DOKUZ EYLUL UNIVERSITY

GRADUATE SCHOOL OF NATURAL AND APPLIED SCIENCES

EVALUATION OF COAL-OIL ASSISTED GOLD FLOTATION AS A NOVEL PROCESSING METHOD FOR GOLD RECOVERY

by

Sezai ŞEN

April, 2007

IZMIR

EVALUATION OF COAL-OIL ASSISTED GOLD FLOTATION AS A NOVEL PROCESSING METHOD FOR GOLD RECOVERY

A Thesis Submitted to the
Graduate School of Natural and Applied Sciences of Dokuz Eylul University
In Partial Fulfillment of the Requirements for the Degree of Doctor of Philosophy in Mining Engineering, Mineral Processing

Sezai ŞEN

April, 2007
IZMIR

Ph.D. THESIS EXAMINATION RESULT FORM

We have read the thesis entitled **"EVALUATION OF COAL-OIL ASSISTED GOLD FLOTATION AS A NOVEL PROCESSING METHOD FOR GOLD RECOVERY"** completed by **SEZAİ ŞEN** under supervision of **PROF. DR. YAŞAR ÇİLİNGİR** and we certify that in our opinion it is fully adequate, in scope and in quality, as a thesis for the degree of Doctor of Philosophy.

Prof. Dr. Yaşar ÇİLİNGİR

Supervisor

Prof. Dr. Üner İPEKOĞLU	Prof. Dr. Ümit CÖCEN
Thesis Committee Member	Thesis Committee Member
Prof. Dr. Halim DEMİREL	Prof. Dr. Mehmet POLAT
Examining Committee Member	Examining Committee Member

Prof.Dr. Cahit HELVACI

Director

Graduate School of Natural and Applied Sciences

ACKNOWLEDGMENTS

I would like to express my sincere gratitude to my advisor Prof. Dr. Yaşar Çilingir for his guidance, advice and help throughout the study.

I would like to thank to the members of my Thesis Progress Committee; Prof. Dr. Ali Akar, Prof. Dr. Üner İpekoğlu, and Prof.Dr. Ümit Cöcen for their useful comments and suggestions.

I am also very much indebted to many people in my work; I would like to thank them all for their valuable support and helpful discussions; Resc. Asst. Doğan Karakuş, Resc. Asst. V. Taylan Engin, Chemist Fatih Turan and my dear colleague Mining Engineer Yavuz Selim Karahan.

I am also thankful to Asst. Prof. Dr. Ufuk Malayoğlu, Asst. Prof. Dr. Uğur Malayoğlu and Resc. Asst. Aylin Deliormanlı for their help.

I wish to thank to Koza Altın Isletmeleri A.Ş. and Tüprag Metal Madencilik San. Tic. A.Ş. for their help in supplying the gold ore samples.

A very special appreciation goes to my sweetheart my wife Belgin Şen, to my mother Mutia Şen and to my father Arif Şen. Thank you so much for your patience and endless support.

There may be some omissions and/or misinterpretations. Thus, I entreat my colleagues and experts in this field to make corrections or additional meanings and send it to my attention through the university.

Izmir / Turkey Sezai ŞEN

April - 2007

EVALUATION OF COAL-OIL ASSISTED GOLD FLOTATION AS A NOVEL PROCESSING METHOD FOR GOLD RECOVERY

ABSTRACT

The study was aimed at investigating the effectiveness of the process of coal-oil agglomeration and gold flotation first individually and then in combination for enhancing gold recovery and for revealing the mechanism of the each processes.

The process is still burdened by the disadvantages of the flotation process but it has been clearly shown that assistance of coal-oil agglomerates improves the gold flotation performance. Relatively high recoveries and superior grades can be obtained by coal-oil assisted gold flotation in comparison to the conventional flotation process. Such recoveries are possible by coal-oil assisted gold flotation even in the absence of any recycling stages (a concentrate assaying about 400 g /t with a recovery about 72 % was obtained by using the optimal parameters). However, recycling of the coal-oil-gold agglomerates further improves the grade of the concentrates while decreasing the coal and oil consumption (a concentrate assaying 1320 g/t with a recovery about 63 % was obtained after four cycles of the loaded agglomerates). The experiments show that employing the scavenger stages in the process enhances the gold recovery. The concentrate gold recovery was improved to 88 % and the grade of the concentrate decreased to 258 g/t by employing a scavenger stage. A concentrate assaying 178.27 g/t Au with 90.25 % gold recovery was obtained by employing secondary scavenger stage.

Our results show that unlike previous work on coal-oil-gold agglomeration, our method requires very small amounts of coal and agglomerating agents. Therefore, it can be called as “coal-oil assisted gold flotation” and seems to be quite effective for gold-ore processing. The test work demonstrated that use of oil and coal particles to assist gold flotation increases the process performance significantly. Presence of flotation collectors in the pulp helps coal-oil-gold agglomeration, which is in turn enhances the flotation and therefore increases the overall recovery.

Keywords: Gold, agglomeration, oil, coal, flotation

ALTIN ZENGİNLEŞTİRMESİNDE YENİ BİR YÖNTEM OLARAK KÖMÜR-YAĞ YARDIMCILI ALTIN FLOTASYONU METODUNUN DEĞERLENDİRİLMESİ

ÖZET

Bu çalışma, kömür-yağ aglomerasyonu ve flotasyon yöntemlerinin kendine özgü çalışma mekanizmaları ve etkinliklerinin öncelikle ayrı ayrı daha sonra da altın kazanımı için birlikte kullanılmaları durumunda ele alınarak belirlenmesini hedeflemiştir.

Yöntem hala geleneksel flotasyon işleminin dezavantajlarını taşımasına rağmen, kömür-yağ aglomeratlarının kullanımının flotasyon işleminin performansını arttırdığı açıklıkla görülmüştür. Geleneksel flotasyon yöntemi ile karşılaştırıldığında yöntem göreceli olarak daha yüksek tenör ve verim değerleri sağlamaktadır. Herhangi bir yeniden yükleme kademesi kullanılmadan 400 g/t tenör değerine sahip bir konsantre % 72 civarı altın verimi ile üretilebilmiştir. Üretilen aglomeratların yeniden yükleme kademelerinde kullanılması ile dördüncü yükleme kademesi sonunda konsantre tenörü 1320 g/t a yükseltilebilmiş, altın kazanma verimi ise % 63 olarak gerçekleşmiştir. İşlem sırasında süpürme kademelerinin kullanılması ile altın verimi daha da yükseltilebilmektedir. Tek kademe süpürme işlemi kullanımı ile konsantre altın kazanma verimi % 88, konsantre tenör değeri ise 258 g/t olarak gerçekleşmiştir. İkinci kademe süpürme işlemi uygulanması ile konsantre altın kazanma verimi % 90 a yükselmiş, konsantre altın tenörü ise 178 g/t olarak belirlenmiştir.

Elde edilen sonuçlar incelendiğinde kömür-yağ-altın aglomerasyonu konulu çalışmalara göre çok daha az miktarda kömür ve yağ kullanıldığı görülmektedir. Bu nedenle yöntem "kömür-yağ yardımcılı altın flotasyonu" olarak adlandırılabilir. Kullanılan yağ ve kömür ile altın flotasyonu için kullanılmakta olan reaktiflerin, işlem sırasında uyum içinde, işlemin toplam etkinliğini artıracak yönde işlev yaptıkları gözlenmiştir. Bu sayede yöntemin uygulanması ile oldukça başarılı sonuçlar elde edilebilmiştir.

Anahtar Sözcükler: Altın, aglomerasyon, yağ, kömür, flotasyon

CONTENTS

Page

CHAPTER ONE
INTRODUCTION

1.1 General

Gold is one of the first metals that have been mined and used by humans due to its native, malleable and ductile metal form. It has been known and highly valued since the prehistoric times. Gold objects as old as 5,000 years have been found in Egypt, it was used for personal ornamentation or in decorating tombs and temples. Throughout the ages mankind have cherished gold, and many have had a compelling desire to amass great quantities of it. This compelling desire to seek and hoard gold has caused great wars and conquers like the petroleum in our time.

Gold is called as a “noble” metal because of its imperishable form under ordinary conditions. Its chemical symbol *Au* is derived from the Latin word “aurum.” It has a metallic luster and is sun yellow in pure form, but it can be found in the form of alloys with other metals, such as silver, copper, nickel, platinum, palladium, tellurium, and iron in various color hues ranging from silver-white to green and orange-red. Gold and its many alloys are most often used in jewelry, coinage, some industrial uses and as a standard for monetary exchange by the International Monetary Fund (IMF) and the Bank for International Settlements (BIS).

Gold mining is vital to the economies, especially for the developing countries. Mining gold and setting mineral processing facilities simply generate tax income to the government. Furthermore, if the country has a significant jewellery manufacturing industry like Turkey, utilizing the native resources carries great importance. Thus, mining gold could generate export revenue but mining and using gold for your own jewellery industry increases profit ratio. Gold production in a country can also bring some improvements to the formal mining industry about how gold is mined and produced around the world at modern operations. It is a fact that gold mining in our time uses some advanced mining techniques, using these

techniques for the production of gold provides technology transfer, worker training and the creation of a skilled workforce at the same time.

The subject of designing an effective gold recovery process for a specific ore sample has always been a challenge. In the last decades, numerous research studies have been done to improve the conventional gold recovery methods and to develop alternative processing techniques. This study aims developing an effective gold recovery process by employing coal and oil in the conventional flotation processes.

1.2 The Scope of the study

The study is concerned with developing a hybrid processing method by combining flotation and coal-oil agglomeration for the gold recovery. The main goal is increasing flotation selectivity without decreasing the gold recovery. Flotation was used to separate previously formed coal-oil-gold agglomerates from the pulp and to recover gold particles which had not penetrated into the agglomerates. Only small amounts of coal and agglomerating agents were added to the pulp to assist gold flotation. Therefore, the method was called as “coal-oil assisted gold flotation”.

1.3 The Method

The study was established on investigating the effectiveness of the methods on their own area first and then combining the methods for the same subject on the same area. It was planned to carry out the experiments on the two different gold ore samples (Bergama-Ovacik and Efem cukuru-Izmir). However, the coal-oil assisted gold flotation process gave very successful results even with its initial tests for the recovery of gold from Izmir-Efem Cukuru ore. Bergama-Ovacik gold ore deposit with its fine gold particle size would be a bigger challenge for the coal-oil-assisted gold flotation process. For this reason, it has been decided to run the further coal-oil assisted gold flotation studies by using only Bergama-Ovacik gold ore samples.

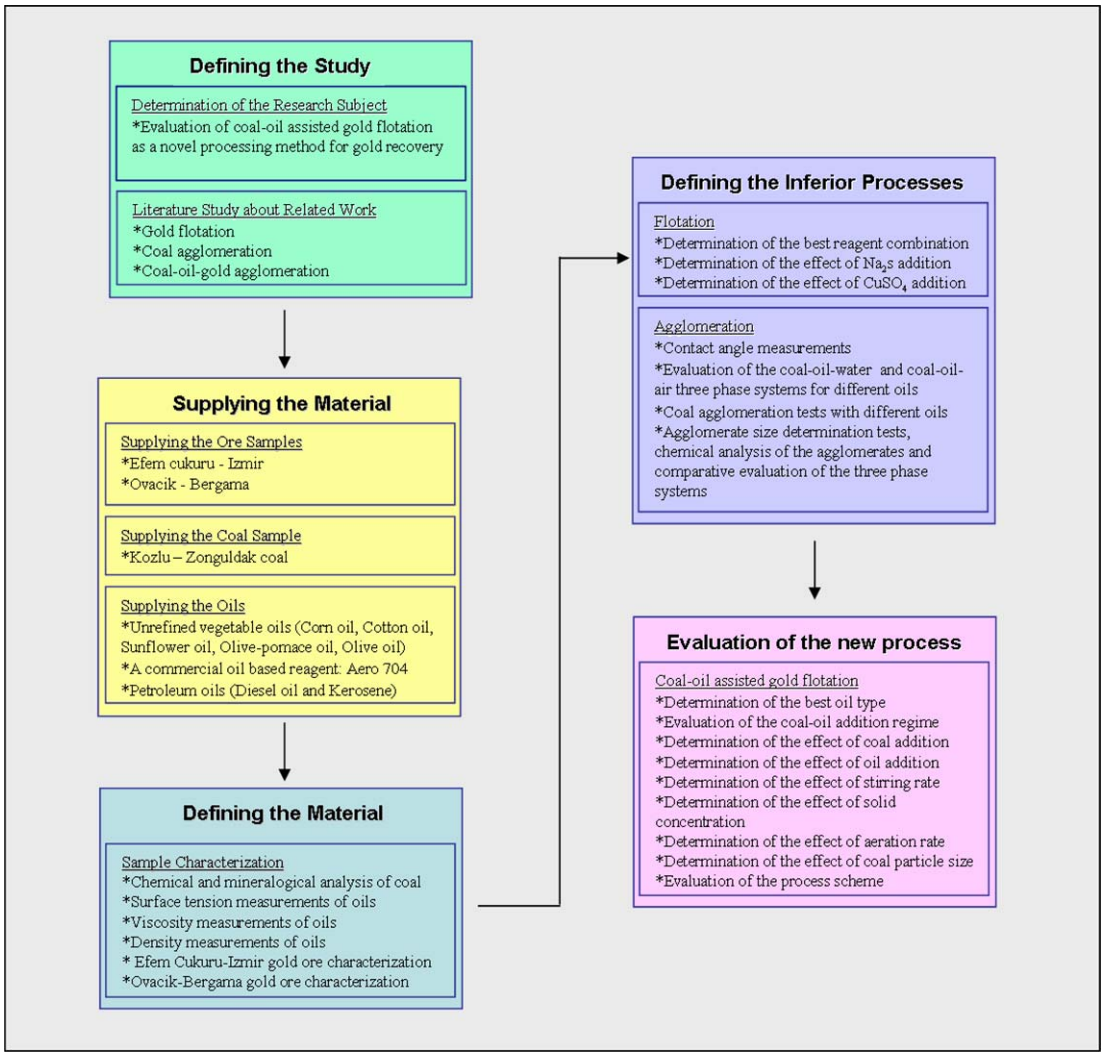

Figure 1.1 The methodology and the roadmap of the study

CHAPTER TWO
GOLD ORE DEPOSITS AND MINEROLOGY

2.1 Gold Ore Deposits and Mineralogy

The average concentration of gold in the world is about 0.005 g/t that are lower than other metals. The low concentration of gold in primary rocks requires an upgrading by a factor of 3000-4000 during ore formation processes to achieve commercial concentrations. Applied mineralogy involves determining mineral characteristics that have great effects on exploration, mineral processing and hydrometallurgy processes for gold. Maximum recovery is generally the main goal for gold recovery operations. Knowledge of ore characteristics affecting gold recoveries can help in designing and redesigning of a flowsheet, and can indicate whether maximum recoveries have been obtained or not. Ore characteristics can be determined by mineralogical techniques, but the techniques need to be designed to analyze gold-bearing materials since gold ores and mill products contain very small quantities of the wanted mineral.

Gold mainly occurs as native metal, often alloyed with up to 15% silver, but some is commonly presents as “invisible” gold, and some as gold tellurides, gold selenides and gold sulphides. Native gold contains up to 99.8% Au but generally varies between 85 and 95% Au content. Pure gold has a density of 19300 kgm^{-3}, but native gold typically has a density of 15000 kgm^{-3}. High density of gold allows it to be recovered easily by gravity methods at particle sizes above 10 μm if it is liberated from its gangue material; the major method of recovery of gold employed throughout history.

If the gold is alloyed with and contain about 25-55% silver, the mineral is named as electrum. Electrum is pale yellow, has lower density (i.e. 13000-16000 kgm^{-3}) than gold. Tellurides are other gold minerals. Common gold-bearing tellurides are; Slyvanite ($(Au, Ag)Te_2$), Calaverite ($(Au, Ag)Te_2$) and Petzite ($(Au, Ag)_2Te$) with Krennerite ($AuAgTe_4$) and Montbroyite (Au_2Te_3) less common. The densities of gold

tellurides (8000-10000 kgm^{-3}) are lower than native gold and the colors are less distinctive shades of white, gray and black.

Common gold bearing minerals are listed in Table 1.1. Gold oxides, silicates, carbonates, sulphates or sulphides are not naturally occurring minerals. Therefore, gold generally occurs in a mineral form different to most other elements which, inter alia, allows selective gold extraction from other mineral mixtures.

The most common ore minerals that gold is associated with are pyrite, arsenopyrite, galena, sphalerite, chalcopyrite, magnetite, pyrrhotite, and tetrahedrite-tennantite.

Quartz, K-feldspar (adularia, microcline, etc.), albite, calcite, dolomite, ankerite, tourmaline, scheelite, fuchsite, muscovite, chlorite, barite, fluorite, graphite, carbonaceous materials, rhodochrosite, and rhodonite are the common gangue minerals of gold ores. The most problematic gangue minerals in gold processing are graphite, carbonaceous materials, secondary copper minerals (malachite, covellite, chalcocite, azurite, brochantite, etc.), bornite, enargite, marcasite, and pyrrhotite. Graphite and carbonaceous materials cause problems in leaching; they re-absorb some of the dissolved gold dissolved. The phenomenon is known as preg-robbing and CIL process was developed to solve this problem. Pyrrhotite, marcasite, and secondary copper minerals (malachite, covellite, chacocite, etc.) are soluble in weak cyanide solutions. They increase the required cyanide consumption, by combining with cyanide and dilute the solution to such an extent that the solution becomes too weak to dissolve the gold (Petruk, 2000).

Maldonite is another gold mineral named after the occurrence at Maldon in Victoria (Australia). The gold occasionally occurs with bismuth in the maldonite (Au_2Bi) has a density of 15500 kgm^{-3} and Mohs' hardness number is lower (1.5-2.0) than gold. Its solubility is low in cyanide solutions.

$AuCu_3$ (auricupride) and AuCu are extremely rare gold and copper inter-metallic compounds. Natural $AuCu_3$ contains 40 % Au instead of the stoichiometric 50.8 %

Au. The crystal system is face-centered cubic with copper at the center and gold at the corners. The copper content imparts a deep red color. (Marsden & House, 1992)

Table 1.1 Gold bearing minerals (Petruk, 2000)

Gold alloys, etc.		**Gold Tellurides**	
Native gold	Au	Syvanite	$(Au,Ag)_2Te_4$
Electrum	(Au,Ag)	Kostovite	$CuAuTe_4$
Gold Alloy	(Au,Ag,Hg)	Calaverite	$AuTe_2$
γ-gold amalgam	(Au,Ag)Hg	Montbrayite	$(Au,Sb)_2Te_3$
Weishanite	$(Au,Ag)_3Hg_2$	Krennerite	$(AuAg)Te_2$
Auricupride	Cu_3Au	Petzite	Ag_3AuTe_2
Tetra-auricupride	CuAu	Bilibinskite	$Au_3Cu_2PbTe_2$
Aurostibite	$AuSb_2$	Muthmannite	(Ag,Au)Te
Anyuiite	$Au(Pb,Sb)_2$	Bezsmertnovite	$Au_4Cu(Te,Pb)$
Maldonite	Au_2Bi	Bogdavonite	$(Au,Te,Pb)_3(Cu,Fe)$
Zvyagintsevite	$(Pd,Pt,Au)_3(Pb,Sn)$	Buckhornite	$AuPb_2BiTe_2S_3$
Gold Sulphides		Gold Selenides	
Nagyagite	$Pb_5Au(Sb,Bi)Te_2S_6$	Fischesserite	Ag_3AuSe_2
Uytenbogaardtite	Ag_3AuS_2	Petrovskaite	AuAg(S,Se)
Criddleite	$TlAg_2Au_3Sb_{10}S_{10}$	Penzhinite	$(Ag,Cu)_4Au(S,Se)_4$
Buckhornite	$AuPb_2BiTe_2S_3$		
Invisible Gold (Au as trace component)		Secondary Gold	
Arsenopyrite	< 0.2 – 15,200 ppm	Secondary gold	Au (??)
Pyrite	< 0.2 – 132 ppm	Aurantimonate	$AuSbO_3$
Loellingite	< 0.2 – 275 ppm		
Tetrahedrite	< 0.2 – 72 ppm		
Chalcopyrite	< 0.2 – 7.7 ppm		

2.1.1 Textures and Microstructures

The most common gold ore textures are (Petruk, 2000);

- Gold in fractures and microfractures in rocks, and in veinlets and microveinlets in minerals,
- Gold in interstitial spaces between mineral grains and at borders of mineral grains,

  a) between grains of the same mineral, and

  b) between grains of two or more different minerals,
- Gold enclosed in a host mineral (encapsulated gold)

And the lesser common textures;

- Submicroscopic gold associated with framboidal pyrite,
- Submicroscopic gold associated with very fine-grained clay minerals,

And the last texture is found only in oxidized zones above ore bodies;

- Gold in oxidized zones

2.1.2 Common Gold Ore Deposits

Primary and secondary gold-bearing materials can be classified in mineral processing-based categories, which are related to their mineralogical and historical characteristics.

- **Primary Ores**
  - Placers
  - Free Milling Ores
  - Oxidized Ores
  - Silver-rich Ores
  - Iron Sulphides
  - Arsenic Sulphides
  - Copper Sulphides
  - Antimony Sulphides
  - Tellurides
  - Carbonaceous Ores
- **Secondary Materials**
  - Gravity Concentrates
  - Flotation Concentrates
  - Tailings
  - Refinery Materials
  - Recycled Gold

Each of these classes of gold-bearing materials has special mineralogical characteristics, which affect their processing and are considered in more detail in following chapters.

The major gold ores originating from exogenetic (formed at the Earth's surface) or endogenetic (formed within the Earth) sources contain gold in its native form.

a) Placers

This type of gold deposits are called as exogenetic type ores and contain alluvial, eluvial or colluvial deposits. They are formed by the liberation of gold and hydraulic transport of gold particles away from a primary deposit. There are several classes of placer gold ores;

- Eluvial (residual) Placers; some of the finer and lighter minerals have been washed away remaining gold at a higher concentration. Gold grade is lower than the other placers due to the absence of mechanical erosion of water.
- Colluvial (deluvial) Placers; Gold has been transported some distance from the main deposit, but where the gold is not located in an established stream system.

- Fluvial (Alluvial) Placers; Deposits are formed by moving water that has eroded gold out of lode deposits, occurs in stream or river systems. The gold tends to concentrate upstream of obstructions and in areas of lower fluid velocity.

The gold mineralization in placers differs from other ore classes. It has been liberated completely or to a large extent by natural processes and found in a particulate or loosely consolidated form and the gold has been liberated to a large extent by natural processes. For this reason it is possible to treat this type of very low grade ores economically.

These types of ores are usually utilized by employing gravity-processing flowsheets. The biggest problem is developing the most suitable process flowsheet for the evaluation of the ore. The evaluation of ore grade is difficult because of low assays and unusually coarse grain size. Extremely large samples of several hundreds of tones must be treated in a mineral concentration sampling plant for the evaluation of a placer ore grade and for flowsheet design (Marsden & House, 1992).

b) Free Milling Ores

Cyanidation can extract approximately 95% of the gold from a free milling ore when the ore is ground below -75 μm (d_{80}). This is a typical cyanidation operation in industrial practice, without incurring prohibitively high reagent consumptions.

The two main classes of free milling ores are palaeo-placers and quartz vein gold ores. Some epithermal deposits may be free milling but more commonly contain significant concentrations of sulfides and are therefore considered in subsequent classes.

Palaeo-placers differ from young alluvial placers. The gold in paleo-placers is unliberated and the ore is consolidated. Common communition practice is therefore required to liberate the gold to an extent, which allows efficient gold extraction. Mining and mineral processing operations costs much greater than a young placer deposit for palaeo-placer gold deposits since they have been mined at depths of up to 3 km. (Marsden & House, 1992).

c) Other Hard-Rock Ores

Various non-placer gold ore types can be classified as free milling. These are usually formed by the deposition from hydrothermal solutions. Epithermal deposits may fall into this category, but they quite often have some refractory component.

d) Oxidized Ores

These types of ores can be characterized by their high permeability. Oxidation and other hydrothermal alteration processes lead to breakdown of rock structure, resulting in increased permeability. High leaching extractions could easily be achieved by heap leaching of run-of mine ore, even though the ore particle size is very coarse. Gold usually occurs either liberated or in the alteration products of pyrite and other sulfide minerals. The most common of these are the iron oxides (such as hematite, magnetite etc.). Carbonate minerals such as Calcite ($CaCO_3$), Dolomite ($CaMg(CO_3)_2$) and Siderite ($FeCO_3$) are also more common (Marsden & House, 1992).

e) Sulphides

In this class of ores, gold is principally unliberated in a sulfide matrix, or the behavior of the sulfides affects process selection or operating conditions. The most important sulfide minerals are; Pyrite (FeS_2), Marcasite (FeS_2), Pyrrhotite (Fe_nS_{n+1}), Arsenopyrite (As_2S_3), Chalcopyrite ($CuFeS_2$), Aurostibnite ($AuSb_3$), Stibnite (SbS_3) etc. Volcanogenic massive sulfide deposits contain about 1 g/t-7 g/t gold. The gold occurs as discrete grains, veinlets, fissure filings and “invisible” gold (Petruk 200)

2.1.3 Gold Ore Deposits in Turkey

The following table gives the information about gold ore deposits of Turkey. Several other companies also are exploring for gold in Turkey; Aldridge Minerals Inc. of Canada on the Derinköy and the Olucak properties (also acquired an option on the Yenipazar property in central Turkey from Anatolia Minerals Development Inc.), Anatolia Minerals in the Çöpler gold prospect, Fronteer Development Group

Inc. of Canada on the Agi Dagi and the Kirazli gold properties in western Turkey and Odyssey Resources Ltd. on the Altintepe and the Tavsan gold properties.

Table 1.2 Gold ore deposits of Turkey

Regions		**Grade**	**Reserve (tons)**
		Au (g/t)	
Izmir-Bergama-Ovacik[1]		10	2,400,000
Gumushane-Mescitli-Mastra[1]		12	1,000,000
Izmir-Efem Cukuru[2]		13.14	1,856,000
Usak-Esme-Kısladag[2]		1.16	135,000,000
Eskisehir-Sivrihisar-Kaymaz[2]		6.04	973,00
Balıkesir-Havran-Kucukdere[2]		6.43	1,410
Çanakkale-Kirazlı-Akbaba[3]		1,25	8,000,000
Artvin Cerattepe[3]	Oxide	4	8,200,000
	Sulphide	1.2	3,900,000

[1] Ovacik gold mine(2007), [2] Eldorado (2007), [3] OIK Raporu (2000)

CHAPTER THREE
GOLD ORE PROCESSING

3.1 General

The optimal processing method for recovering gold is determined by many factors. The optimal processing method for recovering gold is determined by many factors. These include the mineralogy of the gold-bearing and gangue minerals, liberation profile of the gold-bearing minerals upon size reduction, the particle size of the gold itself.

These factors can be summarized as follows (Petruk, 2000; Allan and Woodcock, 2001; Klimpel, 1999):

(a) Mineralogical mode of occurrence of gold

(b) Identities of gold bearing minerals

(c) Identities of associated metallic and non-metallic minerals

(d) Proportion of gold occurring in different host minerals

(e) Gold grain size distribution

(f) Host and gangue mineral grain size distribution

(g) Mineral alterations

(h) Variations of the above within a deposit or with time

(i) Textures of the gold ores

Conventional gold recovery methods include gravity, flotation, amalgamation and leaching processes (Torres et al., 1999) but in the last decades, numerous research studies have been done to improve the conventional gold recovery methods and to find alternative processing techniques.

3.2 Gravity Concentration

Gravity processing is one of the oldest concentration methods used for mineral processing. The principle of gravity processing devices is complex and can not be explained by any single mechanism; generally several mechanisms are involved. Many new generation gravity processing devices are still being developed.

Several gravity processing devices are still being used for the gold recovery;

- Heavy media or dense media separations
- Thick beds (jigs and sluices)
- Thin beds (flowing films); such as shaking tables, spirals and Reicherth cones.
- Multi-Gravity separators (Falcon and Knelson centrifugal separators)

Gravity processing of gold (placer or free milling) has a long history. Sluices, jigs spirals and multi gravity concentrators have been used for a long time for free gold recovery. Jigs were used in North America and Knelson concentrator to a lesser extent the Falcon concentrator have been used (Burt, 1997)

Particles are classified based on their specific density difference in gravity concentration. Gold has a specific gravity of 15.0 - 19.3, and a typical ore usually has a specific gravity of about 2.6 (quartz is 2.7, Pyrite 5.0, Magnetite 4.3 and Feldspar 2.6). Only free gold particles can be recovered by gravity concentration, particles should be liberated by crushing and grinding before gravity processing (Stewart, A., L., 1984).

In jigging, well-liberated ore should be fed to the jig; material is separated through the water pulsations through the jig bed. A layer of heavy particles (steel shot or magnetite) may be placed to assist the separation (ragging). Small heavy particles pass through the ragging and the jig screen into the hutch from removal, while light particles remain at the top of the bed, forming the tailings. Fine gold particles would tend to be held up in the upper part of the bed, and would be lost. For this reason,

feed material should be screened and classified into narrow particle size range and use parallel flows.

Pinched Sluices; are simple inclined launders that narrow from about 200 mm at the feed to 20 mm at the discharge. The Reichert cone was developed on the pinched sluice sluice principle, but is built in circles or segments of circles. It has a high capacity (65-100tph), will accept feeds of up to 3 mm, and can treat down to 30 microns. The concentrate grade is not usually high enough to be considered a final concentrate and further cleaning is needed. Spiral concentrators have been used as coarse roughers in gold plants. Tables, although providing good recoveries are of low capacity (about 1 tph), and they are best used for cleaning rougher concentrates from devices such as cones, sluices and strakes. Tables can treat feeds from 3 mm downwards, and can recover gold down to 10 microns if the feed is properly sized.

A relatively new generation gravity separation technology is the centrifugal concentrators and Kelsey jig. The Knelson and Falcon concentrators are efficient machines for the gold recovery. The machines use the centrifugal force to enhance the gravity force applying on to particles, hence to obtain better metallurgical performance than in other wet gravity separators such as sluices, jigs and tables.

The Knelson concentrator has a stepped bowl that rotates about a vertical axis and develops about 60G. Water is added with the feed and is also injected into the bowl to fluidize the material being treated. It operates in the size range 6mm to 50 microns, and has been used chiefly on gold alluvial and hard rocks ores. Its main competitors are sluices and jigs (Stewart, 1990). The concentrator is being chosen due to their ability to recover coarse and fine gold from primary and placer deposits (Coulter&Subasinghe, 2004)

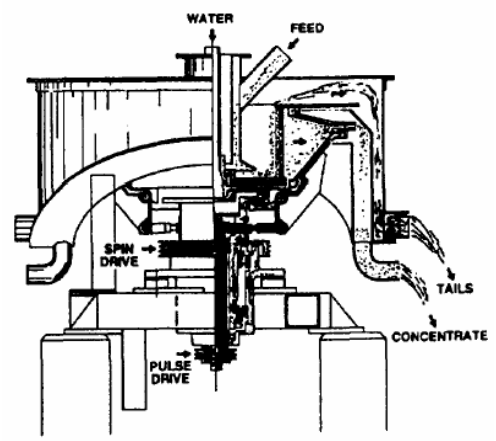

Figure 3.1 The Kelsey centrifugal jig (Napier-Munn, 1997)

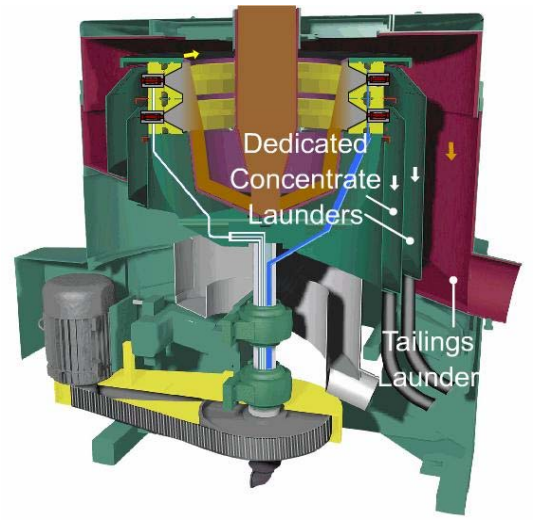

Figure 3.2 Knelson concentrator (Knelson, 2007)

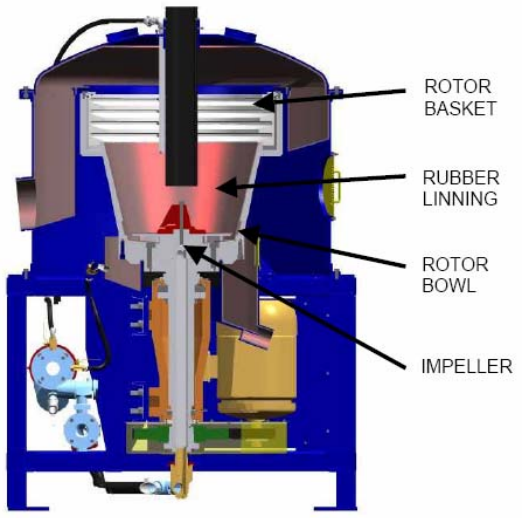

Figure 3.3 Falcon concentrator (Falcon, 2007)

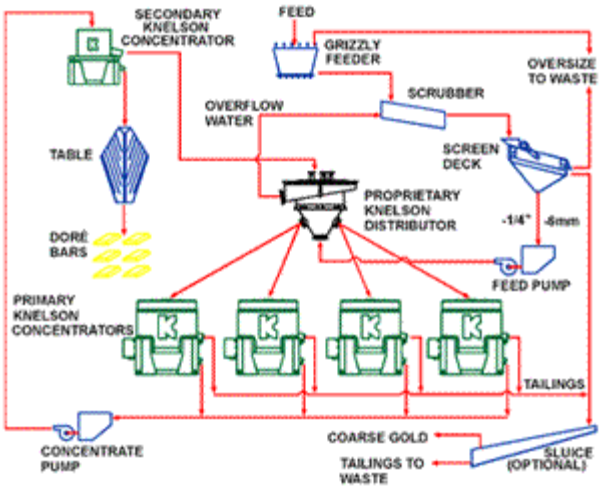

Figure 3.4 Typical alluvial installation for Knelson concentrator (Knelson, 2007)

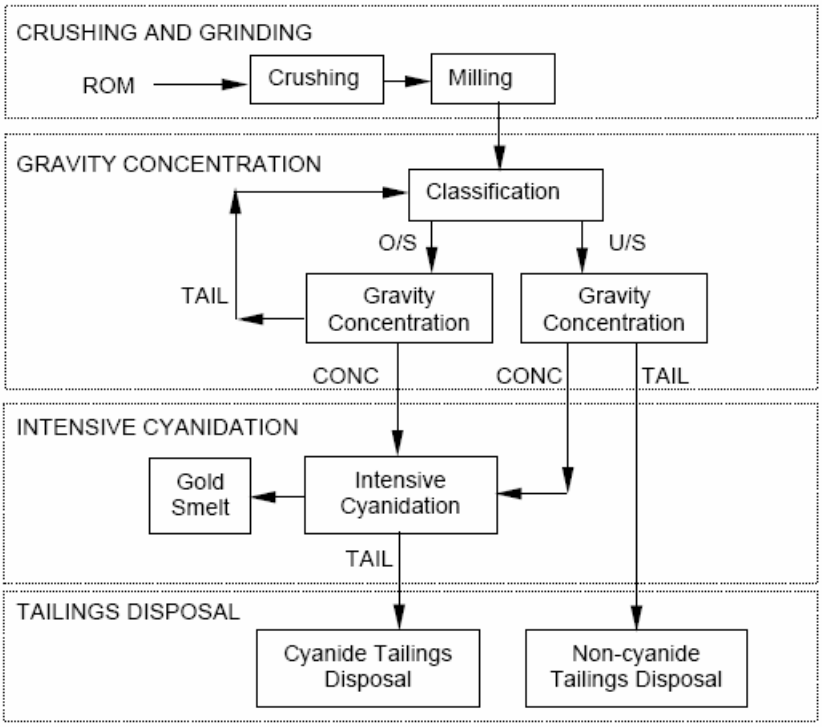

Figure 3.5 A new age gold plant flowsheet for the treatment of high grade ores (Longley et al, 2002)

3.3 Amalgamation

The entire amalgamation process needs little investment and few technical skills. The methods applicability is based on wetting and alloying of mercury and gold (McQuiston, F., W., 1985). In mercury amalgams of gold, there can be two, three, or

four atoms of gold per mercury, giving the compounds Au_2Hg, Au_3Hg, Au_4Hg, respectively (Hoffmann, 1994).

The ore can be subjected to amalgamation directly or a concentrate can be used in the process. Direct amalgamation starts in the grinding stage, and amalgam is recovered by sluicing and panning. The amalgam is half mercury and half gold and mercury is simply roasted off to volatize the remaining mercury, this process must be used with a retorting system to recover the distilled mercury.

Although mercury is still being used in nowadays, it is a highly toxic material. Toxicity depends on the particular chemical form it takes. The worst offenders are organomercurials-compounds such as salts of methyl mercury, CH_3Hg^+. Ionic mercury, Hg^+ (in the form of Hg_2^{2+}) is not very toxic, for it forms the stable Hg_2Cl_2 (Calomel) with stomach chloride. Mercuric ion, Hg^{2+} is dangerous (it can be methylated, forming CH_3Hg^+, by bacteria) but not easily transported across biological membranes. Elemental mercury is very toxic when inhaled (Hoffmann, 1994).

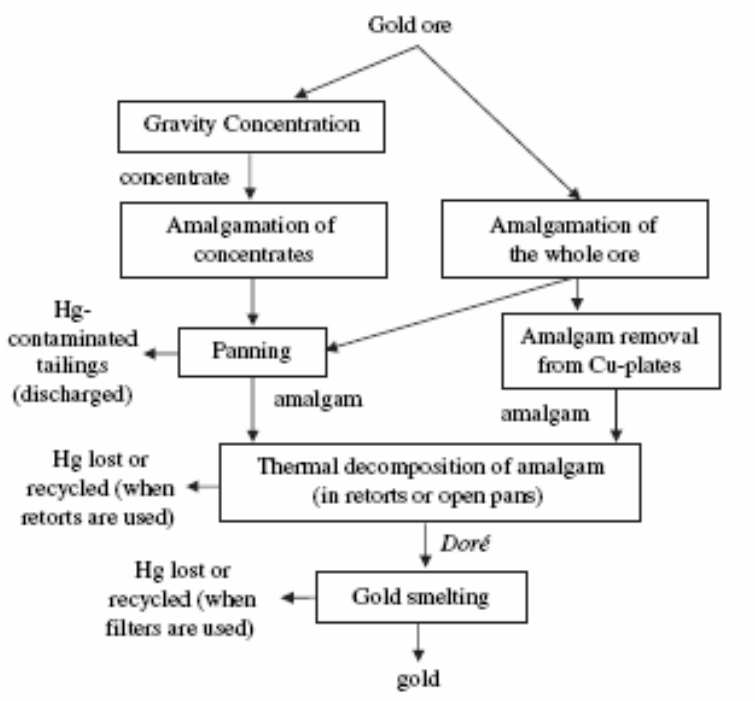

Figure 3.6 Methods used in artisanal mining operations to extract gold with mercury (Veiga, et al., 2006)

3.4 Cyanidation

The process is developed around 1890 and quickly become the most effective method used for the recovery of gold due to high recoveries obtained. Thus; amalgamation plants could recover about 60% of the gold present and cyanide could recover about 90%.

Gold dissolves in cyanide according to following equations;

$$2Au + 4NaCN + 2H_2O + O_2 \rightarrow 2NaAu(CN)_2 + H_2O_2 + 2NaOH \quad (3.1)$$

$$2Au + 4NaCN + H_2O_2 \rightarrow 2NaAu(CN)_2 + 2NaOH \quad (3.2)$$

The main reaction that gold dissolves is the first reaction; H_2O_2 produced in first reaction assists the gold dissolution in second reaction (Cilingir, 1996).

Cyanidation processes may include different operations given as follow:

- Agitated tank leaching
- Heap leaching
- Carbon adsorption recovery
- Zinc precipitation recovery

The standard cyanide leach process consists of grinding the ore to about 80% -200 mesh, mixing the ore/water grinding slurry with about 2 pounds per ton of sodium cyanide and enough quick lime to keep the pH of the solution at about 11.0. At a slurry concentration of 50% solids, the slurry passes through a series of agitated mixing tanks with a residence time of 24 hours. The gold bearing liquid is then separated from the leached solids in thickener tanks or vacuum filters, and the tailings are washed to remove gold and cyanide prior to disposal. The separation and washing take place in a series of units by a process referred to as counter current decantation (CCD). Gold is then recovered from the pregnant solution by zinc precipitation or carbon adsorption recovery method and the solution is recycled for reuse in leaching and grinding.

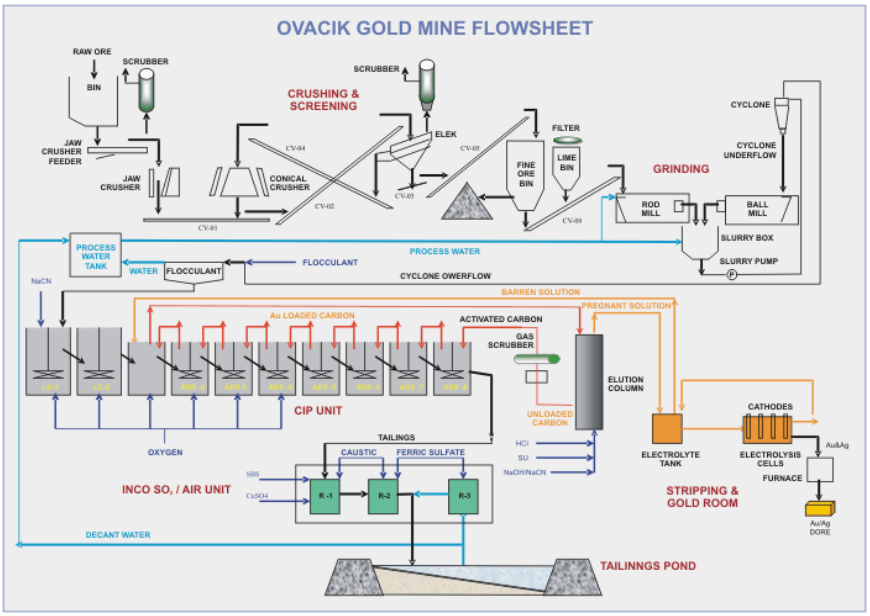

Figure 3.7 A typical cyanidation plant flowsheet (Ovacik Gold mine, 2007)

The Merrill-Crowe process starts with the filtration of pregnant solution in media filters. Filter types used include pressure leaf filters, filter presses, and vacuum leaf filters. Generally, a precoat of diatomaceous earth is used to produce a sparkling clear solution. Clarified solution is then passed through a vacuum deaeration tower where oxygen is removed from the solution. Zinc powder is then added to the solution with a dry chemical feeder and a zinc emulsification cone. The reaction of the special fine powder zinc with the solution is almost instantaneous. Precipitated gold is then typically recovered in a recessed plate or plate and frame filter press.

Granular coconut shell activated carbon is widely used for recovery of gold from cyanide solutions. The process can be applied to clean solutions through fluidized bed adsorption columns, or directly to leached ore slurries by the addition of carbon to agitated slurry tanks, followed by separation of the carbon from the slurry by coarse screening methods.

Gold cyanide is adsorbed into the pores of activated carbon, resulting in a process solution that is devoid of gold. The loaded carbon is heated by a strong solution of hot caustic and cyanide to reverse the adsorption process and strip the carbon of gold. Gold is then removed from the solution by electrowinning. Stripped carbon is returned to adsorption for reuse.

The major advantage of carbon-in-pulp recovery over Merrill Crowe recovery is the elimination of the leached ore solids and liquid separation unit operation. The separation step typically involves a series of expensive gravity separation thickeners or continuous filters arranged for countercurrent washing or filtration of the solids. For ores exhibiting slow settling or filtration rates, such as ores with high clay content, the countercurrent decantation (CCD) step can become cost prohibitive.

3.5 Thiourea Leaching

Thioura has been proposed as an alternative lixiviant to cyanide for the treatment of sulfidic cyanide consuming ores or for use in location where environmental concerns make use cyanide difficult (Yüce, 1994).

Thiourea in an acidic medium forms a single cationic species with gold according to the following simple dissolution reaction;

$$Au^0 + 2CS(NH_2)_2 = Au(CS(NH_2)_2)_2^+ + e^- \quad (3.3)$$

This reaction is considered reversible with an as-written electrode potential (E^o) at 25 C^0 of –38v and the pK value of Au-thiourea complex is 22.0. This indicates that the Au-thiourea complex is more stabile than the Au(I) cyanide complex. In the presence of an oxidizing agent such as hydrogen peroxide or ferric ion, thiourea will oxidize in successive stages to a number of products. The first oxidizing product of thiourea is fomadidine disulphide, which serves as a selective oxidant for Au and Ag;

$$2CS(NH_2)_2 = NH_2(NH)CSSC(NH)NH_2 + 2H^+ + 2e^- \quad (3.4)$$

This is a rapid reaction with an electrode potential of +0.42v. The formadidine disulphide itself is a very active oxidant and is considered to proceed at a sufficient rate to achieve adequate Au extraction (Pyper & Hendrix, 1981).

A combination of (3.3) and (3.4) yields the suggested overall equation;

$$Au^0 + 2CS(NH_2)_2 + NH_2(NH)CSSC(NH)NH_2 + 2H^+ = 2Au(CS(NH_2)_2)_2^+ \quad (3.5)$$

with an E^0= +0.04 v and G^0= -1845 cal/mole (for two electron transfer).

In a sulfate medium containing ferric ion, the Fe^{3+}/Fe^{2+} couple provides oxidation.

$$Fe^{3+} + e^{-} = Fe^{2+} \qquad (3.6)$$

with E^0=+0,77v Combining (3.6) and (3.3) yields the following;

$$Au^0 + Fe^{3+} + 2CS(NH_2)_2 = Au(CS(NH_2)_2)_2^{+} + Fe^{2+} \qquad (3.7)$$

with E^0=+0.39v and G^0=-8994 cal. A comparison of the free energies of (3.5) and (3.7) indicates a greater tendency for (3.7) to proceed; thus the presence of and added oxidant as well as formadidine disulphide is advantageous. It has also been suggested that a ferric ion-thiourea complex is formed which also aids in dissolution of gold.

Leaching kinetic of thiourea is approximately 5 times faster than Cyanidation. (Yüce et. al., 1994)

3.6 Bacterial Leaching

Bacterial leaching is an alternative method for the treatment of bulk flotation concentrates of complex sulfur minerals. This process uses the ability of chemolithotropic autotrophic bacteria, such as those belonging to genus "Thiobacillus", to oxidize metallic sulphides and elemental sulfur into water-soluble sulfates. Especially remarkable are the species "Thiobacillus thioxidans" and "Thibacillus ferrooxidans" which oxidize inorganic sulphide and other reduced valency sulfur compounds to sulfate and sulfuric acid. Accordingly it can be written;

$$S^0 + H_2O + 3/2O_2 \xrightarrow[T.thiooxidans]{T.ferrooxidans} H_2SO_4 \qquad (3.8)$$

Thiobacillus ferrooxidans plays the more important role in bacterial leaching. Thus, these bacteria, apart from the oxidation reaction mentioned above, are able to oxidize Fe^{2+} to Fe^{3+}. The ferric ion is known to be an effective oxidizing agent and further contributes to the dissolution of sulphides (Gomez et al., 1999). The attachment of "Thiobacillus ferrooxidans" was found to be dependant on three factors, particle size, exposure time to the mineral surface of the bacteria and the effect of agitation during incubation (Sampson et al., 1999).

3.7 Thiosulphate Leaching

Thiosulfate is another lixiviant having superior complexing ability and less toxic properties. Thiosulfate is used under neutral or slightly alkaline conditions. Because of its highly complex characteristics, it is required that working under strictly controlled conditions

Gold dissolves in alkaline thiosulfate ($S_2O_3^{2-}$) solution. Dissolved oxygen in the pulp is used as an oxidant and Au- S_2O_3 complex forms, as follows;

$$4Au + 8S_2O_3^{2-} + O_2 + 2H_2O \Rightarrow 4Au(S_2O_3)_2^{3-} + 4OH^- \qquad (3.9)$$

Thiosulfate concentration, dissolved oxygen concentration and temperature are the most important parameters affecting the dissolution rate of gold (Mardsen, & House, 1992). Cupric ion is known having a strong catalytic effect on the rate of oxidation. In the presence of ammonia, the cupric tetrammine enables the dissolution of gold as complex ion as given by the following equation:

$$Au + 5\,S_2O_3^{2-} + Cu(NH_3)_4^{2+} \Rightarrow Au(S_2O_3)_2^{3-} + 4NH_3 + Cu(S_2O_3)_3^{5-} \qquad (3.10)$$

In this equations, cupric ion in $Cu(NH_3)_4^{2+}$ complex is reduced to form $Cu(S_2O_3)_3^{5-}$ while metallic gold is oxidized to $Au(S_2O_3)_2^{3-}$

Under oxidizing conditions, the oxidative degradation of thiosulfate occurs, therefore the proper control of solution pH and redox potential (Eh) is very important.

There are two important decomposition reactions in the thiosulfate media:

Decomposition to sulfite and sulphide in alkaline conditions

$$3S_2O_3^{2-} + 6OH^- \Rightarrow 4SO_3^{2-} + 2S^{2-} + 3H_2O \qquad (3.11)$$

The sulphide formed in this reaction can react with dissolved metals such as Cu and Ag, to form passivating precipitates such as CuS and Ag_2S. Remaining S^{2-} increases the further decomposition of thiosulfate.

Re-formation of thiosulfate from tetrathionate

$$4S_4O_6^{2-} + 6OH^- \Rightarrow 5S_2O_3^{2-} + 2S_3O_6^{2-} + 3H_2O \quad (3.12)$$

Increasing the temperature causes to decompose thiosulfate to S^{2-}, $S_4O_6^{2-}$, SO_3^{2-}. Thus CuS occurred in the solution cover the mineral surfaces, interfering gold dissolution (Wan & Brierley, 1997).

Thiosulfate can be stabilized to some extent by the addition of small amounts of sulphide ions (SO_3^{2-}), which react with S^{2-} and regenerate thiosulfate as follows;

$$3SO_3^{2-} + 2S^{2-} + 3H_2O \Rightarrow 2\,S_2O_3^{2-} + 6OH^- + S \quad (3.13)$$

This also helps to prevent the precipitation of silver as the insoluble sulphide (Mardsen, & House, 1992).

Gold recovery from ammonium thiosulfate solution has some specific problems. Gold can not be recovered effectively from this solution by either anionic exchange resins or activated carbon. Therefore cementation process by Cu and Zn is commonly practiced. When zinc is used for the cementation of gold from ammonium thiosulfate solution some problems are encountered; gold and copper are cemented together increasing the zinc consumption. Second, zinc dissolved in ammoniacal solution can not provide cathodic protection for the gold and eventually, the cemented gold re-dissolves (Wan & Brierley, 1997).

3.8 Flotation

Froth flotation has been a valid industrial gold processing method used industrially for years. The evaluation of a flotation test is based largely on a rougher recovery value obtained at a defined time.

The cumulative recovery of a component in a concentrate is proportional to flotation time. The flotation process can be considered as a time-rate recovery process. Therefore mathematical flotation models that incorporate both a recovery and a rate function can completely describe flotation time- recovery profiles.

Kinetic models are generally used to analyze batch flotation data. The models can be used to evaluate the effect of various parameters like flotation chemicals or operating conditions for flotation process.

First order flotation model considers the general rate equation for flotation and it may be written as:

$$\frac{dC_p(t)}{dt} = -k(t)C_p^m(t)C_b^n(t) \qquad (3.14)$$

Cp(t) and Cb(t) are the concentrations of the particles and bubbles at time t, respectively.

The exponent, m and n are the respective orders for particles and bubbles, and k(t) is a pseudo rate constant. If the material to be floated consist of particles whose k's could be expressed by a continuous distribution function, f(k), the recovery at time t will be equal to:

$$R(t) = R_\infty\left[1 - \int_0^\infty f(k)\exp(-kt)dk\right] \qquad (3.15)$$

Here, R^∞ is the ultimate recovery at long times, R(t) is fractional recovery at time t (Klimpel, 1999)

The incentives for using flotation as a processing step include its high throughput at relatively low capital and operating costs, it's high mineral specificity and its high operating flexibility, e.g., (Klimpel, 1995). The ever increasing availability of effective collectors and frothers for gold flotation along with improved application methodology are also helping to increase the use of froth flotation as an integral step in the overall gold-processing arena.

Previous studies on the flotation of native and free gold have focused on many different factors affecting the gold recovery (Allan and Woodcock, 2001; Teague et al., 1999a;b; Valderrama and Rubio, 1998; Klimpel, 1999; Monte et al., 1997; Forrest et al., 2001). It can be stated that most of the main process variables affecting gold recovery by flotation has been investigated widely.

Although flotation is a very effective process for free and native gold recovery, sometimes it is not possible to obtain satisfactory results. It is inherently a slow kinetic process and additional steps are generally involved to increase product purity (Klimpel, 1999). Flotation of gold particles coarser than 200 μm is very difficult due to their high specific density, floating slower than finer particles and also requires more collectors to recover by flotation (Glembotski et al., 1974; Klimpel, 1999).

Flotation method is an effective method for sulphide ores due to its selectivity. It can be used iron, and refractory gold-bearing and gold-silver-bearing minerals. Flotation can be used in conjunction with other gold processing methods to maximize the recovery of gold (Malhotra and Harris, 1999). Gold-containing ores which may require flotation of native gold in their treatment flowsheet includes placer deposits, hydrothermal deposits, epithermal deposits, volcanic material, sedimentary rocks, metamorphosed rocks and oxidized/ weathered material (Allan and woodcock, 2001).

Flotation could be used for different purposes in mineral processing;

- As a precyanidation process; to remove the interfering elements (stibnite, copper, etc.) and carbon based materials to prepare the ore to cyanidation.
- As a post-cyanidation process; to recover the tellurides this can not be dissolved in cyanidation process.
- To recover gold from the tailings those come from the gravitation and amalgamation processes.
- To produce high-grade preliminary gold concentrates.(Gold associated with sulphide and native gold containing ores)

From the viewpoint of gold mineralogy interacting with reagent chemistry, it is possible to characterize the various possible reagent philosophies into two general categories:

- The flotation of free or native gold species, gold in pure quartz ores can be easily utilized by flotation.

- The recovery of gold associated with a variety of sulphide mineral (Klimpel, 1999). Sulphides are generally inclined to decompose in cyanide solution. For this reason flotation is used to produce sulphide concentrates and then, cyanidation is applied to a relatively small mass of the ore.

3.8.1 Flotation of Native Gold

3.8.1.1 Particle Size and Shape Effects

Gold particles can be found varies over a wide range, but it is easy to recover effectively the gold particles in the range of 20-200μm (Glembotskii et al., 1972).

Fine grinding process produces fine constituents that can coat the gold particles' surfaces and make them less hydrophobic. The recovery of fine gold particles is strongly influenced by the presence of large amount of partly hydrophobic gangue fines such as pyrophyllites.

It is found that native gold flakes have rough flat surfaces, rougher surfaces are less hydrophobic and bubble attachment/ floatability of this type of particles is reduced (Teague et al., 1999).

Particle size plays an important role in the probability of particles for collision, and attachment processes, as well as remaining attached in the pulp phase in flotation. Fine particles typically have decreased particle-bubble collisions and this characteristic can significantly influence and lowers the process recoveries. Coarse particles generally badly float mostly due to the inevitability of the bubble–particle aggregates to prevent the particle detachment from the bubble surface caused by the particle weight and turbulence eddies during flotation. In addition, coarse particles may need longer induction times (Phana et al., 2003; Feng & Aldrich, 1999; Nguyen, 2003; Nguyen et al., 1998 ; Dai et al., 2000; Bravo et al., 2004). In conclusion, the flotation recovery of both fine and coarse particles is usually remarkably low.

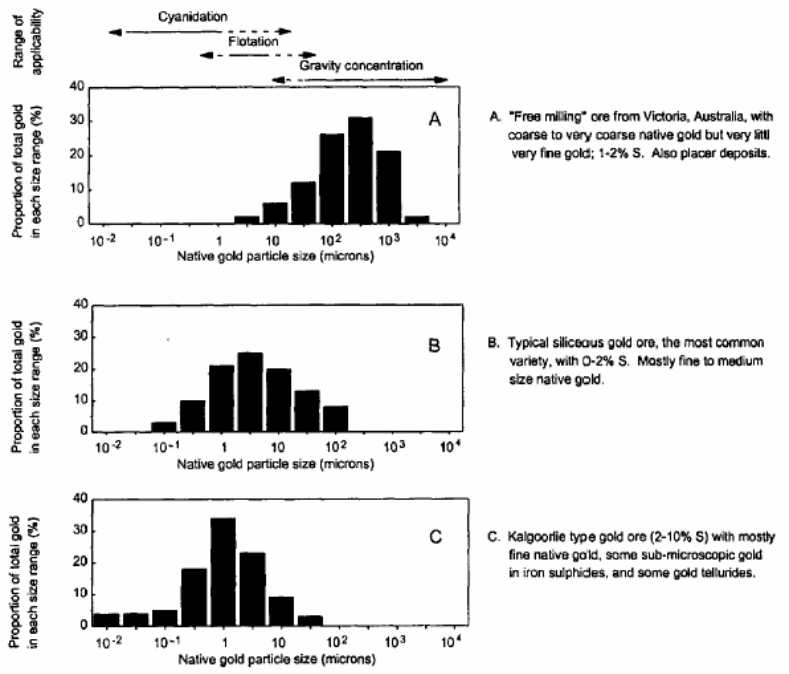

Figure 3.8 Typical size distributions of native gold particles in some gold ores (Allan and Woodcock, 2001)

3.8.1.2 Liberation

Liberation of gold particles is the primary requirement for the flotation, i.e. as free gold grains or that it occurs in composite particle presenting predominantly a gold/sulphide surface. The floatability is lower when oxides or silicate-gold composites and dependent on their respective surface areas and hydrophobicities. If the gold is locked in pyrite or pyrrhotite copper activation will provide an increase in sulphide grades and recoveries and so in gold recoveries.

3.8.1.3 Coatings

The surfaces of native or free gold-containing particles are naturally hydrophobic and if the surfaces of such particles are relatively free of contaminants, they can easily be recovered by flotation. Contaminants on the particles may result from the processing chemicals, small-particle slime or the precipitates of iron compounds (Klimpel, 1999).

Ferric ions in the form of hydrated oxides acted as physical barriers between the air bubbles and the gold surfaces. Humic acid can also shows the same effect.

(Aksoy & Yarar, 1989). Iron contaminant forms from iron containing minerals or grinding media loss (it is commonly between about 0.5-1 kgt^{1-} ore). Teague et al. (1999b) reports; the total gold recovery increases as more metallic iron was added to the ore prior to milling regardless of whether PAX, GX or a combination of the two collectors was used. In most cases, the greatest amount of sulphide material was recovered when 5 kg/t of metallic iron was added to the ore prior to milling which was mainly the result of an increase in froth stability.

3.8.1.4 Flotation Parameters

The mean native or free gold particles (and gold tellurides) have normally hydrophobic surface properties and they tend to float without the addition of collectors. In general, collectors are used at relatively low consumption rates. Free gold particles can be recovered effectively by keeping the particle surfaces as clean of organic species as possible and by removing adhering slime particles. A small amount or no pH regulator is needed, using a small quantity of collectors (25 - 75 gr/t), a frother to stabilize the froth and a small dosage of depressant might be enough.

Forest et al., (2000) reports about previous studies; the concentrate grade increases with decreasing pulp density. High slurry densities reduce the coarse gold particle sedimentation. If the ore contains the gold particles which coarser than 200μm particle size, flotation should be performed at high slurry densities (e.g. 35% solids).

The choice of operating pH is completely depended on the collector to be used. The affectivity or the stability determines the pH range for the flotation.

The use of copper sulphate as an activator for the pyrite is common; the recent studies reports Cu^{2+} ions actually reduced the floatability of gold at concentrations above 400 g/t $CuSO_4$ using xanthate as collector (Forrest et al., 2000).

If the gold is associated in a massive or complex sulphide, operation may need some activators and regulators. Collector adsorption onto particle surface occurs as a

function of surface area that is dominated by the smaller particle sizes. For this reason coarser particle being under collected in a flotation feed mixture, while the finer particles are being over collected. Hence, increasing the collector dosage will increase the coarser particle recovery rate while the finer particle recovery is decreasing (R/K trade off). The studies that have been done in recent years clearly show that gold flotation, especially "native gold" flotation needs to be carried out by working at high pulp densities. It can be stated that if the pulp density is not maintained at 30% or higher (by weight), it is very difficult to keep the coarser free gold in the froth phase (Klimpel, 1999).

If the ore has a high clay content, maximum peptization should be obtained by carefully controlling the hydrogen ion concentration in pulp. Flotation can be performed in a weakly alkaline medium and soda or small amounts of sodium silicate can be used for this purpose. Lime should not be used for this aim, because Ca^{++} is generally a native or free gold depressant and presence of lime slows down the flotation kinetics of gold species. The reagents are fed in to the mill to prevent the particles from slime coating. Flotation is more difficult when the ore contains some micaceous, graphitic or other laminar minerals (Glembotskii et al., 1972). They cause random contamination and readily floats due to their low densities, highly water-repellent surfaces and shapes. It is generally possible to remove them from the pulp by floating before the collector addition stage with a small amount of frother addition and aeration. Sodium silicate or organic colloids such as soluble starch may be used with a small amount of frother addition. Although, $CuSO_4$ is used as activators in gold flotation, mainly it shows an activating effect on gold-bearing pyrite more than free gold. In free gold flotation, $CuSO_4$ appears to be more of a froth modifier than a true activator.

The choice of collector type and dosage is the other important factor in gold flotation. Klimpel (1999) reported that uncharged and water-insoluble collectors are the most effective collectors in free gold flotation applied at or near the natural pH. Thinocarbamates, Xanthogen formates, Mercaptans, Dialkyl sulphides are the useful uncharged and water-insoluble collectors used in gold flotation. Some charged and

water-soluble collectors such as xanthates and dithiophosphates are used in gold flotation too.

Another important point in flotation of native or free gold ores is selection of frothers. Pine oil, glycols (H-$(OC_0H_6)_n$-OH), Polypropylene glycol and methyl ethers (CH_3-$(OC_3H_6)_n$-OH) are usually more effective frothers from a recovery view point than alcohol-based frothers (R-OH).

3.8.2 Flotation of Gold Associated with Sulphide Minerals

There are many types of gold associated sulphide minerals and it can be divided into the groups as following;

- Gold associated with naturally floatable copper ores, such as chalcopyrite and bornite;
- Gold associated with pyrite, pyrrhotite and arsenopyrite;
- Gold associated with Cu/Pb/Ag/Zn sulphide minerals;
- Mixed gold-ore systems in which some of the gold occurs in native form and some is associated with sulphide minerals;
- Gold ores that have high slime characteristics.

Gold grains can either be free or locked in sulphide minerals. In general, some of the factors mentioned in free-gold section can be applied or affect also the recovery of gold associated with sulphide minerals such as frother selection, the effect of pulp densities in flotation, the R/K trade off with increasing collector dosages.

Pyrite tends to oxidize in air which affects its surface properties and flotation behavior. The pH range used for pyrite flotation depends on the ore components and the resulting natural pH of the slurried material. The collector is then chosen to operate close to this pH, e.g. M.B.T at pH 3-4 or xanthate in the range of 9-11. Flotation with xanthates is possible below pH 9; however, reagent decomposition and consumption becomes increasingly significant as pH reduced. In the region pH

4-8 reduced recovery can be demonstrated at low collector additions. The mechanism of pyrite flotation with DTP is similar to xanthate where, in the acidic region (<pH 4-6), dithiophosphate is oxidized to the dithiolate dithiophosphatogen, the most important hydrophobic species (Mardsen & House, 1992).

Copper sulfate ($CuSO_4$) is commonly used as an activator in gold/pyrite flotation particularly in the range 7-9 where silicate gangue flotation is reduced. The effects of ($CuSO_4$) are as following;

- Adsorption of Cu (II) onto pyrite which enhances the rate and/or extent of collector adsorption.

$$FeS^2 + Cu^{2+} \Leftrightarrow Fe^{2+} + S + CuS$$

- Depression of gangue particles.
- Modification of froth structure.
- Complexation with free cyanide, which would otherwise depress pyrite.

Pyrrhotite is stable at lower potentials than pyrites, which leads to a greater oxidation of the sulphide surface and generally lower recoveries, although this can be compensated for by Cu (II) activation. The mechanism of flotation with thiol collectors involves dixanthogen formation as in the case of pyrite. In gold ore treatment, pyrrhotite flotation is undesirable unless it contains gold or if it is intimately associated with other sulphides containing gold.

Arsenopyrite (AsFeS) behaves in a manner similar to other iron sulphides. The Eh-pH diagram for the Fe-As-S-H_2O system shows a region of stability for arsenopyrite which is similar to that of pyrite or pyrrhotite but displaced to a lower potential indicating that arsenopyrite is more prone to oxidation. Arseonopyrite can be floated with xanthate collectors with best results achieved in the pH range 5-8 by activation with Cu (II). In general, the factors which enhance pyrite flotation also improve arsenopyrite flotation. If the ore contains free gold, pyrite and arsenopyrite are depressed by using CaOH (pH of 8-9) and permanganates and native gold particles are floated by the addition of dithiophosphates. After the flotation of free

gold, it can be possible to produce the concentrates that might be divided into the gold containing arsenopyrite and pyrite concentrates.

In general copper sulphate is added as activator xanthates are used for collection and various depressants can be used for gangue control for this type ores.

Gold occurrence in these type ore systems is generally in small amounts. For this reason, the main aim of the process is the recovery of Cu and Zn. The most important problem in this type of ores flotation is the optimization of consumption of the reagents. In general, removal of the Pb and Cu from the pulp firstly is the most used method. Pb and Cu are recovered at moderately alkaline pH of 8.5-9.5 by using KAX. Selectivity against the iron sulphides is always a problem and more selective separation may need a dithiophosphate type collector (such as Aerofloat 241-242) and a alcohol based frother. Zinc flotation follows the first step. After the removal of Pb and Cu, Zn is recovered at higher pH of 9.5-10.5 by using copper sulfate addition as activator and a collector which should provide enough selectivity against iron sulphides. Gold recovery in this type of two or more staged recovery processes is generally poor (about 50%), studies showed that working at lower pH in the first step (Pb-Cu recovery step) may provide to recover gold more effectively. The use of blended collectors (especially phosphinates and dithophosphates) can help gold recovery (Çilingir, 1996).

Chalcopyrite ($CuFeS_2$) and bornite ($CuSFeS_4$) have strong natural (collectorless) flotation tendencies because of their electrochemical response following fresh surface-area creation by grinding (Klimpel, 1999). It has been reported by Klimpel (1997); the use of charged-water soluble collectors in this type of ores flotation generally needs to use excessive amounts of reagents and causes to work at lower recovery rates due to the significant changes in the selectivity of naturally floating materials. For this reason flotation of gold particles in this types ores should be realized by using uncharged-water insoluble collectors. The use of uncharged collectors provides lower reagent costs and more efficiency.

Khazheeva et al., (2001) reports sulphide-cupriferous ore is rapidly oxidized in grinding, and particles are covered surface oxidation. An extremely fine grinding worsens the results of copper-pyritic flotation and flotation of noble metals.

Klimpel (2000) reports that uncharged water-insoluble collectors have some advantages on ores containing naturally floatable sulfide minerals. These advantages may include lower collector dosage, lower operating pH, faster kinetics of flotation, and improved selectivity over iron sulfides.

3.8.3 Carbonaceous Gold Ores

The presence of organic carbon and graphite in the ore may cause to reduce concentrates' grade. These types of constituents are readily floated because of their low density and highly water-repellent surfaces and acquire readily contamination. Flotation of graphite and carbon based materials can be carried out before the collector addition stage by using very low frother feed and moderate aeration. Na_2SiO_3 or an organic colloid such as starch may be used with minimum frother feed. After the selectivity or removal of carbonaceous material was provided, flotation of gold can be applied by using proper amount and type of collectors (Glembotskii et al., 1972).

This type of ores can be treated by either floating carbon while depressing gold and sulphide minerals or floating pyrite and gold and depressing carbonaceous material. Bulotovic (1997) reports its possible to recover gold from a refractory-low grade gold ore by a split flowsheet in which graphitic carbon is pre-floated by hydrocarbon oil and then remained sulphides are recovered by xanthate.

3.9 Coal-Oil Agglomeration

Oil assisted separation processes utilizes an immiscible liquid (oil) to improve the attachment of hydrophobic particles to the bubbles (emulsion flotation), to improve the flotation of fine particles recovery by flotation (agglomerate flotation) or to increase their size to an extent which allows for the separation of the agglomerates by screening (Laskowski, 2001).

In coal-oil agglomeration the suspension of finely ground coal particles in water is strongly agitated with a small amount of oil. Intensive agitation provides for the reduction of the immiscible liquid into fine spherical droplets (Unal and Aktas, 2001). The oil droplets overspread the coal surfaces and the collision of these oil wetted particles causes formation of "liquid bridges" each upon them. As a result, the fine particles in the beginning are converted to the masses which were bounded together by liquid bridges (Keller, 1987). Hydrocarbons or vegetable oils can be used for oil agglomeration of coal. However, high prices of hydrocarbons make their commercial use uncertain. Vegetable oils, especially crude vegetable oils are less expensive than hydrocarbons and they are renewable and nonpolluting resources (Alonso et al., 1999).

Figure 3.9 represents a sessile liquid droplet resting on a flat surface. The values of the work of cohesion of the liquid and the work of adhesion of the liquid to the solid determine the contact angle at the solid/liquid/gas interface.

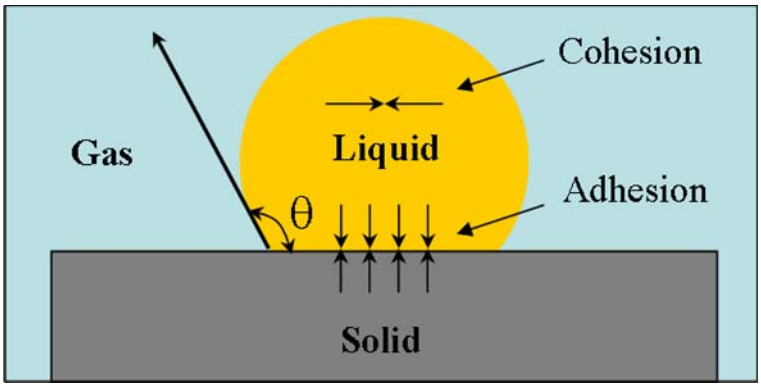

Figure 3.9 Adhesion and cohesion on the solid surface

The classical boundary condition for the hydrophilic-hydrophobic transition is equality of W_{SL}, the work of adhesion of liquid to solid and W_{LL} the work of cohesion of the liquid. Since

$$W_{LL} = 2\gamma_{LV} \tag{3.16}$$

from the equation;

$$\frac{W_{SL}}{W_{LL}} = \frac{\gamma_{LV}(1+\cos\theta)}{2\gamma_{LV}} \qquad (3.17)$$

Which gives $\cos\theta = 2\frac{W_{SL}}{W_{LL}} - 1$ the condition of hydrophobicity follows only for $W_{SL}<W_{LL}$. $\theta \neq 0$

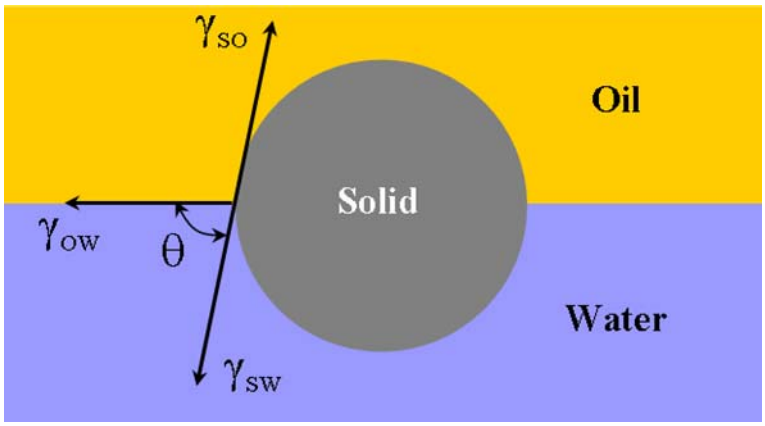

Figure 3.10 Illustration of a solid particle at oil/water interface

When a particle is placed in an oil-water interface (Figure 3.10) may be drawn into the aqueus phase θ<90º or remain concentrated at the interface (θ=90º) or drawn into the oil phase (θ>90º). Oil agglomeration occurs when condition θ>90º is satisfied (Cebeci & Sonmez, 2006)

The major factors affecting agglomeration of coal by an immiscible liquid include wettability of the coal, the type and amount of the bridging oil and type and intensity of conditioning. It is often assumed that oily droplets at a mineral/water interface spread onto the mineral surface and form a thin film (Laskowski. 2001).

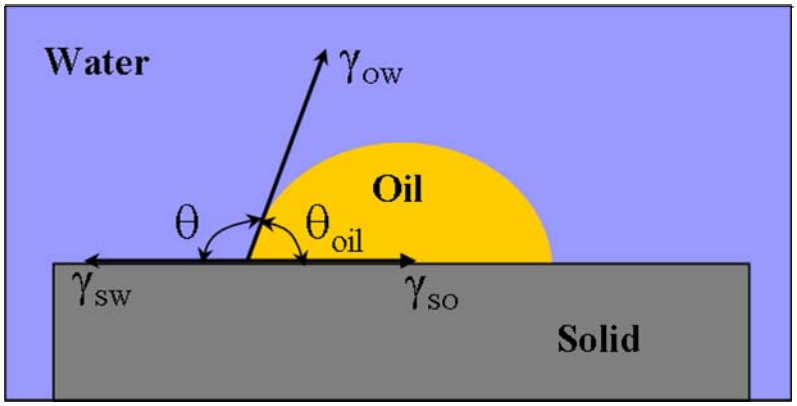

Figure 3.11 Illustration of an oil droplet on a solid in an aqueous phase

$$\Delta G_{spread} = \gamma_{SO} + \gamma_{OW} - \gamma_{W} \quad (3.18)$$

$$\Delta G_{spread} = \gamma_{OW}(1 - Cos\theta_{oil}) \quad (3.19)$$

$$\text{Since } \theta_{oil} = 180 - \theta \quad (3.20)$$

$$\Delta G_{spread} = \gamma_{OW}(1 + Cos\theta) \quad (3.21)$$

$$\Delta G_{attach} = \gamma_{OW}(Cos\theta - 1) \quad (3.22)$$

The attachment of an oil droplet to a mineral particle is likely to; take place only if $\theta > 0$. for $\theta = 0, \cos\theta = 1$ and $\Delta G_{attach} = 0$ and the attachment is impossible. The equations indicate that ΔG_{spread} is always a positive number; thus, thermodynamically the spreading of oil into a thin film at the solid/water interface is always unlikely.

The equation describes the free energy change indicating only the probability of attachment. While $\Delta G_{Attach} = 0$ indicates that particle-oil droplet attachment is impossible, the condition $\Delta G_{Attach} < 0$ indicates only that it is likely that the attachment will take place, but doesn't necessarily have to. An encounter between an oil droplet and coal particle, both suspended in an aqueus solution, may lead to attachment, if their kinetic energy is sufficient to overcome the energy barrier. Burkin and Bramley discuss two energy barriers, one preventing collision and attachment, and the other preventing the spreading of the oil over the coal surface. The first barrier depends on potentials of the approaching surfaces, identified with the electrokinetic potentials of both oil droplets and coal particles and the ionic strength of the aqueous electrolyte. The barrier to spreading is shown by the difference between advancing and receding contact angles; spreading is controlled by

the relevant interfacial tensions, solid porosity, surface texture, heterogeneity, among other things (Laskowski, 2001).

Polat et al. (2003) have summarized particle aggregation in coal flotation. The behavior of the coal particles according to oil concentration, the presence of surfactants and coal rank have been summarized (Table 3.1). .

Table 3.1 Relationship between coal particles and oil droplets as a function of coal rank, oil concentration and the presence of surfactant and its effect on flotation (Polat et al., 2003)

Agent	Rank	Particle aggregation	Recovery	Selectivity	Surfactant effect
No Surfactant Low oil	high	small agglomerates	high	moderate	-
	low	no agglomerates	very low	low	-
No Surfactant High oil	high	large agglomerates (entrapment)	very high	low	-
	low	small agglomerates	moderate	moderate	-
Surfactant Low oil	high	moderate size agglomerates	high	high	surface modifier
	low	small agglomerates	low	low	surface modifier
Surfactant High oil	high	moderate size agglomerates	moderate high	moderate high	modifier/emulsifier
	low	small agglomerates	high	high	modifier/emulsifier

Characterization of the oil utilized processes is shown in Table 3.2. The quantity of the immiscible liquid used, thermodynamic conditions of the process and the final separation technique determine the name of the method (Laskowski, 2001).

Table 3.2 Oil utilized separation processes (Laskowski, 2001)

Method	Presence of ionic collector	Presence of oily collector	Oil consumption	Conditioning	Separation method
Extender flotation	Yes	Yes	0.05-0.5 kg/t	Regular	Flotation
Agglomeration flotation	Yes	Yes	A few kg/t	Intense	Flotation
Emulsion flotation	No	Yes	Up to a few kg/t	Regular	Flotation
Oil agglomeration	Yes	Yes	5-10%	Slow/intense shearing	Sizing
Liquid-liquid extraction	Yes	Yes	N/a	Intense	Phase separation

3.10 Coal-Oil-Gold Agglomeration

Using the coal-oil agglomeration method free or native gold recovery has been a research subject for many researchers since a patent was granted to BP Australia Ltd. (BP, 1984). Though, it is possible to agglomerate gold particles with oil due to its high natural oleophilicity, gold ores do not contain enough gold particles to form agglomerates. For this reason, a highly hydrophobic/oleophilic material is needed to produce agglomerates with oil and gold (Calvez et al., 1998). The coal-oil-gold agglomeration process considers the preferential wetting of coal and gold particles. It takes advantage of the greater hydrophobicity and oleophilicity of coal and gold compared to that of most gangue materials. Coal-oil-gold agglomeration combines flotation with a fine coal treatment process to yield a high-grade concentrate and to minimize processing time while providing maximum efficiency.

Researchers have proposed different applications for the production of coal-oil-gold agglomerates (Wall et al., 1992; Gaidarjiev et al., 1996). Agglomerates may be produced in a two-stage process. In the first stage, an oleophilic material with (30%-70%) is added under high shear conditions to form the micro agglomerates and for the second step, the rest of the material is added under low shear conditions to increase the size of the agglomerates (Marciano et al., 1994). Moses and Petersen's study (2000) indicates that increasing the stirring rate to a certain value results in obtaining higher gold and mass recoveries for the flotation of agglomerates. Wu et al. (2004b) investigated the adhesion behavior of gold particles to coal-oil agglomerates and the study revealed that in addition to gold particles, gold flocs and even micro nuggets can be detected on the agglomerates. The authors say the movement of initially penetrated gold particles probably forms these flocs over the surface of the agglomerates.

Unlike the previous studies (Table 3.3) about coal-oil-gold agglomeration, Sen et al., (2005) used very small amounts of coal and agglomerating agents for coal-oil-gold agglomeration. Flotation was used as primary recovery process; it was used to both separate the coal-oil-gold agglomerates from the pulp and to recover gold particles which had not penetrated into the agglomerates. Only small amounts of coal

and agglomerating agents were added to the pulp to assist gold flotation. Therefore, the method can be called “coal-oil assisted gold flotation”.

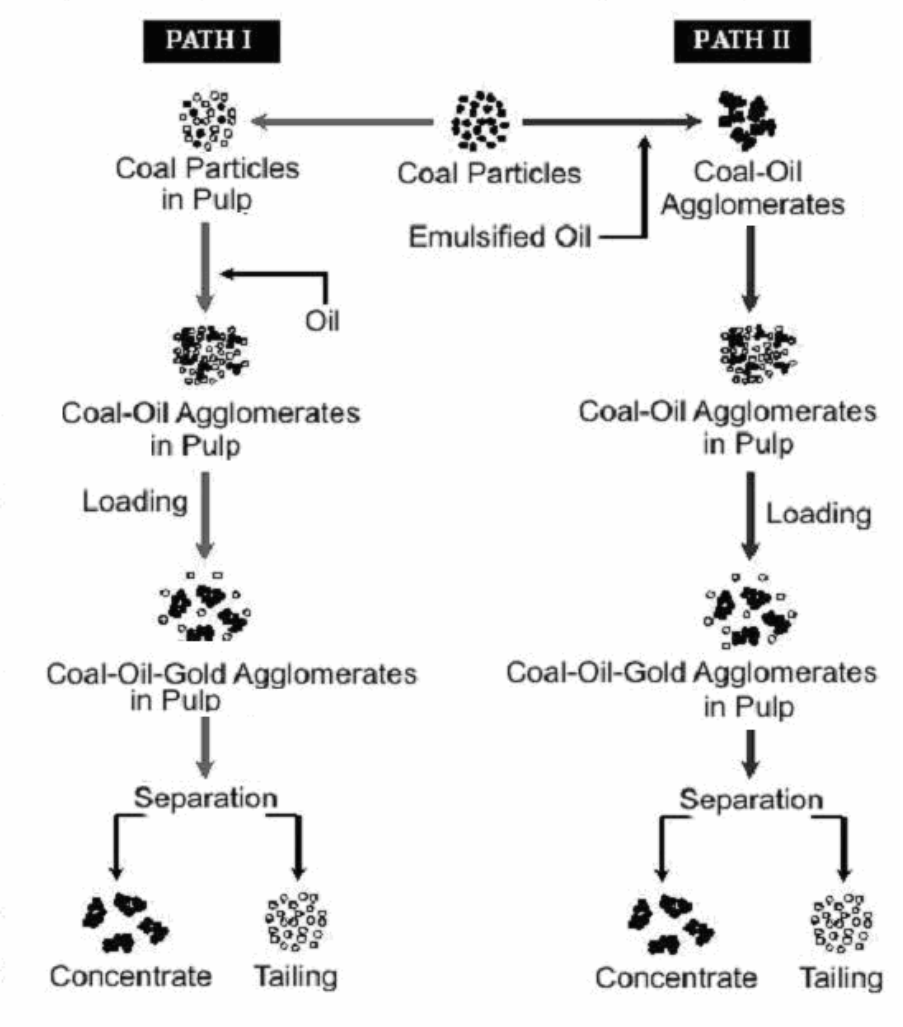

Figure 3.12 Schematic representation of coal-oil-gold agglomeration

However, this method should not be used in some situations where gold is associated with significant quantities of pyrite, arsenopyrite or other sulfide minerals because of their oleophilicity. It has been reported that there is no significant difference in gold recoveries for up to 5% sulfides in the feed (Calvez et al., 1998). These types of gold-bearing ores could be used effectively after they were roasted or leached with oxidized sodium cyanide or by some other leaching methods.

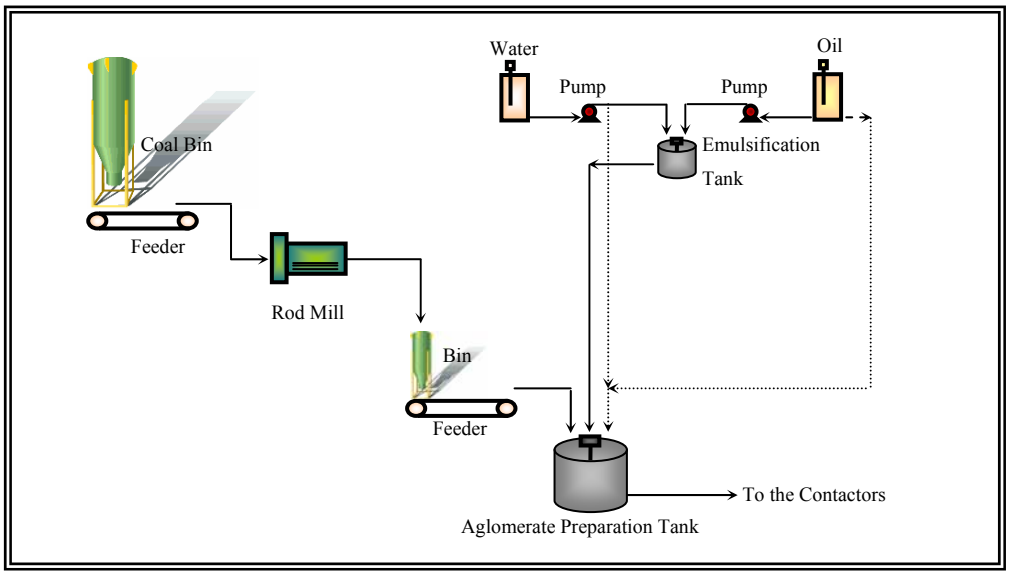

Figure 3.13 Production of the coal-oil agglomerates (Modified from Wall et al., 1992)

After the loaded agglomerates are separated from the slurry by flotation (co-current separation) or screening (current separation), they can be burned to produce a high-grade gold containing coal ash. Selling the gold containing ash to a refinery might be the simplest and cheapest option.

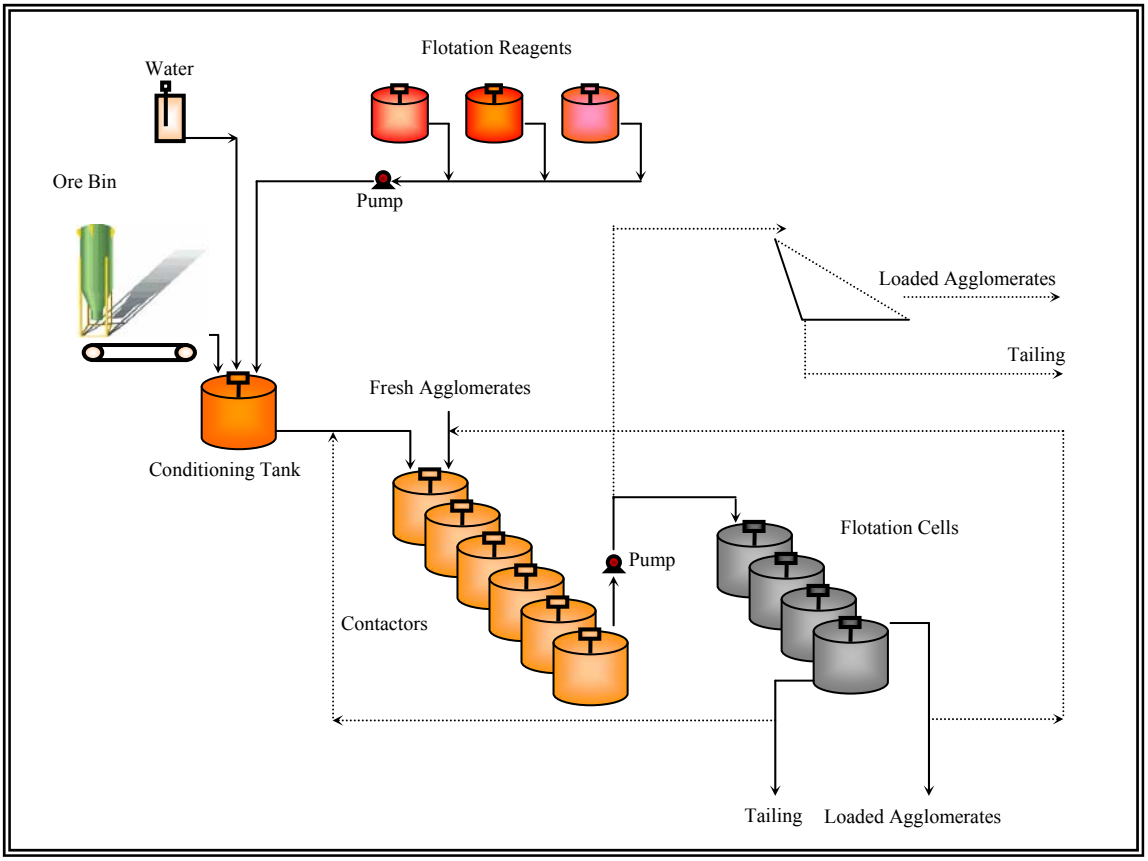

Figure 3.14. Production and separation of the coal-oil-gold agglomerates (Modified from Wall et al., 1992)

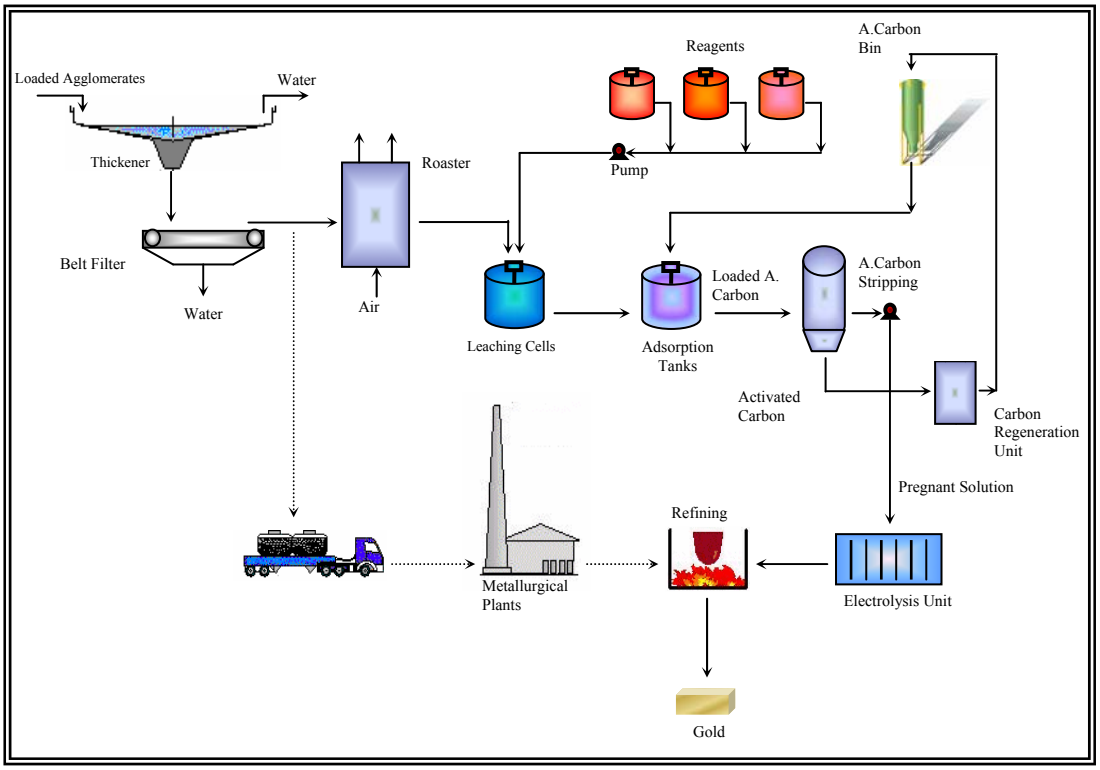

Figure 3.15. Gold production from coal-oil-gold agglomerates (Modified from Wall et al., 1992)

The leaching of the gold provides the possibility of very high extraction. Thiourea or thiosulphate may be used to substitute cyanide, if it can not be used as a leachant for the operation. And it is a very important to note that if cyanidation is used in this stage, the cyanide consumed in the process will be very small, when compared with a conventional gold cyanidation process (House et al., 1988).

Placers, tailings, and free milling ores can be evaluated by coal-oil-gold agglomeration. It has been reported that there is no significant difference in gold recoveries with up to 5% sulfides in the feed (Calvez et al., 1998). The coal-oil-gold agglomeration process consists of two separate parts: production of coal-oil-gold agglomerates and recovery of these agglomerates from pulp. Flotation of the agglomerates or screening can be employed for the recovery stage (House et al., 1988).

The recovery of gold by CGA process depends on many factors: Degree of inter-particle collisions; agglomerate strength and stability; mineralogy of gold (Calvez et al., 1998); type, amount and size of carbonaceous material; type and amount of oil;

type, degree and time of agitation; contact time and number of recycling of the agglomerates (Marciano et al., 1994; Moses and Petersen, 2000; Wu et al., 2004a).

CHAPTER FOUR
MATERIAL AND METHODS

4.1 Coal Samples

The coal samples were received from Kozlu-Zonguldak washery plant feeding material. The bulk samples were transported D.E.U Mining Engineering Department Laboratories and were subjected to characterization studies at Instrumental Analysis Laboratories. Proximate analysis of the samples were done according to the standards TS 4745 (moisture content analysis), TS-ISO 1171 (ash content analysis), TS 363 (total S analysis), TS-ISO 5071-1 (volatile matter), TS 2678 and ISO 1928 (calorific value analysis) the results are given in Table 4.1. The samples were subjected to wet chemical analysis and the assays were determined by using an AnayltikJenaAG novAA 330 Atomic Absorption Spectrometer.

Table 4.1 Proximate analysis of Kozlu-Zonguldak coal sample

Proximate Analysis		**Air dried basis (%)**	**Dry basis (%)**	**Dry ash free basis (%)**
Moisture		0.95	-	-
Ash		13.07	13.20	-
Volatile Matter		27.33	27.59	31.79
Fixed Carbon		58.65	59.21	68.21
Total Sulfur		0.90	0.91	-
Calorific Value (kCal/kg)	Net	6992	7065	8139
	Gross	7224	7293	8402

Table 4.2 Elemental analysis of the Kozlu-Zonguldak coal sample

Elemental Analysis (Ash)	**%**
Na_2O	0.4
K_2O	1.11
Fe_2O_3	18.05
MgO	4.15
CaO	18.35
Al_2O_3	24.30
SiO_2	29.11
K.K	0.30
S	0.46 (SO_3: 1.13)

Characterization studies indicate that the coal is a highly volatile A type bituminous coal.

The samples were subjected to maceral analysis at D.E.U Geological Engineering Department; the results were given in Table 4.3.

Table 4.3 Maceral analysis of Kozlu-Zonguldak coal sample

Macerals	**% (Volumetric)**
Vitrinite	73.42
Semifuzinite	7.24
Inertinite	6.58
Liptinite	3.71
Mineral matter	9.05

4.1.1 Coal Sample Preparation for Agglomeration

The coal samples were crushed below 2 mm by using a jaw crusher and carefully fed to a hammer mill to grind below 300 micron. The ground and sieved coal samples was divided into ready to use samples by using an automatic sample splitter and stored in vacuumed and sealed plastic bags. Each bag was containing 50 grams of ground coal; Figure 4.1 shows the particle size distribution of the samples.

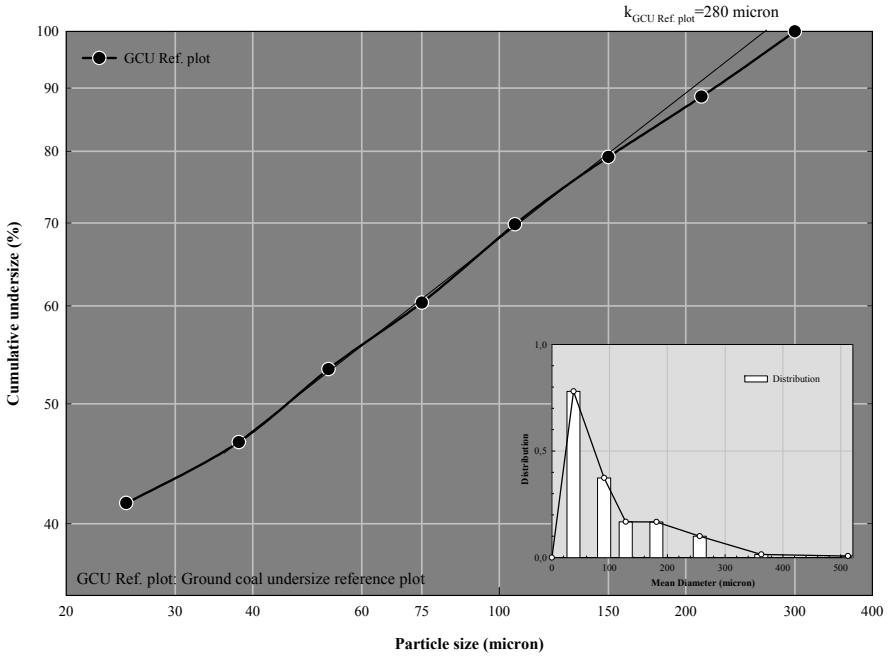

Figure 4.1 Particle size distribution of the Zonguldak - Kozlu coal sample

4.1.2 Coal Sample Preparation for Contact Angle Measurements

The lumps of coal were cut into small pieces (≈2.5 cm x 2.5 cm x 1.5 cm) and placed in a polyester matrix (d: 2 cm). After the polyester matrix was hardened, it was polished using a series of abrasive papers with grit from 60 to 1200, 0.5 μm alumina powders and a fibrous cloth. Abrasive papers and alumina powders were used to flatten and smooth the coal surface. After the polishing step, the samples were stored in vacuumed and sealed plastic bags. The samples were freshly polished on fibrous cloth before taking contact angle measurements.

The physical structure of a solid surface, i.e., its roughness and generally topography, will clearly affect the reactivity of the solid (Yekeler, Ulusoy, Hicyilmaz, 2004). Figure 4.2 represents a natural coal surface. As it is seen from the figure, coal is a very heterogeneous material; it contains organic (macerals), inorganic (minerals) materials and some pores on the surface. This heterogeneity brings different surface characteristics for the same coal surface and pores cause some increase or decreases in the contact angle measurements. Figure 4.3 shows the polishing stages and possible surface disorders.

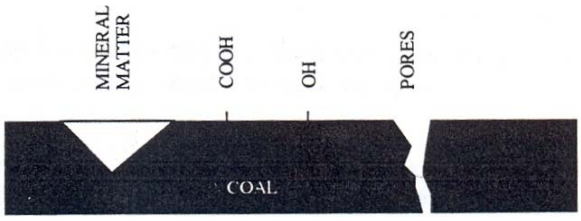

Figure 4.2 Representation of coal surface (Laskowski, 2001)

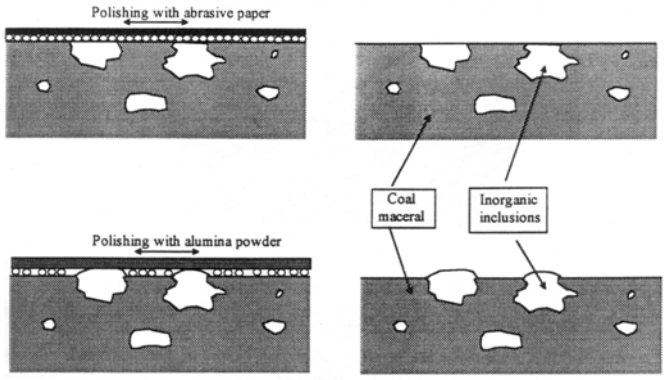

Figure 4.3 Coal Surface polishing and resulting disorders (Laskowski, 2001)

Thus, if the measurement media is liquid and if θ is less than 90º, the liquid will penetrate and fill up most of the hollows and pores in the coal and so form a plane surface, which is effectively part solid and part liquid; since liquid has zero θ with liquid, θ will, therefore, decrease. On the other hand, if θ is greater than 90, the liquid will tend not to penetrate in to the hollows and pores in the coal and can, therefore, be regarded as resting on a plane surface which is effectively part solid and part air; since there is practically no adhesion between the liquid and the entrapped air θ will increase (Shaw, 1989). Surface roughness and heterogeneity are possible causes of contact angle hysteresis (Figure 4.4). For this reason coal lump surfaces should be polished very carefully.

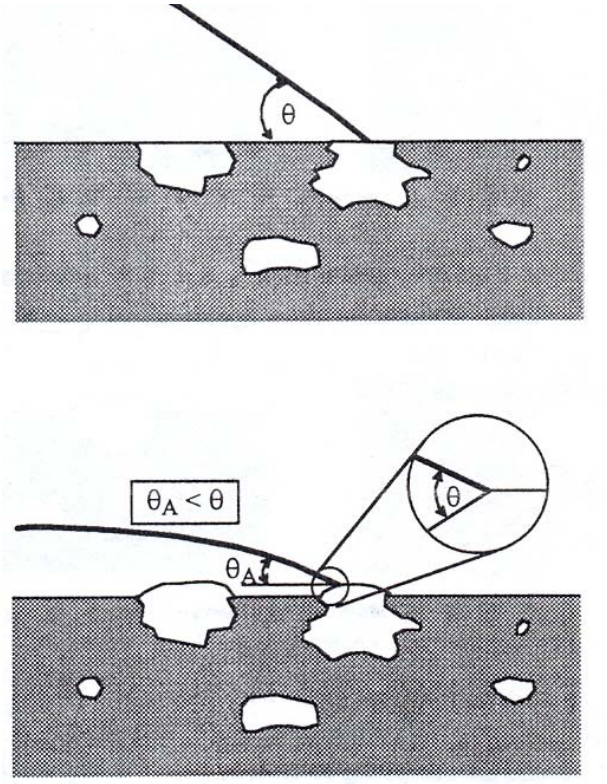

Figure 4.4 A liquid contact with a polished heterogeneous surface (Laskowski, 2001)

Surface roughness values of the polished samples (Ra) have been determined by employing a profilemeter Mitutoyo Surftest SJ 301 device at D.E.U Material and Metallurgical Engineering Department before taking contact angle measurements. The Ra values of the samples are given in Table 4.4. Unfortunately, the Ra values of the samples were found to be relatively high due to the use of a profilometer instead

an atomic force microscope. The coal lumps have some naturally developed cracks in their body and polishing does not remove the natural cracks. Contact angle measurements have been done on the smooth surface areas of the lumps.

Table 4.4 Surface roughness of the polished samples

Sample Number	**Surface roughness (Ra), µm**
1	1,07
2	2,38
3	3,29
4	1,46
5	2,82
6	2,15

4.2 Oil Samples

Six different unrefined vegetable oils, two different petroleum oils and a commercial reagent were used in the study. Density, surface tension and viscosity measurements have been done at Izmir Instute of Technology to characterize the oil samples. The density measurement was done by employing the picnometer method. Brookfield DV-III programmable Rheometer has been used for the viscosity measurements. Sample volume was taken as 15 ml of different oils and the measurements were done at 20 ºC. The viscosity profiles according to rheometer's RPM have been given in Figure 4.5. The viscosity values of different oils at 100 RPM rheometer stirring speed are given in Table 4.5.

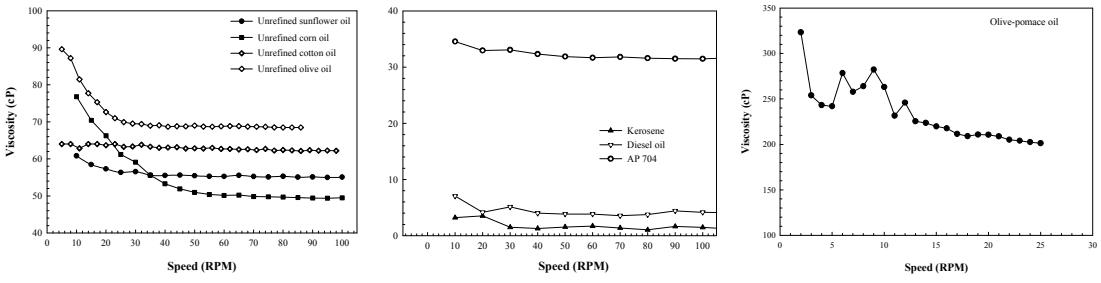

Figure 4.5 Viscosity profiles of the oil samples

Oil/water and air/water interfacial surface tension measurements were done by using a Krüss Contact Angle Measuring System G10 (Goniometer). The

measurements were performed at 20 ºC. The basic properties of the oils were given in Table 4.5.

Table 4.5 Basic properties of the oil samples

Oils	**Density d (g/cm^3)**	**Viscosity η (cP)**	**Surface tension (dyne/cm)**	
			oil/water	**air/oil**
Corn oil	0.946	49.5	4.0	30.7
Cotton oil	0.919	62.0	7.0	31.0
Olive-pomace oil	0.922	201.2	9.2	32.9
Aero promoter 704	0.899	31.5	9.6	33.2
Olive oil	0.947	68.5	13.2	33.3
Sunflower oil	0.930	55.0	16.8	33.9
Diesel oil	0.846	4.16	19.7	28.3
Kerosene	0.811	1.47	29.9	26.7

4.3 Gold Ore Samples

Researchers are aware of problems created by a bias when using non-optimized sampling protocols for precious metals. Poor precision in sampling is responsible for grade reconciliation problems between the mine and the mill for industrial applications and the samples taken from the deposit and the samples prepared for the tests for laboratory studies.

The formula developed by Richard for sampling was used in the study:; $m = k \times d^n$ (Weiss, 1985)

m: Minimum sample weight (libre= 454 g)

d: Diameter of largest particle in the sample (inch= 2.54 cm)

k: 30.000 for gold and

n: 2 for gold.

4.3.1 Efem Cukuru-Izmir Gold Ore Sample

The Efemçukuru project is located in western Turkey, in the Province of Izmir, 20 kilometers southwest of the provincial capital Izmir. The property, held by Tüprag Metal Madencilik San. ve Tic. Ltd. Sti., a wholly owned subsidiary of Eldorado. The ore deposit is a vein-type epithermal gold deposit with related stockwork and replacement mineralization.

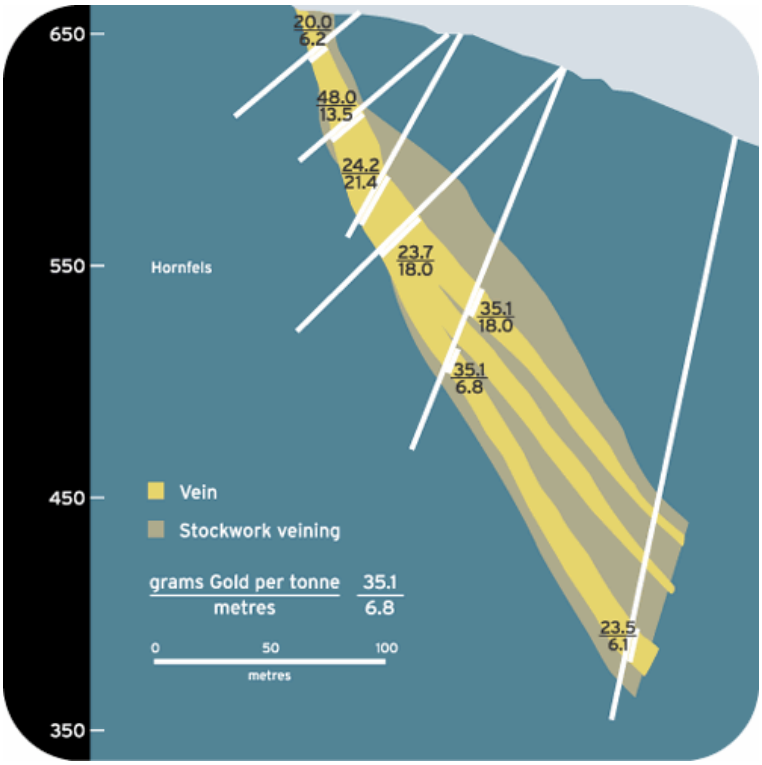

Figure 4.6 Izmir-Efem Çukuru Gold Ore Deposit (El Dorado, 2003)

Oyman et.al. (2003) report that gold mineralization in Efem cukuru district is associated with late pulses of magmatic hydrothermal activity and present zones of hydraulic brecciation adjacent to veins or stockwork zones around a dome-shaped small intrusion. Non-metallic host rock minerals include quartz, rhodonite and rhodochrosite. Associated sulphides include pyrite, pyrrhotite, chalcopyrite, sphalerite and galena, and their oxidized products (Figure 4.7). Most of the gold is very fine (2.5 to 50 microns), occurring as free grains in quartz and carbonate, and as inclusions in sulphide minerals (Oyman et al., 2003).

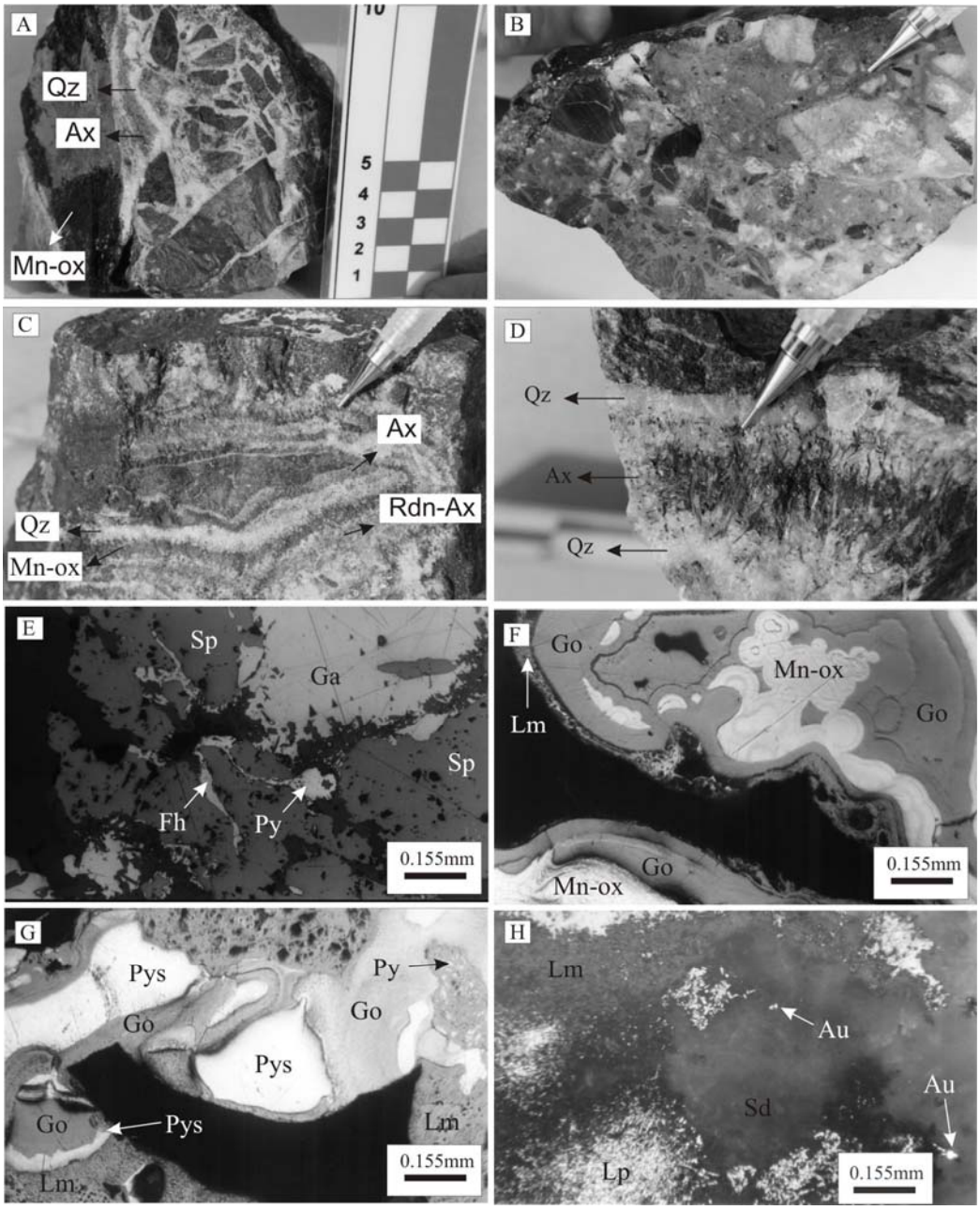

Figure 4.7 Izmir-Efem Cukuru gold ore mineralogy (Produced at D.E.U Geological Engineering Department by Tolga Oyman)

Quartz (Qz), axinite (A), sphalerite (Sp), galena (Ga), pyrite (Py), fahlore group (Fh), manganoxides (MnO), goethite, (Go), limonite, (Lm), pysilomelane (Pys), gold (Au), siderite (Sd), lepidocrocite (Lp)

XRD, XRF and wet chemical analysis of the samples were done at D.E.U Mining Engineering Department. The ore is consisted of the following minerals; Quartz (SiO_2), Pyrite (FeS_2), Hematite (Fe_2O_3), Chlorite $(Mg, Fe, Al)_6(Si, Cr)_4O_{10}(OH)_8$, Rhodochrozite ($MnCO_3$), Rhodonite ($MnSiO_3$), Sphalerite (ZnS), Galena (PbS),

Chalcopyrite ($CuFeS_2$), Sericite (Complex Silicates), Goethite ($HFeO_2$) and various clay minerals.

The sample was subjected to particle size analysis; fractional gold distribution was determined and presented in Figure 4.8.

Table 4.7 Izmir-Efem Cukuru gold ore sample

Element (g/ton)	
Au	10.80
Ag	13.4
Cu	193
Zn	1065
Pb	1125
Element (%)	
MnO %	16.62
Fe_2O_3 %	5.6
SiO_2 %	65.52
CaO %	3.35
S %	1.08
Al_2O_3 %	2.37
LOI %	3.43

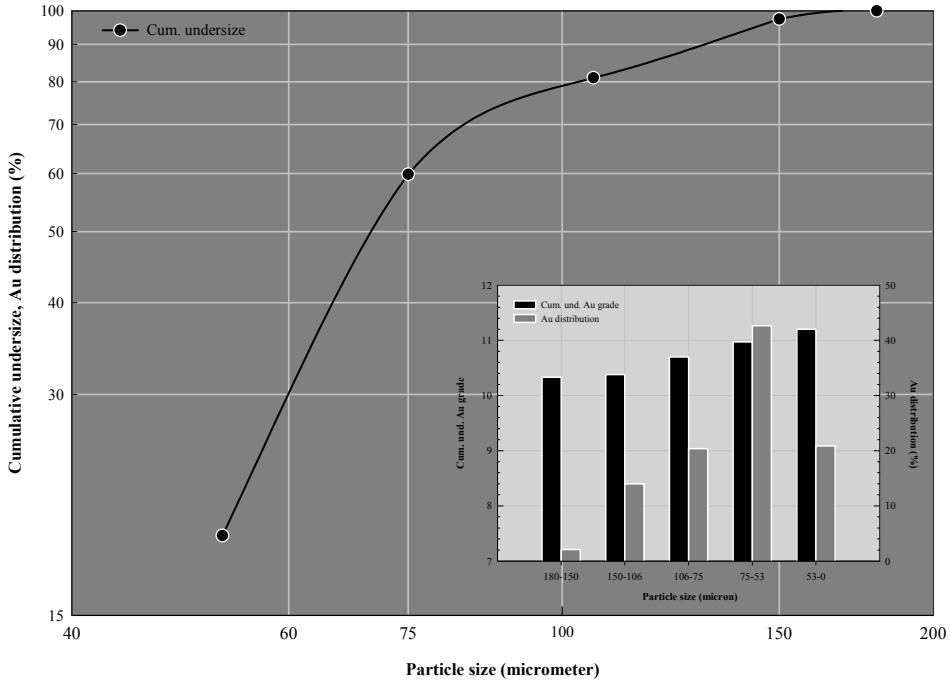

Figure 4.8 Izmir-Efem Cukuru gold ore sample size and Au distribution curves

For a gold ore sample reduced to below 0.18 mm;

$m = 30.000 \times (0.0071)^2 = 0.687$ kg sample should be taken to provide an acceptable accuracy.

The sample weighing 320.67 kg was crushed below 5 mm by a jaw crusher, split and 150 kg of the sample was archived. The rest of the material was crushed below 2 mm and ground below 0.18 mm by a ball mill. The ground ore split into the new samples weighing 1000 grams by an automatic sample splitter.

4.3.2 Bergama-Ovacik Gold Ore Sample Preparation

Ovacık Gold Mine is in the Western Anatolia, 100 km north of İzmir. It is located 20 km east of Dikili.and 12 km west of Bergama. The total reserve established by the exploration activities so far in the Ovacik Gold Mine is 4.2 million tons. The measured and indicated gold reserve suitable for the operation within this resource is 2.4 million tons at a grade of 10 grams/ton (Ovacik gold mine, 2007).

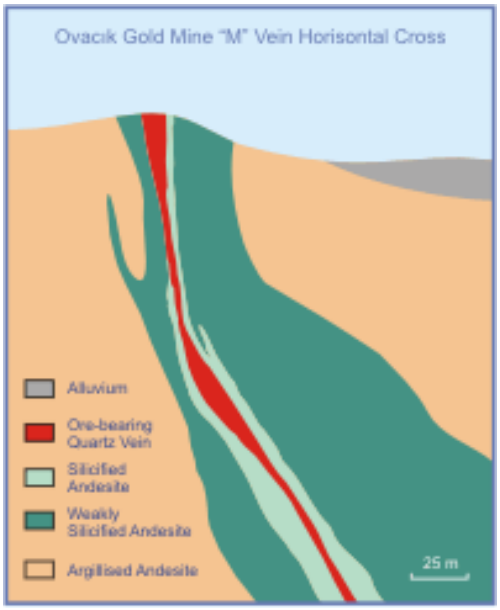

Figure 4.9 Ovacik gold mine horizontal cross (Ovacik gold mine, 2007)

The ore zone opaque minerals consist mainly of disseminated marcasite, pyrite and native silver; these are associated with relatively high Sb values, although no Sb

minerals have been identified. The ore minerals formed mostly as disseminations within subordinate veinlets, subsequent to the silica alteration across the deposit as a whole. Ore mineral assemblages identified in the Ovacik and Narlica deposits are typical of low- and intermediate- sulfidation epithermal systems pyrite. Gold occurs as fine <0.05 mm grains of electrum (72%Au, 25% Ag, 3% Hg, Cu, Zn, Pb, As and Fe) or in native form, precipitated between interstices of quartz crystals (Figure 4.10). High-grade Au mineralization is typically accompanied by sulfide assemblages including base metal sulfides, the distribution of which is confined to the center of the M vein. In decreasing order of abundance, the major opaque minerals are pyrite, chalcopyrite, arsenopyrite, sphalerite, galena, (Ag-rich) tetrahedrite, tennantite, bornite, native gold, electrum and achantite (Yilmaz et al., 2006).

XRD, XRF and wet chemical analysis of the samples were done at D.E.U Mining Engineering Department, the results were given in Table 4.8.

Table 4.8 Bergama-Ovacik gold ore sample

Element (g/ton)	
Au	8-10
Ag	48.2
Cu	26.98
Zn	295.31
Pb	253.43
Mn	39.29
Fe	7102
Mg	361.46
Co	9
Cr	111
Element (%)	
SiO_2	95.23
S	0.42

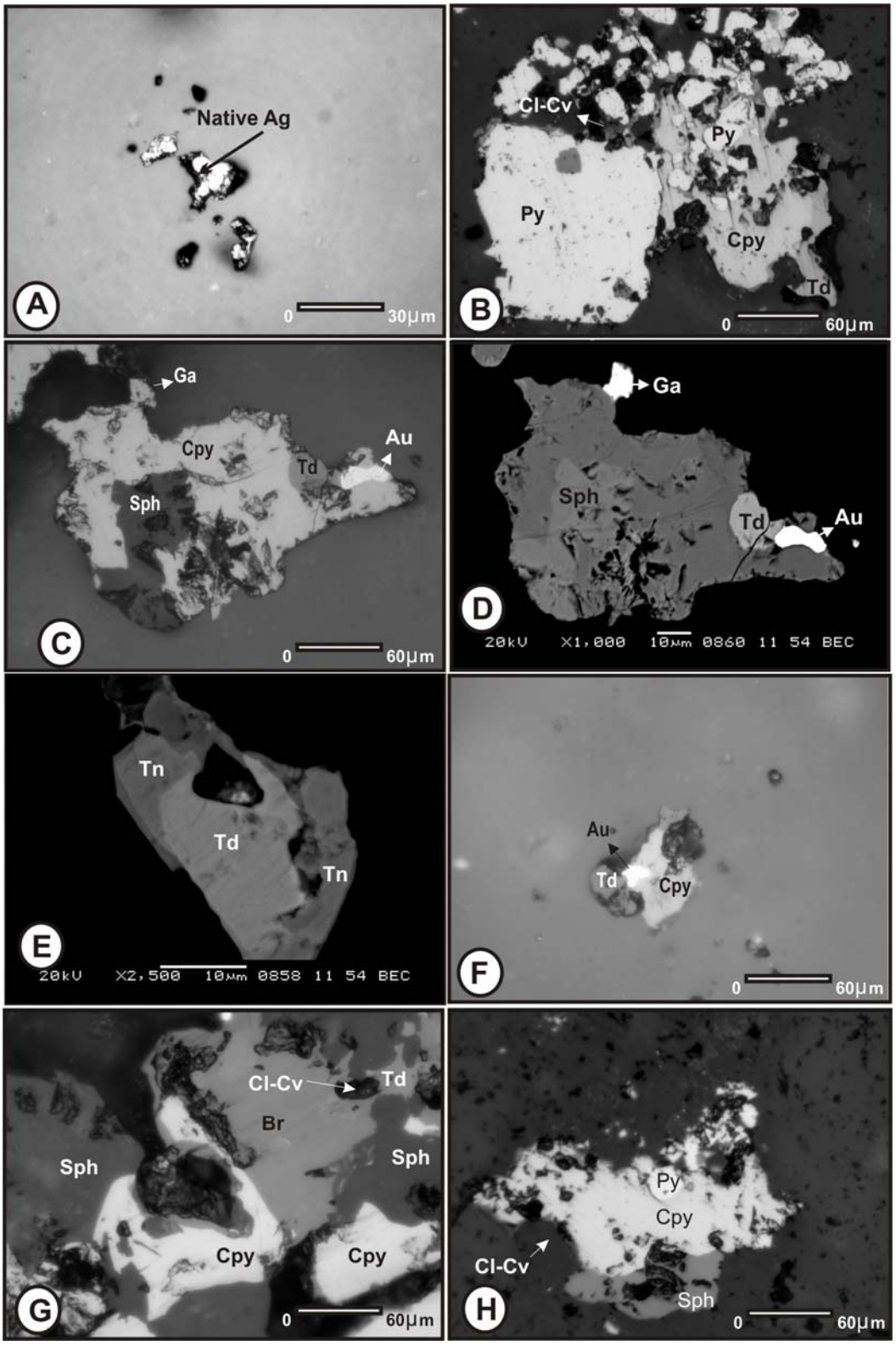

Figure 4.10 Bergama - Ovacik gold ore mineralogy (reflected plane-polarized light except D and E, which are SEM-BSE images) (Produced at D.E.U Geological Engineering Department by Tolga Oyman)

*Euhedral to subhedral pyrites (Py), chalcopyrite (Cpy), tetrahedrite (Td), Chalcocite–Covellite (Cl–Cv), Native silver (Ag), Native gold (Au), Sphalerite (Sph), galena (Ga), Tennantite (Tn), bornite (Br)

For a gold ore sample reduced to below 0.06 mm;

$$m = 30.000 \times (0.0039)^2 = 0.167 \text{ Libre}$$

At least $m = 76$ g sample should be taken to provide an acceptable accuracy.

The sample weighing 251.32 kg was crushed below 5 mm by a jaw crusher, split and 125.93 kg of the sample was archived. The rest of the material was crushed by a jaw crusher and ground below 0.06 mm by a ball mill. The ground ore split into the new samples weighing 400 grams by an automatic sample splitter.

The sample was subjected to particle size analysis; fractional gold distribution was determined and presented in Figure 4.11.

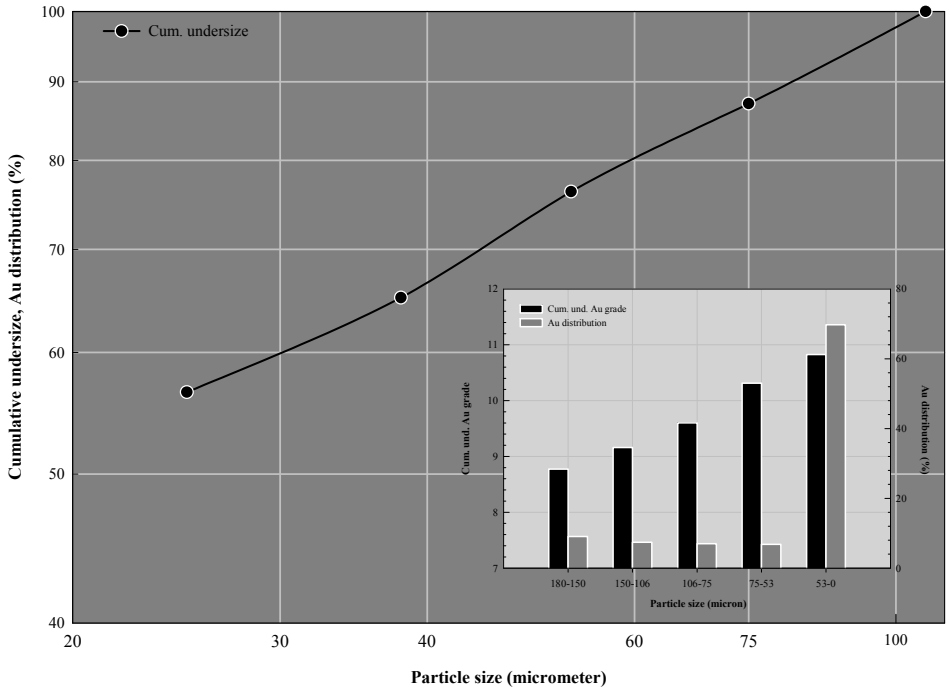

Figure 4.11 Bergama-Ovacik gold ore sample size and Au distribution curves

CHAPTER FIVE
EXPERIMENTAL STUDIES

5.1 Coal-Oil Agglomeration

5.1.1 Material

Agglomeration studies have been run on the coal samples characterized in section 4.1 (See Table 1, Table 2, Table 3 and Table 4) using oil samples characterized in section 4.2 (See Figure 4.5 and Table 4.6).

5.1.2 Method

Contact angle and advancing-receding contact angle measurements were done on the polished coal samples. Advancing and receding contact angles were determined to reveal the hysteresis of the measurements at first step and contact angle measurements have been determined at the second step.

After the determination of the contact angles, coal-oil agglomeration studies were done using the coal the oils given in section 4.1 and section 4.2. Theoretical and experimental evaluations of the results were completed together.

5.1.2.1 Contact Angle Measurements

Advancing and receding contact angles have been done to reveal the hysteresis of the measurements for the coal/air/water and coal/air/oil systems with a goniometer/microscope setup (Rame Hart NRL contact angle goniometer model-100) using sessile drop method at D.E.U Mining Engineering Department. A micro-droplet of double-distilled water (or oils) was generated with a micrometer syringe and placed on a specific portion of the coal lump. The volume of the liquid was increased or decreased in order to produce the movement at the air/water/coal (air/oil/coal for oils) interface was measured using a microscope–goniometer system for the droplet. Figure 5.1 and 5.2 shows the results of the advancing and receding contact angle measurements. All the measurements were done at 20 ºC.

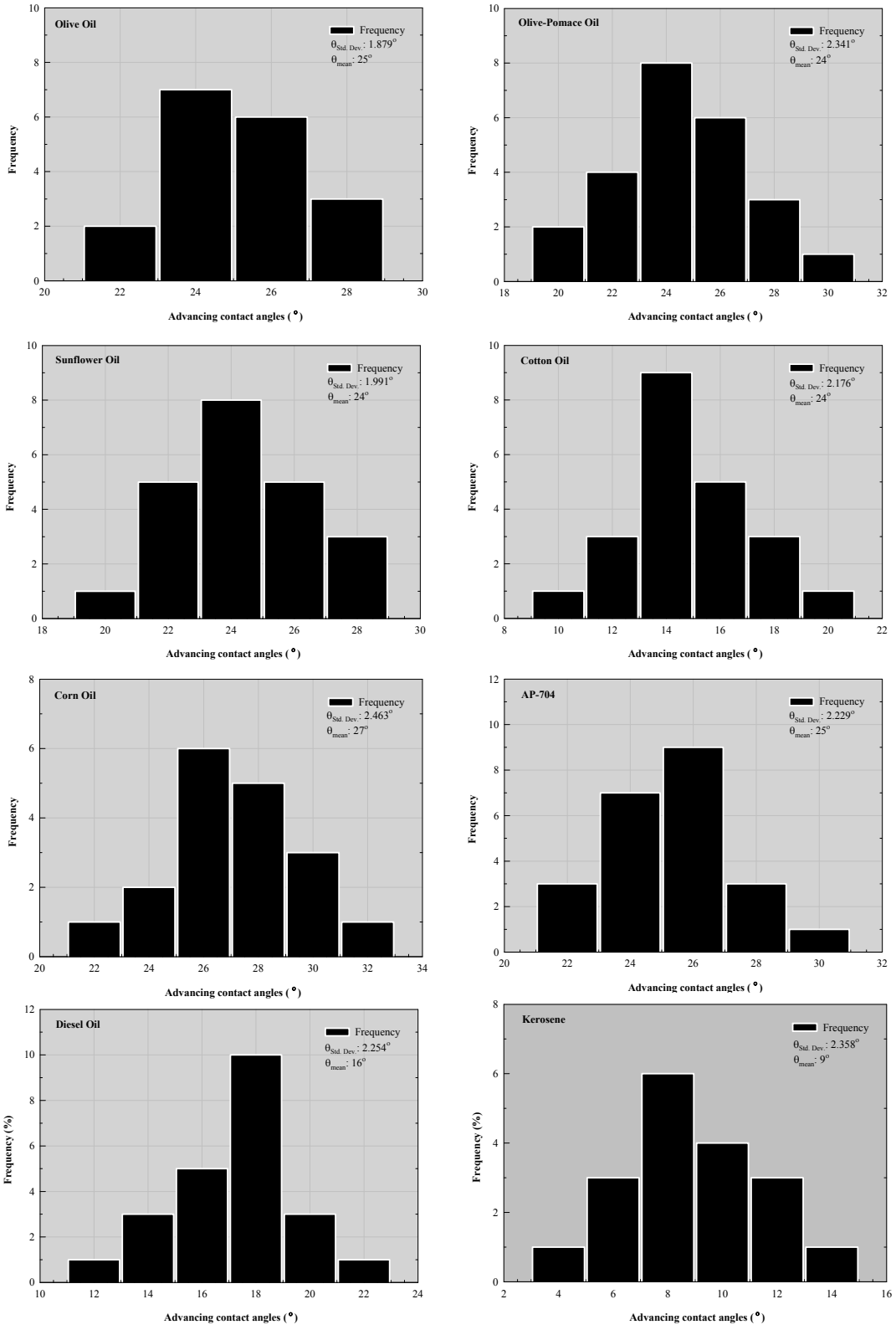

Figure 5.1 Results of the advancing angle measurements

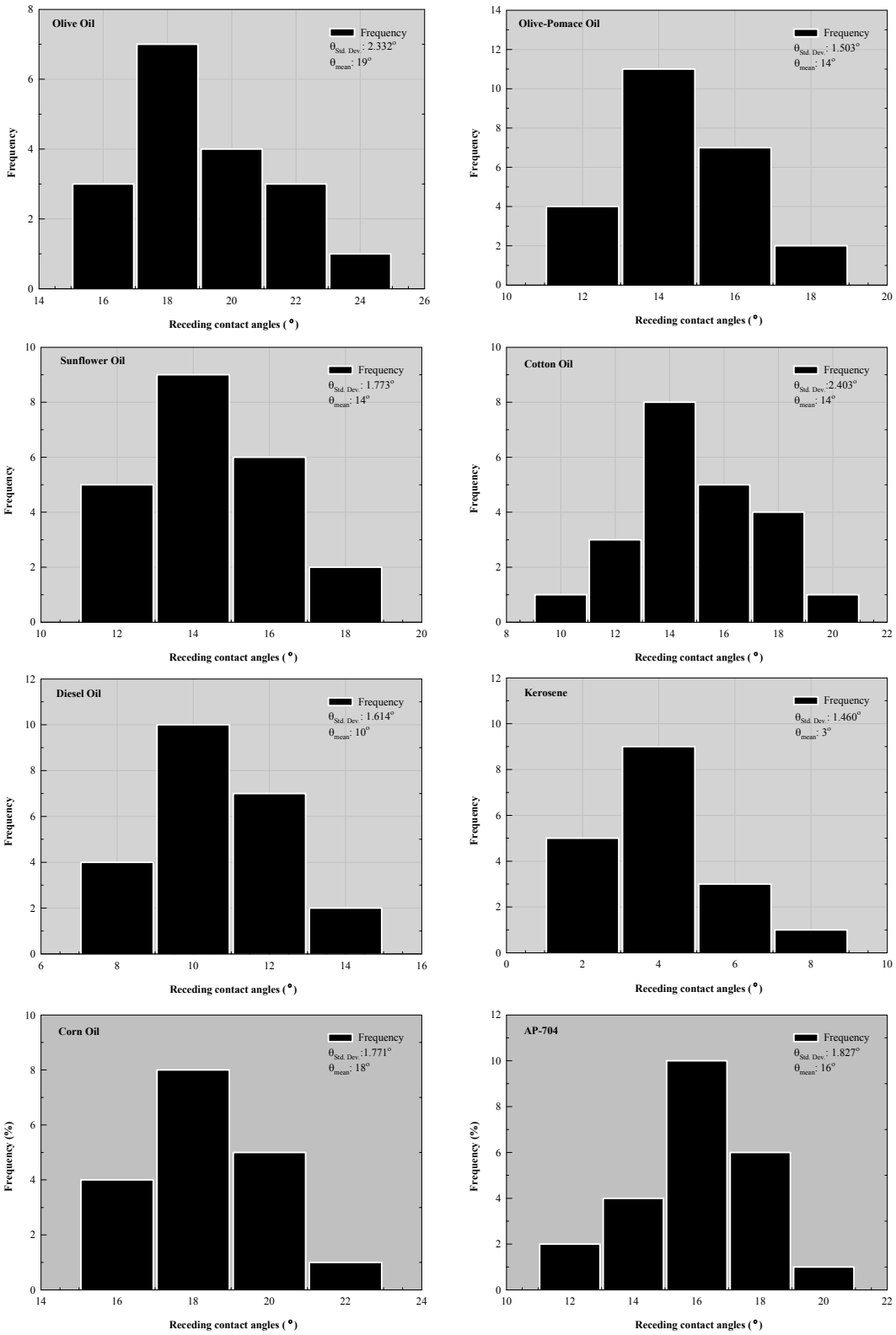

Figure 5.2 Results of the receding angle measurements

The measurements were repeated for several times and hysteresis values of the measurements were determined for each coal lump. Figure 5.3 represents the hysteresis values via surface roughness (Ra) (See section 4.1; Table 4.5).

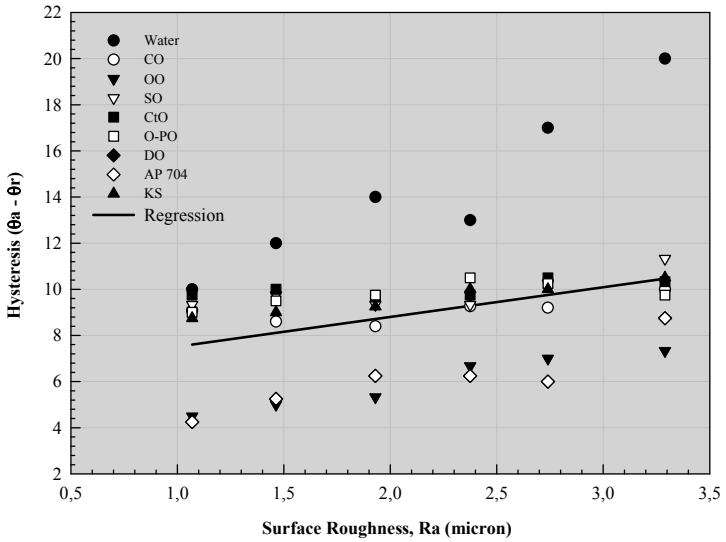

Figure 5.3 Hysteresis values versus surface roughness (Ra)

Figure 5.3 implies that the increase in the roughness values of the polished surfaces increases the hysteresis values (θa- θr) of the measurements. Calculated hysteresis values seems to be acceptable values, thus the highest hysteresis value of the measurements was 11°.

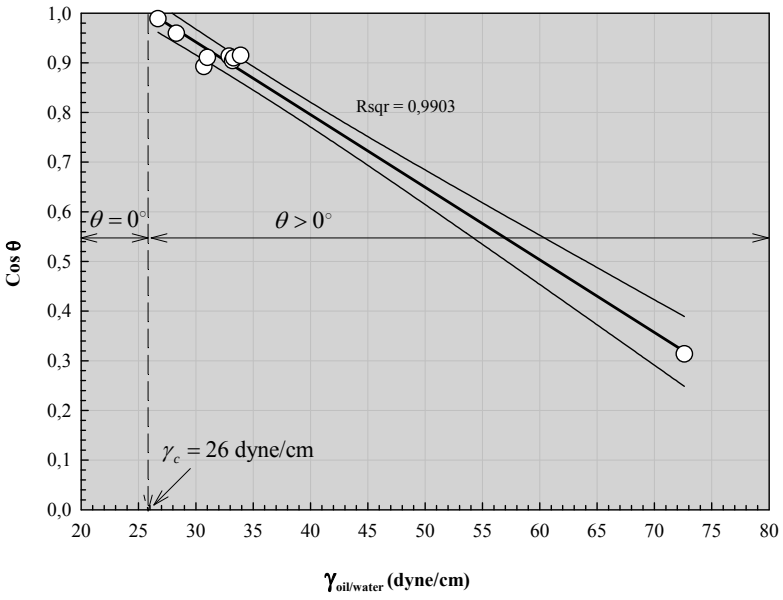

Figure 5.4 Zisman's plot for the Zonguldak-Kozlu coal sample

Cos θ vs. surface tension of the oils and pure water was plotted to obtain the value of γ_{LV} at the intercept where Cos θ=1 (critical surface tension of wetting). Figure 5.4 indicates that the coal surface is completely wetted when immersed in liquids of $\gamma_{LV} < \gamma_C$ (26dyne/cm). Wettability decreases for increasing the surface tension values (γ_{LV}). The surface tensions of the oils used were all around the critical surface tension value of the coal sample.

Contact angle measurements were done in a square glass chamber. The chamber was filled with double-distilled water. The coal lump was placed on stable supports with the flat surface facing downward. A micro-droplet of different oils was generated with a U- shaped needle using a microsyringe and placed on a specific portion of the polished coal lump surface from below. The temperature remained at about 20 ºC in all the measurements. Figure 5.5 illustrates the experimental setup.

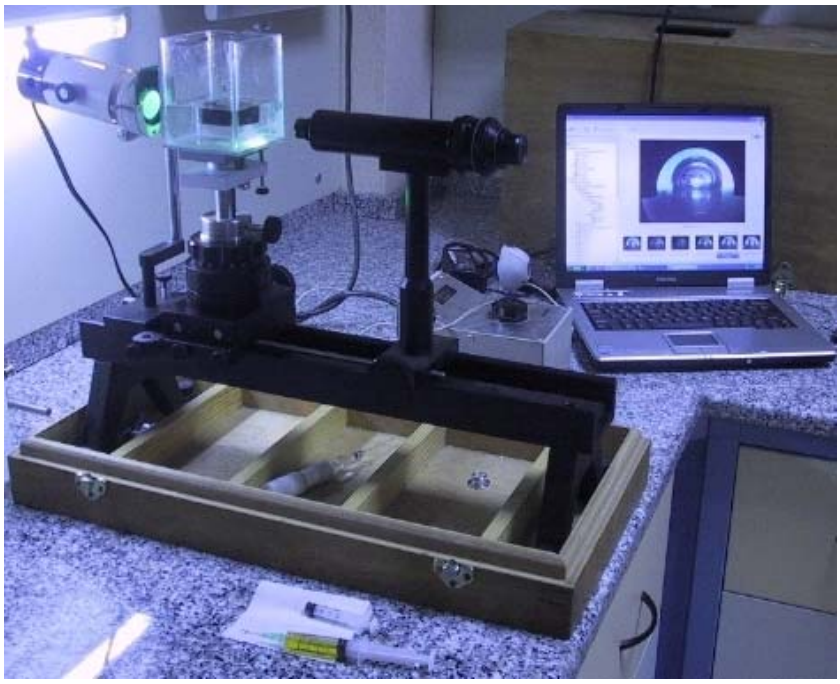

Figure 5.5 Illustration of the experimental setup for contact angle measurements

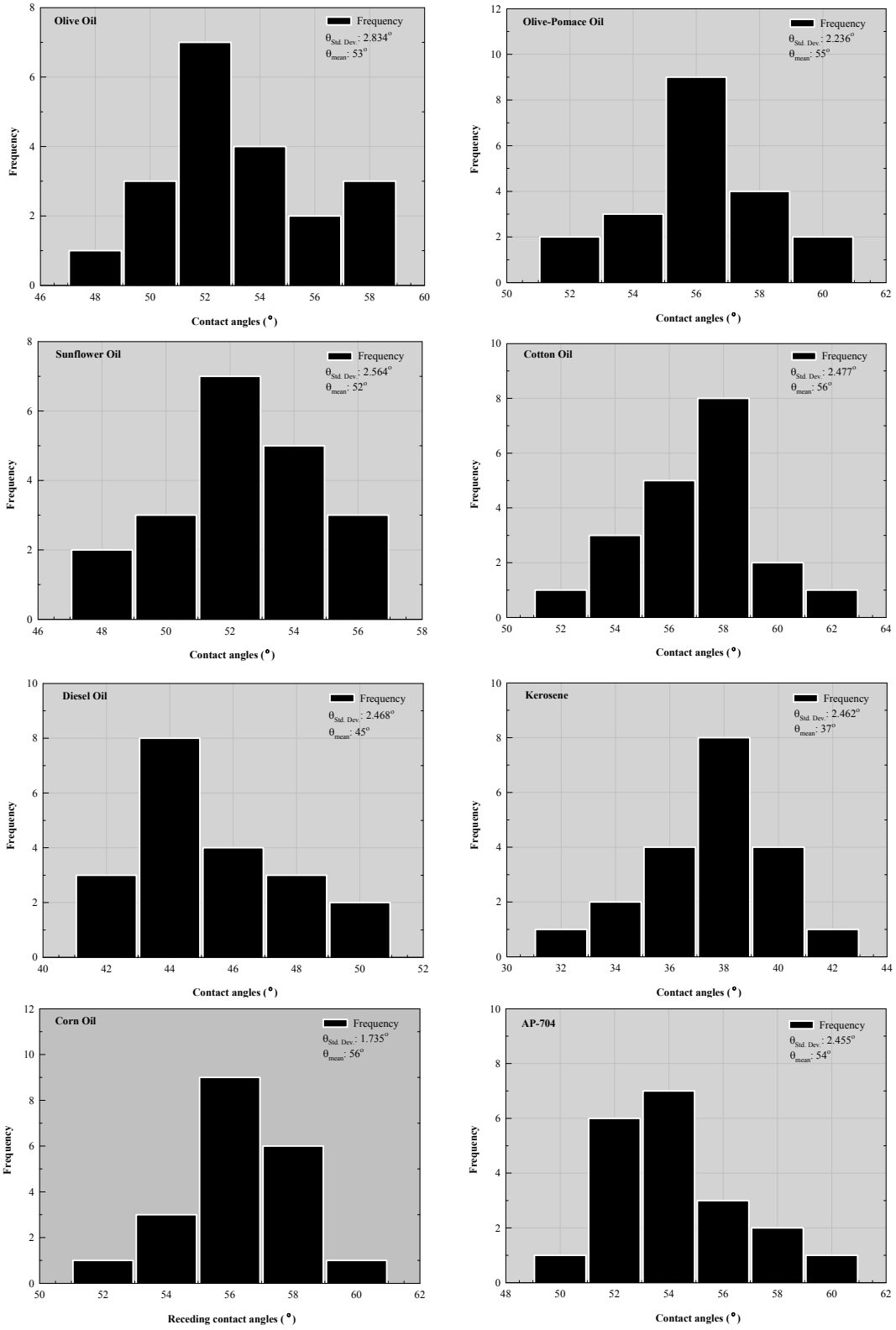

Figure 5.6 Results of the contact angle measurements

Interesting to note that vegetable oils gave very close contact angle values. The petroleum oils have given the lowest contact angles. The results were given in Table 1.

Table 5.1 Results of the contact angle measurements

Oils	**Contact angles, (°)**	
	oil/water (θ_{oil})	**air/oil (θ_{oil})**
Corn oil	56	27
Cotton oil	56	24
Olive-pomace oil	55	24
Aero promoter 704	54	25
Olive oil	53	25
Sunflower oil	52	24
Diesel oil	45	16
Kerosene	37	9

5.1.2.2 Evaluation of the Processes at Water/Oil/Coal Interface

Figure 5.7 represents a sessile liquid droplet resting on a flat surface. The values of the work of cohesion of the liquid and the work of adhesion of the liquid to the solid determine the contact angle at the solid/liquid/gas interface.

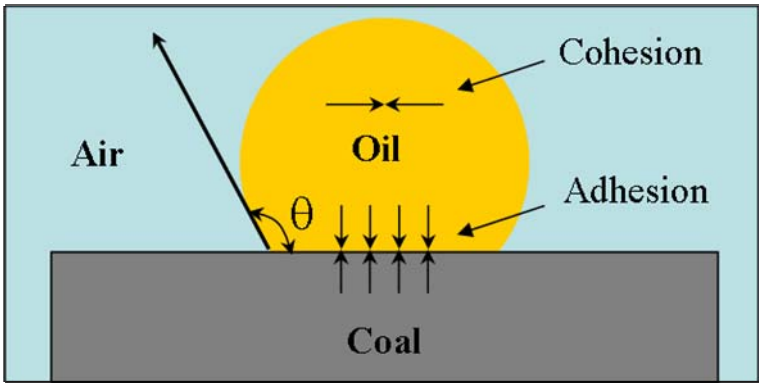

Figure 5.7 Adhesion and cohesion on the solid surface

$$W_{LL} = 2\gamma_{LV}$$
$$W_{LL} = 2 \times 72.8$$
$$W_{LL} = 145.2 \text{ dyne/cm}$$

$$\frac{W_{SL}}{145,2} = \frac{72,6 \times (1 + \cos 72)}{2 \times 72,6} \quad W_{SL} = 95.11 \text{ dyne/cm}$$

$W_{SL} < W_{LL}$. $\theta \neq 0$ hydrophobicity occurs on the coal surface.

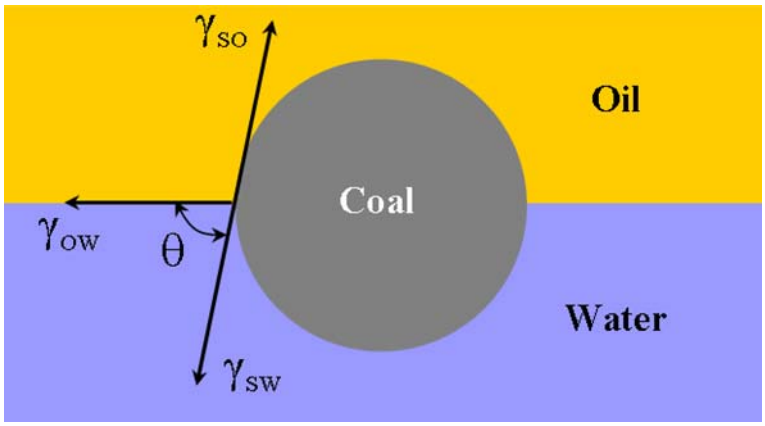

Figure 5.8 Illustration of a solid particle at oil/water interface

When a coal particle is placed in an oil-water interface (Figure 5.8) may be drawn into the aqueus phase (θ<90º) or remain concentrated at the interface (θ=90º) or drawn into the oil phase (θ>90º). Oil agglomeration occurs when condition θ>90º is satisfied (Cebeci & Sonmez. 2006)

The major factors affecting agglomeration of coal by an immiscible liquid include wettability of the coal, the type and amount of the bridging oil and type and intensity of conditioning. It is often assumed that oily droplets at a mineral/water interface spread onto the mineral surface and form a thin film (Laskowski. 2001).

$$\Delta G_{spread} = \gamma_{SO} + \gamma_{OW} - \gamma_{SW}$$
$$\Delta G_{spread} = \gamma_{OW}(1 - Cos\theta_{oil})$$
Since $\theta_{oil} = 180 - \theta$
$$\Delta G_{spread} = \gamma_{OW}(1 + Cos\theta)$$
$$\Delta G_{attach} = \gamma_{OW}(Cos\theta - 1)$$

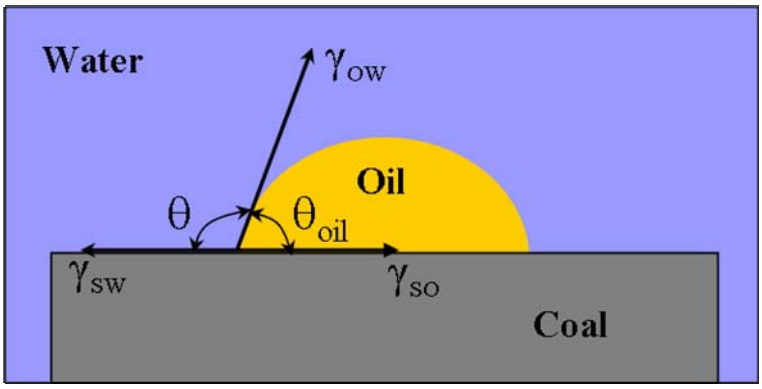

Figure 5.9 Illustration of an oil droplet on a solid in an aqueous phase

The attachment of an oil droplet to a mineral particle is likely to; take place only if $\theta > 0$. for $\theta = 0, \cos\theta = 1$ and $\Delta G_{attach} = 0$ and the attachment is impossible. The equations indicate that ΔG_{spread} is always a positive number; thus, thermodynamically the spreading of oil into a thin film at the solid/water interface is always unlikely.

The collected data (See section 4.2; Table 4.6 and section 5.1; Figure 5.6) have been evaluated to determine ΔG_{spread} and ΔG_{attach} values. The results have been given in Table 5.2.

Table 5.2 Evaluation of the processes at water/oil/coal interface

Oils	**Density d (g/cm^3)**	**Viscosity η (cP)**	**Surface tension (dyne/cm)**		**Contact angle (°) (θ_{oil})**		**ΔG_{spread} $\gamma_{OW}(1+Cos\theta)$ (dyne/cm)**	**ΔG_{attach} $\gamma_{OW}(Cos\theta-1)$ (dyne/cm)**
			oil/water	**air/oil**	**oil/water**	**air/oil**		
Corn oil	0.946	49.5	4.0	30.7	56	27	1.76	-6.24
Cotton oil	0.919	62.0	7.0	31.0	56	24	3.09	-10.91
Olive-pomace oil	0.922	101.2	9.2	32.9	55	24	3.92	-14.48
Aero promoter 704	0.899	31.5	9.6	33.2	54	25	3.96	-15.24
Olive oil	0.947	68.5	13.2	33.3	53	25	5.26	-21.14
Sunflower oil	0.930	55.0	16.8	33.9	52	24	6.46	-27.14
Diesel oil	0.846	4.16	19.7	28.3	45	16	5.77	-33.63
Kerosene	0.811	1.47	29.9	26.7	37	9	6.02	-53.78

It can be said that; thermodynamically the oils having higher γ_{OW} interfacial tensions are having more chance to attach the coal particles. Although, the spreading behaviors of the oils were very similar, the oils having lower interfacial tensions

have relatively higher chance to spread over the coal particles (Figure 5.10 and Figure 5.11).

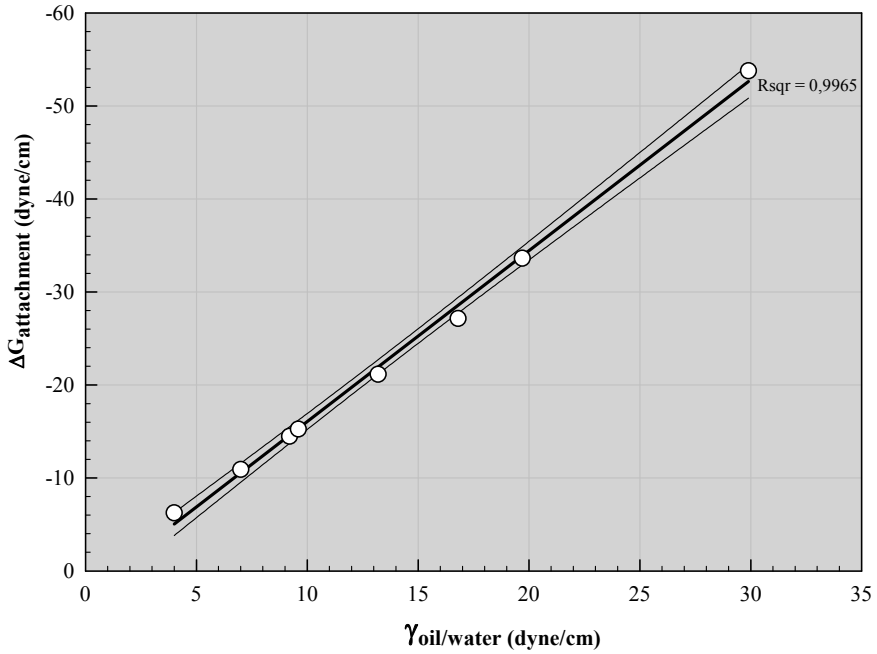

Figure 5.10 Calculated ΔG_{attach} values for the oils having different interfacial tensions

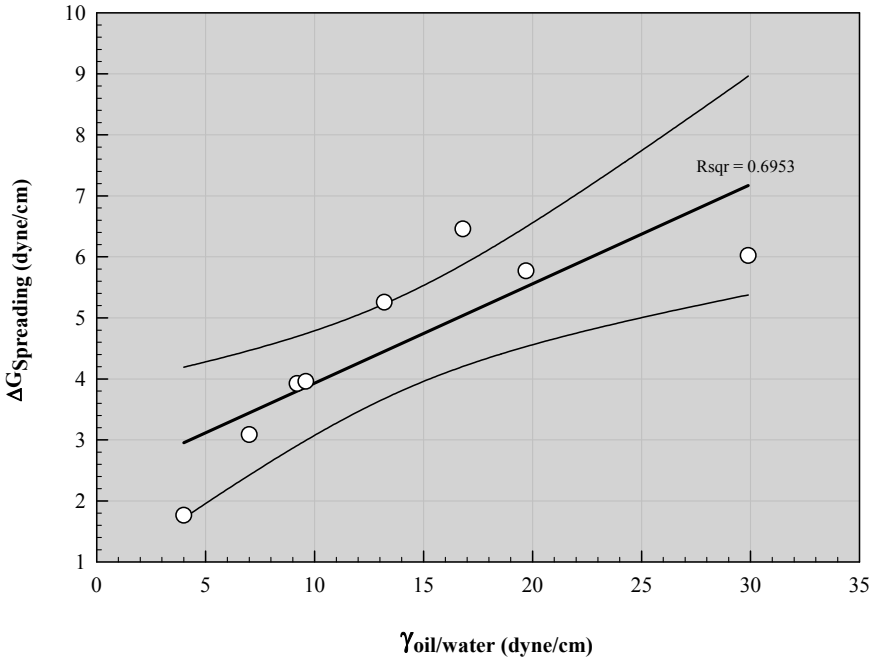

Figure 5.11 Calculated ΔG_{spread} values for the oils having different interfacial tensions

5.1.3 Agglomeration experiments

Agglomeration experiments were done using an IKA Eurostar (Power control-visc) stirrer. Total agglomeration volume used for agglomeration was 500 ml. A four baffled agglomeration cell (Figure 5.12) and a turbine impeller (four bladed) were used for emulsification. A standard agglomeration procedure was applied in all tests. Thus, different oils (1/25 coal/oil ratio) was added to the emulsifying cell, agitated at 2000 rpm for 5 minutes. Coal particles (10 % by weight) were added to the oil-water emulsion and agitated at 2000 rpm for 3 minutes. The revolution speed was decreased to 1300 rpm and pulp was agitated for another 2 minutes.

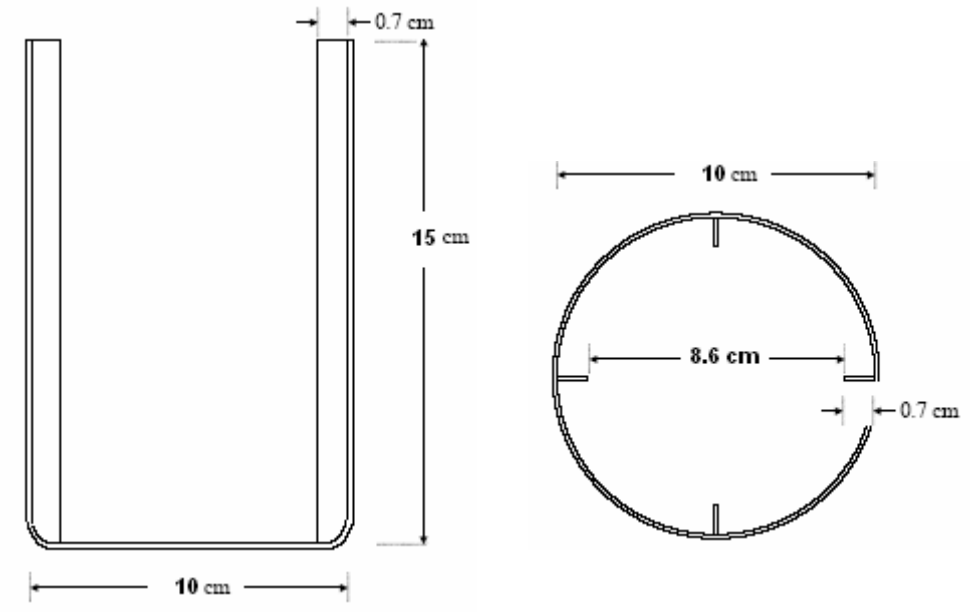

Figure 5.12 Illustration of the agglomeration cell

Two criterions were taken into account to evaluate the agglomeration performance;

- The first one involves the size of the produced agglomerates (it could affect the available surface area of the agglomerates for gold penetration in coal-oil assisted gold flotation).

- The other involves ash rejection by agglomeration, proximate analysis of the agglomerates were done for this purpose (it could represent selectivity against the gangue mineral in coal-oil assisted gold flotation).

The produced agglomerates were analyzed using a series of test sieves. A special sieving method was developed for this purpose. The method involves avoiding the mechanical effects of sizing while getting the same success in sieving. Figure 5.13 represents the validity of the method according to the formal size analysis method.

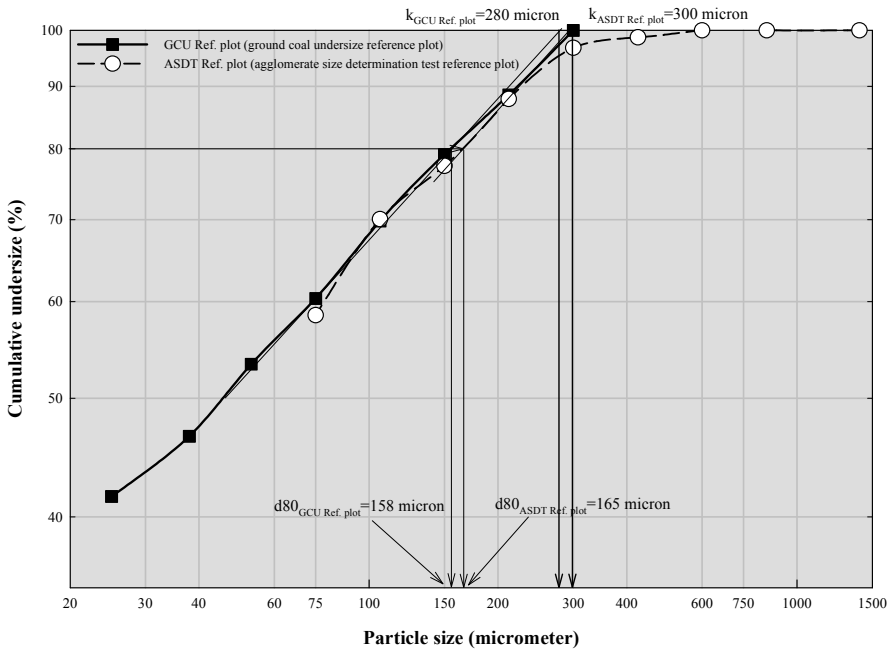

Figure 5.13 The Schuhmann plots of formal size analysis and agglomerate size determination test

Very similar results were obtained for k and d_{80} values. Formal size analysis method has been indicated the values as k= 280 microns and d_{80}=158 microns for the ground coal sample, while the agglomerate size determination test indicated the same values as k= 300 microns, and d_{80}=165 microns. So it has been verified that "agglomerate size determination test (ASDT)" was capable for determining the agglomerate size distribution.

Agglomeration studies were done with different oils. Agglomerated samples were subjected to agglomerate size determination tests; the results are given by the following figures (Figure 5.14, see Appendix 1)

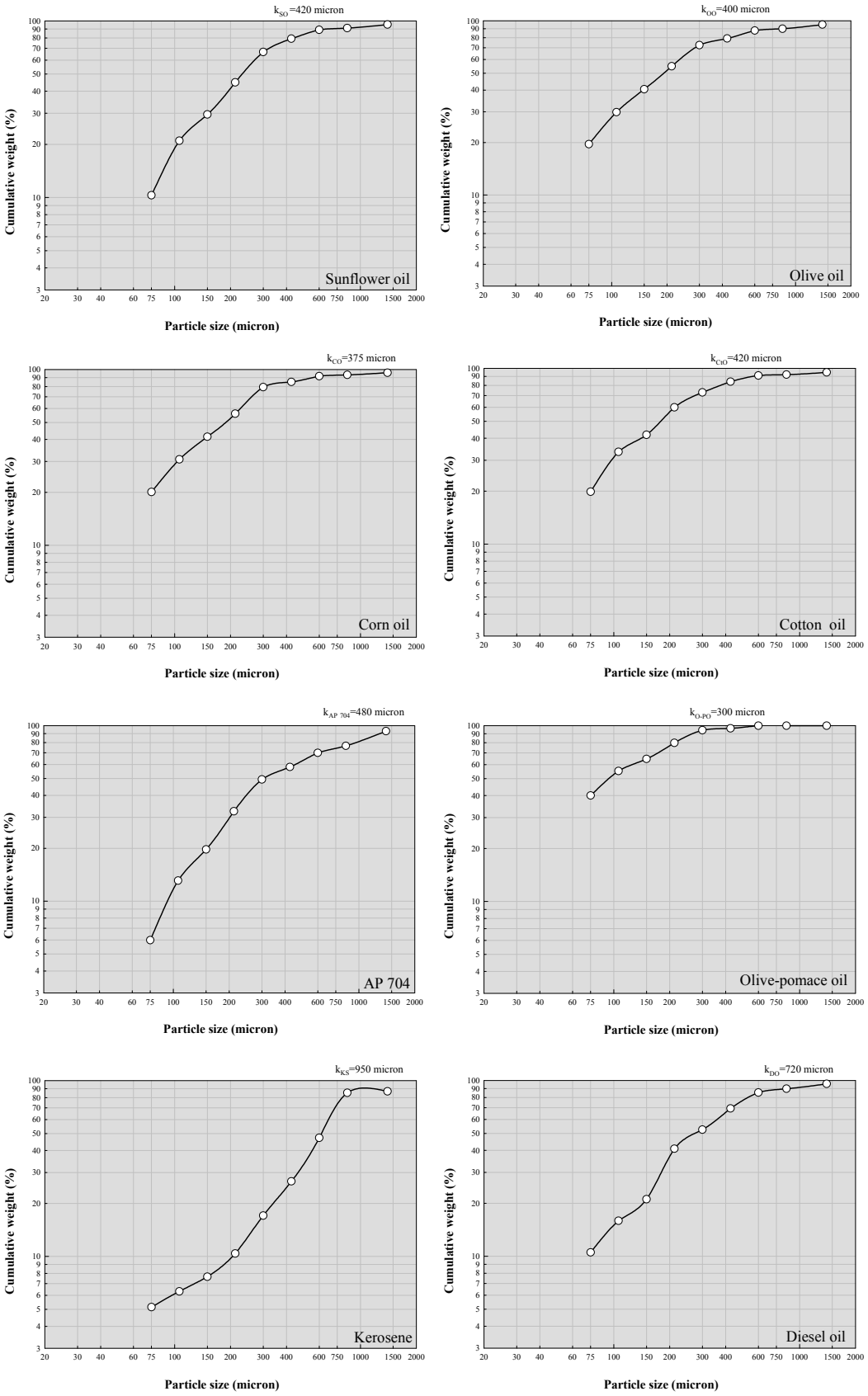

Figure 5.14 Schuhmann size distribution curves of the agglomerates

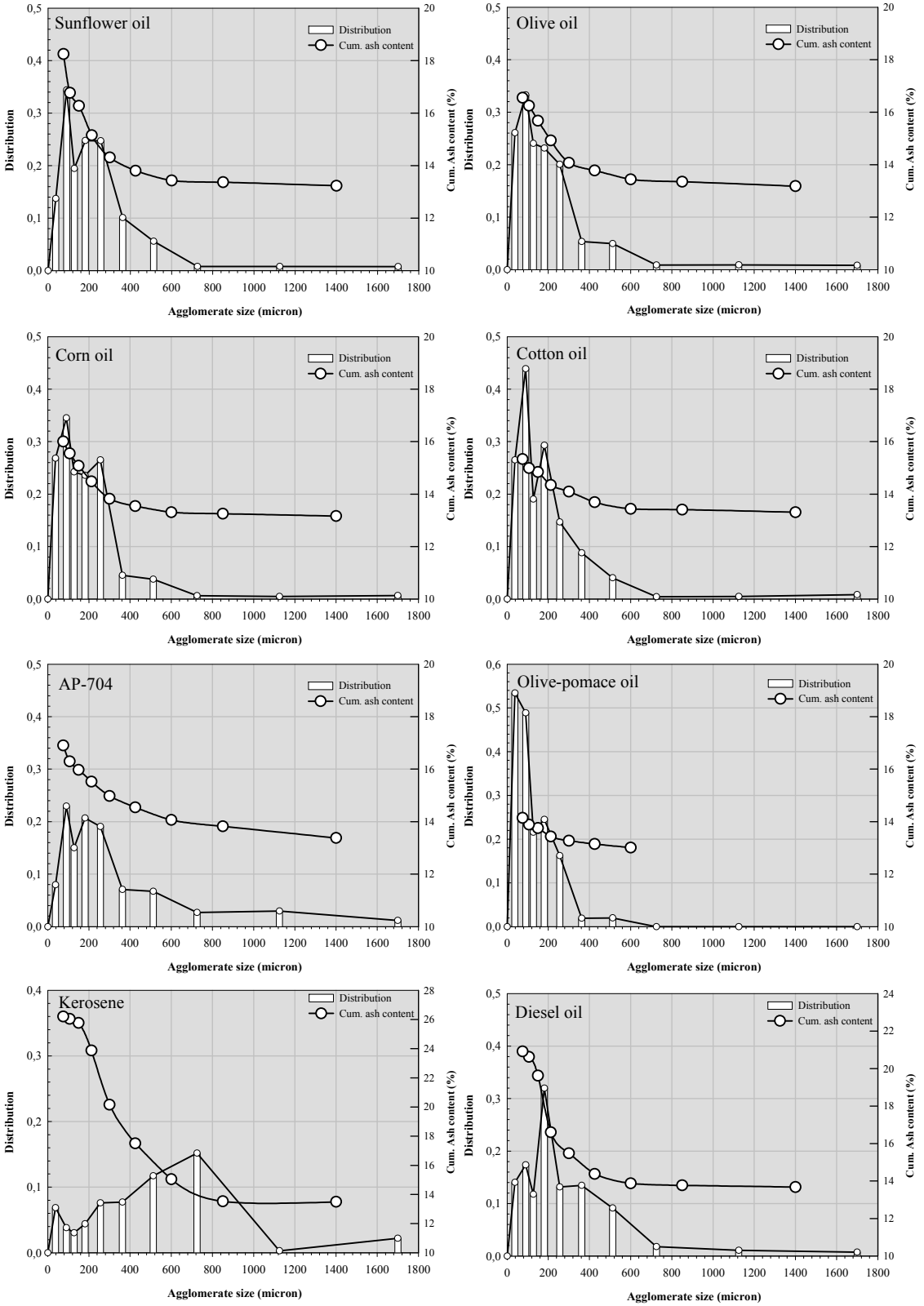

Figure 5.15 Normal distribution curves and cumulative ash content of the agglomerates

Schuhmann plots were drawn to find out the k values of the produced agglomerates. The plots showed that petroleum oils produced the coarsest agglomerates (950 micron for kerosene and 720 micron for diesel oil). Vegetable oils generate agglomerates having k values about 400 micron (Figure 5.15).

Normal distribution curves and cumulative ash content of the agglomerates were plotted via particle size (Figure 5.15). Figure 5.15 shows the same findings; petroleum oils make the agglomerates coarser and particle size of the agglomerates accumulated above 200 micron.

Ash rejection of the Kerosene and diesel oil was also higher than the vegetable oils. Petroleum oils produced the agglomerates having the highest ash content at the finest size ranges (-75 micron) which was also indicating the selectivity against the gangue material.

The further examination of the agglomeration processes with different oils was done by evaluating k, d_{80} and d_{50} values of the agglomerates with corresponding surface tension, viscosity and density values of the oils (Figure 5.16, Figure 5.17 and Figure 5.18). The figures imply that oils having higher oil/water interfacial surface tension values produced coarser agglomerates and oils having higher density values and oils having higher viscosity values produced finer agglomerates.

Table 5.3 Updated presentation of the evaluation of the processes at water/oil/coal interface

Oils	**Density d (g/cm^3)**	**Viscosity η (cP)**	**Surface tension (dyne/cm)**		**Contact angle (°) (θ_{oil})**		**ΔG_{spread}** $\gamma_{OW}(1+Cos\theta)$ **(dyne/cm)**	**ΔG_{attach}** $\gamma_{OW}(Cos\theta-1)$ **(dyne/cm)**	**Agglomerate particle size**			
			oil/water	**air/oil**	**oil/water**	**air/oil**			**d_{50}**	**d_{80}**	**k**	**ά**
Corn oil	0.946	49.5	4.0	30.7	56	27	1.76	-6.24	200	300	375	1.10
Cotton oil	0.919	62.0	7.0	31.0	56	24	3.09	-10.91	175	310	420	0.79
Olive-pomace oil	0.922	101.2	9.2	32.9	55	24	3.92	-14.48	90	210	300	0.58
Aero promoter 704	0.899	31.5	9.6	33.2	54	25	3.96	-15.24	300	400	480	1.47
Olive oil	0.947	68.5	13.2	33.3	53	25	5.26	-21.14	200	320	400	1.00
Sunflower oil	0.930	55.0	16.8	33.9	52	24	6.46	-27.14	235	350	420	1.19
Diesel oil	0.846	4.16	19.7	28.3	45	16	5.77	-33.63	300	560	720	0.79
Kerosene	0.811	1.47	29.9	26.7	37	9	6.02	-53.78	610	825	950	1.56

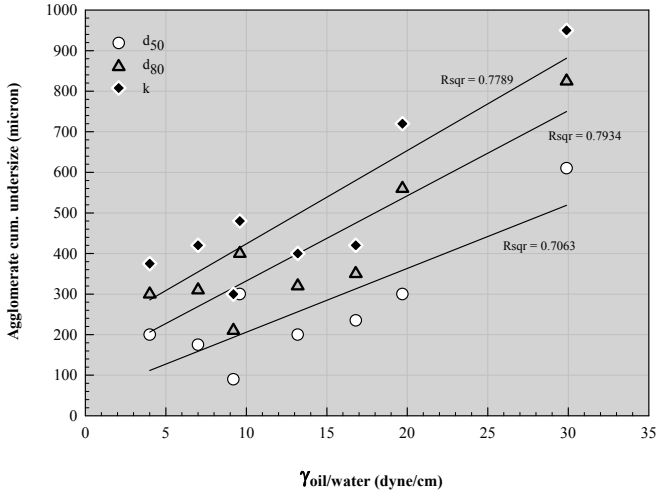

Figure 5.16 The correlation between the oil/water interfacial tension of oils and produced agglomerate particle size

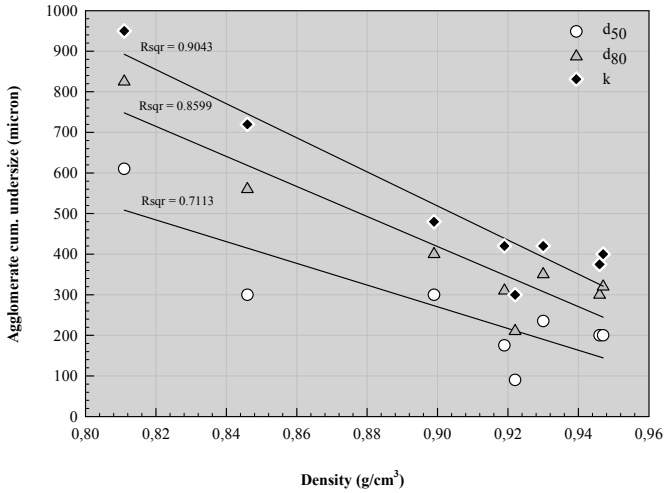

Figure 5.17 The correlation between density of oils and agglomerate particle size

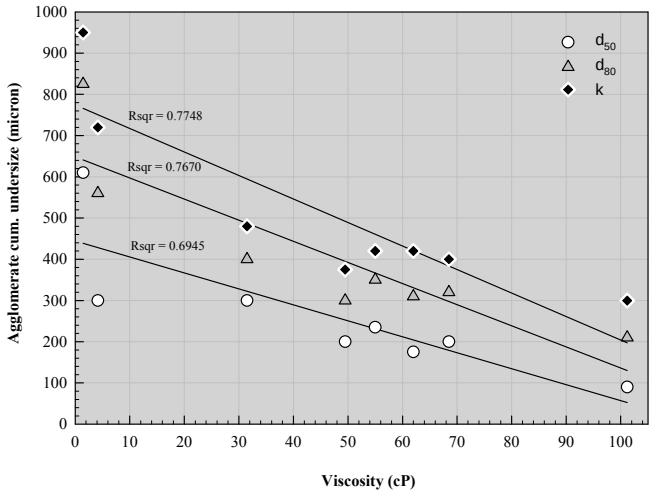

Figure 5.18 The correlation between viscosity of oils and agglomerate particle size

5.2 Flotation Studies

Flotation studies have been run to investigate the effect of reagent synergism on the recovery of gold from Izmir-Efem Cukuru (characterized in section 4.3.1) gold ore and Bergama-Ovacik gold ore (characterized in section 4.3.2) samples.

5.2.1 Flotation Studies of Efem Cukuru-Izmir Gold Ore

5.2.1.1 Material and Method

- Flotation studies have been carried out at natural pH of the gold ore. Outokumpu laboratory scale flotation machine has been used for the flotation tests
- 2.5 l/min. aeration rate has been applied for all the tests
- Batch flotation tests were carried out in a round shaped flotation cell (2 l)
- 1000 g of Izmir-Efem Cukuru gold ore sample ground below 0.180 mm and averaging 11 g/ton gold grade has been subjected to the flotation experiments
- KAX (50 g/t) and Aerofloat 208 (50 g/t) and one more different collector (50 g/t) were used in each experiment
- Na_2S (70 g/t) was used
- Copper sulfate ($CuSO_4$) was used as an activator (150 g/t)
- Na_2SiO_3 (400 g/t) was used as a depressant
- Aerofroth 70 (25 g/t) was used as a frother
- 25 minutes of total conditioning time and 8 minutes of total flotation time have been applied for kinetic flotation tests.
- The results have been evaluated by classical first-order flotation model (Klimpel, 1999)

5.2.1.2 The Effect of Reagent Synergism

Flotation studies have been run to investigate the effect of reagent synergism on the gold recovery. The effect of using different reagent combinations has been given in the following figures.

Figure 5.19 shows that concentrate gold grade tends to decrease after 0.5 minute while concentrate gold recovery increased with flotation time. The gold recovery was poor and the grade of the concentrate was very low. Ultimate recovery and flotation rate were 54.68 % and 0.55 respectively. (Appendix 2 – Table 1).

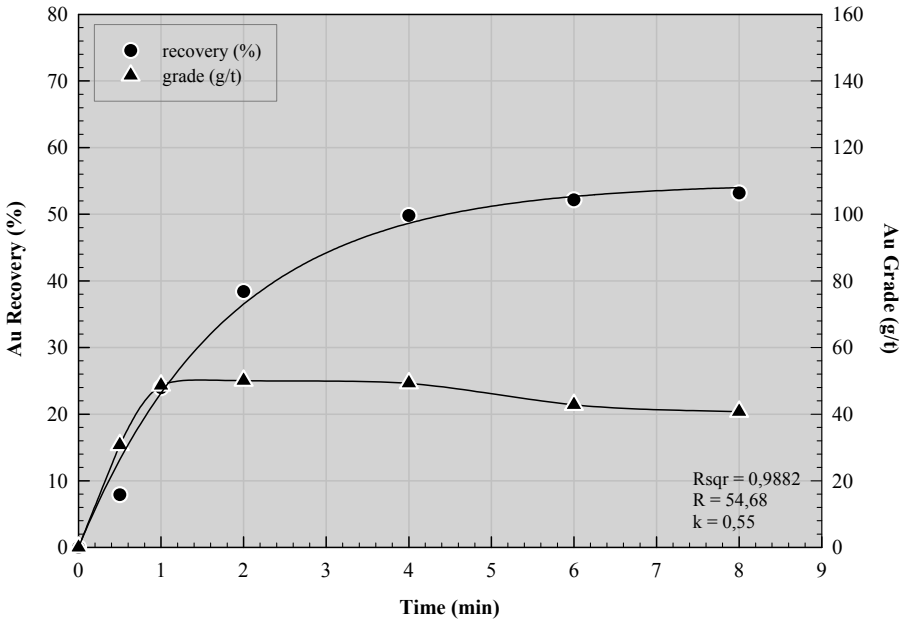

Figure 5.19 The effect of 150 g/t $CuSO_4$ addition on the time-recovery curve for gold (collector combination: KAX-Aerofloat 208 and Aero 6697)

Figure 5.20 implies that gold grade decreases with flotation time while recovery increases. By comparing Figure 19 with Figure 5.20, it is observed that concentrate gold grade and ultimate recovery values increased from 40.77 g/t to 46.62 g/t and 54.68 % to 64.54 % respectively. The flotation rate value was the same, 0.55. The results in these experiments show that Na_2S addition to the pulp increases the concentrate grade and ultimate recovery values of the flotation with KAX-Aerofloat 208, Aero 6697 and $CuSO_4$ combination but does not affect the flotation rate (Appendix 2 – Table 2).

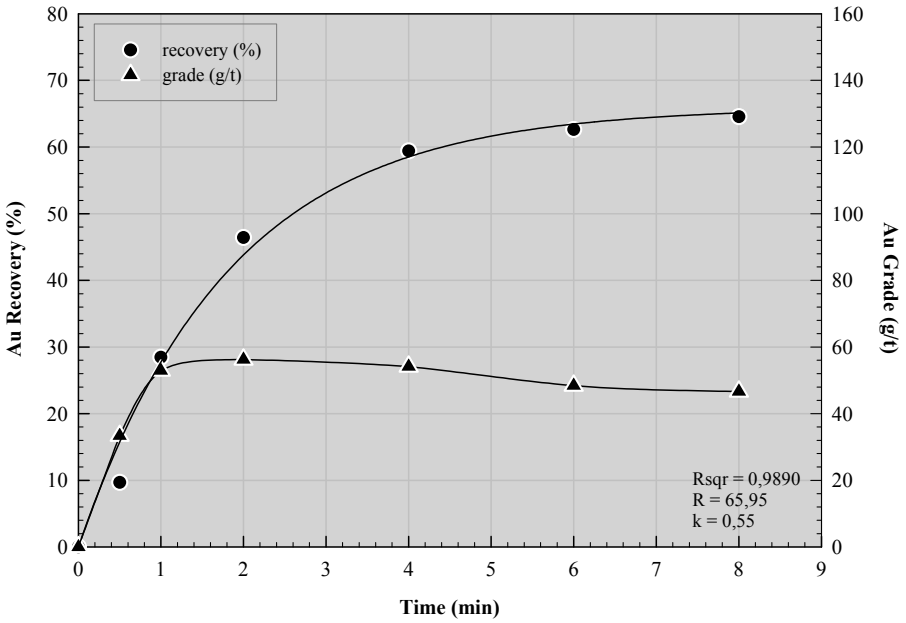

Figure 5.20 The effect of 70 g/t Na_2S addition on the time-recovery curve for gold (collector combination: KAX-Aerofloat 208 and Aero 6697)

Concentrate grade was affected slightly by the flotation time while the recovery values of the concentrates. KAX-Aerofloat 208 and Aero 412 combination produced a concentrate assaying 68.33 g/t Au with 63.76 % ultimate recovery (Figure 5.21, Appendix 2 – Table 3).

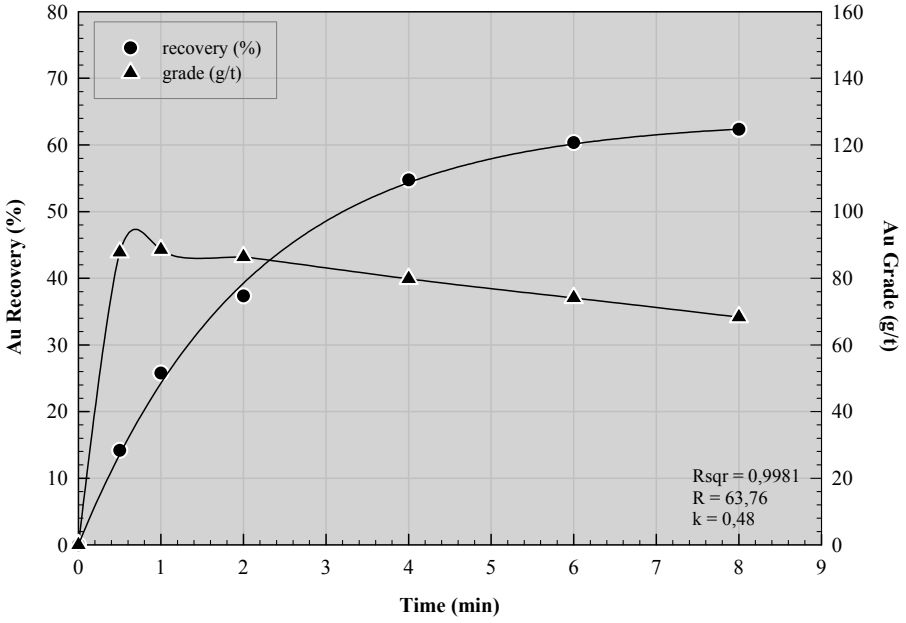

Figure 5.21 The effect of 150 g/t $CuSO_4$ addition on the time-recovery curve for gold (collector combination: KAX-Aerofloat 208 and Aero 412)

Gold grade and ultimate recovery values were higher when Na_2S added to the pulp. Na_2S addition also enhanced the flotation rate to 0.82. By comparing Figure 4 with Figure 5.22, it can be seen that concentrate gold grade increased from 68.33 g/t to 73.58 g/t and ultimate gold recovery was also increased from 63.76 % to 65.04 % (Appendix 2 – Table 4).

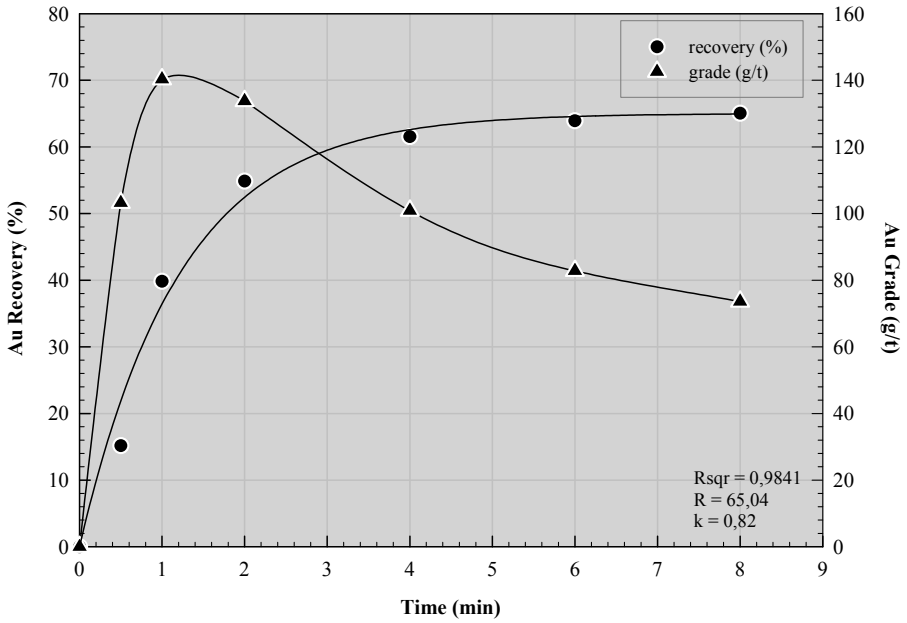

Figure 5.22 The effect of 70 g/t Na_2S Na_2S addition on the time-recovery curve for gold (collector combination: KAX-Aerofloat 208 and Aero 412)

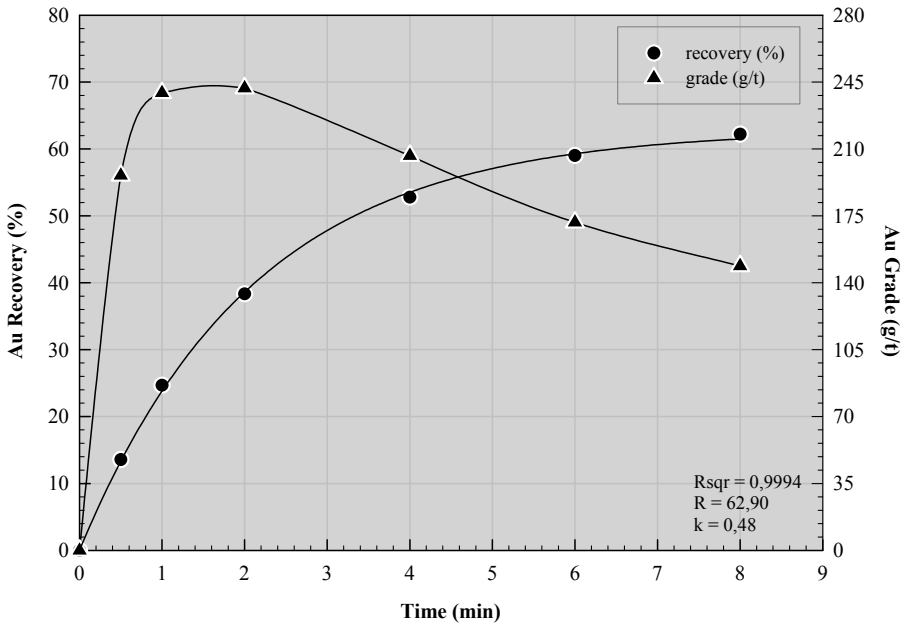

Figure 5.23 The effect of 150 g/t $CuSO_4$ addition on the time-recovery curve for gold (collector combination: KAX-Aerofloat 208 and Aerofloat 242)

Figure 5.23 illustrates that addition of Aerofloat 242 to the combination increased the grade and the recovery of the gold concentrate. A concentrate assaying 148.81 g/t Au was recovered with 62.90 % gold recovery (Appendix 2 – Table 5).

Figure 5.24 shows the effect of Na_2S addition to the KAX, Aerofloat 208 and Aerofloat 242 reagent combination. Na_2S addition to the pulp increased the ultimate recovery to 67.89 %. Flotation rate increased from 0.48 to 0.65. A concentrate with 92.22 g/t Au grade was obtained (Appendix 2 – Table 6).

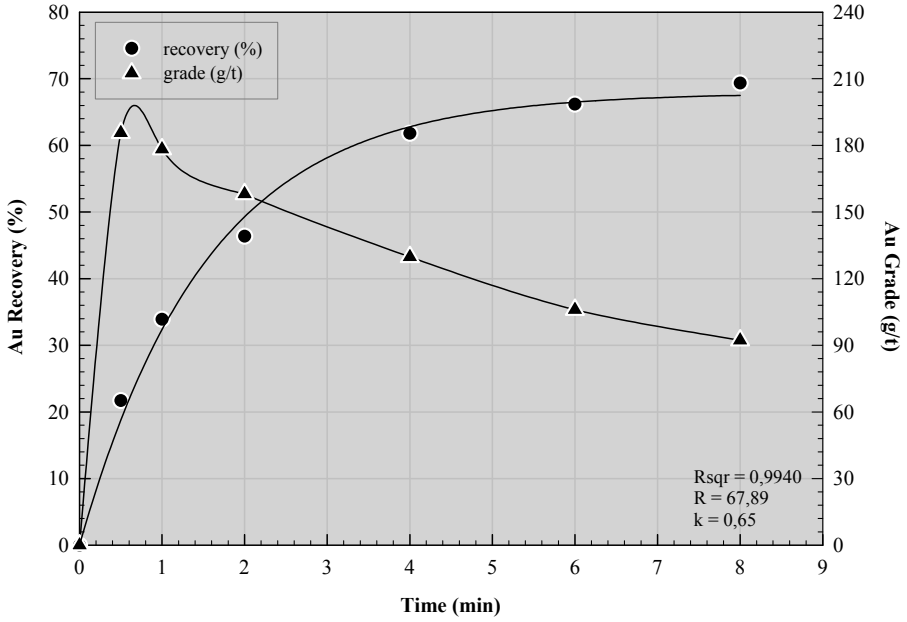

Figure 5.24 The effect of 70 g/t Na_2S Na_2S addition on the time-recovery curve for gold (collector combination: KAX-Aerofloat 208 and Aerofloat 242)

A high-grade concentrate obtained at 0.5 minutes flotation time and a tendency to decrease with flotation time could be observed in concentrate grades (Figure 5.25). A concentrate containing 89.02 g/t gold was produced at the end of the flotation time. Ultimate recovery was calculated as 61.19% and flotation rate was achieved as 0.71 (Appendix 2 – Table 7).

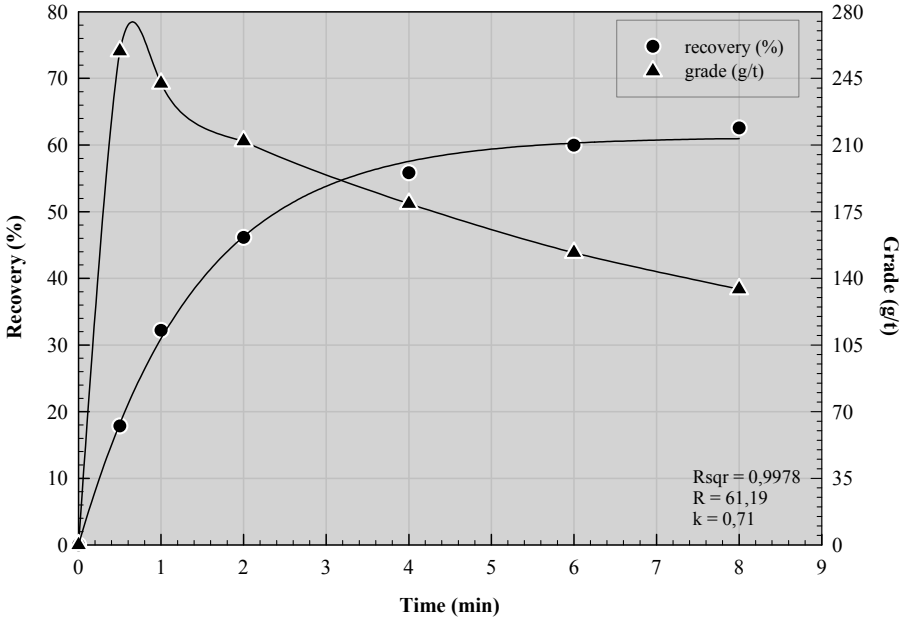

Figure 5.25 The effect of 150 g/t $CuSO_4$ addition on the time-recovery curve for gold (collector combination: KAX-Aerofloat 208 and Aerophine 3418A)

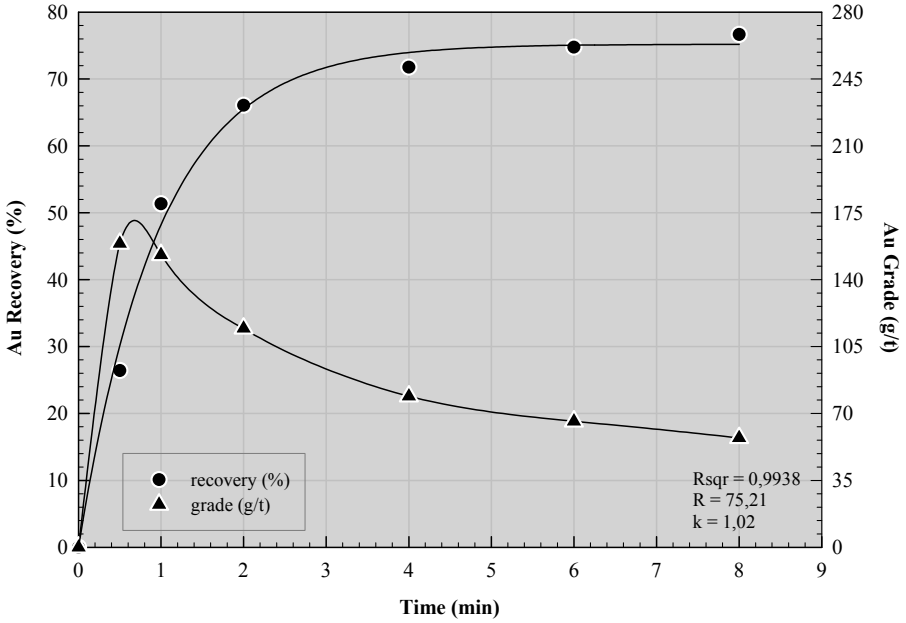

Figure 5.26 The effect of 70 g/t Na_2S Na_2S addition on the time-recovery curve for gold (collector combination: KAX-Aerofloat 208 and Aerophine 3418A)

Comparison of Figures 5.25 and 5.26 shows that concentrate gold grade decreased to 57.20 and recovery value increased from 61.19 % to 75.21 %. The flotation rate value was increased to 1.02. Na_2S addition to the pulp enhanced the gold recovery while speeding up the gold particles attachment to the gas bubbles.

Table 5.4 and Figure 5.27 and Figure 5.28 represent the overall results of the experiments done to reveal the effect of reagent synergism on the flotation of Efem Cukuru gold ore.

Table 5.4 The overall presentation of the effect of reagent synergism on the recovery of gold from Efem Çukuru ore

Reagents	**Concentrate (8 min. flot. Time)**			
	Weight (%)	**Grade (g/t)**	**R (%)**	**k**
KAX-Aerofloat 208-Aero 6697	8.89	40.77	54.68	0.55
KAX-Aerofloat 208-Aero 6697-Na_2S	9.35	46.62	65.95	0.55
KAX-Aerofloat 208-Aero 412	6.47	68.33	63.76	0.48
KAX-Aerofloat 208-Aero 412-Na_2S	7.34	62.34	65.04	0.82
KAX-Aerofloat 208-Aerofloat 242	4.71	148.81	62.90	0.48
KAX-Aerofloat 208-Aerofloat 242-Na_2S	6.19	92.22	67.89	0.65
KAX-Aerofloat 208-Aerophine 3418A	7.88	134.29	61.19	0.71
KAX-Aerofloat 208-Aerophine 3418A -Na_2S	8.26	57.20	75.21	1.02

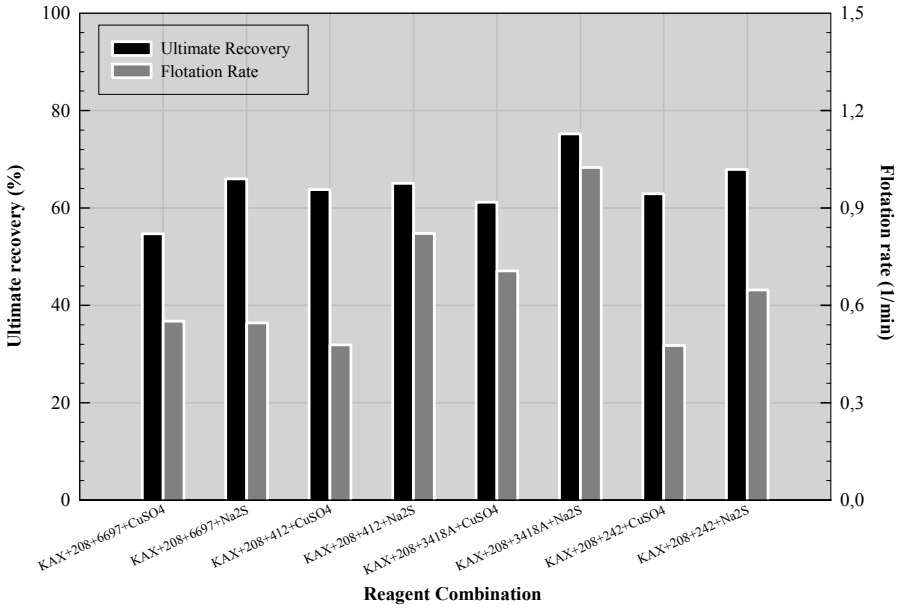

Figure 5.27 The overall presentation of the effect of reagent synergism on the recovery of gold for Izmir-Efem Cukuru ore

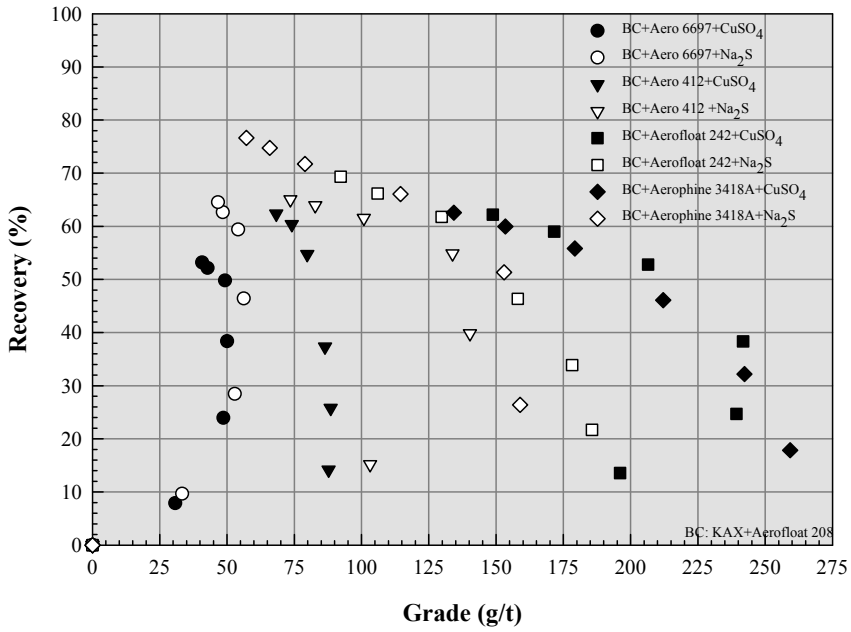

Figure 5.28 Grade vs. recovery curves of different reagent combinations for Izmir-Efem Cukuru ore

According to Table 5.4, Figure 5.27 and Figure 5.28 overall results can be summarized as follow;

- KAX - Aerofloat 208 - Aerofloat 242 and $CuSO_4$ combination is the most selective combination (148.81g/t Au grade) for the recovery of gold from Efem Cukuru gold ore.
- KAX - Aerofloat 208 - Aerophine 3418A - Na_2S combination is the most effective combination for the recovery of gold from Efem Cukuru gold ore.
- KAX - Aerofloat 208 - Aerophine 3418A - Na_2S combination has the highest flotation rate for the recovery of gold from Efem Cukuru gold ore.

After the flotation experiments were completed, some initial coal-oil assisted gold flotation studies have been run on the Efem Cukuru gold ore. Coal particles were agglomerated using oil prior to mixing with the gold-bearing ore pulp. An IKA-Eurostar (Power control-visc) stirrer was used for emulsification and a Denver sub-A flotation machine with speed control was used for the agglomerate separation.

Pre-agglomeration of coal particles and conditioning of flotation pulp were started simultaneously. Thus, oil emulsion was added to the emulsifying cell, agitated at 2000 rpm for 5 minutes. Coal particles (10 % by weight) were added to the oil-water

emulsion and agitated at 2000 rpm for another 5 minutes. Flotation reagents were added to the pulp that was agitated at 2000 rpm for 10 minutes. Pre-agglomerated coal particles-oil-water suspension was added to the gold-bearing pulp and agitated at 2000 rpm for 10 minutes. Stirring speed was decreased to 850 rpm and pulp was conditioned for 10 minutes for agglomerate formation. The produced coal-oil-gold agglomerates were separated from the pulp by flotation at a natural pH level.

Table 5.5 Flotation reagents

Collector	**Consumption (g/ton)**
KAX	50
Aerofloat 242	50
Aerofloat 208	50
Na_2SiO_3	1000

Table 5.6 Agglomeration reagents

Collector	**Consumption (g/ton)**
Cotton oil	400
Diesel oil	1200
Amyl Alcohol	200
Dowfroth 1012	200

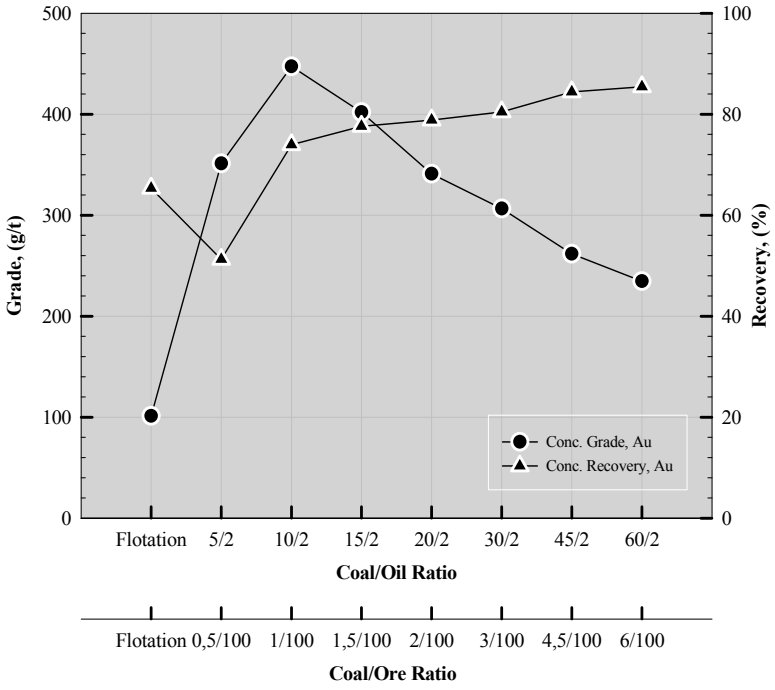

Figure 5.29 Gold concentrate grade and recoveries vs. different coal/oil ratios for Efem Çukuru gold ore sample

The best results were obtained by using a 30/1 coal-oil ratio (Figure 5.29). Coal-oil assisted gold flotation produced concentrates with higher recovery values for higher coal/oil ratios. Increasing the coal/oil ratio decreased the grade of the concentrates.

The process seems to be very successful even with its initial tests for the recovery of gold from Izmir-Efem Cukuru ore. Bergama-Ovacik gold ore deposit with its fine gold particle size would be a bigger challenge for the coal-oil-assisted gold flotation process. So, it has been decided to run the further coal-oil assisted gold flotation studies on the Bergama-Ovacik gold ore sample.

5.2.2 Flotation Studies of Bergama-Ovacik Gold Ore

5.2.2.1 Material and Method

- Flotation studies have been run at natural pH of the gold ore (8.1). The Outokumpu laboratory scale flotation machine has been used for the flotation tests.
- 2.5 l/min. aeration rate has been applied
- Batch flotation tests were carried out in a round shaped flotation cell (2 l)
- 400 g of Bergama-Ovacik gold ore sample ground below 0.060 mm and averaging 9 g/ton gold grade has been subjected to the flotation experiments
- KAX (50 g/t) and Aerofloat 208 (50 g/t) and one more different collector (50 g/t) were used in each experiment
- Na_2S (70 g/t) was used
- Copper sulfate ($CuSO_4$) was used as an activator
- Na_2SiO_3 (400 g/t) was used as a depressant
- Aerofroth 70 (25 g/t) was used as a frother

- 25 minutes of total conditioning time and 8 minutes of total flotation time have been applied for kinetic flotation tests.
- The results have been evaluated by classical first-order flotation model (Klimpel, 1999)

5.2.2.2 The Effect of Reagent Synergism

Flotation studies have been run to investigate the effect of reagent synergism on the gold recovery. The effect of using different reagent combinations has been given in the following figures.

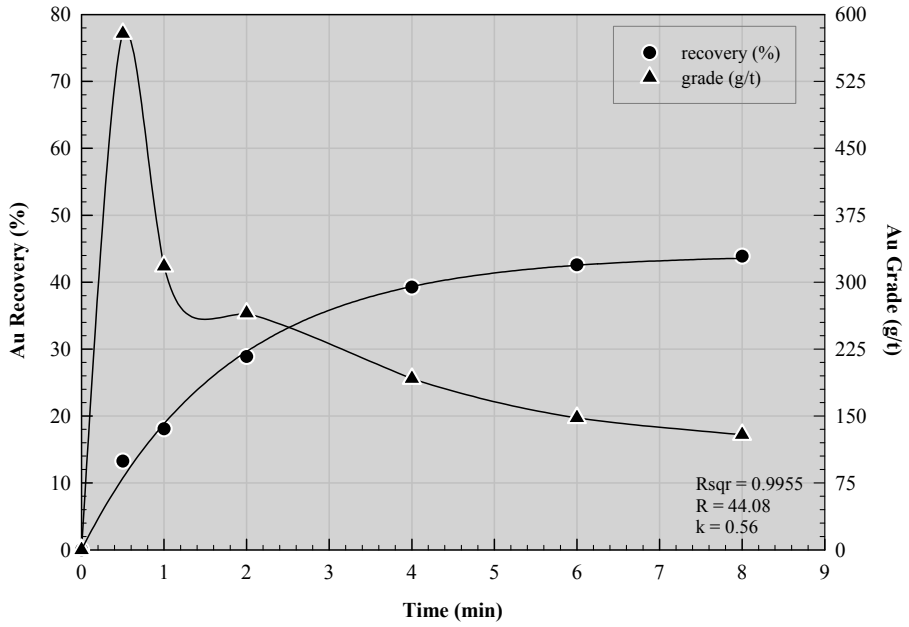

Figure 5.30 The effect of 150 g/t $CuSO_4$ addition on the time-recovery and time-grade curves for gold (collector combination: KAX-Aerofloat 208 and Aero 6697)

It can be seen from Figure 5.30 that concentrate gold grade was 578.63 g/t for 0.5 min. and tends to decrease after 0.5 minute. The concentrate gold recovery was 13.23 % for 0.5 min. and tends to increase by the flotation time; it reached to 43.86 % at 8 min. of flotation time. The gold recovery was poor and the grade of the concentrate was moderate. Ultimate recovery and flotation rate were 44.08 % and 0.56 respectively (Appendix 3 – Table 1).

Figure 5.31 shows the effect of using Na_2S instead using $CuSO_4$. Concentrate gold grade was lower but the recovery was improved (Appendix 3 – Table 2).

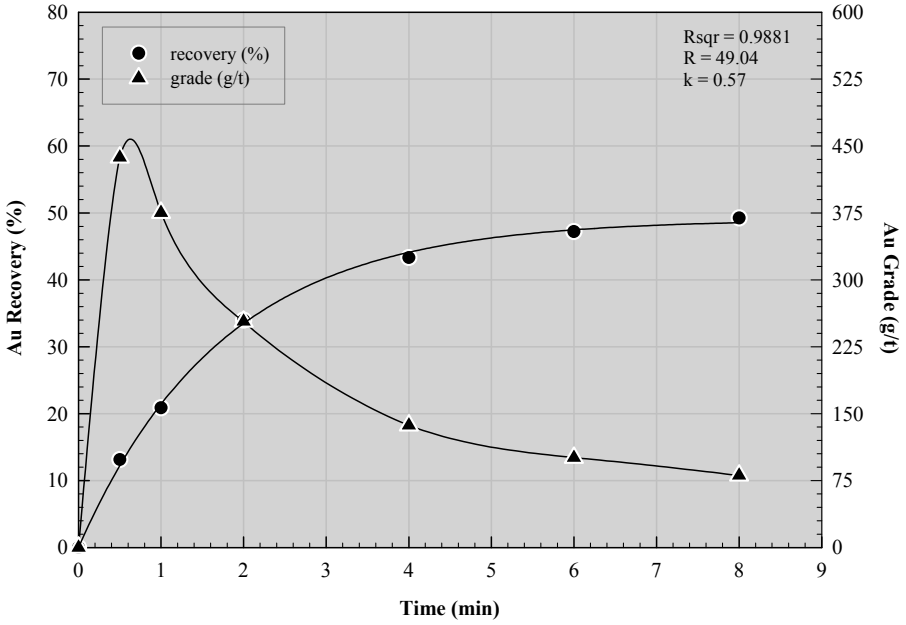

Figure 5.31 The effect of 70 g/t Na_2S addition on the time-recovery and time-grade curves for gold (collector combination: KAX-Aerofloat 208 and Aero 6697)

By comparing Figure 5.30 with Figure 5.31, it can be noticed that concentrate gold grade decreased to 80.64 g/t from 129.02 g/t. The gold recovery was enhanced to 49.25 % from 43.86 %. The flotation rate value was about the same (0.57) and the ultimate recovery was 49.04 %.

Figure 5.32 illustrates the effect of the combination of KAX, Aerofloat 208 and Aero 412 - $CuSO_4$ reagent combination. The figure shows that concentrate gold grade was very high (569.80 g/t) for 0.5 min. and rapidly decreased to 128.29 g/t at the end of the flotation time. The ultimate recovery was 54.36 % for 8 min. and flotation rate was low (0.37) (Appendix 3 – Table 3).

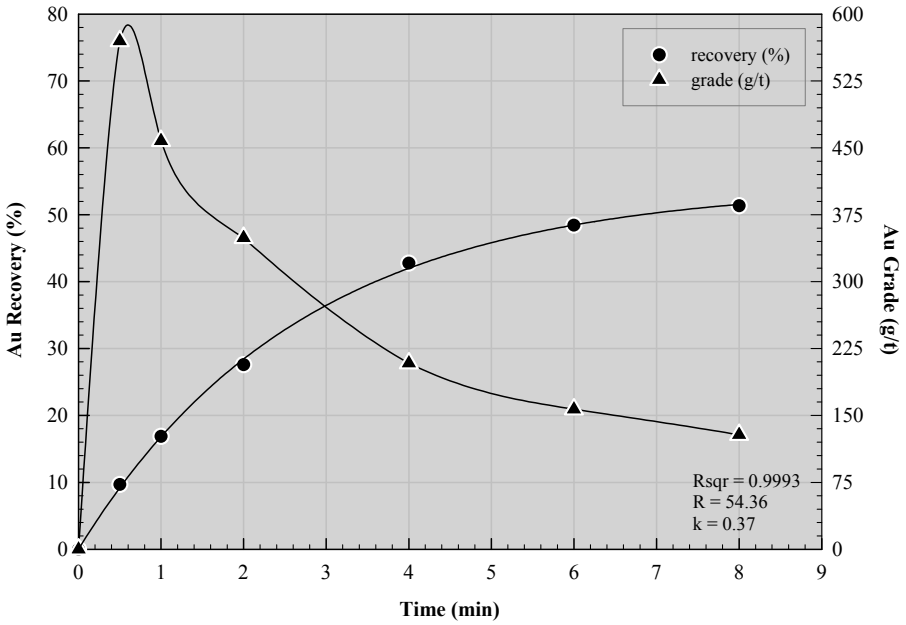

Figure 5.32 The effect of 150 g/t $CuSO_4$ addition on the time-recovery and time-grade curves for gold (collector combination: KAX-Aerofloat 208 and Aero 412)

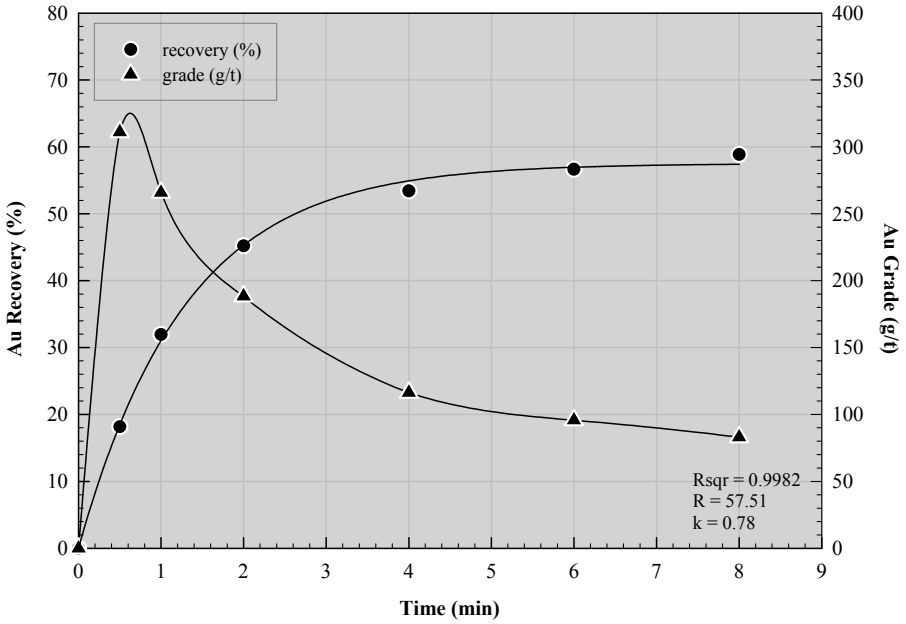

Figure 5.33 The effect of 70 g/t Na_2S addition on the time-recovery and time-grade curves for gold (collector combination: KAX-Aerofloat 208 and Aero 412)

Figure 5.33 shows the effect of Na_2S addition to the pulp. A concentrate assaying 83.03 g/t Au was produced with 58.32 % gold recovery at the end of the 8 min. of flotation time (Appendix 3 – Table 4).

The graph 5.33 implies that Na_2S addition improved the ultimate gold recovery from 54.36 % to 57.51 % while decreasing the gold grade of the concentrate to 83.01 g/t. The flotation rate value was also enhanced to 0.78.

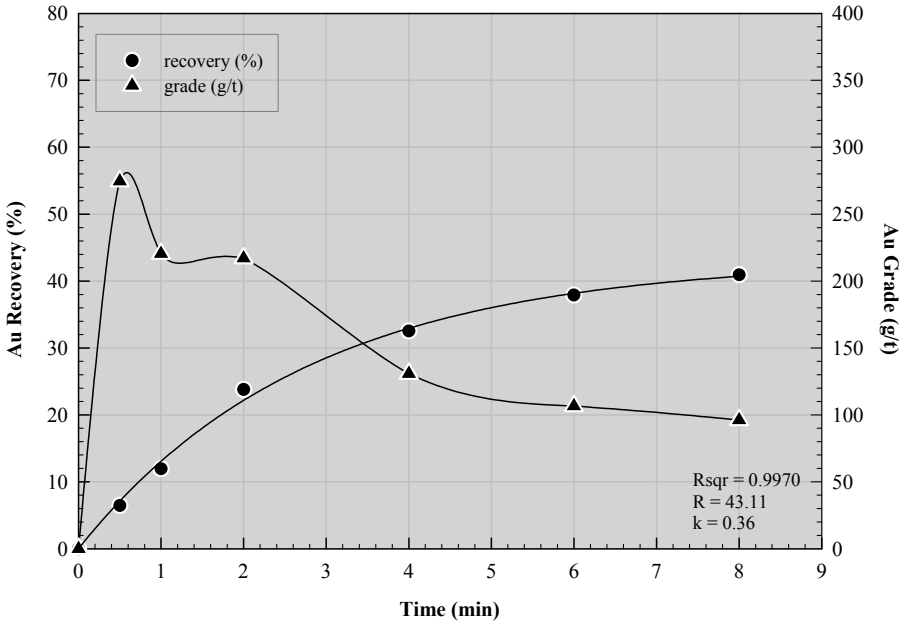

Figure 5.34 The effect of 150 g/t $CuSO_4$ addition on the time-recovery and time-grade curves for gold (collector combination: KAX-Aerofloat 208 and Aerofloat 242)

Figure 5.34 demonstrates the effect of Aerofloat 242 and $CuSO_4$ addition to the regular collector combination (KAX-Aerofloat 208). The use of Aerofloat 242 produced a concentrate assaying 96.10 g/t Au with 43.11 % ultimate recovery (Appendix 3 – Table 5).

Figure 5.35 emphasizes that Na_2S addition improved the flotation rate of the combination (0.63). Ultimate recovery value was also increased from 43.11 % to 48.06 %. The concentrate gold grade was 365.52 g/t for 0.5 min. and rapidly decreases by the flotation time. It was 91.87 g/t for 8 min. flotation time (Appendix 3 – Table 6).

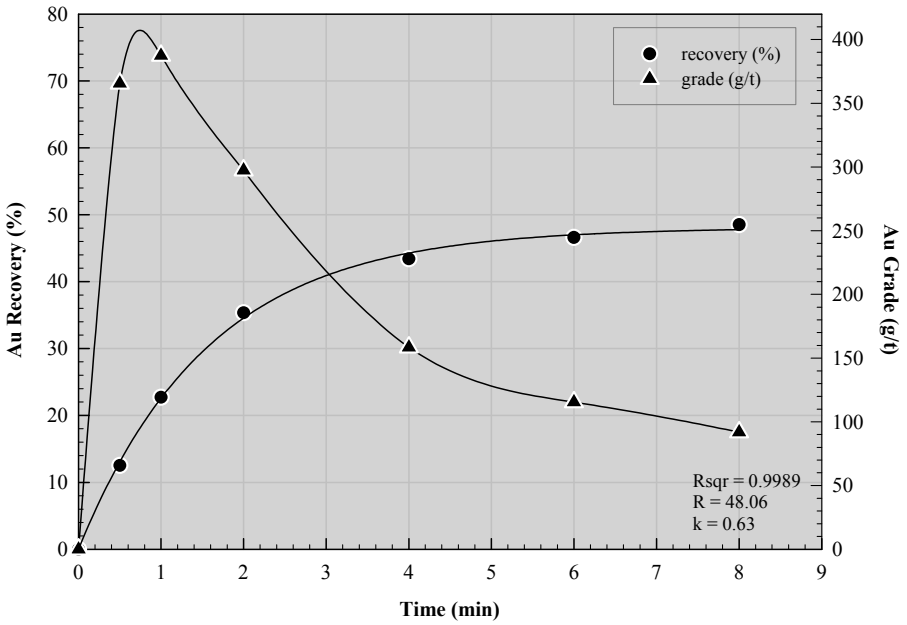

Figure 5.35 The effect of 70 g/t Na_2S addition on the time-recovery and time-grade curves for gold (collector combination: KAX-Aerofloat 208 and Aerofloat 242)

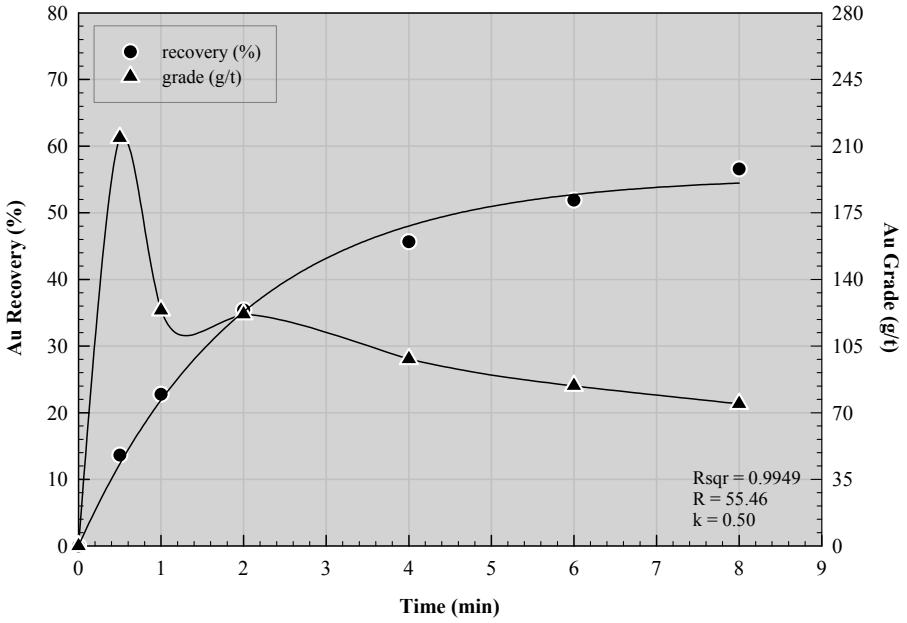

Figure 5.36 The effect of 150 g/t $CuSO_4$ addition on the time-recovery and time-grade curves for gold (collector combination: KAX-Aerofloat 208 and Aerophine 3418A)

The graph 5.36 represents the effect of Aerophine 3418A addition to the KAX-Aerofloat 208 - $CuSO_4$ combination. The new combination produced a concentrate containing 74.42 g/t Au with 55.46 % ultimate recovery. Flotation rate was 0.50 (Appendix 3 – Table 7).

The effect of Na_2S addition to this combination was given in the Figure 5.37. By comparing Figure 5.36 with Figure 5.37, we can see that concentrate gold grade and ultimate recovery values increased to 110.48 g/t and 60.13 % respectively. The flotation rate value was also increased from 0.50 to 0.64 (Appendix 3 – Table 8).

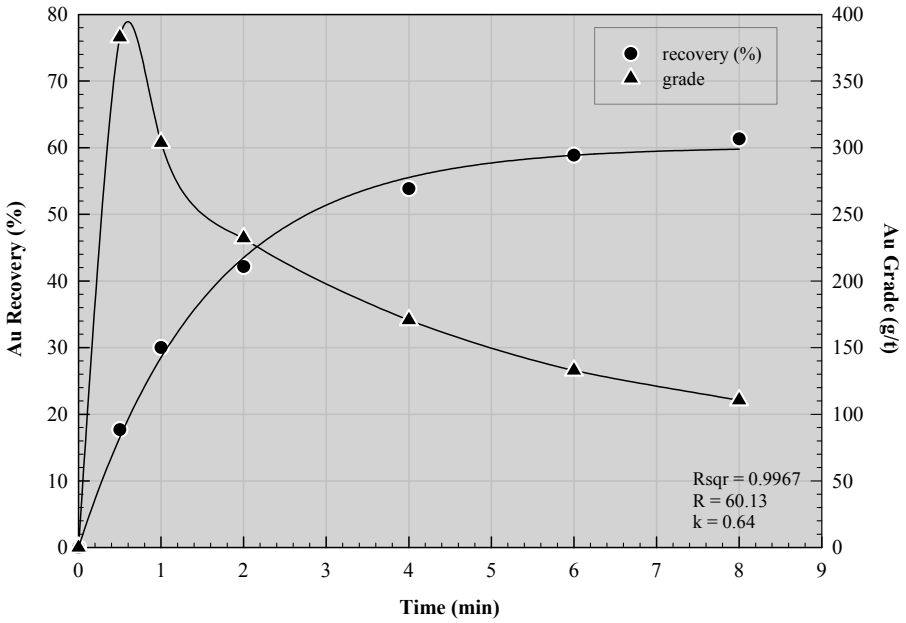

Figure 5.37 The effect of 70 g/t Na_2S addition on the time-recovery and time-grade curves for gold (collector combination: KAX-Aerofloat 208 and Aerophine 3418A)

Table 5.7, Figure 5.38 and Figure 5.39 represent the overall results of the experiments done to reveal the effect of reagent synergism on the flotation of Bergama-Ovacik gold ore.

Table 5.7 Overall evaluation of the experiments done to reveal the effect of reagent synergism

Reagents	**Concentrate (8 min. flot. Time)**			
	Weight (%)	**Grade (g/t)**	**R (%)**	**k**
KAX-Aerofloat 208-Aero 6697-$CuSO_4$	4.37	129.02	44.08	0.56
KAX-Aerofloat 208-Aero 6697-Na_2S	6.19	80.64	49.04	0.57
KAX-Aerofloat 208-Aero 412- $CuSO_4$	4.16	128.29	54.36	0.37
KAX-Aerofloat 208-Aero 412-Na_2S	7.08	83.03	57.51	0.78
KAX-Aerofloat 208-Aerofloat 242- $CuSO_4$	4.29	96.10	43.11	0.36
KAX-Aerofloat 208-Aerofloat 242-Na_2S	5.33	91.86	48.06	0.63
KAX-Aerofloat 208-Aerophine 3418A- $CuSO_4$	6.07	74.62	55.46	0.50
KAX-Aerofloat 208-Aerophine 3418A -Na_2S	5.82	110.48	60.13	0.64

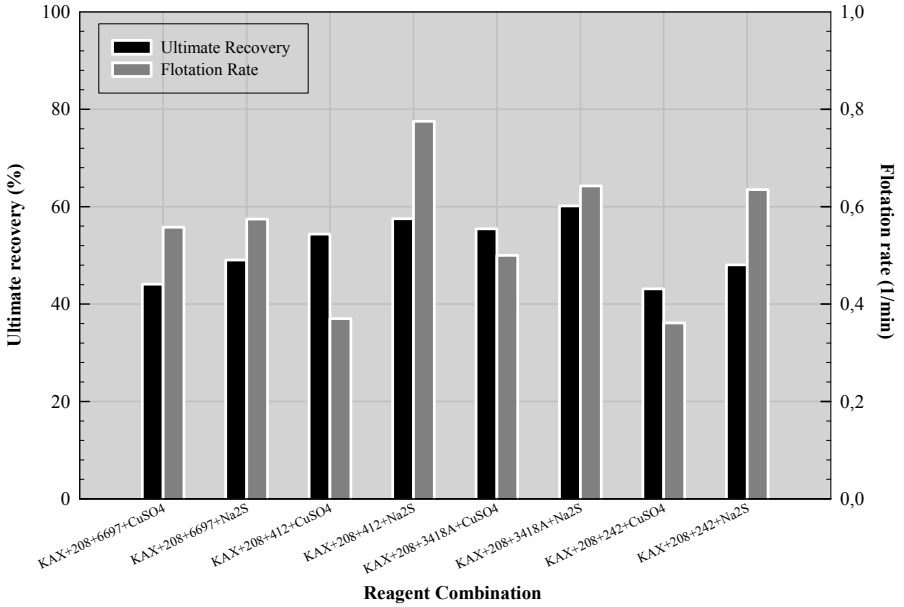

Figure 5.38 The overall presentation of the effect of reagent synergism on the recovery of gold for Bergama-Ovacik ore

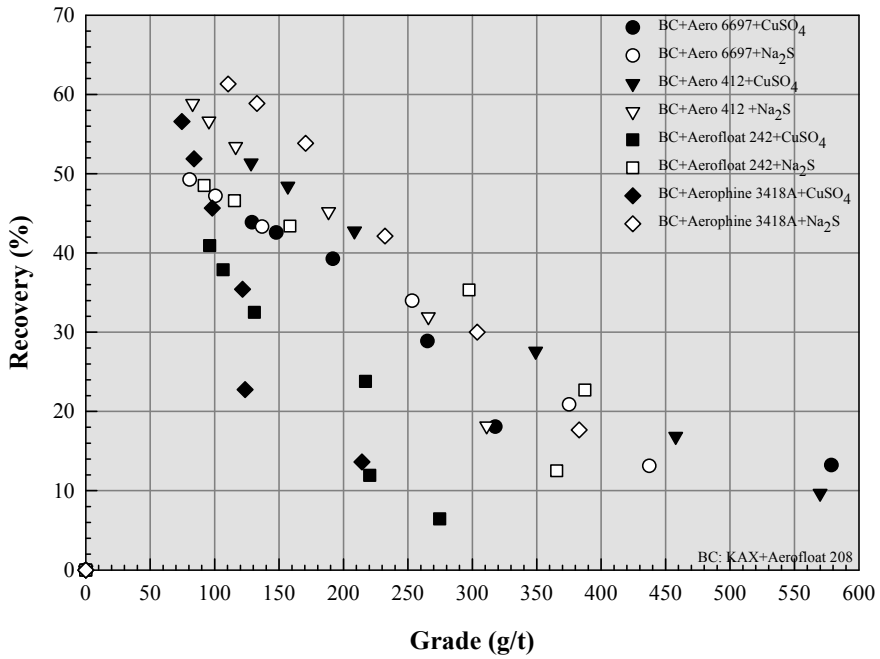

Figure 5.39 Grade vs. recovery curves of different reagent combinations for Bergama-Ovacik ore

According to Table 5.7, Figure 5.38 and Figure 5.39 overall results can be summarized as follow;

- KAX-Aerofloat 208-Aero 6697-$CuSO_4$ reagent combination was the most selective combination. It produced a concentrate assaying 129.02 Au with 44.08 % ultimate recovery, flotation rate was 0.56.

- KAX-Aerofloat 208-Aero 412-Na_2S reagent combination has the highest flotation rate for the recovery of gold from Bergama-Ovacik gold ore. A concentrate assaying 83.03 g/t Au with 57.71 ultimate recovery was obtained. Flotation rate was 0.78.

- KAX-Aerofloat 208-Aerophine 3418A -Na_2S is the most effective combination for the recovery of gold from Bergama-Ovacik gold ore. A concentrate assaying 110.48 g/t Au with 60.13 % ultimate recovery was obtained. Flotation rate was 0.64.

5.2.2.3 The Effect of Na_2S and $CuSO_4$ Dosages on the Flotation of Gold

Sodium sulphide (Na_2S) used in the correct concentration, it can function as an activator; however, there is an optimum addition. Otherwise, sodium sulphide can possibly act by competitive adsorption, thus preventing the adsorption of xanthate or other collector on the gold surface.

Copper sulphate ($CuSO_4$) oxidizes xanthate to dixanthogen. It has been noted in past research that dixanthogen is a more powerful collector than metal xanthates but is less selective. If the dixanthogen formation is in significant amounts, it can affect the frothing characteristics of the pulp. Increasing the amount of dixanthogen in the pulp may result some increase for particles entrained into the concentrate.

Flotation studies have been done to investigate the effect of Na_2S dosage on the gold recovery. KAX, Aerofloat 208 and Aerophine 3418A reagents were used in conjunction; the effect of using Na_2S and $CuSO_4$ at different dosages has been given in the following figures.

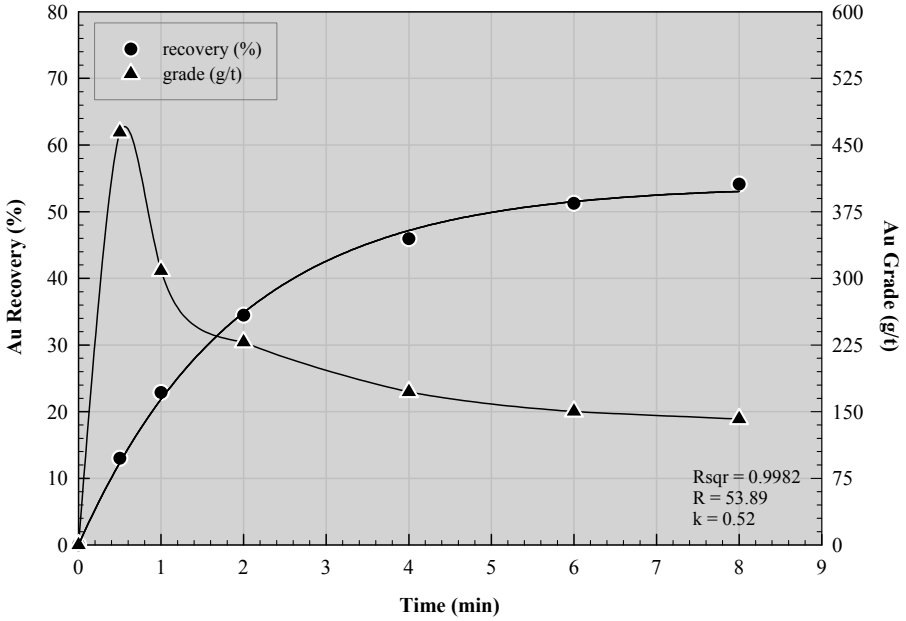

Figure 5.40 The time-recovery and time-grade curves for gold with 20 g/t Na_2S

The graph 5.40 illustrates the effect of 20 g/t Na_2S on the gold concentrate grade and recoveries. The concentrate gold grade was 464.29 g/t at beginning of the flotation (0.5 min.) and decreased to 141.65 g/t at the end of the flotation (8 min.). The ultimate recovery of the flotation with 20 g/t Na_2S was 53.89 %. Flotation rate value has been determined as 0.52 (Appendix 3 – Table 9).

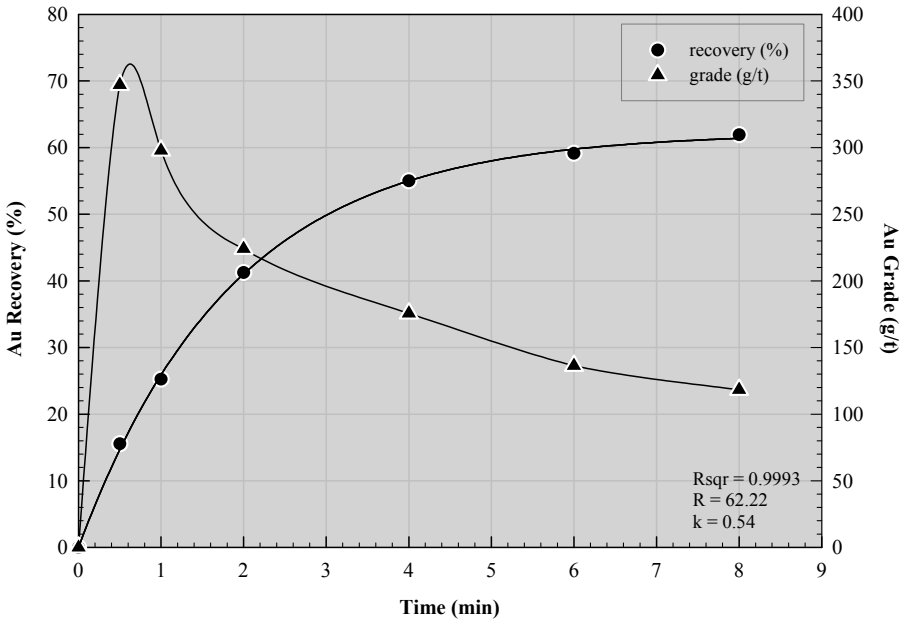

Figure 5.41 The time-recovery and time-grade curves for gold with 40 g/t Na_2S

Figure 5.41 show that increasing the sodium sulphide concentration to 40 g/t increased the ultimate recovery of the concentrate to 62.22 %. Concentrate gold grade decreased to 347.10 g/t for 0.5 min. and 118.29 g/t for 8 minutes of flotation time. Flotation rate value was increased to 0.54 (Appendix 3 – Table 10).

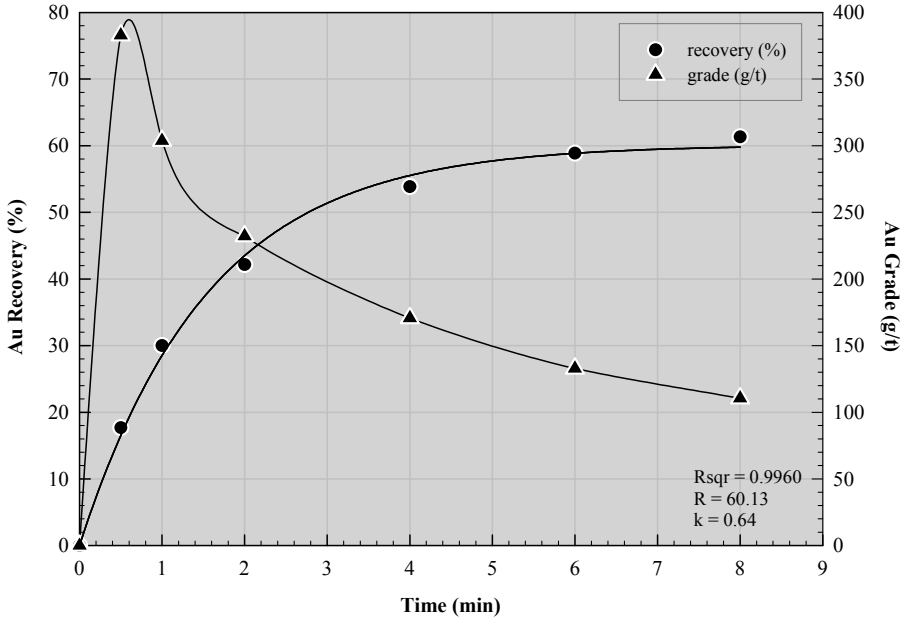

Figure 5.42 The time-recovery and time-grade curves for gold with 70 g/t Na_2S

It can be seen from the graphic 5.42 that using 70 g/t sodium sulphide resulted an increase in the flotation rate; it was increased to 0.64 by increasing the sodium sulphide dosage. The concentrate gold grade for the beginning and the end of the flotation time (0.5 and 8 min.) were 382.86 g/t and 110.48 g/t respectively. Ultimate recovery was 60.13 % (Appendix 3 – Table 11).

Figure 5.43 presents the concentrate gold grade and recoveries with 85 g/t sodium sulphide. A concentrate assaying 403.85 g/t for 0.5 min. and 98.38 g/t for 8 min. flotation time was produced with 61.79 % ultimate recovery. Flotation rate value was 0.66 (Appendix 3 – Table 12).

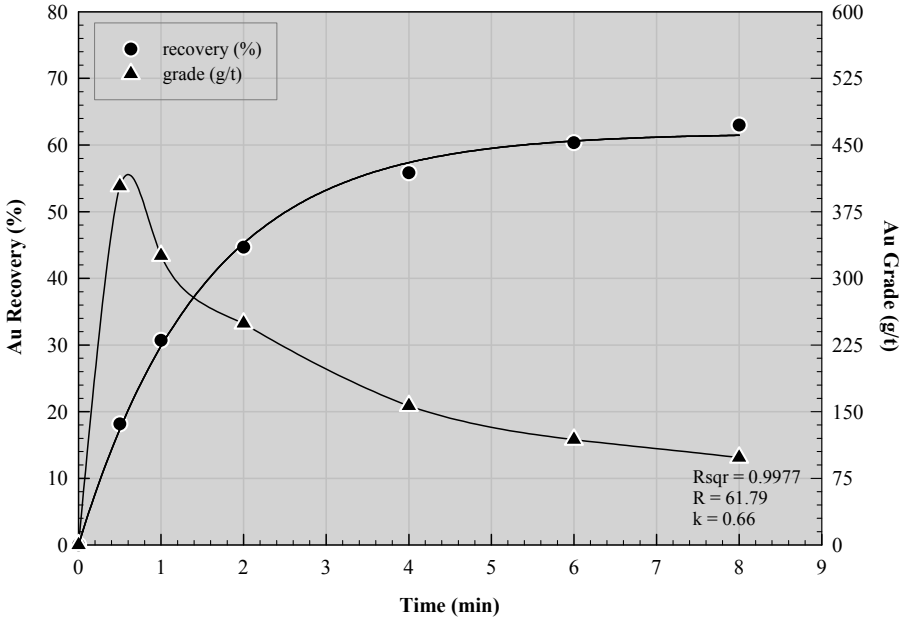

Figure 5.43 The time-recovery and time-grade curves for gold with 85 g/t Na_2S

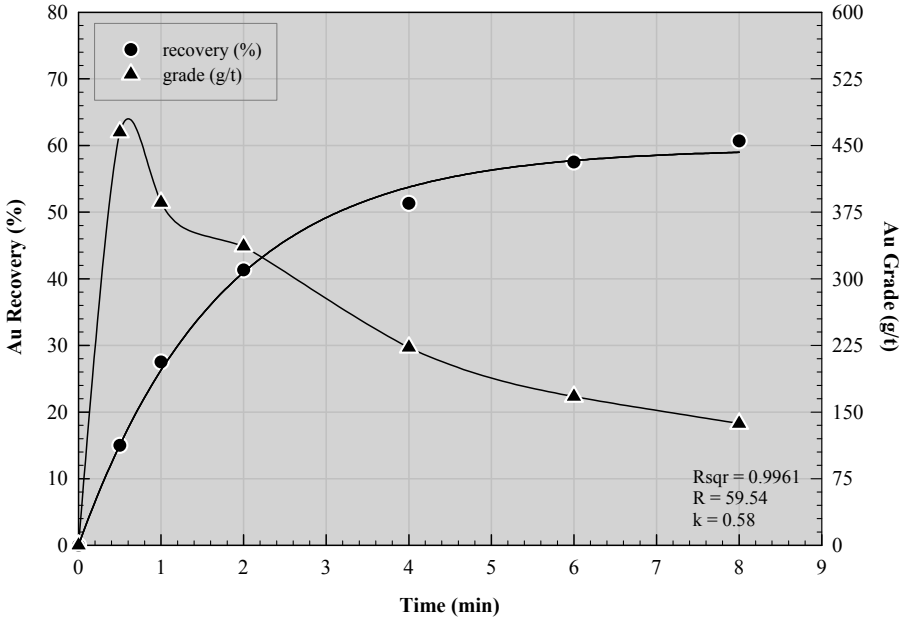

Figure 5.44 The time-recovery and time-grade curves for gold with 100 g/t Na_2S

The graph 5.44 demonstrates the results obtained by using 100 g/t Na_2S. Ultimate recovery value has been determined as 59.54%. The concentrate gold grade was 464.96 g/t for 0.5 min. and it decreased to 136.99 at the end of the flotation time (8 min.). The flotation rate value was found to be as 0.58 (Appendix 3 – Table 13).

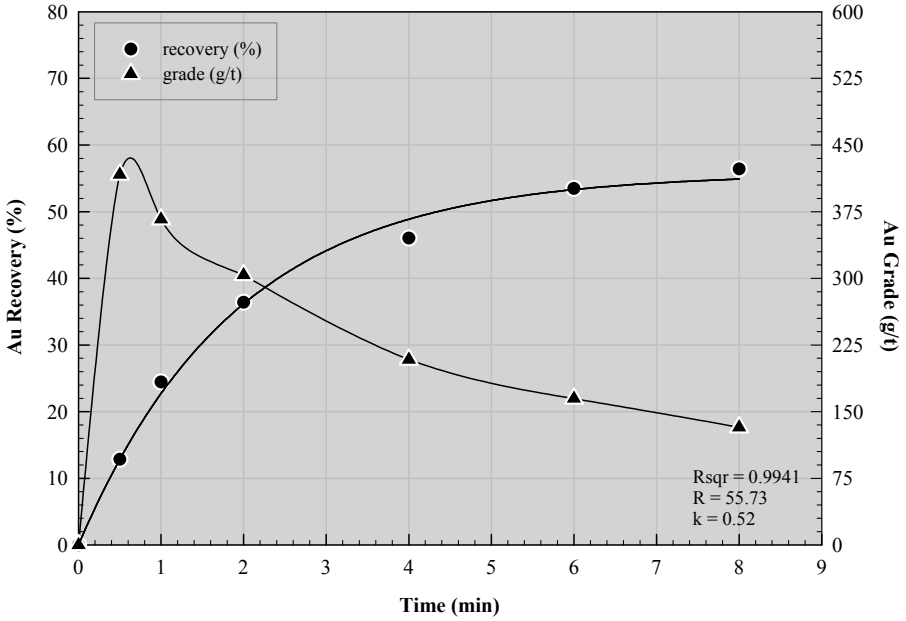

Figure 5.45 The time-recovery and time-grade curves for gold with 150 g/t Na_2S

Increasing the sodium sulphide dosage to 150 g/t caused to decrease ultimate recovery value (Figure 5.45). The recovery has been determined as 55.73 % while concentrate gold grade was 416.67 g/t for 0.5 min. and 132.32 g/t for 8 min. flotation time. Flotation rate value was determined as 0.52 (Appendix 3 – Table 14).

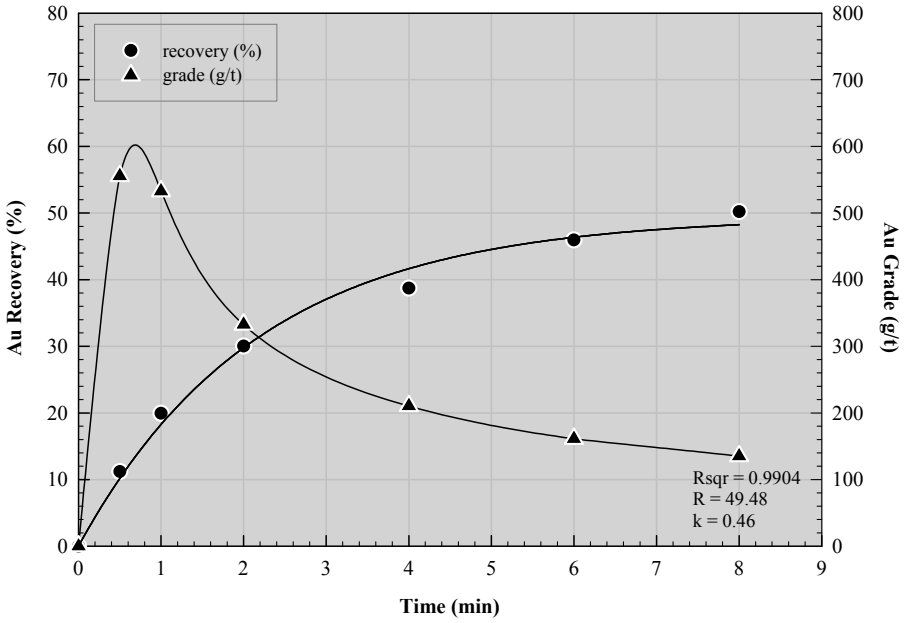

Figure 5.46 The time-recovery and time-grade curves for gold with 350 g/t Na_2S

Figure 5.46 shows the effect of using 350 g/t Na2S. A concentrate assaying 135.13 g/t Au has been obtained with 49.48 % ultimate recovery. The flotation rate value decreased to 0.46 (Appendix 3 – Table 15).

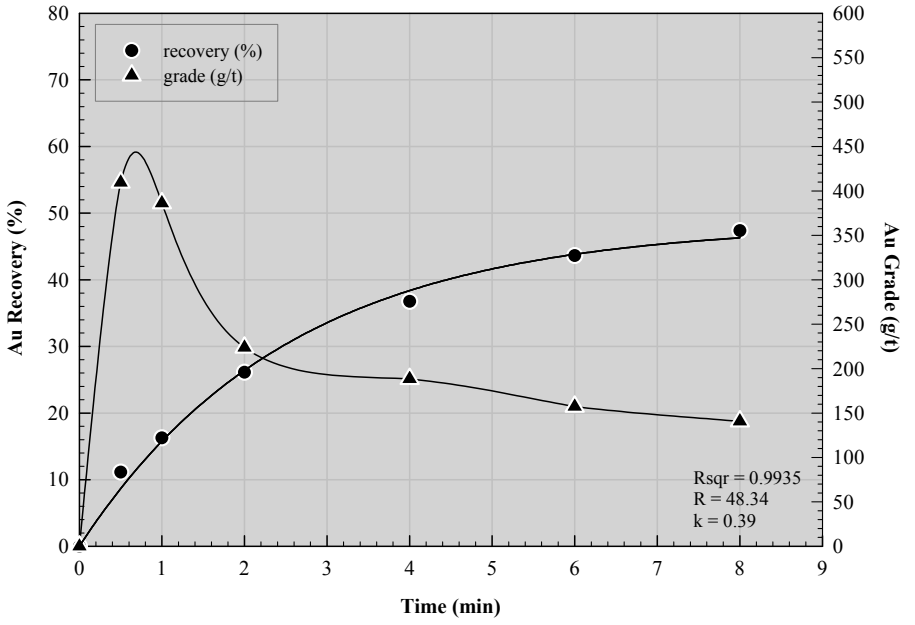

Figure 5.47 The time-recovery and time-grade curves for gold with 750 g/t Na_2S

Figure 5.47 represents the grade and recovery curves obtained by using 750 g/t sodium sulphide. It can be noted that increasing the Na_2S concentration to 750 g/t decreased the flotation rate to 0.39. Ultimate recovery was slightly lowered (48.34 %). Concentrate gold grade was 409.45 g/t for 0.5 min. and 140.69 g/t for 8 min. of flotation time (Appendix 3 – Table 16).

Figure 5.48 show the effect of $CuSO_4$ on the concentrate gold grade and recovery. A concentrate assaying 88.01 g/t Au for 0.5 min. and 282.11 g/t for 8 min. were produced by the use of 75 g/t copper sulphate. The concentrate was obtained with 58.02 % ultimate recovery and 0.51 flotation rate (Appendix 3 – Table 17).

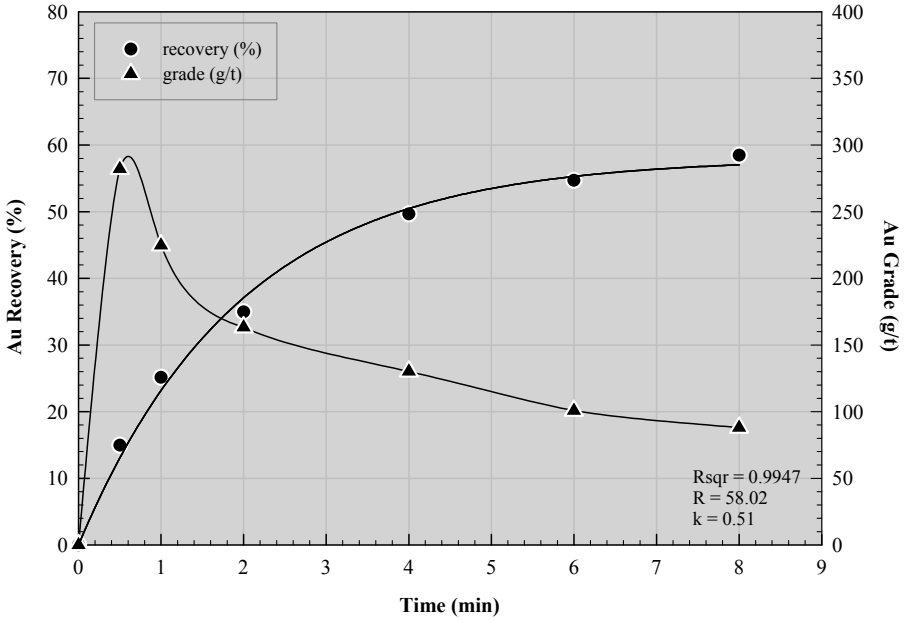

Figure 5.48 The time-recovery and time-grade curves for gold with 75 g/t $CuSO_4$

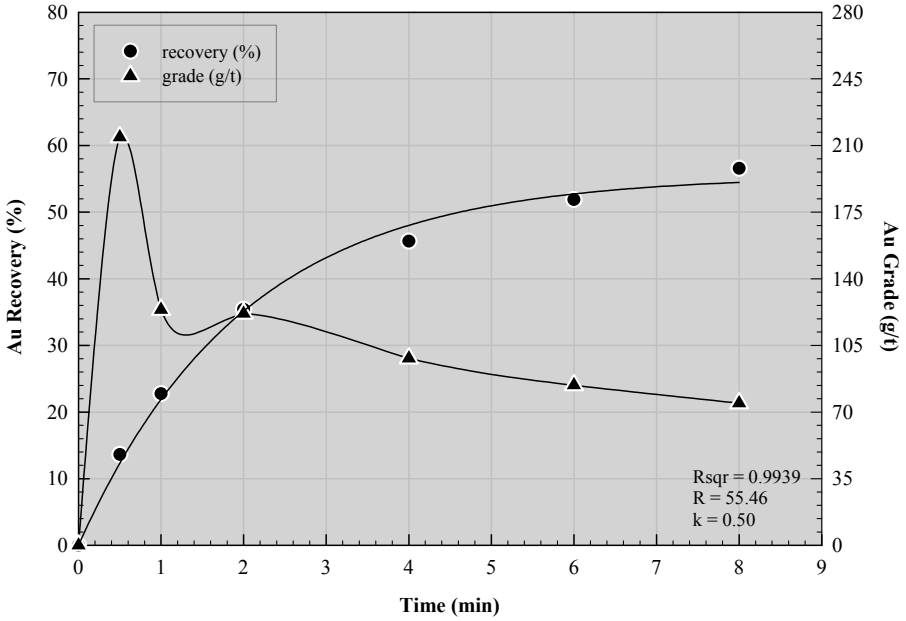

Figure 5.49 The time-recovery and time-grade curves for gold with 150 g/t $CuSO_4$

It can be seen from Figure 5.49 that increasing the $CuSO_4$ dosage to 150 g/t produced a concentrate containing 214.38 g/t for 0.5 min. and 74.62 g/t Au for 8 min. of flotation time with 55.46 % ultimate recovery. Flotation rate was determined as 0.5 (Appendix 3 – Table 18).

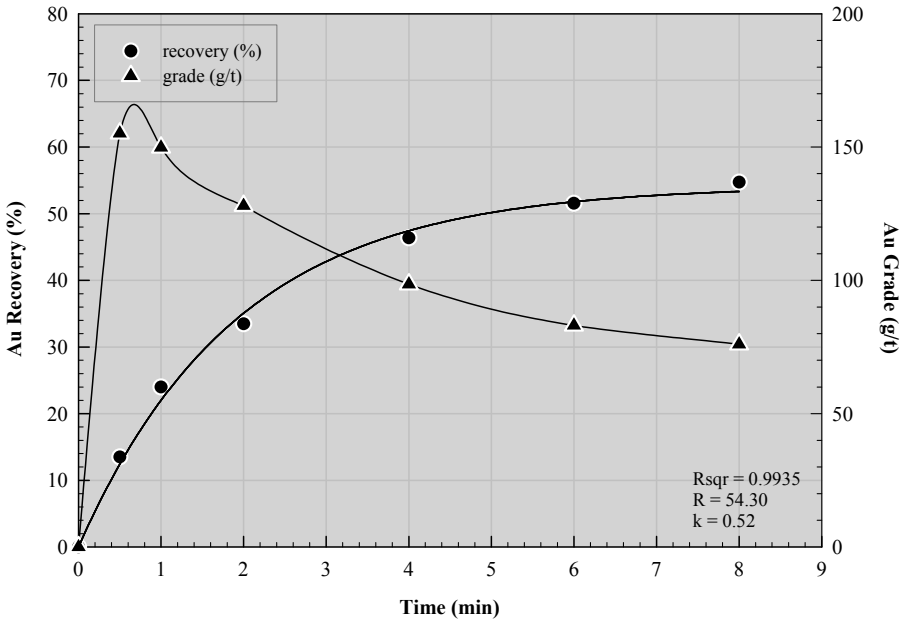

Figure 5.50 The time-recovery and time-grade curves for gold with 300 g/t $CuSO_4$

The concentrate gold grade was 155.11 g/t at beginning of the flotation (0.5 min.) and decreased to 76.03 g/t at the end of the flotation (8 min.). The ultimate recovery of the flotation with 300 g/t $CuSO_4$ was 54.17 %. Flotation rate value has been determined to be as 0.52 (Appendix 3 – Table 19).

Table 5.8 and Figure 5.51 emphasize the overall results of the experiments done to reveal the effect of using different amount of Na_2S and $CuSO_4$ addition on the flotation of Bergama-Ovacik gold ore.

Table 5.8 Overall evaluation of the experiments done to reveal the effect of using different amount of Na_2S and $CuSO_4$ on the flotation of Bergama-Ovacik gold ore (collector combination: KAX-Aerofloat 208 and Aerophine 3418A)

Reagents	**Concentrate (8 min. flot. Time)**			
	Weight (%)	**Grade (g/t)**	**R (%)**	**k**
20 g/t Na_2S	5.09	141.65	53.89	0.52
40 g/t Na_2S	5.63	118.29	62.22	0.54
70 g/t Na_2S	5.82	110.48	60.13	0.64
85 g/t Na_2S	5.13	98.38	61.79	0.66
100 g/t Na_2S	4.62	136.99	59.54	0.58
150 g/t Na_2S	4.21	132.32	55.73	0.52
350 g/t Na_2S	4.23	135.13	49.48	0.46
750 g/t Na_2S	3.33	140.69	48.34	0.39
75 $CuSO_4$	5.23	88.01	58.02	0.51
150 $CuSO_4$	6.07	74.62	55.46	0.50
300 $CuSO_4$	5.56	76.03	54.30	0.52

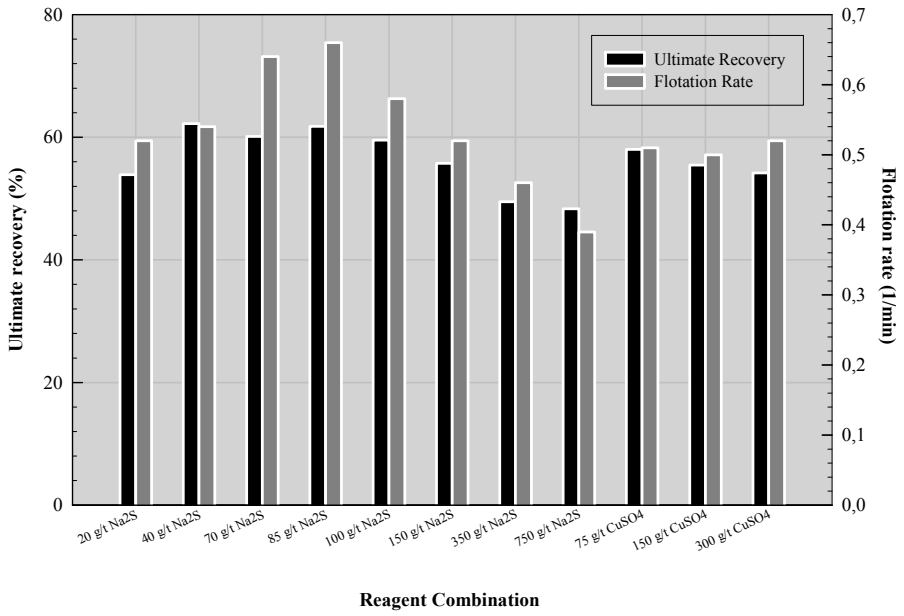

Figure 5.51 Overall evaluation of the experiments done to reveal the effect of using different amount of Na_2S and $CuSO_4$ on the flotation of Bergama-Ovacik gold ore (collector combination: KAX-Aerofloat 208 and Aerophine 3418A)

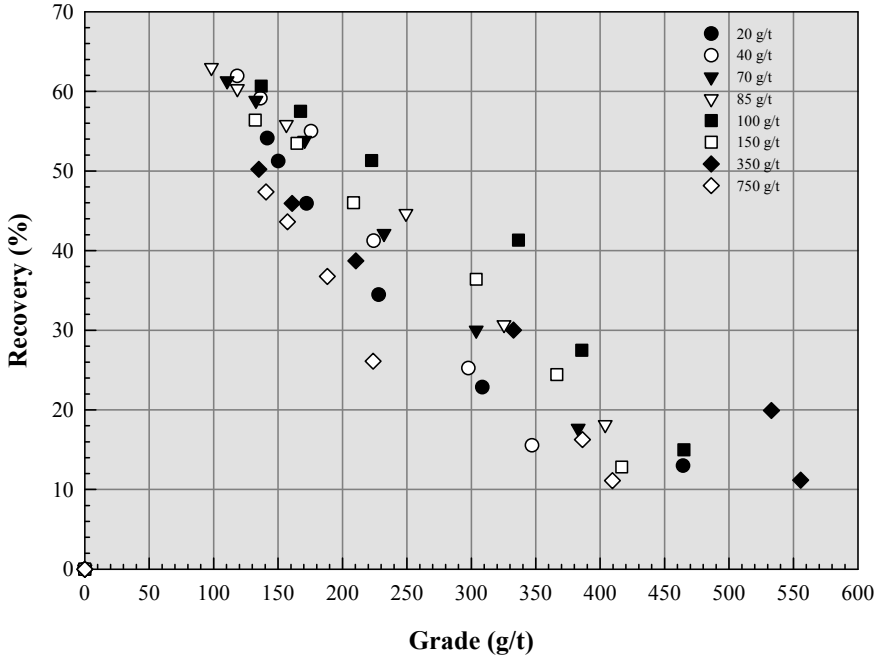

Figure 5.52 Grade vs. recovery curves of the experiments done to reveal the effect of using different amount of Na_2S on the flotation of Bergama-Ovacik gold ore (collector combination: KAX-Aerofloat 208 and Aerophine 3418A)

According to Table 5.8, Figure 5.51 and Figure 5.52 the results can be summarized as follow;

- Increasing Na_2S dosage to 85 g/t increased the ultimate recovery to 61.79 %. Flotation rate value was also increased, it was found to be as 0.66. A concentrate assaying 98.38 g/t was produced. Increasing the amount of Na_2S used further caused an adverse effect on the flotation recovery and rates.

- There were not any noticeable results obtained from the investigation of the effects of $CuSO_4$ addition to the pulp. There are minor differences in concentrate grade, ultimate recovery and flotation rate values obtained by using different amount of $CuSO_4$. But it can be noted that increasing copper sulphate caused minor decreases in the ultimate recoveries

5.3 Coal-Oil Assisted Gold Flotation

5.3.1 Material

Coal-oil assisted gold flotation studies have been conducted using the Bergama-Ovacik gold ore samples characterized in section 4.3.2, using coal samples (see Table 4.1, Table 2, Table 3 and Table 4), oil samples characterized in section 4.2 (See Figure 4.5 and Table 4.6) and the flotation reagents.

5.3.2 Method

- Flotation studies have been run at natural pH of the gold ore (8.1). The Outokumpu laboratory scale flotation machine has been used for the flotation tests.
- 2.5 l/min. aeration rate has been applied
- Batch flotation tests were carried out in a round shaped flotation cell (2 l)
- 400 g of Bergama-Ovacik gold ore samples ground below 0.060 mm and averaging 9 g/ton gold grade has been subjected to the flotation experiments
- KAX (50 g/t) and Aerofloat 208 (50 g/t) and one more different collector (50 g/t) were used in each experiment
- Na_2S (85 g/t) was used
- Na_2SiO_3 (400 g/t) was used as a depressant
- Aerofroth 70 (25 g/t) was used as a frother

Coal particles were agglomerated using oil prior to mixing with the gold-bearing ore pulp. An IKA-Eurostar (Power control-visc) stirrer was used for emulsification and an Outokumpu flotation machine with speed control was used for the coal-oil assisted gold flotation. Pre-agglomeration of coal particles and conditioning of flotation pulp were started simultaneously. 1/25 coal-oil ratio was used for standard tests. Oil was added to the emulsifying cell, agitated at 2000 rpm for 5 minutes. Coal particles (10 % by weight) were added to the oil-water emulsion and agitated at 2000

rpm for another 3 minutes. Agitation speed was decreased to 1300 rpm and agglomeration finished after 2 minutes.

Depressant and sulphidizer was added to the flotation pulp and conditioned at 1500 rpm agitation speed. Flotation reagents were added after 5 minutes conditioning and the pulp was conditioned for another five minutes. Previously agglomerated coal particles was added to the gold-bearing pulp and agitated at 1500 rpm for 10 minutes. Stirring speed was decreased to 1300 RPM and pulp was conditioned for 5 minutes for agglomerate formation, frother was added to the pulp and conditioned for another 5 minutes. The produced coal-oil-gold agglomerates were separated from the pulp by flotation at natural pH level.

5.3.3 Coal-Oil Assisted Flotation Studies of Bergama-Ovacik Gold Ore

5.3.3.1 Determination of the Agglomeration Oil

50 g of coal ground below 300 micron was used as agglomerating material in the experiments (amount of coal was kept constant). 1/25 coal/oil ratio and eight different oils were used to determine the best agglomerating oil for the recovery of gold. The experiments conducted by two different methods;

- The method involves adding pre-agglomerated coal particles and oil-water emulsion to the gold-bearing pulp.
- The other method involves the screening of previously formed agglomerates to remove the oil-water emulsion and adding the agglomerates to the gold-bearing pulp.

The following code names were used for the rest of the study instead of the full names of the agglomerating oils.

Code	Oils	Code	Oils
SO	Unrefined sunflower oil	OO	Olive oil
CO	Unrefined corn oil	DO	Diesel oil
O-PO	Olive-pomace oil	KS	Kerosene
CtO	Cotton oil	AP-704	Aero promoter 704

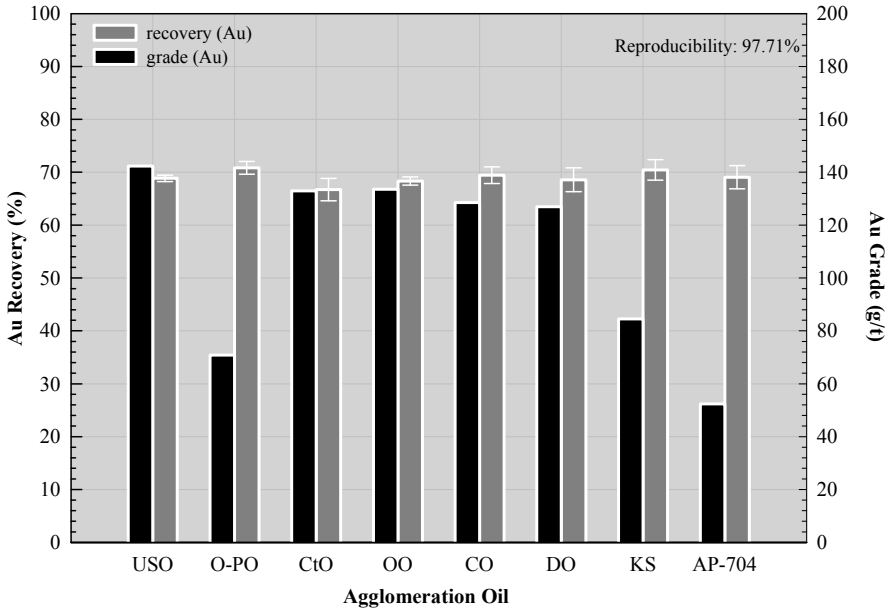

Figure 5.53 The effect of different agglomerating oils and oil-water emulsions

Figure 5.53 implies that vegetable oils produced very similar results. Olive-pomace oil, Aero promoter 704 and petroleum oils differ from the vegetable oils (Appendix 4 – Table 1and Table 2).

Vegetable oils produced concentrates assaying between 125 g/t – 145 g/t Au with about 65 % - 70 % recovery. Olive-pomace oil is the oil obtained by treating olive pomace with solvents or other physical treatments, that is the reason why differs from the other vegetable oils. It produced a concentrate containing 70.81 g/t Au with 70.84 % recovery.

Diesel oil produced a concentrate assaying 126.94 g/t Au with 70.09 % recovery and kerosene produced a concentrate having 84.53 g/t Au grade with 69.03 % recovery.

It was observed that adding oil-water emulsion to the pulp affects the froth stability and worsens it, decreases the selectivity and recovery.

It has been decided to employ screening of the previously formed agglomerates before the addition to the pulp. Agglomerates produced and screened using a 75 micron sieve before contacting with gold ore pulp.

As it can be seen from Figure 5.54, when screening of the agglomerates employed in the process, the grade and the recovery of the concentrates have been improved (Appendix 4 – Table 3).

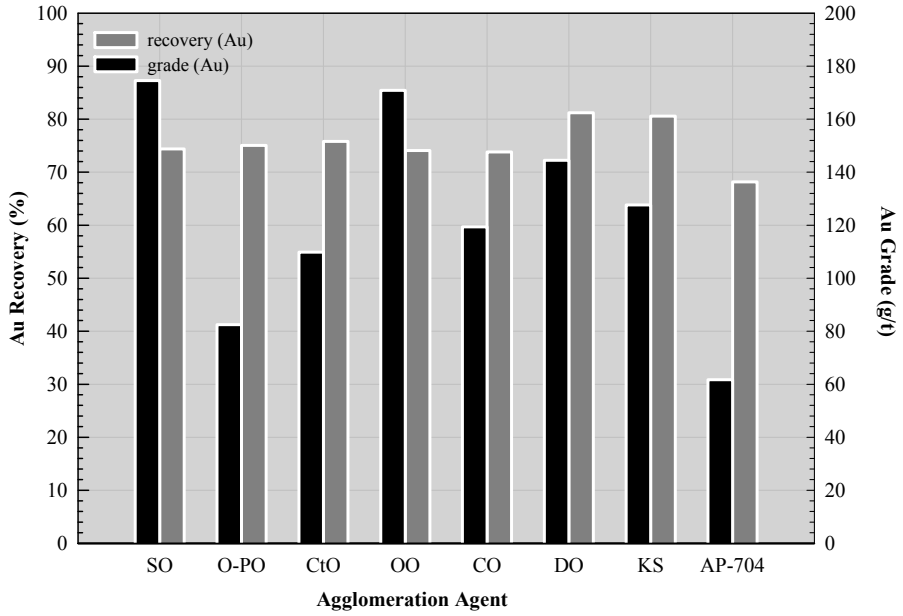

Figure 5.54 The effect of different agglomerating oils (water-oil emulsion removed by screening)

Olive and sunflower oils were the best vegetable oils and produced concentrates containing 170.83 g/t and 174.59 g/t Au respectively. The gold recoveries were also very close 74.09 % and 74.37 %.

Olive-pomace oil and Aero promoter 704 were the least selective agglomerating oils. The concentrates having 82.51 g/t Au and 61.72 g/t Au were obtained with 75.05 % and 68.18 % gold recoveries respectively.

Petroleum oils were the most effective agglomerating agents. A concentrate assaying 127.65 g/t Au was obtained with 80.56 % gold recovery by using kerosene. Diesel oil was more selective than kerosene, the concentrate gold grade was 144.50 g /t Au, and 81.22 % gold recovery was achieved.

It has been decided to employ screening of agglomerates before the addition to the pulp and to use diesel oil as the agglomerating oil for the rest of the study.

5.3.3.2 The Effect of Ore/Coal Ratio

25 g of coal ground below 300 micron, 1/25 coal/oil ratio was kept constant in the experiments. The effect of changing the amount of coal on the grade and the recovery of gold concentrates was investigated.

Figure 5.55 demonstrates the result of the experiments. It was observed that increasing the ore/coal ratio (decreasing the amount of coal used) caused to produce concentrates having higher gold grades with decreasing recoveries. It can be noted that for a constant coal/oil ratio, increasing the coal amount used for agglomeration should cause to generate more agglomerates, provide increased surface area and as a result, it should provide higher recoveries.

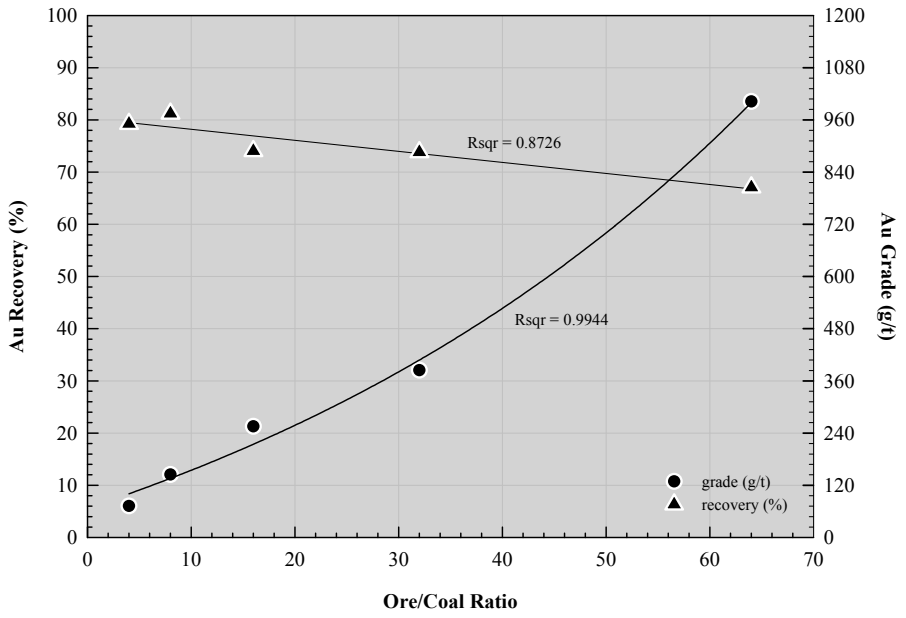

Figure 5.55 The effect of coal/oil ratio on the coal-oil assisted gold flotation

A concentrate assaying 72.15 g/t Au with 79.23 % was produced using the lowest ore/coal ratio (4). Concentrate gold grades tend to increase by increasing the ore/coal ratio and a concentrate containing 1002.23 g/t Au with 67.08 % recovery was obtained by using the highest ore/coal ratio (Figure 5.55, Appendix 4 – Table 4).

5.3.3.3 The Effect of Coal/Oil Ratio

25 g of coal ground below 300 micron was used as agglomerating material in the experiments (amount of coal was kept constant). Tests were done by using different amount of diesel oil for each experiment.

According to Wu et al. (2004b) producing smaller agglomerates provides increased collision rates, so higher K values and faster adhesion, whereas larger agglomerates bestows slightly larger attachment probability in coal-oil-gold agglomeration. Calvez et al. (1998) obtained higher gold recovery values by using lower oil/coal (higher coal/oil) ratios, increasing the available surface area of the agglomerates for gold penetration increased the gold recovery.

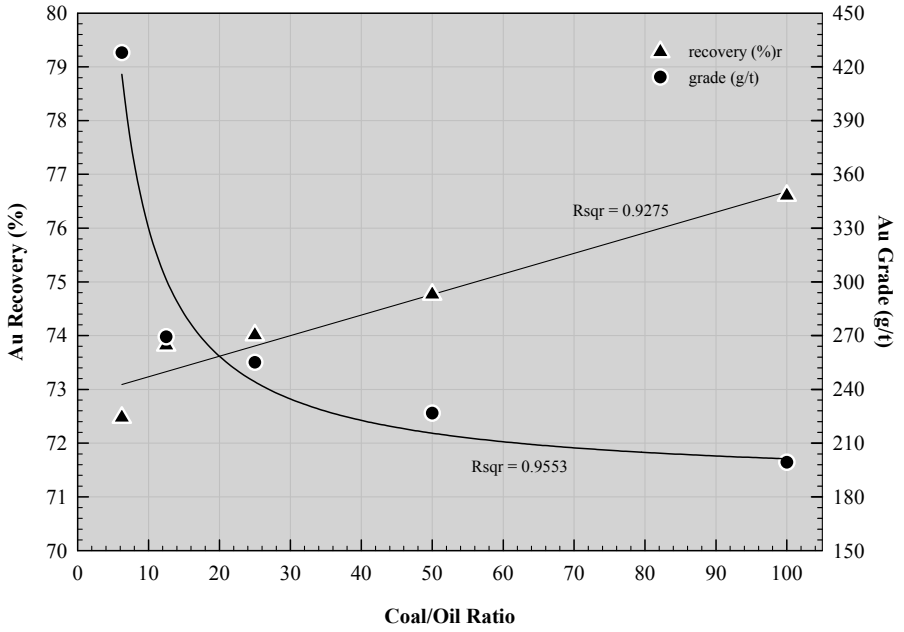

Figure 5.56 The effect of coal/oil ratio on the coal-oil assisted gold flotation

Figure 5.56 demonstrates that decreasing the coal/oil ratio by increasing the amount of oil used; decreased the gold recovery and increased the gold grade of the concentrates. It was most probably due to the difference in the size and the integrity of the formed agglomerates (Appendix 4 – Table 5).

Similarly, higher gold recovery values were achieved by using higher coal/oil ratios in coal-oil assisted gold flotation. Nevertheless, larger agglomerate surface

area caused fine gangue entrainment into the agglomerates. Therefore, increasing the coal/oil ratio decreases the grade of the concentrates.

Micro agglomerates were generated at low oil concentrations, they have larger surface area and recover more gold but also more vulnerable against gangue penetration. Low-grade concentrates with relatively high recoveries were obtained at low oil concentrations. A gold concentrate containing 199.24 g/t Au was obtained with 76.60 % gold recovery by using the highest coal/oil ratio.

Increasing the oil concentration decreased the surface area of the agglomerates while strengthening the formation. The structure of the agglomerates became more compact. Penetration into the agglomerates is harder for gangue particles and the gold grade of the concentrates enhanced, gold recovery relatively decreased. Using the highest oil concentration a concentrate assaying 427.88 g/t Au with 72.47 % gold recovery was obtained.

5.3.3.4 The Effect of Solid Ratio

This is a physical parameter affecting the flotation performance. Generally, it is believed that higher solid ratios provide lower concentrate grades and higher gold recoveries. High density of gold requires working at high solid concentrations especially when the ore contain coarse gold particles. 25 g of coal ground below 300 micron was used and 2/25 coal/oil ratio was kept constant in the experiments.

Figure 5.57 presents the relationship between the grade and recovery of the concentrates and solid concentration of the pulp. It can be seen that gold recovery is greater and the concentrate gold grade is lower at the highest solid ratio. The higher the solid ratios, the lower the grades were obtained while the concentrate recoveries were slightly enhanced. The use of the lowest solid ratio (20 %) produced a concentrate containing 269.27 g/t Au with 73.82 % gold recovery. A concentrate having 161.67 g/t with 75.09 % gold recovery was obtained by the use of 35 % solid ratio (Appendix 4 – Table 6).

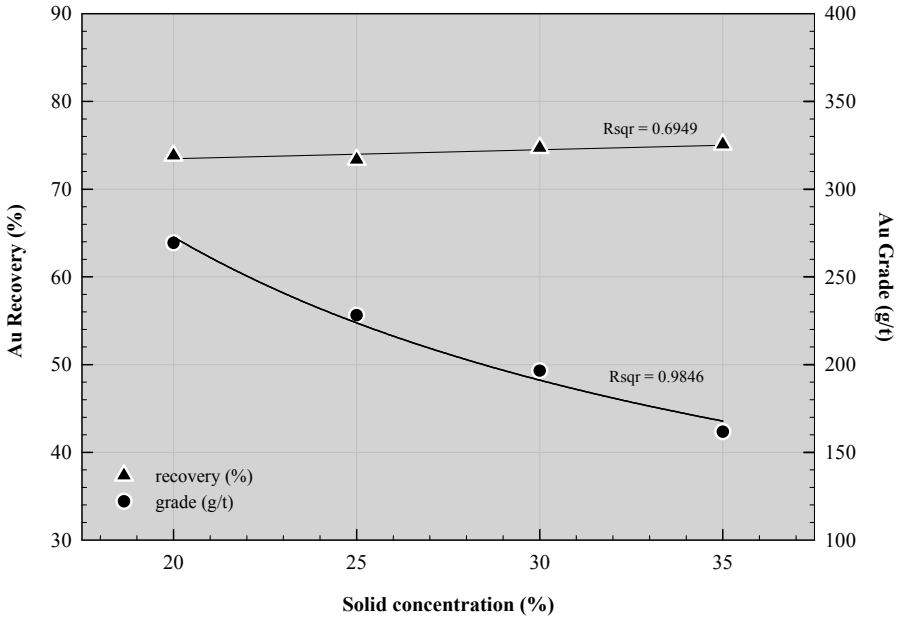

Figure 5.57 The effect of solid concentration on the coal-oil assisted gold flotation

5.3.3.5 The Effect Coal-Particle Size

The effect of the particle size on coal-oil-gold agglomeration is not clear. The literature seems to be in a contradiction about this subject. Calvez et al., (1998) reports that finer coal particles generate smaller agglomerates, larger surface area and increase the recovery of gold. On the contrary, Moses and Petersen (1999) have worked with the coal samples having 45 % -90 micron, 300-500 micron and 500-650 micron particles and found that the coarsest size fraction gave the highest recoveries. 25 g of coal was used and 2/25 coal/oil ratio was kept constant in the experiments.

The success of recovering gold particles seems to be dependent on the surface area, stability and floatability of the agglomerates. Five different size fractions were used to investigate the effect of particle size range on the process efficiency (Figure 5.58). Figure 5.59, Figure 5.60 and Figure 5.61 should be carefully examined to understand the effect of the coal particle size on the coal-oil assisted gold flotation. As it can be clearly seen from the pictures, using finer coal particles makes the agglomerates more compact, and decreases the number of individual agglomerates and the cavities between the particles.

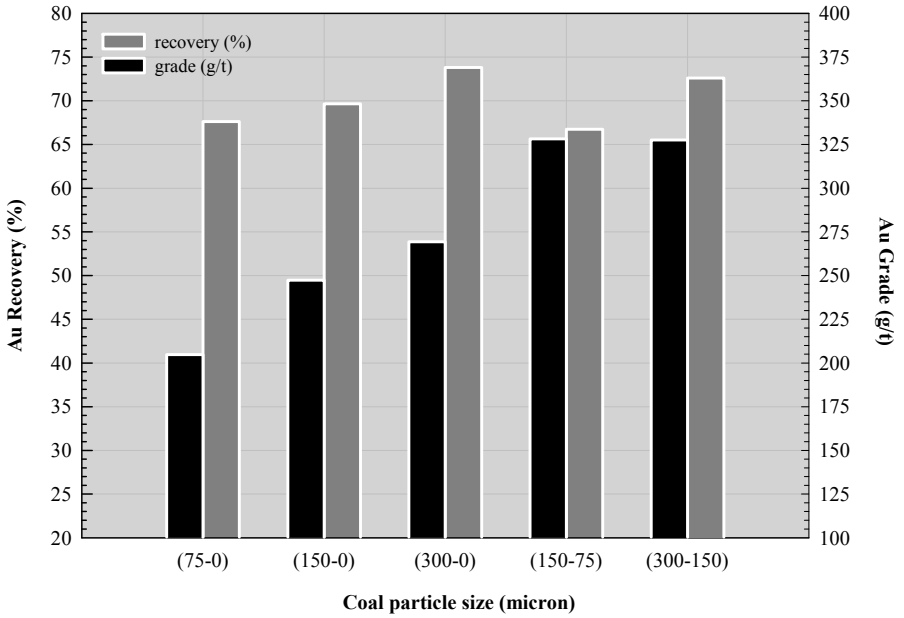

Figure 5.58 The effect of coal particle size on the coal-oil assisted gold flotation

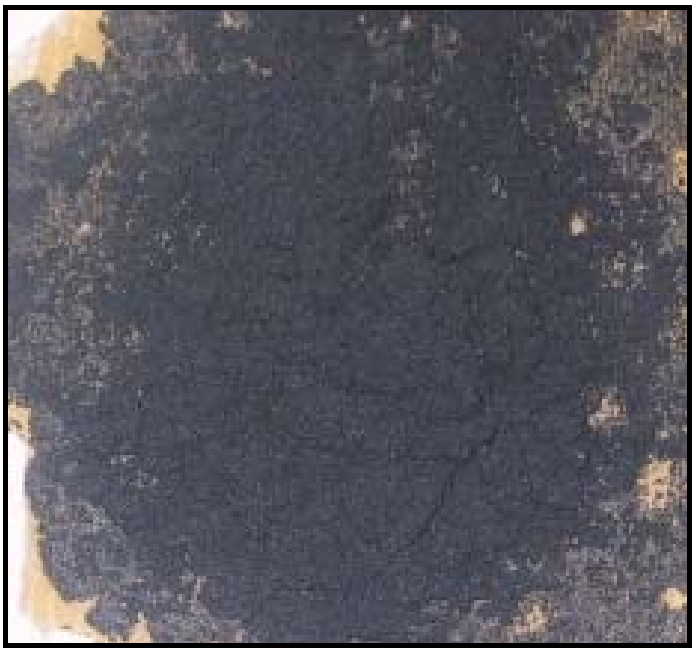

Figure 5.59 The picture of the coal-oil-gold agglomerates (300-150 micron)

Figure 5.60 The picture of the coal-oil-gold agglomerates (150-75 micron)

Figure 5.61 The picture of the coal-oil-gold agglomerates (-75 micron)

The concentrate recoveries were 67.61 %, 69.66 %, and 73.82 % and the concentrate grades were 204.80 g/t, 247.39 g/t and 269.27 g/t for the agglomeration

materials having -75 micron, -150 micron, and -300 micron coal particles respectively (Appendix 4 – Table 7).

A concentrate assaying 327.56 g/t with 72.61 % recovery was obtained by using coal particles in a size range of 300-150 microns. Concentrate gold grade decreased to 328.4 g/t and the recovery was 66.75 % when a material classified into 150-75 micron size range was used.

By comparing the results, it can be noticed that using an agglomeration material having coal particles in a wide size range produced highly recovered gold containing concentrates. However, using coarser coal particles in a narrow size range increased the selectivity.

5.3.3.6 The Effect of Aeration Rate

Aeration is an important physical parameter for flotation. It can be varied within limits; either too much or too little can be detrimental. The amount of air and the number and size of air bubbles forced or induced in the pulp characterize the aeration conditions of the flotation.

Moses and Petersen (1999) reports that reducing or increasing the air rate both had a negative effect on the coal-oil-gold agglomeration.

Researchers reports that increasing the amount of aeration to a higher level increases flotation recovery at first but a maximum is reached. Further increase in aeration gives an unacceptable increase in gangue entrainment in flotation (Allan & Woodman, 2001).

25 g of coal ground below 300 micron was used and 2/25 coal/oil ratio was kept constant in the experiments.

The graphic 5.62 demonstrates the effect of aeration rate on the coal-oil assisted gold flotation. As it can be seen from the figure, concentrate gold grades tend to decrease by increasing the aeration rate (Appendix 4 – Table 8).

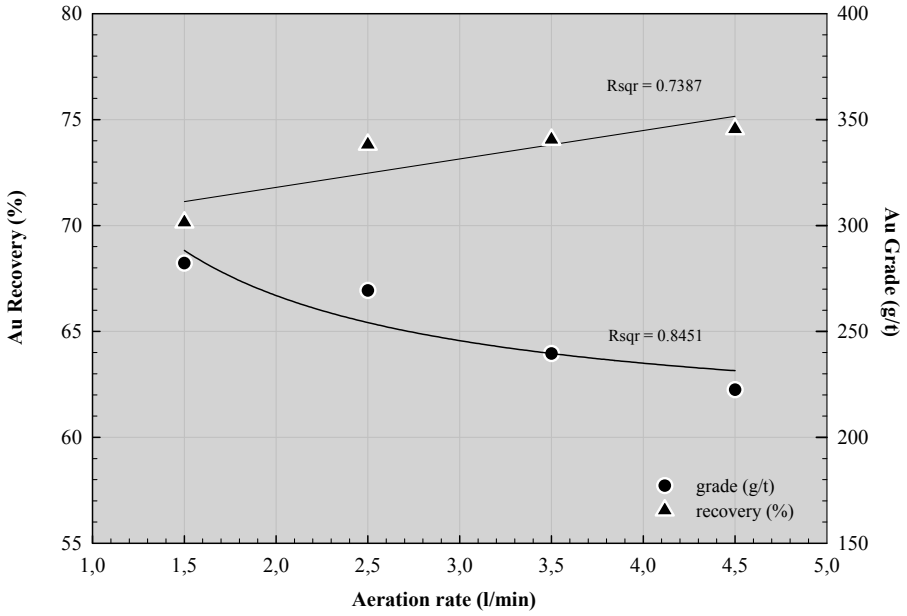

Figure 5.62 The effect of aeration rate on the coal-oil assisted gold flotation

The results indicated that increasing the amount of aeration to a higher level (from 1.5 l/min. to 2.5 l/min.) achieved higher flotation recovery at first, but no more improvement has been observed while concentrate gold grades tending to decrease by the further increases. The recovery values were 70.15 % and 73.82 % and the grade of the concentrates were 282.21 g/t and 269.27 g/t respectively. There was a slight improvement in the recovery between 2.5 l/min. and 3.5 l/min, it was increased from 73.82 % to 74.05 %, but the grade of the concentrate was decreased from 269.27 g/t to 239.52 g/t. When the aeration rate was increased from 3.5 l/min. to 4.5 l/min., the recovery was enhanced to 75.47 %. However, the gold grade of the concentrate decreased dramatically to 198.26 g/t.

5.3.3.7 The Effect of Stirring Speed

The experiments involve changing the stirring speed for conditioning and agglomerate formation stages together. Two different agitation speeds in conditioning and different stirring speeds for agglomerate formation stage were applied. 25 g of coal ground below 300 micron was used and 2/25 coal/oil ratio was kept constant in the experiments. Theoretically, increasing the stirring speed in flotation creates a mechanical assisting effect on the process by lifting the heavier

gold particles in the flotation cell to the upward and giving a chance to entrain into the froth phase. The gold recovery by entrainment increases while true flotation of free gold decreases. Thus the concentrate gold grade tends to decrease due to the gangue co-entrainment but higher gold recoveries could be obtained.

It can be assumed for coal-oil assisted gold flotation that higher agitation speeds for conditioning (contacting) of agglomerates-ore pulp would increase gold particle-coal/oil agglomerate collision and attachment probabilities. On the other hand over a critical stirring speed, formed coal-oil-gold agglomerates may become to loose their strength and stability, and lowers the gold recovery. The results verified the assumptions at the beginning of the study; increasing the stirring speed in contacting stage provided better recoveries and decreased the concentrate gold grades. But, as it can be seen from the graph that recovery and grade values were changed slightly for 1500 and 1800 rpm. It may be explained by the counter-relationship between the agglomerate stability and mechanical effect of stirring, thus, higher stirring speed might increase gold recovery by lifting but also caused agglomerate breakage and lowered it. A concentrate containing 277.10 g/t with 80.60 % recovery was obtained by using 1800 rpm stirring speed for conditioning and 1500 rpm stirring speed for contacting stages (Appendix 4 – Table 9).

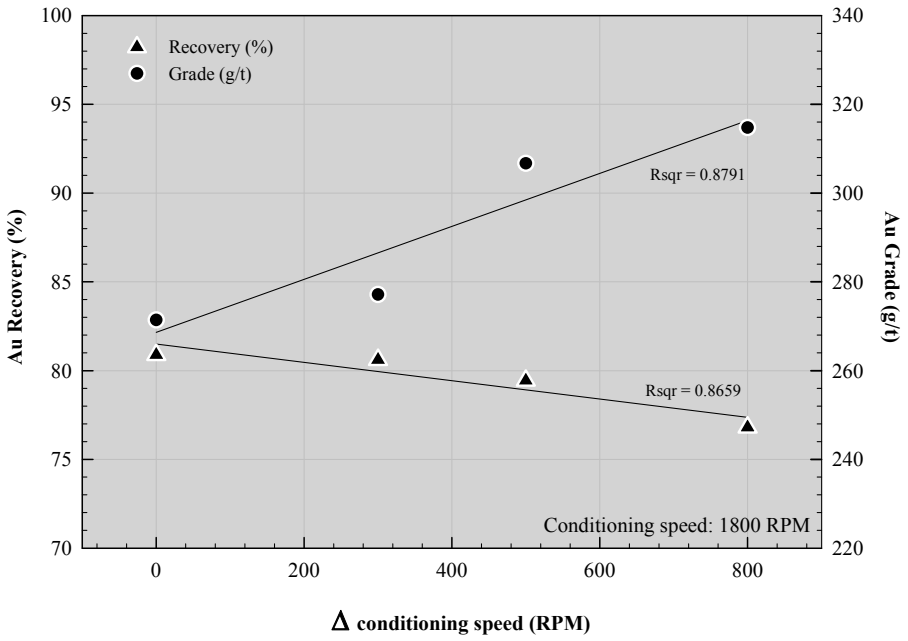

Figure 5.63 The effect of stirring speed in agglomerate formation stage on the process

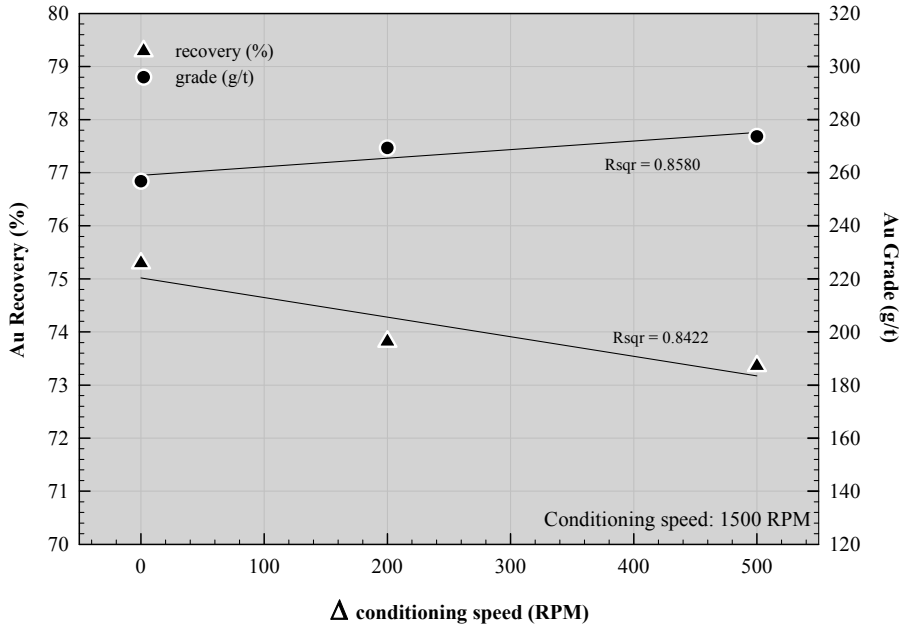

Figure 5.64 The effect of stirring speed in agglomerate formation stage on the process

The experiments were repeated by applying 1500 rpm for conditioning stage. The same relationship has been found that the lowest stirring speed provided the highest concentrate grade and the highest stirring speed produced the concentrate with maximum efficiency. A concentrate having 273.65 g/t with 73.36 % recovery has been produced by using 1000 rpm stirring speed. Increasing the speed to 1300 rpm increased the recovery to 73.82 % and the grade of the concentrate was 269.27 g/t. The concentrate gold grade decreased to 256.76 g/t and the recovery was 75.29 % at 1500 rpm.

5.3.3.8 The Effect of Agglomerate Cycles

The experiments were done to investigate the adsorption capacity of the agglomerates. Loaded agglomerates were used in multiple cycles to reveal the effect of recycling of agglomerates on coal-oil-assisted gold flotation. The best results obtained from the previous experiments were applied; 400 g of fresh ore, -300 micron coal particles, and 6.25 coal/oil and 16 ore/coal ratios were used for the experiments. Stirring speeds for conditioning and contacting were 1800/1500 rpm and the aeration rate was kept constant at 2.5 l/min.

Wu et al., (2003) and House et al., (1988) report the same finding for coal-oil-gold agglomeration that recycling did not reduce the adsorption capacity of the agglomerates.

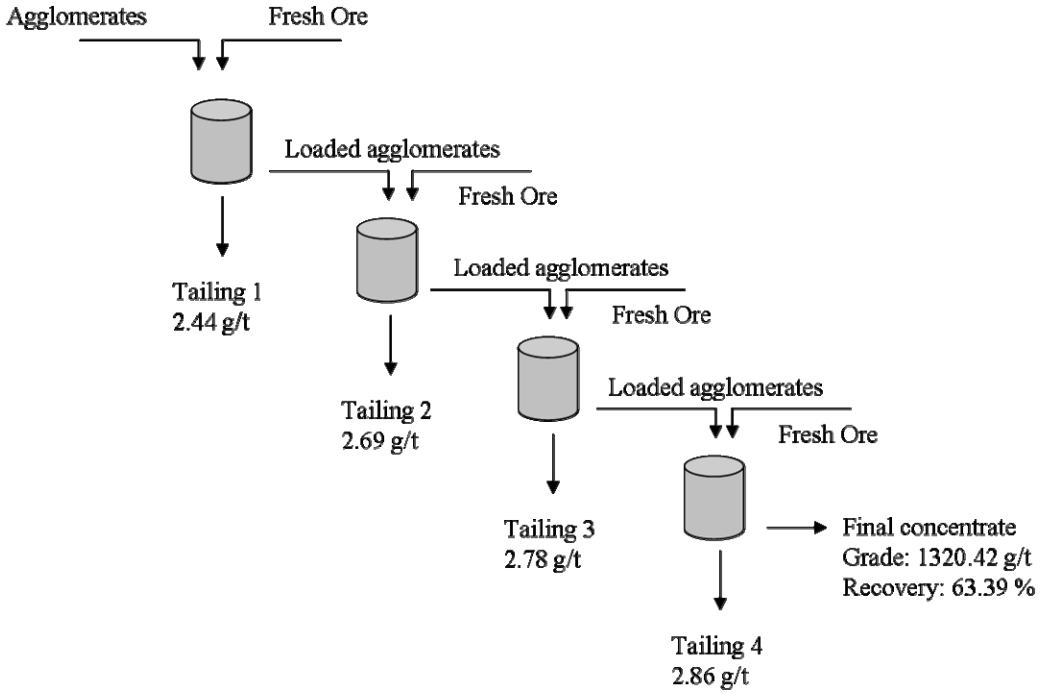

Figure 5.65 Schematic representation of agglomerate recycling test

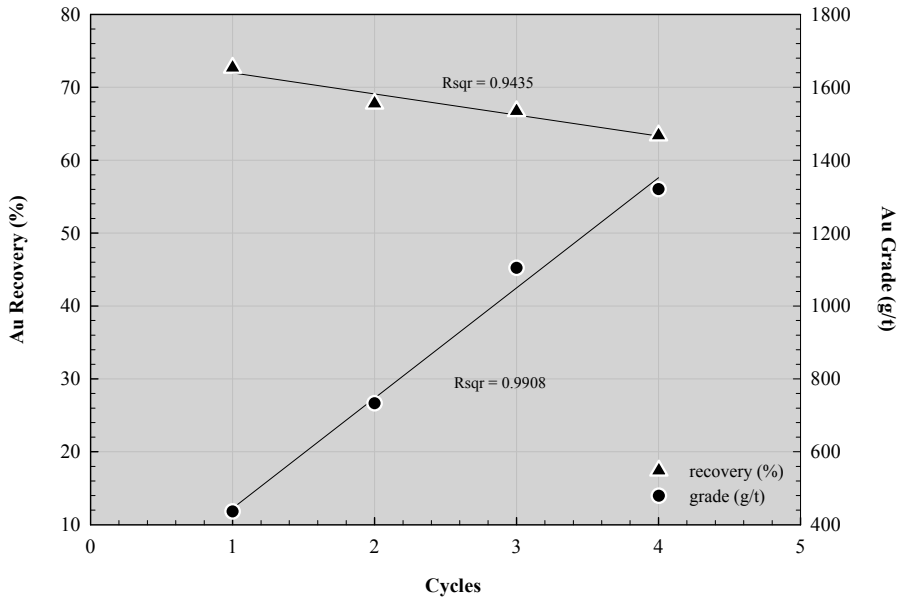

Figure 5.66 The effect of recycling the coal-oil-gold agglomerates

The gold grade of the concentrate was 436.30 g/t and the recovery was 72.68 % for the first run of the agglomerates. The gold grade was increased to 732.76 g/t while the gold recovery was decreased to 67.77 % for the second run. At third cycle of the agglomerates, the gold content of the agglomerates was increased to 1104.24 g/t and the gold recovery was 66.70 %. The last run of the agglomerates produced a concentrate assaying 1320.42 g/t with 63.39 % recovery. The results dictate that the gold content of the agglomerates increases by increasing the number of cycles while recovery was decreased in a linear fashion. The reason of that can be explained by the breakage of the formed agglomerates in each cycle. The agglomerates were treated with the fresh ore pulps and lost its adsorption capacity slightly in each cycle (Appendix 4 – Table 10).

5.3.3.9 The Effect of Scavenging

The previous experiments showed that a high grade concentrate could be produced with 80.60 % recovery by using a stirring speed combination like 1800 rpm for conditioning and 1500 rpm for agglomerate formation stages (25 g of coal ground below 300 micron was used and 2/25 coal/oil ratio was kept constant in the experiments). However, the recovery value was decreased to 72.68 % when the optimal conditions applied (referring the experiments done to reveal the effect of agglomerate recycling / the first run of the agglomerates). The reason of that can be explained by the micro agglomerate formation. Low oil concentration and high stirring/conditioning speeds provided the convenient conditions for the formation of micro agglomerates. Increasing the oil amount in the system caused to produce coarser and more compact agglomerates even in strongly agitating conditions (at high stirring/conditioning speeds) and it has caused to obtain lower recovery values.

Regarding the above explanations, it has been decided to investigate enhancing the recovery by scavenging the tailings. The best results obtained from the previous experiments were applied; 400 g of fresh ore, -300 micron coal particles, and 6.25 coal/oil and 16 ore/coal ratios were used for the experiments. Stirring speeds for conditioning and contacting were 1800/1500 rpm and the aeration rate was kept constant at 2.5 l/min.

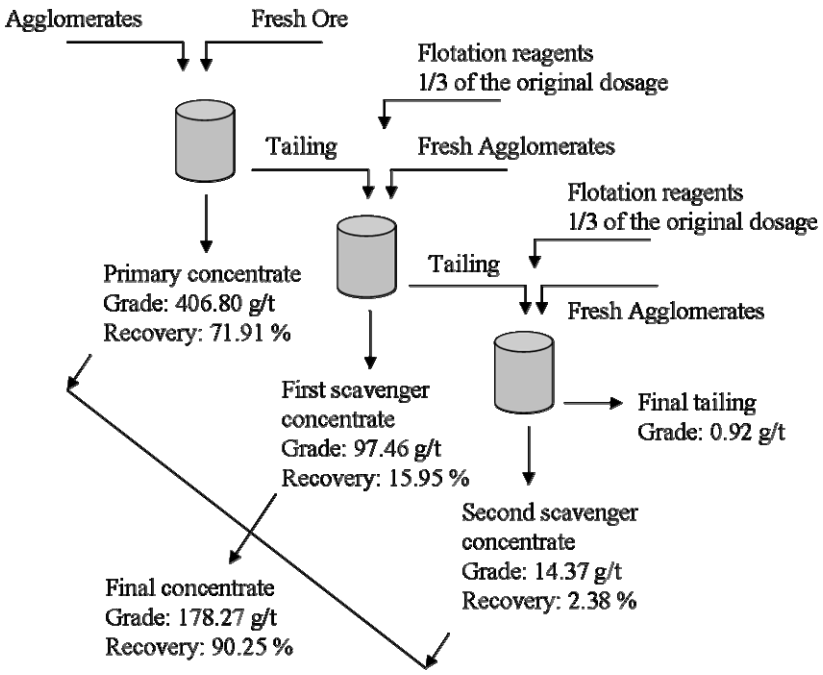

Figure 5.67 Schematic representation of scavenging the tailings (-60 micron feed)

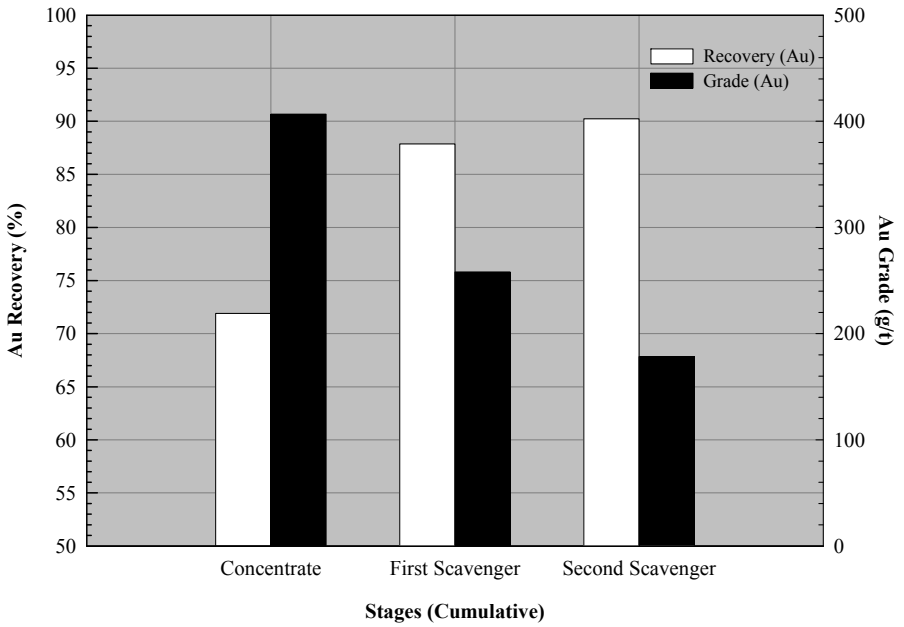

Figure 5.68 The effect of scavenging on the coal-oil assisted gold flotation (-60 micron)

The experiments show that employing the scavenger stages in the process enhances the gold recovery. A concentrate 406.80 g/t gold grade was obtained with

71.91 % recovery. The concentrate gold recovery was improved to 87.86 % and grade of the concentrate decreased to 258.07 g/t by the addition of the first scavenger concentrate in to the primary concentrate. Employing the secondary scavenger stage improved the gold recovery further. A concentrate assaying 178.27 g/t Au with 90.25 % recovery was obtained by combining the concentrates (Appendix 4 – Table 11).

The same experiment was repeated by using a feed material screened at 25 micron. The material below 25 micron particle size was subjected to the same test. The results are shown in Figure 5.69.

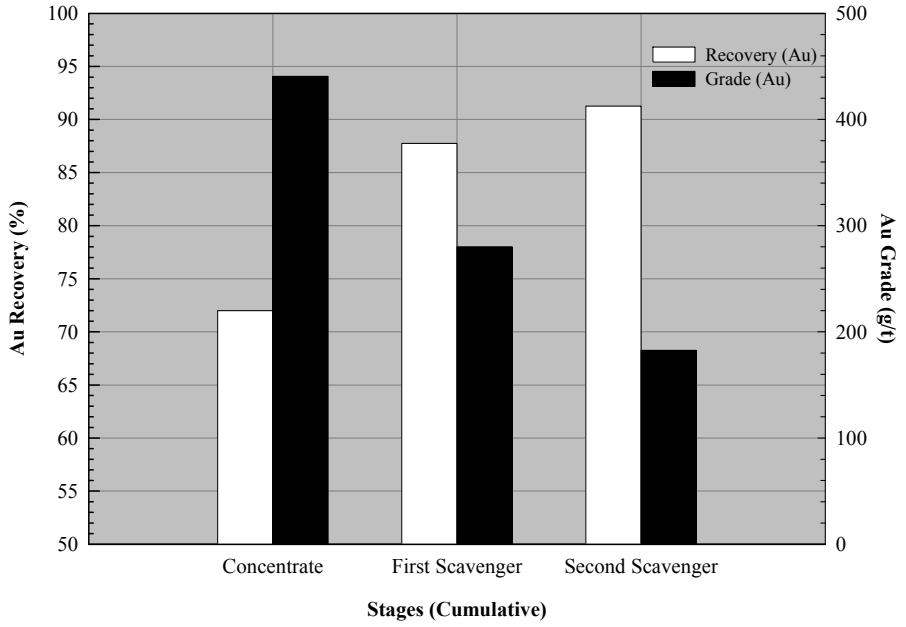

Figure 5.69 The effect of scavenging on the coal-oil assisted gold flotation (-25 micron)

The use of – 25 micron feed material did not make any difference on the process efficiency. This is most probably due to the mineralogical characteristic of the gold ore. A concentrate 440.71 g/t gold grade was obtained with 71.99 % recovery. The concentrate gold recovery was improved to 87.75 % and grade of the concentrate decreased to 280 g/t by the addition of the first scavenger concentrate in to the primary concentrate. Employing the secondary scavenger stage improved the gold recovery further. A concentrate assaying 182.50 g/t Au with 91.26 % recovery was obtained by combining the concentrates (Appendix 4 – Table 12).

5.4 Comparison with the Conventional Methods

Flotation and coal-oil assisted gold flotation tests were done on the same samples (-60 micron particle size and assaying about 8 g/t Au). The results are summarized in the Table 5.9.

Table 5.9 Comparison of the flotation and coal oil assisted gold flotation

Methods	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Flotation (8 min. flotation time)	5.13	98.38	61.79
Coal-Oil Assisted Gold Flotation	1.90	406.80	71.91
Coal-Oil Assisted Gold Flotation (1 scavenger stages)	3.07	258.07	87.86
Coal-Oil Assisted Gold Flotation (2 scavenger stages)	4.56	178.27	90.25
Coal-Oil Assisted Gold Flotation (4 times agglomerate recycling)	0.37	1320.42	63.39

Bergama-Ovacik gold ore has been treated by leaching since may 2001. The leaching data was taken from the Koza Gold Company. Knelson multi-gravity concentration tests were done at the company using the samples taken from the hyrocyclone underflow, 900 kg of the ore was passed through the device and 12 psi water pressure was applied. Coal-oil assisted gold flotation test was run on the same sample after screening at 300 micron (27.79 g/t Au). The results were summarized in Table 5.10.

Table 5.10 Comparison of Knelson concentrator, cyanide leaching and coal oil assisted gold flotation

Methods	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Coal-Oil Assisted Gold Flotation	1.07	2164.32	83.25
Cyanide Leaching	-	-	≈ 95.00
Knelson Concentrator	0.16	15000	≈ 32

The results show that coal-oil assisted gold flotation gave better performance than flotation and multi gravity separation and it is possible to use it before cyanidation to reduce the amount of the ore feed to the cyanidation plant.

CHAPTER SIX
SUMMARY OF THE RESULTS

The study has been conducted to develop coal-oil assisted gold flotation as an alternative gold recovery process. For this purpose, the experiments were done to reveal the important parameters and their possible effects on the efficiency of the process.

Coal-oil assisted gold flotation is mainly consisted of two different inferior processes; flotation and coal-oil agglomeration. For this reason, it is important to know the basic kinetics of these sub-processes before studying the coal-oil assisted gold flotation. Therefore, the mechanism of coal-oil agglomeration and flotation conditions of the specific gold ores were investigated individually at the beginning of the study.

Contact angle measurements were done to reveal the agglomeration behavior of the coal sample; Advancing and receding contact angles were determined to reveal the hysteresis of the measurements. The roughness values of the polished surfaces have been measured and the results show that higher roughness values caused higher hysteresis values ($\theta a - \theta r$). The highest hysteresis value for the oils was 11° (Figure 5.3)

Interesting to note that petroleum oils have given the lowest contact angle values and the results of the contact angle measurements of vegetable oils were very close. Critical surface tension of wetting has been determined as 26 dyne/cm by plotting $\cos\theta$ vs. surface tension of the oils and pure water.

ΔG_{spread} and ΔG_{attach} values have been calculated, the results show that petroleum oils would be better agglomerating agents (Figure 5.10 and Figure 5.11).

Coal-oil agglomeration tests were done to reveal the oil agglomeration behavior of the coal sample using coal samples. Schuhmann plots were drawn to find out the k values of the produced agglomerates. The plots showed that petroleum oils produced the coarsest agglomerates (950 micron with kerosene and 720 micron with diesel

oil). Vegetable oils generate agglomerates having k values about 400 micron (Figure 5.15). Normal distribution curves and cumulative ash content of the agglomerates were plotted via particle size (Figure 5.15). Figure 5.15 shows the same findings; petroleum oils make the agglomerates coarser and particle size of the agglomerates accumulated above 200 micron. Ash rejection with petroleum oils was also higher than the vegetable oils. Petroleum oils produced the agglomerates having the highest ash content at the finest size ranges (-75 micron) which was also indicating the selectivity against the gangue material.

The further examination of the agglomeration processes with different oils was done by plotting k, d_{80} and d_{50} values of the agglomerates via corresponding surface tension, viscosity and density values of the oils (Figure 5.16, 5.17 and 5.18). The indicators (d_{50}, d_{80}, k values) show the same tendencies;

- Oils having higher oil/water interfacial surface tension values produced coarser agglomerates

- Oils having higher density values generated finer agglomerates

- Oils having higher viscosity values produced finer agglomerates

- Petroleum oils seems to be the best agglomerating oils as it was observed from the contact angle measurements.

Flotation studies have been run to investigate the effect of reagent synergism on the gold recovery.

The effect of reagent synergism on the flotation of Efem Cukuru-Izmir gold ore was investigated using the method given in section 5.2.1.1 and the results were given in the related figures. Table 5.4, Figure 5.27, and Figure 5.28 represent the overall results of the experiments. Thus;

It can be concluded that the use of copper sulphate in the combinations produced a stabilized froth by providing a clean separation zone in the froth phase and increased the selectivity. However, the weight and ultimate recovery values and the flotation rate of the concentrates were low. Employing sodium sulphide in the combinations

enhanced the concentrate ultimate recovery and flotation rates. The weight recovery of the concentrates was also improved but the selectivity decreased eventually. The use of sodium sulphide in the combinations provided better recovery values than using copper sulphate. The reason of that can be explained by the mineralogy of the ore deposit. The deposit contains mostly fine gold particles (2.5 to 50 microns), occurring mostly as free grains in quartz and carbonate, and, in less amount, as inclusions in sulphide minerals (Oyman et.al, 2003).

- KAX - Aerofloat 208 - Aerofloat 242 and $CuSO_4$ combination is the most selective combination for the recovery of gold from Efem Cukuru gold ore (148.81g/t Au grade and 62.90 % ultimate recovery). Flotation rate was low and determined as 0.48. KAX –

- Aerofloat 208 - Aerophine 3418A - Na_2S combination is the most effective combination for the gold recover. The combination produced a concentrate containing 57.20 g/t Au with 75.21 % ultimate recovery.

- The same combination (KAX - Aerofloat 208 Aerophine 3418A - Na_2S) also produced the concentrate with the highest flotation rate value (1.02).

After the flotation experiments were completed, some initial coal-oil assisted gold flotation studies have been run on the Efem Cukuru gold ore. The process seems to be very successful even with its initial tests for the recovery of gold from Izmir-Efem Cukuru ore. Bergama-Ovacik gold ore deposit with its fine gold particle size would be a bigger challenge for the coal-oil-assisted gold flotation process. So, it has been decided to run the further coal-oil assisted gold flotation studies on the Bergama-Ovacik gold ore sample.

The effect of reagent synergism on the flotation of Bergama-Ovacik gold ore was investigated and overall results can be summarized as follow;

- KAX-Aerofloat 208-Aero 6697-$CuSO_4$ reagent combination was the most selective combination. It produced a concentrate assaying 129.02 Au with 44.08 % ultimate recovery.

- KAX-Aerofloat 208-Aero 412-Na_2S reagent combination has the highest flotation rate for the recovery of gold from Bergama-Ovacik gold ore (83.03 g/t gold grade, 57.51 % ultimate recovery, flotation rate was 0.78)

- KAX-Aerofloat 208-Aerophine 3418A -Na_2S is the most effective combination for the recovery of gold from Bergama-Ovacik gold ore (110.48 g/t gold grade, 60.13 %, flotation rate was 0.64)

Table 5.8 and Figure 5.51 and Figure 5.52 represent the overall results of the experiments done to reveal the effect of using Na_2S and $CuSO_4$ reagent synergism on the flotation of Bergama-Ovacik gold ore when KAX – Aerfloat 208 and Aerophine 3418A reagent combination used;

- Increasing Na_2S dosage to 85 g/t increased the ultimate recovery (61.79 %). Flotation rate value was also increased, it was found to be as 0.66. A concentrate assaying 98.38 g/t was produced. Increasing the amount of Na_2S used further caused an adverse effect and lowered the ultimate recovery and flotation rate values.

- There were not any considerable results obtained from the investigation of the effects of $CuSO_4$ addition to the pulp. Nevertheless, it can be noted that increasing copper sulphate caused minor decreases in the ultimate recoveries.

It can be concluded that the use of copper sulphate in the combinations produced a stabilized froth by providing a clean separation zone in the froth phase and increased the selectivity. However, the weight and ultimate recovery values and the flotation rate of the concentrates were low. Employing sodium sulphide in the combinations enhanced the concentrate ultimate recovery and flotation rates. The weight recovery of the concentrates was also improved but the selectivity decreased eventually. The use of sodium sulphide in the combinations provided better recovery values than using copper sulphate. The reason of that can be explained by the mineralogy of the ore deposits. The deposits contains mostly fine gold particles, occurring mostly as free grains in quartz and carbonates, and, in less amount, as inclusions in sulphide minerals.

The previously characterized oils have been used to determine the best oil type for coal-oil assisted gold flotation and the results summarized below;

- Vegetable oils produced concentrates assaying between 125 g/t – 145 g/t Au with about 65% - 70 % recovery. Olive-pomace oil produced a concentrate containing 70.81 g/t Au with 70.84 % recovery. Diesel oil produced a concentrate assaying 126.94 g/t Au with 70.09 % recovery and kerosene produced a concentrate having 84.53 g/t Au grade with 69.03 % recovery. It was observed that adding oil-water emulsion to the pulp affects the froth stability and worsens it, decreases the selectivity and recovery. It has been decided to employ screening of the previously formed agglomerates before the addition to the pulp. The grade and the recovery of the concentrates have been improved by applying screening of the agglomerates using a 75 micron sieve before contacting with gold ore pulp.

- When screening of the agglomerates employed in the process, - Olive and sunflower oils were the best vegetable oils and produced concentrates containing 170.83 g/t and 174.59 g/t Au respectively. The gold recoveries were also very close 74.09 % and 74.37 %.

- Olive-pomace oil and Aero promoter 704 were the least selective agglomerating oils. The concentrates having 82.51 g/t Au and 61.72 g/t Au were obtained with 75.05 % and 68.18 % gold recoveries respectively.

- Petroleum oils were the most effective agglomerating agents. A concentrate assaying 127.65 g/t Au was obtained with 80.56 % gold recovery by using kerosene. Diesel oil was more selective than kerosene, the concentrate gold grade was 144.50 g /t Au, and 81.22 % gold recovery was achieved.

Some experiments were done to determine the effect of ore/coal ratio; A concentrate assaying 72.15 g/t Au with 79.23 % was produced using the lowest ore/coal ratio (4). The gold grades of the concentrates tend to increase by increasing the ore/coal ratio and a concentrate containing 1002.23 g/t Au with 67.08 % recovery was obtained by using the highest ore/coal ratio (Figure 5.55). Increasing the coal

amount for a constant coal/oil ratio increased the number of the agglomerates and provided higher surface area, as a result, the recovery has been increased.

The effect of coal/oil ratio on the efficiency of the coal-oil assisted gold flotation has been investigated; a gold concentrate containing 199.24 g/t Au was obtained with 76.60 % gold recovery by using the highest coal/oil ratio. Increasing the oil concentration for a constant amount of coal decreased the surface area of the agglomerates. The structure of the agglomerates became more compact and particle penetration into the agglomerates became harder for gangue particles and even for the gold particles. The gold grade of the concentrates increased but gold recovery relatively decreased. Using the highest oil concentration a concentrate assaying 427.88 g/t Au with 72.47 % gold recovery was obtained (Figure 5.56).

The effect of solid ratio on the process was investigated; the use of the lowest solid ratio (20 %) produced a concentrate containing 269.27 g/t Au with 73.82 % gold recovery. A concentrate having 161.67 g/t with 75.09 % gold recovery was obtained by the use of 35 % solid ratio. The increased water transfer at high solid ratios can explain the reason of that. Higher water transfer brings more fine gold as well as more gangue minerals into the froth and enhances the recovery while reducing the grade of the concentrates (Figure 5.57).

The effect coal-particle size on the coal-oil assisted gold flotation was investigated and the experiments revealed that the use of agglomeration material containing finer coal particles produced compact agglomerates, and using an agglomeration material having the coarsest coal particles provided higher selectivity. By comparing the results, it can be noticed that using an agglomeration material having coal particles in a wide size range produced highly recovered gold containing concentrates. The concentrate recoveries were 67.61 %, 69.66 %, and 73.82 % and the concentrate grades were 204.80 g/t, 247.39 g/t and 269.27 g/t for the agglomeration materials having -75 micron, -150 micron, and -300 micron coal particles respectively (Figure 5.58).

The effect of aeration rate on the process was determined, the results indicated that increasing the amount of aeration to a higher level (from 1.5 l/min. to 2.5 l/min.)

achieved higher flotation recovery at first, but concentrate gold grades tend to decrease by the further increases in the aeration rate. The recovery values were 70.15 % and 73.82 % and the grade of the concentrates were 282.21 g/t and 269.27 g/t respectively. When the aeration rate was increased from 3.5 l/min. to 4.5 l/min., the recovery has increased from 74.05 % to 75.47 %. However, the gold grade of the concentrate dramatically decreased to 198.26 g/t (Figure 5.62).

The effect of stirring speed on the gold recovery has been investigated; Two different agitation speeds in conditioning and different stirring speeds for agglomerate formation stage were applied. Increasing the stirring speed in contacting stage provided better recoveries and decreased the concentrate gold grades (Figure 5.63 and Figure 5.64).

Loaded agglomerates were used in multiple cycles to reveal the effect of recycling of agglomerates on coal-oil-assisted gold flotation. The best results obtained from the previous experiments were applied; 400 g of fresh ore, -300 micron coal particles, and 6.25 coal/oil and 16 ore/coal ratios were used for the experiment. Stirring speeds for conditioning and contacting were 1800/1500 rpm and the aeration rate was kept constant at 2.5 l/min. The gold grade of the concentrate was 436.30 g/t and the recovery was 72.68 % for the first run of the agglomerates. Agglomerates were recycled for four times and the last run of the agglomerates produced a concentrate assaying 1320.42 g/t with 63.39 % recovery. The results dictate that the gold content of the agglomerates increases with the number of cycles while recovery was decreased in a linear fashion. The reason of that can be explained by the breakage of the formed agglomerates in each cycle. The agglomerates were treated with the ore pulps and lost its adsorption capacity slightly in each cycle (Figure 5.65 and Figure 5.66).

The experiments show that employing the scavenger stages in the process enhances the gold recovery. A concentrate 406.80 g/t gold grade was obtained with 71.91 % recovery. The concentrate gold recovery was improved to 87.86 % and grade of the concentrate decreased to 258.07 g/t by the addition of the first scavenger concentrate in to the primary concentrate. Employing the secondary scavenger stage improved the gold recovery further. A concentrate assaying 178.27 g/t Au with 90.25

% recovery was obtained by combining the concentrates. The same test was applied to an ore sample screened below 25 micron, there was no improvement (Figure 5.67, Figure 5.68 and Figure 5.69).

Comparison of the results of coal-oil assisted gold flotation with flotation and Knelson concentrator show that coal-oil assisted gold flotation gave better performance than flotation and multi gravity separation and it is possible to use it before cyanidation to reduce the amount of the ore feeding to the cyanidation plant by using this method.

CHAPTER SEVEN
CONCLUSIONS

The study has been conducted to develop coal-oil assisted gold flotation as an alternative gold recovery process. For this purpose, the experiments were done to reveal the important parameters and their possible effects on the efficiency of the process. The method is consisted of two different inferior processes; flotation and coal-oil agglomeration. It is important to determine the basic kinetics of these sub-processes before studying the coal-oil assisted gold flotation. Therefore, the mechanism of coal-oil agglomeration and flotation conditions of the specific gold ores were also investigated at the beginning of the study.

Theoretical and practical evaluation of coal-oil agglomeration were done by determining the viscosity and density values of the oils and measuring the contact angles of coal-oil-water system. The theoretical predictions were all overlapped with the practical results. Petroleum oils were determined to be the best agglomerating oils and vegetable oils gave very similar responses in agglomeration tests.

Flotation studies have been done to produce a bulk gold concentrate. The main aim is to determine the best reagent combination maximizing the gold recovery. It has been revealed that it is possible to produce gold concentrates from both Izmir-Efem Cukuru gold ore and Bergama-Ovacik gold ore samples with moderate recoveries. Nevertheless, the flotation rate of gold particles is inherently slow. It was easier to recover gold from Efem Cukuru ore samples due to the native gold content of the ore. Gold recovery from Bergama Ovacik ore by flotation is more complex because of the mode of the occurrence of gold, it is very fine free granules in quartz and some of the gold is hosted by electrum. The use of sodium sulphide in flotation of Ovacik ore achieved higher flotation recoveries at first, but concentrate ultimate recovery and flotation rate values tend to decrease by the further increases in the sodium sulphide dosage.

Coal-oil assisted flotation studies were completed using the best operating parameters obtained from the initial tests (coal-oil agglomeration and flotation studies). The method has given very successful results for both Izmir-Efem Cukuru gold ore and Bergama-Ovacik gold ore samples.

Comparison of the results of coal-oil assisted gold flotation with conventional gold recovery processes indicated that the performance of the method is higher than flotation and multi gravity separation of Bergama-Ovacik gold ore. It can also be proposed that it is possible to concentrate the gold content of the ore into 5% by weight of the feed material and it would reduce the cyanide and detoxification chemicals consumption.

The developed process needs small amounts of oil and coal particles in the flotation slurry to achieve higher gold recoveries, which makes the method economically viable. The results of this investigation are quite encouraging for the development of the coal-oil assisted gold flotation technology as an alternative process for the recovery of the gold.

REFERENCES

Akar, A., Seyrankaya, A., Akcil, A. (1998). Gold recovery by flotation and oil agglomeration. *VII International Mineral Processing Symposium*.

Akcil, A., & Akar, A. (2000). Some factors influencing the gold recovery by oil agglomeration. *Geosound*, (37), 171–178.

Akcil, A., (1998). *Recovery of gold from Izmir-Karsíyaka (Arapdagí) epithermal deposits*, Izmir: Dokuz Eylul University, Graduate School of Natural and Applied Sciences.

Aksoy B. S, Yarar B. (1989). *Natural hydrophobicity of native gold flakes and their flotation under different conditions*. Halifax: Proceedings of Complex Ores.

Allan, G. C., Woodcock, J. T. (2001). A review of the flotation of native gold and electrum. *Minerals Engineering*, 14, 931-962.

Alonso, M. I, Valdes, A. F., Martinez-Tarazona, R. M., Garcia, A. B. (1999). Coal recovery from coal fines cleaning wastes by agglomeration with vegetable oils: effects of oil type and concentration. *Fuel*, (78), 753-759.

Bulatovic, S. M., (1997). Flotation behavior of gold during processing of porphyry copper-gold ores and refractory gold-bearing sulphides. *Minerals Engineering*, (10), 895-908.

Bravo, S. V. C., Torem, M. L., Monte, M. B. M., Dutra, A. J. B., & Tondo, L.A. (2005). The influence of particle size and collector on the flotation of a very low grade auriferous ore. *Minerals Engineering*, (18), 459-461.

Bellamy, S. R., House, C. I., Townsend, I. G., & Veal, C.J. (1989). Recovery of fine gold by coal gold agglomeration, Reno: *Gold Forum on Technology and Practices—"World Gold '89"*.

BP Australia Limited, (1984). *Recovery of metal values from mineral ores by incorporation in coal–oil agglomerates*. Australia: Int. Patent pct/au 84/00071.

Burt R. (1997). The role of gravity concentration in modern processing plants. *Minerals Engineering*, (12), 1291-1300.

Calvez, J. P. S., Kim, M. J., Wong, P. L. M., Tran, T. (1998). Use of coal-oil-gold agglomerates for particulate gold recovery. Minerals Engineering, (11), 803-812.

Chan, S. K., Mozley, R. H. (1987). *Enhanced Gravity Separation for the Benefication of Fine and Ultrafines*, Cornwall: Richard Mozley Limited.

Cebeci, S., Sonmez, I. (2006). Application of the Box-Wilson experimental design method fo the spherical oil agglomeration of coal,. *Fuel*, (85), 289-297.

Coulter, T., Subasinghe, G. K. N. (2004). A mechanistic approach to modeling Knelson concentrators. *Minerals Engineering,* (18), 9-17.

Dai, Z., Fornasiero, D., Ralston, J. (2000). Particle-bubble collision models-a review. *Advances in Colloid and Interface Science*, (85), 231-256.

Eldorado Gold Corp. (n.d.). *The data about Izmir-Efem Cukuru gold ore*. Retrieved February 10, 2007, from http://www.eldoradogold.com/s/EfemcukuruProject.asp?ReportID=37159

Feng, D., & Aldrich, C., (1999). Effect of particle size on flotation performance of complex sulphide ores. *Minerals Engineering*, (7), 721-731.

Forrest, K., Yan, D., Dunne, R. (2001). Optimisation of gold recovery by selective gold flotation for copper-gold-pyrite ores. *Minerals Engineering*, (2), 227-241

Gaidarjiev, S., Stoev, S., Kireva, R. 1996. Coal-gold agglomeration of alluvial gold. 5th International Mineral Processing Symposium, 577-581.

Glembotski, V.A., Klassen, V.I., Plaksin, I.N., (1974). *Flotation*. New York: Primary Sources.

Gomez, C., Blazquez, M., L., Ballester, A. (1999). Bioleaching of a Spanish Complex Sulfide Ore Bulk Concentrate, *Minerals Engineering*, (12), 93-106.

Cyanamid Company. (1986). USA: *Mining Chemicals Handbook*.

Hoffmann, R. (1994). Winning Gold, *American Scientist*, (82), 15-17.

House, C.I., Townsend, I.G., Veal, C.J. (1988). *Coal Gold Agglomeration—Pilot Scale Retreatment of Tailings*. Perth: International Gold Conference.

Phana, C. M., Nguyena, A. V., Millerb, J. D., Evansa, G. M., & Jamesona G.J. (2003). Investigations of bubble–particle interactions. *International Journal of Mineral Processing*, (72), 239– 254.

Falcon Concentrators (n.d.). *The data about falcon concentrators*. Retrieved February 10, 2007, from http://www.concentrators.net/

Keller, D.V., & Burry, W. (1987). An investigation of a separation process involving liquid-water-coal systems. *Colloids and Surfaces*, 22, 37-50.

Khazheeva, Z., I., Urbazaeva, S., D., Nikiforov, K., A., & Zoltoev K., A. (2001). Selective flotation of gold-containing ore. *Journal of Mining Science*, 37, 643-645.

Klimpel, R., R., (1995). Technology Trends in Froth Flotation Chemistry, *Mining Engineering*, (47), 933-942.

Klimpel, R., R., (1999). Industrial Experiences in the Evaluation of Various Flotation Reagent Schemes for the Recovery of Gold, *Minerals & Metallurgical Processing*, (16), 1-11.

Klimpel, R.R. (2000). Optimizing the industrial flotation performance of sulfide minerals having some natural floatability. *International Journal of Mineral Processing*, (58), 77–84.

Knelson Gravity Soutions. (n.d.). *The data about Knelson concentrators*. Retrieved February 10, 2007, from http://www.knelsongravitysolutions.com/page418.htm

Laskowski, J.S., (2001). *Coal flotation and fine coal utilization*. Amsterdam: Elsevier Publications.

Longley, R. J., Katsikaros N., Hillman C., (2002). *New age gold plant. flowsheet for the treatment of high grade ores*. Perth: AusIMM - Metallurgical Plant Design and Operating Strategies Conference.

Malhotra, D., Harris, L. (1999). Review of plant practice of gold and silver ores. Parekkh, B.K., & Miller, J.D. (Eds.). *Advances in flotation technology* (167-181). SME: SME Press.

Marciano, A., Costa, L.S.N., Lins, F.F. (1994). Utilization of coal-oil agglomerates to recover gold particles. *Minerals Engineering*, (7), 1401-1409.

Mardsen, J., House, I. (1992). *Chemistry of Gold Extraction*, New York: Ellis Horwood Ltd. Publication.

McQuiston, F., W. (1985). *Properties of Gold and Recovery Methods*. Weiss, N. L. (Eds.). SME Mineral Processing Handbook, (18), 8-36.

Madencilik Ozel Ihtisas Komisyonu Metal Madenler Alt Komisyonu. (2000). *Degerli Metaller Çalisma Grubu Raporu*, DPT: 2623 - ÖİK: 634.

Monte, M.B.M., Lins, F.F., Olivieara, J., F., 1997. Selective flotation of gold from pyrite under oxidising conditions. *International Journal of Mineral Processing*, (51), 255-267.

Moses, L.B., Petersen, F.W., (2000). Flotation as a separation technique in the coal gold agglomeration process. *Minerals Engineering*, (3), 255-264.

Nguyen, A.V. 2003. New method and equations for determining attachment tenacity and particle size limit in flotation. *International Journal of Mineral Processing*, (68) 167–182.

Nguyen, A.V., Ralston, J., Schulze, H.J. (1998). On modelling of bubble–particle attachment probability in flotation. *International Journal of Mineral Processing*, (53), 225–249.

Ovacik Gold Mine. (n.d.). *The data about Bergama-Ovacik gold mine*. Retrieved February 10, 2007, from http://www.ovacik-altin.com/koze/koze/english.htm

Oyman, T., Minareci, F., Pişkin, Ö., (2003). Efem çukuru B-rich epithermal gold deposit (Izmir-Turkey). *Ore Geology Reviews*. (438), 1-19.

Yilmaz, H., Oyman, T., Arehart, G. B., Colakoglu A. R., & Billor, Z. (2006) Low-sulfidation type Au–Ag mineralization at Bergama, Izmir, Turkey. *Ore Geology Reviews*. doi:10.1016/j.oregeorev.2006.10.007.

Petruk, W., (2000). *Applied minerology in the mining industry*. Amsterdam: Elsevier Science.

Pyper, R., A., Hendrix, J., L., (1981). *Extraction of Gold from a Carling-Type Ore Using Thiourea*, Newyork: 110th Annual AIME Meeting.

Polat, M., Polat, H., Chander, S. (2003). Physical and chemical interactions in coal flotation. *International Journal of Mineral Processing*, (72), 199-213.

Sampson, M.I., Phillips, C.C., & Ball, A.S. 1999. Investigation of the attachment of thiobacillus ferrooxidans to mineral sulfides using scanning electron microscopy analysis. *Minerals Engineering*, (13), 643-656.

Sen, S. (2000). *An investigation on the beneficiation of gold ores by flotation and coal-oil-gold agglomerate flotation (MSc thesis)*. Izmir: Dokuz Eylul University, Graduate School of Natural and Applied Sciences.

Sen, S., Seyrankaya, A., Cilingir, Y. (2005). Coal oil assisted flotation for the gold recovery. *Minerals Engineering*, (18), 1086-1092.

Stewart, A., L., (1984). Gold Ore Processing Today-Part I, *International Mining,* (04), 21-31.

Stewart, A., L., (1990). Non-toxic Recovery, *Mining Engineering*, (11), 33-37.

Tanrıverdi, M., Mordoğan H., İpekoğlu Ü., & Şen S. (2000). *Thiosulfate: An Alternative Lixiviant for gold ore*, Antalya: 7th International Mineral Processing Symposium.

Teague, A.J., Van Deventer, J.S.J., Swaminathan, C., (1999a). A conceptual model for gold flotation. *Minerals Engineering*, (9), 1001-1019.

Teague, A.J., Van Deventer, J.S.J., & Swaminathan, C. (1999b). The effect of galvanic interaction on the behaviour of free and refractory gold during froth flotation. *International Journal of Mineral Processing*, (57), 243-263.

Torres, V.M., Chaves, A.P., & Meech, J.A. (1999). Process design for gold ores: A diagnostic approach. *Minerals Engineering*, (3), 245-254.

Ünal, İ., Aktaş, Z., (2001). Effect of various bridging liquids on coal fines agglomeration performance. *Fuel Processing Technology*, (69), 141-155.

Valderrama, L., Rubio, J., (1998). High intensity conditioning and the carrier flotation of gold fine particles. *International Journal of Mineral Processing*, (52), 273-285.

Veiga, M. M., Maxson, P. A., & Hylander, L. D. (2006). Origin and consumption of mercury in small-scale gold mining. *Journal of Cleaner Production*, (16), 436-447.

Wall N. C., Hughes-Narborough C., & Willey G, (1992). *Coal Gold Agglomeration: an Innovative Approach to the Recovery of Gold in*

Environmentally Sensitive Areas, Antalya: 4th International Mineral Processing Symposium.

Wan, R.Y., Brierley J.A. (1997). Thiosulfate leaching following biooxidation pretreatment for gold recovery from refractory carbonaceous - sulfidic ore, *Mining Engineering*, (49) 76-79.

Wills, B. A. (1985). *Mineral Processing Technology*, New York: Pergamon Press.

Yüce, A. E., Önal, G., Hendrix, J. L., (1994). Thiourea: *An alternative lixiviant for gold and silver extraction*, Rotterdam: Progress in mineral processing technology.

Wu, X. Q., Gochin, R. J., & Monhemius, A. J. (2004a). The adhesion of gold to oil-carbon agglomerates. Minerals Engineering, (17), 33-38.

Wu, X. Q., Gochin, R. J., Monhemius, A. J., (2004b). Modelling gold particle adhesion to oil–carbon agglomerates. *International Journal of Mineral Processing*, (74), 327– 336.

APPENDIX ONE

RESULTS OF THE COAL-OIL AGGLOMERATION TESTS

Table 1 Results of the agglomeration test done by using no oil

Size Fractions (micron)	Weight (%)	Ash (%)	Ash Cum. Undersize (%)	Ash Cum. Oversize (%)	Combustible Matter (%)	Recovery (com. matt.) Cum. Undersize (%)	Recovery (com. matt.) Cum. Oversize (%)	ND
(+ 1400)								
(1400-850)								
(850-600)								
(600-425)	1.32	9.44	13.64	9.44	90.56	100.00	1.38	0.01
(425-300)	1.93	7.55	13.70	8.31	92.45	98.62	3.44	0.02
(300-212)	8.89	15.44	13.82	13.54	84.56	96.56	12.14	0.10
(212-150)	10.40	13.82	13.66	13.67	86.18	87.86	22.52	0.17
(150-106)	7.39	13.46	13.64	13.62	86.54	77.48	29.93	0.17
(106-75)	11.60	14.27	13.65	13.80	85.73	70.07	41.44	0.37
(75-0)	58.48	13.53	13.53	13.64	86.47	58.56	100.00	0.78
	100.00	13.64						

Table 2 Results of the agglomeration test done by using sunflower oil

Size Fractions (micron)	Weight (%)	Ash (%)	Ash Cum. Undersize (%)	Ash Cum. Oversize (%)	Combustible Matter (%)	Recovery (com. matt.) Cum. Undersize (%)	Recovery (com. matt.) Cum. Oversize (%)	ND
(+ 1400)	4.63	10.03	13.08	10.03	89.97	100.00	4.80	0.01
(1400-850)	4.36	10.47	13.23	10.24	89.53	95.20	9.29	0.01
(850-600)	1.98	10.44	13.36	10.28	89.56	90.71	11.33	0.01
(600-425)	9.77	10.39	13.43	10.33	89.61	88.67	21.40	0.06
(425-300)	12.63	11.15	13.80	10.64	88.85	78.60	34.31	0.10
(300-212)	21.76	12.56	14.31	11.40	87.44	65.69	56.20	0.25
(212-150)	15.35	13	15.15	11.75	87.00	43.80	71.57	0.25
(150-106)	8.57	15.06	16.27	12.11	84.94	28.43	79.94	0.19
(106-75)	10.66	15.34	16.77	12.49	84.66	20.06	90.32	0.34
(75-0)	10.29	18.25	18.25	13.08	81.75	9.68	100.00	0.14
	100.00	13.08						

Table 3 Results of the agglomeration test done by using kerosene

Size Fractions (micron)	Weight (%)	Ash (%)	Ash Cum. Undersize (%)	Ash Cum. Oversize (%)	Combustible Matter (%)	Recovery (com. matt.) Cum. Undersize (%)	Recovery (com. matt.) Cum. Oversize (%)	ND
(+ 1400)	13.19	11.31	13.20	11.31	88.69	100.00	13.48	0.02
(1400-850)	1.61	11.79	13.49	11.36	88.21	86.52	15.11	0.00
(850-600)	37.95	11.64	13.52	11.56	88.36	84.89	53.75	0.15
(600-425)	20.53	11.82	15.03	11.63	88.18	46.25	74.61	0.12
(425-300)	9.64	12.80	17.50	11.77	87.20	25.39	84.29	0.08
(300-212)	6.69	14.38	20.15	11.96	85.62	15.71	90.89	0.08
(212-150)	2.72	18.55	23.87	12.16	81.45	9.11	93.45	0.04
(150-106)	1.34	24.47	25.76	12.33	75.53	6.55	94.61	0.03
(106-75)	1.18	25.30	26.04	12.50	74.70	5.39	95.63	0.04
(75-0)	5.14	26.20	26.20	13.20	73.80	4.37	100.00	0.07
	100.00	13.20						

Table 4 Results of the agglomeration test done by using diesel oil

Size Fractions (micron)	Weight (%)	Ash (%)	Ash Cum. Undersize (%)	Ash Cum. Oversize (%)	Combustible Matter (%)	Recovery (com. matt.) Cum. Undersize (%)	Recovery (com. matt.) Cum. Oversize (%)	ND
(+ 1400)	4.34	11.57	13.59	11.57	88.43	100.00	4.45	0.01
(1400-850)	5.94	12.34	13.68	12.01	87.66	95.55	10.47	0.01
(850-600)	4.41	11.66	13.77	11.91	88.34	89.53	14.98	0.02
(600-425)	15.99	11.70	13.87	11.80	88.30	85.02	31.32	0.09
(425-300)	16.82	10.93	14.38	11.49	89.07	68.68	48.66	0.13
(300-212)	11.58	11.55	15.48	11.50	88.45	51.34	60.52	0.13
(212-150)	19.83	13.38	16.60	11.97	86.62	39.48	80.39	0.32
(150-106)	5.17	16.53	19.62	12.25	83.47	19.61	85.39	0.12
(106-75)	5.39	20.07	20.63	12.72	79.93	14.61	90.37	0.17
(75-0)	10.52	20.91	20.91	13.59	79.09	9.63	100.00	0.14
	100.00	13.59						

Table 5 Results of the agglomeration test done by using AP-704

Size Fractions (micron)	Weight (%)	Ash (%)	Ash Cum. Undersize (%)	Ash Cum. Oversize (%)	Combustible Matter (%)	Recovery (com. matt.) Cum. Undersize (%)	Recovery (com. matt.) Cum. Oversize (%)	ND
(+ 1400)	7.00	11.11	13.22	11.11	88.89	100.00	7.17	0.01
(1400-850)	16.35	11.3	13.38	11.24	88.70	92.83	23.89	0.03
(850-600)	6.75	11.3	13.82	11.26	88.70	76.11	30.78	0.03
(600-425)	11.76	11.7	14.06	11.38	88.30	69.22	42.75	0.07
(425-300)	8.84	12.1	14.54	11.51	87.90	57.25	51.71	0.07
(300-212)	16.78	13.92	14.98	12.11	86.08	48.29	68.35	0.19
(212-150)	12.82	14.83	15.52	12.54	85.17	31.65	80.93	0.21
(150-106)	6.60	15.35	15.97	12.75	84.65	19.07	87.37	0.15
(106-75)	7.11	15.78	16.29	12.98	84.22	12.63	94.27	0.23
(75-0)	5.99	16.90	16.90	13.22	83.10	5.73	100.00	0.08
	100.00	13.22						

Table 6 Results of the agglomeration test done by using corn oil

Size Fractions (micron)	Weight (%)	Ash (%)	Ash Cum. Undersize (%)	Ash Cum. Oversize (%)	Combustible Matter (%)	Recovery (com. matt.) Cum. Undersize (%)	Recovery (com. matt.) Cum. Oversize (%)	ND
(+ 1400)	4.11	10.03	13.03	10.03	89.97	100.00	4.25	0.01
(1400-850)	2.69	10.07	13.16	10.05	89.93	95.75	7.04	0.00
(850-600)	1.58	9.98	13.25	10.04	90.02	92.96	8.67	0.01
(600-425)	6.61	10.31	13.31	10.16	89.69	91.33	15.48	0.04
(425-300)	5.64	9.57	13.54	10.00	90.43	84.52	21.35	0.05
(300-212)	23.30	12.24	13.82	11.19	87.76	78.65	44.86	0.26
(212-150)	14.61	12.78	14.48	11.59	87.22	55.14	59.52	0.24
(150-106)	10.65	13.71	15.08	11.91	86.29	40.48	70.09	0.24
(106-75)	10.70	14.70	15.55	12.29	85.30	29.91	80.58	0.35
(75-0)	20.11	16.00	16.00	13.03	84.00	19.42	100.00	0.27
	100.00	13.03						

Table 7 Results of the agglomeration test done by using olive-pomace oil

Size Fractions (micron)	Weight (%)	Ash (%)	Ash Cum. Undersize (%)	Ash Cum. Oversize (%)	Combustible Matter (%)	Recovery (com. matt.) Cum. Undersize (%)	Recovery (com. matt.) Cum. Oversize (%)	ND
(+ 1400)								
(1400-850)								
(850-600)								
(600-425)	3.49	9.29	13.01	9.29	90.71	100.00	3.64	0.02
(425-300)	2.37	8.06	13.14	8.79	91.94	96.36	6.14	0.02
(300-212)	14.27	12.38	13.27	11.33	87.62	93.86	20.52	0.16
(212-150)	15.18	12.07	13.43	11.65	87.93	79.48	35.86	0.24
(150-106)	9.49	12.98	13.75	11.93	87.02	64.14	45.35	0.22
(106-75)	15.14	13.21	13.88	12.26	86.79	54.65	60.46	0.49
(75-0)	40.06	14.14	14.14	13.01	85.86	39.54	100.00	0.53
	100.00	13.01						

Table 8 Results of the agglomeration test done by using olive oil

Size Fractions (micron)	Weight (%)	Ash (%)	Ash Cum. Undersize (%)	Ash Cum. Oversize (%)	Combustible Matter (%)	Recovery (com. matt.) Cum. Undersize (%)	Recovery (com. matt.) Cum. Oversize (%)	ND
(+ 1400)	4.95	10.23	13.03	10.23	89.77	100.00	5.11	0.01
(1400-850)	5.01	10.14	13.18	10.19	89.86	94.89	10.29	0.01
(850-600)	2.19	9.78	13.35	10.11	90.22	89.71	12.55	0.01
(600-425)	8.71	10.36	13.44	10.22	89.64	87.45	21.53	0.05
(425-300)	6.69	10.60	13.78	10.31	89.40	78.47	28.41	0.05
(300-212)	17.65	11.43	14.07	10.75	88.57	71.59	46.38	0.20
(212-150)	14.36	12.80	14.92	11.24	87.20	53.62	60.77	0.23
(150-106)	10.58	14.05	15.67	11.66	85.95	39.23	71.22	0.24
(106-75)	10.32	15.67	16.25	12.18	84.33	28.78	81.23	0.33
(75-0)	19.56	16.55	16.55	13.03	83.45	18.77	100.00	0.26
	100.00	13.03						

Table 9 Results of the agglomeration test done by using cotton oil

Size Fractions (micron)	Weight (%)	Ash (%)	Ash Cum. Undersize (%)	Ash Cum. Oversize (%)	Combustible Matter (%)	Recovery (com. matt.) Cum. Undersize (%)	Recovery (com. matt.) Cum. Oversize (%)	ND
(+ 1400)	5.16	9.98	13.13	9.98	90.02	100.00	5.35	0.01
(1400-850)	2.77	10.01	13.31	9.99	89.99	94.65	8.22	0.01
(850-600)	1.05	10.43	13.41	10.04	89.57	91.78	9.30	0.00
(600-425)	7.07	10.48	13.44	10.23	89.52	90.70	16.59	0.04
(425-300)	11.02	11.00	13.69	10.55	89.00	83.41	27.88	0.09
(300-212)	12.93	12.94	14.10	11.32	87.06	72.12	40.84	0.15
(212-150)	18.15	13.20	14.34	11.91	86.80	59.16	58.98	0.29
(150-106)	8.37	14.25	14.84	12.20	85.75	41.02	67.24	0.19
(106-75)	13.60	14.49	14.99	12.59	85.51	32.76	80.63	0.44
(75-0)	19.87	15.33	15.33	13.13	84.67	19.37	100.00	0.26
	100.00	13.13						

APPENDIX TWO
FLOTATION STUDIES OF IZMIR-EFEM CUKURU GOLD ORE

Table 1 The effect of 150 g/t $CuSO_4$ addition on the time-recovery curve for gold (collector combination: KAX-Aerofloat 208 and Aero 6697)

Product	Time (min)	Weight (%)	Cum. Weight (%)	Grade, Au (g/t)	Recovery, Au (%)	Cum. Grade, Au (%)	Cum. Recovery, Au (%)
Concentrates	0.5	1.7502	1.7502	30.7449	7.8997	30.7449	7.8997
	1	1.6054	3.3556	68.1240	16.0556	48.6278	23.9553
	2	1.8643	5.2199	52.6562	14.4116	50.0666	38.3668
	4	1.6636	6.8835	46.7702	11.4225	49.2699	49.7894
	6	1.4158	8.2993	11.2992	2.3486	42.7923	52.1379
	8	0.5874	8.8867	12.1326	1.0462	40.7659	53.1841
Tailing		91.1133	100.0000	3.5000	46.8159		
Feed		100.0000		6.8117			

Table 2 The effect of 70 g/t Na_2S addition on the time-recovery curve for gold (collector combination: KAX-Aerofloat 208 and Aero 6697)

Product	Time (min)	Weight (%)	Cum. Weight (%)	Grade, Au (g/t)	Recovery, Au (%)	Cum. Grade, Au (%)	Cum. Recovery, Au (%)
Concentrates	0.5	1.9563	1.9563	33.3413	9.6746	33.3413	9.6746
	1	1.6691	3.6255	75.8487	18.7775	52.9111	28.4521
	2	1.9370	5.5625	62.5047	17.9578	56.2519	46.4099
	4	1.8377	7.4001	47.6028	12.9749	54.1041	59.3848
	6	1.3249	8.7250	16.5209	3.2464	48.3972	62.6312
	8	0.6096	9.3346	21.1126	1.9090	46.6153	64.5403
Tailing		90.6654	100.0000	2.6369	35.4597		
Feed		100.0000		6.7421			

Table 3 The effect of 150 g/t $CuSO_4$ addition on the time-recovery curve for gold (collector combination: KAX-Aerofloat 208 and Aero 412)

Product	**Time (min)**	**Weight (%)**	**Cum. Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**	**Cum. Grade, Au (%)**	**Cum. Recovery, Au (%)**
Concentrates	0.5	1.1449	1.1449	87.7722	14.1584	87.7722	14.1584
	1	0.9218	2.0667	89.4941	11.6237	88.5402	25.7822
	2	0.9994	3.0661	82.0388	11.5521	86.4211	37.3342
	4	1.8007	4.8668	68.6209	17.4100	79.8351	54.7442
	6	0.9132	5.7800	43.5378	5.6019	74.1003	60.3462
	8	0.6933	6.4733	20.2327	1.9765	68.3306	62.3227
Tailing		93.5267	100.0000	2.8592	37.6773		
Feed		100.0000		7.0973			

Table 4 The effect of 70 g/t Na_2S addition on the time-recovery curve for gold (collector combination: KAX-Aerofloat 208 and Aero 412)

Product	**Time (min)**	**Weight (%)**	**Cum. Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**	**Cum. Grade, Au (%)**	**Cum. Recovery, Au (%)**
Concentrates	0.5	0.9985	0.9985	103.1964	15.1724	103.1964	15.1724
	1	0.9284	1.9269	180.2316	24.6372	140.3120	39.8095
	2	0.8576	2.7845	119.1734	15.0492	133.8014	54.8587
	4	1.3603	4.1448	33.4160	6.6931	100.8555	61.5518
	6	1.0978	5.2426	14.6067	2.3611	82.7952	63.9129
	8	0.7613	6.0039	10.0740	1.1292	73.5745	65.0421
Tailing		93.9961	100.0000	2.5258	34.9579		
Feed		100.0000		6.7915			

Table 5 The effect of 150 g/t $CuSO_4$ addition on the time-recovery curve for gold (collector combination: KAX-Aerofloat 208 and Aerophine 3418A)

Product	**Time (min)**	**Weight (%)**	**Cum. Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**	**Cum. Grade, Au (%)**	**Cum. Recovery, Au (%)**
Concentrates	0.5	0.7908	0.7908	259.2867	17.8495	259.2867	17.8495
	1	0.7350	1.5258	223.9501	14.3287	242.2648	32.1782
	2	0.9720	2.4978	164.6952	13.9357	212.0789	46.1138
	4	1.0801	3.5779	103.2741	9.7108	179.2316	55.8246
	6	0.9105	4.4884	52.0635	4.1267	153.4347	59.9513
	8	0.8626	5.3510	34.6750	2.6037	134.2907	62.5550
Tailing		94.6490	100.0000	4.5446	37.4450		
Feed		100.0000		11.4873			

Table 6 The effect of 70 g/t Na_2S addition on the time-recovery curve for gold (collector combination: KAX-Aerofloat 208 and Aerophine 3418A)

Product	**Time (min)**	**Weight (%)**	**Cum. Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**	**Cum. Grade, Au (%)**	**Cum. Recovery, Au (%)**
Concentrates	0.5	1.0242	1.0242	158.9066	26.4118	158.9066	26.4118
	1	1.0434	2.0676	147.1942	24.9244	152.9959	51.3362
	2	1.4893	3.5569	60.9429	14.7292	114.4529	66.0654
	4	2.0386	5.5955	17.1484	5.6734	79.0015	71.7387
	6	1.3947	6.9902	13.3542	3.0226	65.9034	74.7613
	8	1.2672	8.2574	9.2155	1.8951	57.2041	76.6565
Tailing		91.7426	100.0000	1.5679	23.3435		
Feed		100.0000		6.1620			

Table 7 The effect of 150 g/t $CuSO_4$ addition on the time-recovery curve for gold (collector combination: KAX-Aerofloat 208 and Aerofloat 242)

Product	**Time (min)**	**Weight (%)**	**Cum. Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**	**Cum. Grade, Au (%)**	**Cum. Recovery, Au (%)**
Concentrates	0.5	0.7784	0.7784	196.0894	13.5585	196.0894	13.5585
	1	0.3823	1.1608	327.3640	11.1180	239.3297	24.6764
	2	0.6243	1.7850	246.3674	13.6618	241.7910	38.3382
	4	1.0934	2.8784	148.7735	14.4495	206.4573	52.7877
	6	0.9922	3.8707	70.5432	6.2175	171.6160	59.0052
	8	0.8350	4.7056	43.0899	3.1959	148.8102	62.2011
Tailing		95.2944	100.0000	4.4654	37.7989		
Feed		100.0000		11.2578			

Table 8 The effect of 70 g/t Na_2S addition on the time-recovery curve for gold (collector combination: KAX-Aerofloat 208 and Aerofloat 242)

Product	**Time (min)**	**Weight (%)**	**Cum. Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**	**Cum. Grade, Au (%)**	**Cum. Recovery, Au (%)**
Concentrates	0.5	0.9623	0.9623	185.6356	21.6990	185.6356	21.6990
	1	0.6024	1.5647	166.3846	12.1752	178.2240	33.8741
	2	0.8504	2.4150	120.9071	12.4891	158.0421	46.3633
	4	1.5044	3.9194	84.3777	15.4195	129.7674	61.7828
	6	1.2203	5.1398	29.6526	4.3956	105.9972	66.1784
	8	1.0496	6.1894	24.7623	3.1572	92.2212	69.3356
Tailing		93.8106	100.0000	2.6909	30.6644		
Feed		100.0000		8.2323			

APPENDIX THREE
FLOTATION STUDIES OF BERGAMA-OVACIK GOLD ORE

Table 1 The effect of 150 g/t $CuSO_4$ addition on the time-recovery and time-grade curves for gold (collector combination: KAX-Aerofloat 208 and Aero 6697)

Product	**Time (min)**	**Weight (%)**	**Cum. Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**	**Cum. Grade, Au (%)**	**Cum. Recovery, Au (%)**
Concentrates	0.50	0.2941	0.2941	578.6325	13.2334	578.6325	13.2334
	1.00	0.4374	0.7315	142.4138	4.8438	317.8007	18.0772
	2.00	0.6686	1.4001	207.5188	10.7900	265.1346	28.8673
	4.00	1.2317	2.6318	108.3673	10.3796	191.7670	39.2468
	6.00	1.0708	3.7026	39.9648	3.3279	147.8649	42.5747
	8.00	0.6686	4.3712	24.6429	1.2813	129.0167	43.8561
Tailing		95.6288	100.0000	7.5498	56.1439		
Feed		100.0000		12.8593			

Table 2 The effect of 70 g/t Na_2S addition on the time-recovery and time-grade curves for gold (collector combination: KAX-Aerofloat 208 and Aero 6697)

Product	**Time (min)**	**Weight (%)**	**Cum. Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**	**Cum. Grade, Au (%)**	**Cum. Recovery, Au (%)**
Concentrates	0.50	0.3042	0.3042	437.2521	13.1333	437.2521	13.1333
	1.00	0.2592	0.5633	302.2444	7.7345	375.1434	20.8678
	2.00	0.7949	1.3582	166.8699	13.0971	253.2555	33.9649
	4.00	1.8469	3.2051	51.2650	9.3490	136.8616	43.3139
	6.00	1.5408	4.7459	25.5034	3.8800	100.7090	47.1939
	8.00	1.4388	6.1847	14.4215	2.0489	80.6348	49.2429
Tailing		93.8153	100.0000	5.4793	50.7571		
Feed		100.0000		10.1274			

Table 3 The effect of 150 g/t $CuSO_4$ addition on the time-recovery and time-grade curves for gold (collector combination: KAX-Aerofloat 208 and Aero 412)

Product	**Time (min)**	**Weight (%)**	**Cum. Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**	**Cum. Grade, Au (%)**	**Cum. Recovery, Au (%)**
Concentrates	0.50	0.1768	0.1768	569.7917	9.6786	569.7917	9.6786
	1.00	0.2062	0.3830	361.7857	7.1696	457.7885	16.8482
	2.00	0.4394	0.8224	254.3855	10.7426	349.1045	27.5908
	4.00	1.3110	2.1334	120.4120	15.1696	208.5731	42.7604
	6.00	1.0778	3.2112	54.6697	5.6621	156.9190	48.4225
	8.00	0.9525	4.1637	31.7784	2.9089	128.2901	51.3314
Tailing		95.8363	100.0000	5.2846	48.6686		
Feed		100.0000		10.4061			

Table 4 The effect of 70 g/t Na_2S addition on the time-recovery and time-grade curves for gold (collector combination: KAX-Aerofloat 208 and Aero 412)

Product	**Time (min)**	**Weight (%)**	**Cum. Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**	**Cum. Grade, Au (%)**	**Cum. Recovery, Au (%)**
Concentrates	0.50	0.5964	0.5964	311.0870	18.4088	311.0870	18.4088
	1.00	0.5809	1.1773	219.4196	12.6456	265.8590	31.0544
	2.00	1.0658	2.2430	102.7981	10.8703	188.3815	41.9247
	4.00	2.1082	4.3512	39.7540	8.3155	116.3707	50.2402
	6.00	1.5584	5.9097	37.7371	5.8352	95.6341	56.0754
	8.00	1.1695	7.0791	19.3459	2.2448	83.0311	58.3202
Tailing		92.9209	100.0000	4.5208	41.6798		
Feed		100.00		12.78			

Table 5 The effect of 150 g/t $CuSO_4$ addition on the time-recovery and time-grade curves for gold (collector combination: KAX-Aerofloat 208 and Aerofloat 242)

Product	**Time (min)**	**Weight (%)**	**Cum. Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**	**Cum. Grade, Au (%)**	**Cum. Recovery, Au (%)**
Concentrates	0.50	0.2364	0.2364	274.6296	6.4501	274.6296	6.4501
	1.00	0.3087	0.5451	178.9362	5.4867	220.4418	11.9367
	2.00	0.5582	1.1034	213.7255	11.8519	217.0437	23.7887
	4.00	1.3989	2.5022	62.8326	8.7313	130.8311	32.5200
	6.00	1.0749	3.5771	50.2851	5.3693	106.6279	37.8892
	8.00	0.7093	4.2864	43.0247	3.0315	96.1032	40.9207
Tailing		95.7136	100.0000	6.2137	59.0793		
Feed		100.0000		10.0668			

Table 6 The effect of 70 g/t Na_2S addition on the time-recovery and time-grade curves for gold (collector combination: KAX-Aerofloat 208 and Aerofloat 242)

Product	**Time (min)**	**Weight (%)**	**Cum. Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**	**Cum. Grade, Au (%)**	**Cum. Recovery, Au (%)**
Concentrates	0.50	0.3456	0.3456	365.5238	12.5113	365.5238	12.5113
	1.00	0.2462	0.5918	417.9150	10.1917	387.3212	22.7030
	2.00	0.6080	1.1998	209.8570	12.6364	297.3952	35.3394
	4.00	1.5668	2.7666	51.9543	8.0623	158.3946	43.4017
	6.00	1.3089	4.0755	24.6294	3.1929	115.4340	46.5946
	8.00	1.2560	5.3315	15.3843	1.9137	91.8645	48.5083
Tailing		94.6685	100.0000	5.4917	51.4917		
Feed		100.0000		10.0967			

Table 7 The effect of 150 g/t $CuSO_4$ addition on the time-recovery and time-grade curves for gold (collector combination: KAX-Aerofloat 208 and Aerophine 3418A)

Product	**Time (min)**	**Weight (%)**	**Cum. Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**	**Cum. Grade, Au (%)**	**Cum. Recovery, Au (%)**
Concentrates	0.50	0.5075	0.5075	214.3750	13.6199	214.3750	13.6199
	1.00	0.9622	1.4698	75.8571	9.1369	123.6906	22.7568
	2.00	0.8544	2.3242	118.3911	12.6616	121.7425	35.4185
	4.00	1.3894	3.7136	58.7747	10.2222	98.1834	45.6407
	6.00	1.2139	4.9275	41.0017	6.2302	84.0966	51.8709
	8.00	1.1293	6.0568	33.2772	4.7041	74.6212	56.5750
Tailing		93.9432	100.0000	3.6928	43.4250		
Feed		100.0000		7.9887			

Table 8 The effect of 70 g/t Na_2S addition on the time-recovery and time-grade curves for gold (collector combination: KAX-Aerofloat 208 and Aerophine 3418A)

Product	**Time (min)**	**Weight (%)**	**Cum. Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**	**Cum. Grade, Au (%)**	**Cum. Recovery, Au (%)**
Concentrates	0.50	0.4835	0.4835	382.8638	17.6633	382.8638	17.6633
	1.00	0.5516	1.0351	234.3621	12.3351	303.7281	29.9984
	2.00	0.8672	1.9023	146.7277	12.1401	232.1599	42.1385
	4.00	1.4052	3.3075	87.2375	11.6961	170.5903	53.8346
	6.00	1.3393	4.6468	39.4407	5.0402	132.7894	58.8748
	8.00	1.1691	5.8159	21.8058	2.4324	110.4801	61.3071
Tailing		94.1841	100.0000	4.3057	38.6929		
Feed		100.0000		10.4806			

Table 9 The time-recovery and time-grade curves for gold with 20 g/t Na_2S

Product	Time (min)	Weight (%)	Cum. Weight (%)	Grade, Au (g/t)	Recovery, Au (%)	Cum. Grade, Au (%)	Cum. Recovery, Au (%)
Concentrates	0.50	0.37	0.37	464.29	13.00	464.29	13.00
	1.00	0.61	0.99	213.84	9.86	308.48	22.86
	2.00	1.02	2.01	150.74	11.60	228.12	34.46
	4.00	1.54	3.55	99.10	11.48	172.13	45.93
	6.00	0.99	4.54	71.24	5.31	150.11	51.24
	8.00	0.54	5.09	70.72	2.88	141.65	54.12
Tailing		94.91	100.00	6.43	45.88		
Feed		100.00		13.31			

Table 10 The time-recovery and time-grade curves for gold with 40 g/t Na_2S

Product	Time (min)	Weight (%)	Cum. Weight (%)	Grade, Au (g/t)	Recovery, Au (%)	Cum. Grade, Au (%)	Cum. Recovery, Au (%)
Concentrates	0.50	0.48	0.48	347.10	15.54	347.10	15.54
	1.00	0.43	0.91	242.49	9.70	297.73	25.24
	2.00	1.07	1.98	161.22	16.00	224.10	41.25
	4.00	1.39	3.37	106.45	13.74	175.59	54.99
	6.00	1.30	4.67	34.33	4.14	136.35	59.13
	8.00	0.97	5.63	31.23	2.81	118.29	61.93
Tailing		94.37	100.00	4.34	38.07		
Feed		100.00		10.76			

Table 11 The time-recovery and time-grade curves for gold with 70 g/t Na_2S

Product	**Time (min)**	**Weight (%)**	**Cum. Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**	**Cum. Grade, Au (%)**	**Cum. Recovery, Au (%)**
Concentrates	0.50	0.48	0.48	382.86	17.66	382.86	17.66
	1.00	0.55	1.04	234.36	12.34	303.73	30.00
	2.00	0.87	1.90	146.73	12.14	232.16	42.14
	4.00	1.41	3.31	87.24	11.70	170.59	53.83
	6.00	1.34	4.65	39.44	5.04	132.79	58.87
	8.00	1.17	5.82	21.81	2.43	110.48	61.31
Tailing		94.18	100.00	4.31	38.69		
Feed		100.00		10.48			

Table 12 The time-recovery and time-grade curves for gold with 85 g/t Na_2S

Product	**Time (min)**	**Weight (%)**	**Cum. Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**	**Cum. Grade, Au (%)**	**Cum. Recovery, Au (%)**
Concentrates	0.50	0.36	0.36	403.85	18.14	403.85	18.14
	1.00	0.40	0.76	254.03	12.56	325.35	30.70
	2.00	0.68	1.43	164.58	13.96	249.26	44.66
	4.00	1.42	2.86	62.87	11.16	156.48	55.82
	6.00	1.22	4.08	29.59	4.51	118.47	60.34
	8.00	1.05	5.13	20.17	2.64	98.38	62.98
Tailing		94.87	100.00	3.13	37.02		
Feed		100.00		8.01			

Table 13 The time-recovery and time-grade curves for gold with 100 g/t Na_2S

Product	**Time (min)**	**Weight (%)**	**Cum. Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**	**Cum. Grade, Au (%)**	**Cum. Recovery, Au (%)**
Concentrates	0.50	0.34	0.34	464.96	14.98	464.96	14.98
	1.00	0.41	0.74	320.45	12.52	385.77	27.49
	2.00	0.54	1.28	268.47	13.82	336.57	41.31
	4.00	1.13	2.41	92.82	10.01	222.61	51.32
	6.00	1.18	3.59	54.83	6.19	167.47	57.51
	8.00	1.04	4.62	31.72	3.15	136.99	60.66
Tailing		95.38	100.00	4.31	39.34		
Feed		100.00		10.44			

Table 14 The time-recovery and time-grade curves for gold with 150 g/t Na_2S

Product	**Time (min)**	**Weight (%)**	**Cum. Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**	**Cum. Grade, Au (%)**	**Cum. Recovery, Au (%)**
Concentrates	0.50	0.30	0.30	416.67	12.83	416.67	12.83
	1.00	0.35	0.66	323.14	11.61	366.31	24.44
	2.00	0.52	1.18	225.27	11.96	303.79	36.40
	4.00	1.00	2.18	95.50	9.63	208.60	46.03
	6.00	1.03	3.21	71.51	7.45	164.64	53.48
	8.00	1.00	4.21	28.74	2.91	132.32	56.40
Tailing		95.79	100.00	4.49	43.60		
Feed		100.00		9.87			

Table 15 The time-recovery and time-grade curves for gold with 350 g/t Na_2S

Product	Time (min)	Weight (%)	Cum. Weight (%)	Grade, Au (g/t)	Recovery, Au (%)	Cum. Grade, Au (%)	Cum. Recovery, Au (%)
Concentrates	0.50	0.23	0.23	555.56	11.18	555.56	11.18
	1.00	0.20	0.43	506.47	8.75	532.88	19.93
	2.00	0.60	1.03	190.96	10.09	332.66	30.02
	4.00	1.07	2.09	92.68	8.68	210.41	38.70
	6.00	1.15	3.25	71.36	7.22	161.05	45.93
	8.00	0.98	4.23	49.55	4.28	135.13	50.21
Tailing		95.77	100.00	5.92	49.79		
Feed		100.00		11.38			

Table 16 The time-recovery and time-grade curves for gold with 750 g/t Na_2S

Product	Time (min)	Weight (%)	Cum. Weight (%)	Grade, Au (g/t)	Recovery, Au (%)	Cum. Grade, Au (%)	Cum. Recovery, Au (%)
Concentrates	0.50	0.27	0.27	409.45	11.12	409.45	11.12
	1.00	0.15	0.42	344.00	5.15	386.19	16.26
	2.00	0.74	1.16	132.12	9.86	223.79	26.12
	4.00	0.78	1.93	135.56	10.63	188.32	36.75
	6.00	0.81	2.74	83.59	6.86	157.32	43.61
	8.00	0.59	3.33	63.09	3.75	140.69	47.36
Tailing		96.67	100.00	5.39	52.64		
Feed		100.00		9.90			

Table 17 The time-recovery and time-grade curves for gold with 75 g/t $CuSO_4$

Product	Time (min)	Weight (%)	Cum. Weight (%)	Grade, Au (g/t)	Recovery, Au (%)	Cum. Grade, Au (%)	Cum. Recovery, Au (%)
Concentrates	0.50	0.42	0.42	282.11	14.97	282.11	14.97
	1.00	0.46	0.88	172.98	10.18	224.72	25.15
	2.00	0.80	1.68	96.02	9.81	163.30	34.96
	4.00	1.32	3.00	87.72	14.70	130.11	49.66
	6.00	1.27	4.27	31.28	5.04	100.79	54.70
	8.00	0.96	5.23	31.01	3.77	88.01	58.47
Tailing		94.77	100.00	3.45	41.53		
Feed		100.00		7.87			

Table 18 The time-recovery and time-grade curves for gold with 150 g/t $CuSO_4$

Product	Time (min)	Weight (%)	Cum. Weight (%)	Grade, Au (g/t)	Recovery, Au (%)	Cum. Grade, Au (%)	Cum. Recovery, Au (%)
Concentrates	0.50	0.5075	0.5075	214.3750	13.6199	214.3750	13.6199
	1.00	0.9622	1.4698	75.8571	9.1369	123.6906	22.7568
	2.00	0.8544	2.3242	118.3911	12.6616	121.7425	35.4185
	4.00	1.3894	3.7136	58.7747	10.2222	98.1834	45.6407
	6.00	1.2139	4.9275	41.0017	6.2302	84.0966	51.8709
	8.00	1.1293	6.0568	33.2772	4.7041	74.6212	56.5750
Tailing		93.9432	100.0000	3.6928	43.4250		
Feed		100.0000		7.9887			

Table 19 The time-recovery and time-grade curves for gold with 300 g/t $CuSO_4$

Product	**Time (min)**	**Weight (%)**	**Cum. Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**	**Cum. Grade, Au (%)**	**Cum. Recovery, Au (%)**
Concentrates	0.50	0.73	0.73	139.32	13.50	139.32	13.50
	1.00	0.55	1.28	142.86	10.50	140.85	23.99
	2.00	0.78	2.06	91.99	9.49	122.42	33.48
	4.00	1.60	3.65	60.61	12.89	95.39	46.37
	6.00	1.15	4.80	33.93	5.20	80.66	51.57
	8.00	0.75	5.56	31.36	3.15	73.96	54.72
Tailing		94.44	100.00	3.60	45.28		
Feed		100.00		7.51			

APPENDIX FOUR

COAL-OIL ASSISTED GOLD FLOTATION STUDIES OF BERGAMA-OVACIK GOLD ORE

Table 1 The coal-oil assisted gold flotation concentrates obtained with different agglomerating oils (Agglomerates + oil Emulsion)

AP 704	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	8.08	61.72	68.18
Tailing	91.92	2.53	31.82
Total	100.00	7.32	100.00
Sunflower Oil	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	3.99	174.59	74.37
Tailing	96.01	2.50	25.63
Total	100.00	9.37	100.00
Olive-Pom.Oil	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	8.24	82.51	75.05
Tailing	91.76	2.46	24.95
Total	100.00	9.05	
Diesel Oil	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	4.31	144.50	81.22
Tailing	95.69	1.50	18.78
Total	100.00	7.67	
Cotton Oil	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	5.19	109.83	75.79
Tailing	94.81	1.92	24.21
Total	100.00	7.53	
Olive Oil	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	3.59	170.83	74.09
Tailing	96.41	2.23	25.91
Total	100.00	8.28	
Kerosene	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	4.67	127.65	80.56
Tailing	95.33	1.51	19.44
Total	100.00	7.40	
Corn Oil	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	4.67	127.65	80.56
Tailing	95.33	1.51	19.44
Total	100.00	7.40	

Table 2 The coal-oil assisted gold flotation concentrates obtained with different agglomerating oils (Agglomerates + oil Emulsion) – experiment done for the determination of reproducibility.

AP 704	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	8.85	51.66	67.58
Tailing	91.15	2.41	32.42
Total	100.00	6.77	
Sunflower Oil	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	4.53	122.80	68.43
Tailing	95.47	2.69	31.57
Total	100.00	8.13	
Olive-Pom.Oil	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	7.53	70.48	71.71
Tailing	92.47	2.26	28.29
Total	100.00	7.40	
Diesel Oil	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	4.62	113.10	67.05
Tailing	95.38	2.69	32.95
Total	100.00	7.80	
Cotton Oil	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	4.60	122.83	67.67
Tailing	95.40	2.83	32.33
Total	100.00	8.34	
Olive Oil	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	4.86	121.46	67.80
Tailing	95.14	2.95	32.20
Total	100.00	8.71	
Kerosene	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	6.72	82.66	71.83
Tailing	93.28	2.34	28.17
Total	100.00	7.74	
Corn Oil	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	4.87	110.08	70.56
Tailing	95.13	2.35	29.44
Total	100.00	7.60	

Table 3 The coal-oil assisted gold flotation concentrates obtained with different agglomerating oils (Agglomerates) – screened of the agglomerates was employed.

AP 704	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	10.98	52.39	70.55
Tailing	89.02	2.70	29.45
Total	100.00	8.15	
Sunflower Oil	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	4.14	142.32	69.25
Tailing	95.86	2.73	30.75
Total	100.00	8.50	
Olive-Pom.Oil	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	7.84	70.81	69.96
Tailing	92.16	2.59	30.04
Total	100.00	7.94	
Diesel Oil	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	4.58	126.94	70.09
Tailing	95.42	2.60	29.91
Total	100.00	8.29	
Cotton Oil	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	3.98	132.94	65.75
Tailing	96.02	2.87	34.25
Total	100.00	8.04	
Olive Oil	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	4.33	133.51	68.85
Tailing	95.67	2.73	31.15
Total	100.00	8.39	
Kerosene	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	6.26	84.53	69.03
Tailing	93.74	2.53	30.97
Total	100.00	7.66	
Corn Oil	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	4.11	128.50	68.32
Tailing	95.89	2.55	31.68
Total	100.00	7.73	

Table 4 The effect of ore/coal ratio

6.25	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	1.51	427.88	72.47
Tailing	98.49	2.49	27.53
Total	100.00	8.92	
12.5	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.42	269.27	73.82
Tailing	97.58	2.37	26.18
Total	100.00	8.82	
25	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.69	255.05	74.01
Tailing	97.31	2.48	25.99
Total	100.00	9.28	
50	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.42	226.71	74.76
Tailing	97.58	1.90	25.24
Total	100.00	7.33	
100	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	3.05	199.24	76.60
Tailing	96.95	1.91	23.40
Total	100.00	7.93	

Table 5 The effect of coal/oil ratio

4	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	9.26	72.15	79.23
Tailing	90.74	1.93	20.77
Total	100.00	8.43	
8	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	4.31	144.50	81.22
Tailing	95.69	1.50	18.78
Total	100.00	7.67	
16	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.69	255.05	74.01
Tailing	97.31	2.48	25.99
Total	100.00	9.28	
32	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	1.73	384.56	73.83
Tailing	98.27	2.39	26.17
Total	100.00	8.99	
64	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	0.57	1002.23	67.08
Tailing	99.43	2.82	32.92
Total	100.00	8.51	

Table 6 The effect of solid ratio

20%	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.42	269.27	73.82
Tailing	97.58	2.37	26.18
Total	100.00	8.82	
25%	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.48	228.03	73.34
Tailing	97.52	2.10	26.66
Total	100.00	7.70	
30%	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	3.06	196.50	74.70
Tailing	96.94	2.10	25.30
Total	100.00	8.04	
35%	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	3.72	161.67	75.09
Tailing	96.28	2.07	24.91
Total	100.00	8.01	

Table 7 The effect of coal particle size

-75 micron	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.67	204.80	67.61
Tailing	97.33	2.69	32.39
Total	100.00	8.10	
150-75 micron	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	1.64	328.24	66.75
Tailing	98.36	2.72	33.25
Total	100.00	8.05	
-150 micron	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.28	247.39	69.66
Tailing	97.72	2.51	30.34
Total	100.00	8.10	
300-150 micron	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.02	327.56	72.61
Tailing	97.98	2.54	27.39
Total	100.00	9.10	
-300.00 micron	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.42	269.27	73.82
Tailing	97.58	2.37	26.18
Total	100.00	8.82	

Table 8 The effect of aeration rate

1.50 l/min	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.22	282.21	70.15
Tailing	97.78	2.73	29.85
Total	100.00	8.94	
2.50 l/min	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.42	269.27	73.82
Tailing	97.58	2.37	26.18
Total	100.00	8.82	
3.50 l/min	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.58	239.52	74.05
Tailing	97.42	2.22	25.95
Total	100.00	8.35	
4.5 l/min	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	3.34	198.26	75.47
Tailing	96.66	2.23	24.53
Total	100.00	8.78	

Table 9 The effect of stirring speed (rpm)

1500-1000	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.44	273.65	73.36
Tailing	97.56	2.48	26.64
Total	100.00	9.09	
1500-1300	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.42	269.27	73.82
Tailing	97.58	2.37	26.18
Total	100.00	8.82	
1500-1500	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.40	256.76	75.29
Tailing	97.60	2.07	24.71
Total	100.00	8.19	
1800-1000	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	1.86	314.75	76.80
Tailing	98.14	1.81	23.20
Total	100.00	7.64	
1800-1300	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.05	306.68	79.44
Tailing	97.95	1.66	20.56
Total	100.00	7.92	
1800-1500	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	1.90	277.10	80.60
Tailing	98.10	1.29	19.40
Total	100.00	6.52	
1800-1800	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	1.99	271.41	80.89
Tailing	98.01	1.30	19.11
Total	100.00	6.69	

Table 10 The effect of aeration rate

1.50 l/min	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.22	282.21	70.15
Tailing	97.78	2.73	29.85
Total	100.00	8.94	
2.50 l/min	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.42	269.27	73.82
Tailing	97.58	2.37	26.18
Total	100.00	8.82	
3.50 l/min	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	2.58	239.52	74.05
Tailing	97.42	2.22	25.95
Total	100.00	8.35	
4.5 l/min	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	3.34	198.26	75.47
Tailing	96.66	2.23	24.53
Total	100.00	8.78	

Table 11 The effect of agglomerate recycling

First cycle	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	1.47	436.30	72.68
Tailing	98.53	2.44	27.32
Total	100.00	8.82	
Second cycle	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	0.77	732.76	67.77
Tailing	99.23	2.69	32.23
Total	100.00	8.28	
Thirth cycle	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	0.50	1104.24	66.70
Tailing	99.50	2.78	33.30
Total	100.00	8.30	
Fourth cycle	**Weight (%)**	**Grade, Au (g/t)**	**Recovery, Au (%)**
Concentrate	0.37	1320.42	63.39
Tailing	99.63	2.86	36.61
Total	100.00	7.78	

Table 12 The effect of scavenging (-60 micron)

Products	**Cum. Weight (%)**	**Cum. Grade, Au (g/t)**	**Cum. Recovery, Au (%)**
Concentrate	1.59	406.81	71.91
Scav. Conc. 1	3.07	258.07	87.86
Scav. Conc. 2	4.56	178.27	90.25
Tailing	100.00	9.01	100.00

Table 13 The effect of scavenging (-25 micron)

Products	**Cum. Weight (%)**	**Cum. Grade, Au (g/t)**	**Cum. Recovery, Au (%)**
Concentrate	1,52	440,71	71,99
Scav. Conc. 1	2,92	280,00	87,75
Scav. Conc. 2	4,65	182,50	91,26
Tailing	100,00	9,31	100,00