Gold Mineralization in the Sabie-Pilgrim's Rest Goldfields, Eastern Transvaal, *South Africa, Explor. Mining Geol*, 3, 3, 231-246.

Harris, D. (1984). The Knelson concentrator - applications in Australia, AUSIMM (Perth and Kalgoorlie), *Regional Conference on "Gold—Mining, Metallurgy and Geology"*, October 1984.

Healy, R. E. & Petruk, W. (1990). The Mineral Characteristics that Affect Metal Recoveries from the Cu, Zn, Pb, and Ag Ores from Manitoba. *CANMET, Dept. Energy, Mines and Resources, Canada, Internal Report MSL*, 68.

Healy, T. W. (1984). Pulp chemistry, surface chemistry and flotation. In: *Principles of Mineral Flotation, Wark Symposium, Series No 40. Australasian Institute of Mining and Metallurgy, Melbourne*, 43–56.

Hofineyer, P. K. & Potgeiter, G. A. (1983). The Occurrence of Psuedo-Hydrothermal Gold in Unisel Gold Mines, *Geol Soc. S.Africa, Spec. Publ. 7*, 271-274.

Honan S. G. & Luinstra W. F. (1996). Gravity Gold Concentrator at Hemlo Gold Mines Inc. Golden Giant Mine, *Proceedings, 28th Annual General Meeting, Canadian Mineral Processors, compiled by M. Mular, CIM, Mineral Processors Division, Ottawa, Canada*, 83-96.

Hoşten, Ç. (2002). Gravity Concentration. METU MinE 310 Lecture notes.

JKMRC CO. (2006). Procedure for BMWI Test.

Jones, M. H. & Woodcock, J. T. (1984). Application of pulp chemistry to regulation of chemical environment in sulfide mineral flotation. *In: Principles of Mineral Flotation, Wark Symposium, Series No. 40. Australasian Institute of Mining and Metallurgy, Melbourne*, 147–174.

Klimpel, R. R. (1997). An approach to the flotation of complex gold ores containing some free gold and/or some gold associated with easily floatable sulfide minerals. *In: World Gold '97. Australasian Institute of Mining and Metallurgy, Melbourne*, 109–113.

Klimpel, R. R. (1999). Industrial experiences in the evaluation of various flotation reagent schemes for the recovery of gold. *Min. Met. Process. 18(1)*, 1–11.

Knelson, (2010). How it works, Retrieved January 11, 2010 from www.knelsongravitysolutions.com/page417.htm

Kojonen, K. & Johanson, B. (1999). Determination of Refractory Gold Distribution by Microanalysis, Diagnostic Leaching and Image Analysis, *Mineralogy and Petrology*, 67, 1-19.

- Kojonen K, B., Johanson, H. E., O'Brien & L. Pakkanen (1993). Mineralogy of Gold occurrences in the Late Archean Hattu Schist Belt, Ilomantsi, Eastern Finland, *Geol. Survey of Finland, Special paper 17*, 233-271.
- Kucha, H., Plimer, I. R., (2001). Au-silicates from mesothermal gold deposits. In: Piestrzynski, E. (Ed.). *Mineral Deposits at the Beginning of the 21st Century*. Swets and Zeitlinger Publ., Lisse, Belgium, 775–778.
- Lakshmanan, V. I. & McCool, I. G. (1987). Heap Leaching of Gold Ores in Permafrost Conditions, *In Proceedings, Canadian Mineral Processors, Compiled by G. McDonald, CIM, Mineral Processors Division, Ottawa, Canada*, 590-614.
- Laplante A. R., Vincent, F. & Luinstra, W. F. (1996). A Laboratory Procedure to Determine the Amount of Gravity Recoverable Gold - A Case Study at Hemlo Gold Mines, *In Proceedings 28th Annual Meeting, Canadian Mineral Processors, compiled by M. Mular, CIM, Mineral Processors Division, Ottawa, Canada*, 69-82.
- Laplante, A. R., Woodcock, F. & Noaparast, M. (1995), Predicting gravity separation gold recoveries. *Minerals and Metallurgical Processing*, 74-79.
- Laurence E. B. (2009). Gold prices, cost of carry, and expected inflation. *Journal of Economics and Business*, 62 (2010), 35–47.
- Leaver, E. S. & Woolf, J. A. (1935). Flotation of gold, effect of sodium sulfide, U.S. Bureau of Mines Report of Investigation No. 3275, *Progress Report-Metallurgical Division, 11 Studies on the Recovery of Gold and Silver*, 23–38.
- Lins, P. J. D. & Adamian, R. (1993). Some chemical aspects of gold particles flotation. *In: XVIII International Mineral Processing Congress, Sydney, vol. 5. Australasian Institute of Mining and Metallurgy, Melbourne*, 1119–1122.

Livermore, J. S. (1996). The Jackling Lecture: Carlin Gold Exploration in Nevada Since the Newmont Discovery in 1961, *Mining Engineering*, 48, 6, 69-73.

Lmine, (2009). Denver D12 Laboratory Flotation Machine, Retrieved January 11, 2010 from http://www.lmine.com/Merchant2/merchant.mvc?Screen=PROD&Product_Code=24960&Category_Code=flotation_parts

Magdalinovi, N. (1989). A procedure for rapid determination of the Bond work index. *International Journal of Mineral Processing* 27 (1-2), 125–132.

Man, Y. T. (2002). Why is the Bond Ball Mill Grindability Test done the way it is done? *European journal of mineral processing and environmental protection*(1).

Marsden, J. & House, I. (1992). The Chemistry of Gold Extraction. *Ellis Horwood, London*, 597.

Meza S. L. A., Hartmann W. & Escobar C.A. (1994). Recovery of Placer Gold Using the Knelson Concentrator.

Micheletti, J. E. & Weitz, T. J. (1997). Winter Heap Leaching at Pegasus Gold's Beal Mountain Mine, in *Global Exploration of Heap Leachable Gold Deposits* (eds. D.M. Hansen, W. Petruk and R.D. Hagni), TMS, Warrendale, PA, 177.

Minerals Zone (2005). Industrial Minerals - Gold, Retrieved December 11, 2009 from <http://www.mineralszone.com/minerals/gold.html>

Mitrofanov, S. I. & Kushnikova, J. (1959). Adsorption of butyl xanthate and Cu^{2+} ion by pyrrhotite. *Min. Quarry Eng.* 25, 362–364.

Monte, M. B. M., Lins, F. F. & Oliveira, J. F. (1997). Selective flotation of gold from pyrite under oxidizing conditions. *Int. J. Miner. Process.* 51, 255–267.

Nicol, M. J. (1984). An electrochemical study of the interaction of copper (II) ions with sulphide minerals. In: Richardson, P.E., Srinivasan, S., Woods, R. (Eds.), *Electrochemistry in Minerals and Metal Processing*. Electrochem. Soc., Pennington, NJ.

Northwest Territorial Mint. (2005). Gold Historical Background, Retrieved January 13, 2010 from http://bullion.nwtmint.com/gold_history.php

Oberbilling, E. (1964). Flotation of antimony ores. *Min. Mag.* 111, 35–45.

Oberthür, T. & Frey, M. (1991). Mineralogical and Geochemical Investigation of the Ore Components from Witwatersrand-type Deposits: Their Genetic and Exploration Significance, *Proceedings ICAM91*, paper 39.

O'Connor, C. T., Bradshaw, D. J. & Upton, A. E. (1990). The use of dithiophosphates and dithiocarbamates for the flotation of arsenopyrite. *Miner. Eng.* 3(5), 447–459.

O'Connor, C. T. & Dunne, R. C. (1991). The practice of pyrite flotation in South Africa and Australia. *Miner. Eng.* 4 (7–11), 1057–1069.

Only Gold (2009). A Brief History of Gold, Retrieved January 13, 2010 from <http://www.onlygold.com/tutorialpages/historyfs.htm>

Oudenne, P. D. & de Cuyper, J. (1986). Reagents and flotation flow-sheet selection for the beneficiation of a complex sulfide ore containing copper and gold. In: *Proceedings, 2nd International Symposium on Beneficiation and Agglomeration, Bhubaneswar, India*, 358–364.

Oyman, T., Minerace, F. & Piskin, O. (2003). Efemcukuru B-rich epithermal gold deposit (Izmir, Turkey), *Ore Geology Reviews*, Vol. 23, 35-53.

- Pevzner, M. L., Shcherbakov, V. I. & Kosova, L. Y. (1966). Behaviour of gold during grinding (in Russian). *Tsvetn. Metall*, 39(5), 11–12.
- Petruk, W. (Ed.) (2000). Applied Mineralogy in the mining industry. *Chapter 6, Elsevier Science B.V.*, 111-133.
- Petruk, W. & Wilson, J. M. (1993). Silver and Gold in some Canadian Volcanogenic Ores, *Proceedings 8th IAGOD Meeting*, 105-117.
- Ramsay, E. (1978). Kerr Addison Mines Limited. In: *Pickett, D.E. (Ed.), Milling Practice in Canada, CIM Special Vol. 16. Canadian Institute of Mining and Metallurgy, Montreal*, 65–67.
- Rickard, T.A. (1917). The flotation of gold and silver mineral. In: *Rickard, T. A., Ralston, O. C. (Eds.), Flotation. Mining and Scientific Press, San Francisco*, 379–396.
- Silva, M. (1987). Placer Gold Recovery Methods. *Vol. 23, Colorado School of Mines (Special Publications)*.
- Simmons, G. L., Orlich, J. N., Lenz, J. C. & Cole, J. A. (1999). Implementation and start-up of N2TEC flotation at the Lone Tree Mine. In: *Parekh, B.K., Miller, J.D. (Eds.), Advances in Flotation Technology. The Society for Mining, Metallurgy and Exploration, Inc., Littleton, Colorado*, 183–195.
- Taggart, A. F. (1945). Handbook of Mineral Dressing, *Section 12. Wiley, New York*, 116–119.
- Teague, A. J., Van Deventer, J. S. J. & Swaminathan, C. (1999). A conceptual model for gold flotation, *Minerals Engineering*, 12(9), 1001-1019.

- Thomas, K.G., (1997). Barrick and gold characterization by SIMS and related techniques. In: Gillen, G., Lareau, R., Bennett, J., Stevie, J. (Eds.). *Secondary Ion Mass Spectrometry (SIMS XI)*. J. Wiley & Sons, Chichester, 793–797.
- Xu, J., Sun, W. & Jia, S. (1994). Mineralogical and Wallhock Alteration at the Jinqingding Gold Deposit in Jiaodong Penmsula, China, *Exploration and Mining Geol.*, 3, 1, 1-8.
- Van Aswegen, P. C., Haines, A. K. & Marais, A.J. (1988). Design and Operation of a Conmiercial Bacterial Oxidation Plant at Fairview, *Proceedings Randol Gold Symposium, Perth, Australia, Randol International Ltd., Lakewood, CO, USA*, 144-147.
- Yigit, O. (2004). Gold in Turkey - a missing link in Tethyan metallogeny. *Ore Geology Reviews* 28 (2006), 147-179.