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LEACHING BEHAVIOR OF A LOW GRADE GOLD ORE : A PERFORMANCE COMPARISON OF COLUMN AND TANK LEACHING

MASTER OF SCIENCE

GÖKSEL KAYGISIZ ŞENDUR

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APPROVAL OF THE THESIS

The thesis submitted by **GÖKSEL KAYGISIZ ŞENDUR with the title of "LEACHING BEHAVIOR OF A LOW GRADE GOLD ORE: A PERFORMANCE COMPARISON OF COLUMN AND TANK LEACHING"** has been unanimously accepted by the jury members on the 24th of January, 2020 to fulfill the requirements for the degree of Master of Science in the Department of Mining Engineering.

THESIS JURY MEMBERS

Prof. Dr. Taki GÜLER **(Head of Jury)**
Department of, Mining Engineering,
Muğla Sıtkı Koçman University, Muğla
Signature:

Assist. Prof. Dr. Mustafa ÇIRAK **(Supervisor)**
Department of, Mining Engineering,
Muğla Sıtkı Koçman University, Muğla
Signature:

Assoc. Prof. Dr. Savaş ÖZÜN **(Member)**
Department of, Mining Engineering,
Süleyman Demirel University, Isparta
Signature:

APPROVAL OF HEAD OF THE DEPARTMENT

Assoc. Prof. Dr. Avni GÜNEY
Head of Department, Mining Engineering,
Muğla Sıtkı Koçman University, Muğla
Signature:

Assist. Prof. Dr. Mustafa ÇIRAK
Supervisor, Department of Mining Engineering,
Muğla Sıtkı Koçman University, Muğla
Signature:

Date of Defence: 24/01/2020

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Göksel KAYGISIZ ŞENDUR
24/01/2020

ABSTRACT

LEACHING BEHAVIOR OF A LOW GRADE GOLD ORE : A PERFORMANCE COMPARISON OF COLUMN AND TANK LEACHING

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Today, with the rise of gold prices, exploration activities in Turkey have increased. As a result of this increase, the number of gold mines opened has increased day by day, and gold production has gained a great speed. As a result of these, the start of the operation of low-grade gold mines has also come to the agenda. The operation of low-grade or high-grade gold mines is made possible by the application of many chemical processes.

This study was conducted by taking samples from Balıköy area of Tavşanlı District of Kütahya. Firstly, samples were sent to MTA laboratories for mineralogical-petrographic analysis in order to determine mineralogy. According to the laboratory results, the particle size ranges of which gold is heavily present were determined as -1000+500, -500+300 and -212+106 µm. In order to study the behavior of low-grade gold reserves belonging to the region in leaching tests, 2 different laboratory tests were applied and these were compared. These tests were column and tank leaching tests and examinations were carried out for different particle sizes. Column tests were applied for -20 mm and -12.5 mm breaking particle size and tank leaching tests were applied for -0.075 mm. NaCN was used in the leaching tests and $Ca(OH)_2$ (slaked lime) was preferred as the pH regulator. As a result of tests performed in different particle sizes, consumption values for both chemicals were calculated and comparisons were made. As a result of this research, gold recovery rates for low-grade gold ore were calculated and the maximum gold yield was realized at a particle size of -0.075 mm. As a result of this research, leaching behavior in different tests for low-grade ore was examined and appropriate methods were proposed for the gold acquisition process to be applied.

Keywords: Gold Ore, Column Leaching, Tank Leaching, NaCN, $Ca(OH)_2$, Recovery.

ÖZET

DÜŞÜK TENÖRLÜ ALTIN CEVHERİNİN LİÇ DAVRANIŞI: KOLON VE TANK LİÇİNİN PERFORMANSLARININ KARŞILAŞTIRILMASI

Göksel KAYGISIZ ŞENDUR
Yüksek Lisans Tezi
Fen Bilimleri Enstitüsü
Maden Mühendisliği Anabilim Dalı
Danışman: Dr. Öğr. Üyesi Mustafa ÇIRAK
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Günümüzde, altın fiyatlarının artmasıyla, Türkiye'deki arama faaliyetleri artış göstermiştir. Bu artışa bağlı olarak, açılan altın madenlerinin sayısı gün geçtikçe artmış olup, altın üretimi büyük bir hız kazanmıştır. Bunların sonucu olarak da düşük tenörlü altın madenlerinin de işletilmeye başlanması gündeme gelmiştir. Düşük tenörlü veya yüksek tenörlü altın madenlerinin işletilebilmesi, birçok kimyasal prosesinin uygulanması ile mümkün olmaktadır.

Bu çalışma Kütahya'nın Tavşanlı ilçesinin Balıköy mevkiinden, örnekler alınarak yapılmıştır. İlk olarak mineralojiyi belirlemek amacıyla, numuneler mineralojik-petrografik analiz için, MTA laboratuvarlarına gönderilmiştir. Laboratuvar sonuçlarına göre, altının yoğun olarak bulunduğu tane boyut aralıkları -1000+500, -500+300 ve -212+106 mikron olarak tespit edilmiştir. Bölgeye ait olan düşük tenörlü altın rezervinin liç testlerindeki davranışlarını incelemek için, 2 farklı laboratuvar testi uygulanarak bunların karşılaştırılması yapılmıştır. Bu testler kolon ve tank liç testleri olup, farklı tane boyutları için incelemeler yapılmıştır. -20 mm ve -12,5 mm kırma tane boyutu için kolon testleri uygulanmış olup, -0,075 mm için tank liç testleri uygulanmıştır. Uygulanan liç testlerinde NaCN kullanılmış olup, pH düzenleyici olarak $Ca(OH)_2$ (sönmüş sanayi kireci) tercih edilmiştir. Farklı tane boyutlarında gerçekleştirilen testlerin sonucunda her iki kimyasal için, tüketim değerleri hesaplanmış olup, karşılaştırmaları yapılmıştır. Bu araştırma sonucunda düşük tenörlü altın cevheri için elde edilen altın kazanım oranları hesaplanmış olup, maksimum altın verimi -0,075 mm tane boyutunda gerçekleşmiştir. Yapılan bu araştırma sonucunda, düşük tenörlü cevher için, farklı testlerdeki liç davranışları incelenmiş ve uygulanacak altın kazanım prosesi için uygun yöntemler önerilmiştir.

Anahtar Kelimeler: Altın Cevheri, Kolon Liçi, Tank Liçi, NaCN, $Ca(OH)_2$, Verim.

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1. INTRODUCTION

Natural gold is thought to be the first metal to be used by the human. It has attracted attention in every period due to its intactness, sparkling and rareness. Native gold is an element and a mineral. The gold element is formed by the stratification of the molten rocks, called magma, in the solid rock. During the cooling of the magma, the water is separated from the magma by the effect of high pressure in the water and other evaporable substances. As a result of high pressure, cracks get formed on the solid rocks. The melts in these cracks start tailing, and this sedimentation occurs quartz settles like veins. As the gold has a low melting temperature, it can solidify in quartz veins. The first place to look for gold are the quartz veins where the magma body is stratified (Altın, MTA, 2019).

Chemical symbols of this element is Au, atomic number is 79 and atomic mass is 196.97 g / mol. Its hardness is 2.5-3.0 and its density is 19.3 g / cm^3. Melting point is 1064°C, boiling point is approximately 2700°C. The gold is not affected by the weather, any atmospheric conditions (temperature, humidity). Additionally, gold is not affected by the oxygen, ozone, nitrogen, hydrogen gases (Wallis, T. M., N. Nilius, and W. Ho., 2002).

Gold is a precious metal, has a special color and is rare in the nature. For this reason, the price of gold is higher than the other metals. Gold is found in many regions, but in large deposits in some regions. In the nature, there are nearly 20 gold minerals. However, most gold found in nature is in the form of the native metal.

1.1. Information About Gold Mineral

Gold is common in nature; however, due to the low percentages, there are few productive areas. The amount of gold in seawater varies from 1 to 10 mg per tonne of water, depending on the region. Gold deposits are divided into two: primary deposits, secondary deposits. Primary deposits are generally composed of quartz-rich rocks; these rocks include silver and gold tellurides (silvanite, calaverite, krennerite, petzite, etc.) as well as various sulphides (pyrite, blend, stibnite, galena, etc.). Secondary deposits are alluvial deposits due to the erosion of primary deposits, and gold particles, which do not exceed 1/100 of a millimeter in length, can aggregate due to their density, forming masses of 25-30 kg (Zhang, Xianfeng, Micha Pazner, and Norman Duke, 2007).

Natural gold is crystallized in a cubic system. Its crystals are sparse, mostly in the form of plates, wires, flakes, particles and ingots. The most important gold minerals are:

Native Gold	Au(± other metals)
Electrum	Au-Ag alloy
Kustelit	Ag(± Au)
Aurocuprite	Au_2Cu_3
Aurostibit	$AuSb_2$
Rodit	Au(Pt, Rh, Fr, Pd)
Callaverite	$Au(Ag)Te_2$
Silvanit	$(AuAg)Te_4$
Tetsit	(Au,Ag)Te
Kennerit	$(Au,Ag)Te_2$
Petzit	Ag_3AuTe_2

1.2. Physical Properties of Gold

Pure gold is very soft and for this reason it becomes easily sheet. Gold has very high polishability due to its softness and its bright color can be changed more or less depending on the amount of alloy elements, in which it contains. Very thin foils show blue-green color through the light passing through them (Bernstein, 2008).

Gold, which cannot be used in its pure form because of its softness, is used in alloy with copper (red gold), silver and many elements. It does not have a crystalline appearance since gold is found in nature.

1.3. Chemical Properties of Gold

Gold does not react normally with water, dry and humid air, ozone, nitrogen, hydrogen, fluorine, iodine, sulfur, hydrogen sulphide and oxygen even at high temperatures.

Sulfuric acid, hydrofluoric acid, phosphoric acid, halogen-free nitric acid, except in very high concentrations, and practically all dilute or concentrated solutions of organic acids do not affect gold, even at boiling temperatures (Sadler, P. J., 1982).

When the hydrohalogenic acid, nitric acid, hydrogen peroxide and chromic acid are combined with an oxidant such as free halogen, the solution dissolves gold. Gold can form an alloy with many metals. In the formation of gold and silver alloys, many metals act as collectors for gold (such as: zinc, lead and copper) (Habashi, 2005).

1.4. Types of Gold Minerals

1.4.1 Electrum

Electrum is a metal alloy containing gold and silver. This alloy was used to make coins, vessels and coatings in ancient time. A silver content is between 25% and 55%. The density of this metal between 13.000 kg / m3 and 16.000 kg / m3 (Allan, G. C., and J. T. Woodcock., 2001).

Figure 1.1. Electrum Crystals (Round Mountain-America)

1.4.2 Pyrite, Marcasite, Pirotin and Arsenopyrite / Gold Ores

Gold is found in free or very fine particles in sulfide minerals. In fact, pyrite is a sulfide mineral observed in many gold mines. In addition, arsenopyrite and other arsenic minerals create a group except for other sulfide minerals. The reason is arsenic compounds which cause environmental pollution problems.

Pyrite

Marcasite

Pyrrhotite

Figure 1.2. Type of Gold Sulfide Minerals

1.4.3 Native Gold

The gold content of native gold generally between 85% to 95% and the highest knowledge content is 99,8%. The density of the pure gold is 19.300 kg / m^3 and the native gold is generally mean of 15.000 kg / m^3. If the gangue is separated from the minerals, the native gold can be taken with a difference in gold density above the 10 microns.

Figure 1.3. Native Gold (Eagles Nest Mine-California)

1.4.4 Copper Minerals / Gold Ores

Chalcopyrite-bound gold is generally derived from anode sludge that occurs during the refining of copper. However, some gold beds may be related to chalcopyrite and pyrite.

Figure 1.4. Gold and Copper Ore Samples (Cowichan Lake-British Columbia)

1.4.5 Gold Tellurium Ores

The tellurium is the most important ore after the native gold. Some tellerium minerals contain such as; Calaverite and Krennerite contain 40% of gold, Silvanite and petzite contain 25% of gold. It is known that the gold tellurium ores are sometimes associated with free gold particles and sulfurous minerals. The density of tellurium is between 8.000 kg / m3 and 10.000 kg / m3.

Figure 1.5. Krennerite Mineral – Arizona

1.5. Recovery Methods of Gold

Almost all of the gold produced in the world is extracted from quartz or schist veins and gold sands. Minerals are mechanically or chemically treated to separate gold. In the mechanical process, the mineral is ground to a fine powder, then washed in a mixture of water and oily material. In the second method, the operating possibilities of gold deposits are very diverse. During the grinding or washing of the mineral, the material can be applied to the alloy (mercury alloy). The resulting material is distilled off and the gold is separated. It also allows the treatment of chlorine water (especially after amalgamation) (chlorination). Nowadays, cyanidation is generally applied; The metal is dissolved in potassium cyanide or sodium cyanide in large pools and then treated with another metal (zinc, aluminum) to precipitate the gold. Metals which are easily oxidized in processing some complex gold minarels can be eliminated either by differential oxidation or by the action of certain acids or vapors. To purify the gold, the metal is first converted to gold chloride and this compound is separated by electrolysis; foreign metals accumulate in mud (Bayraktar, İ., & Yarar, B., 1981).

Ores that cannot be economically recovered with cyanide are called “refractory” ores. Ores that can be economically recovered by the addition of higher cyanide and oxygen are called “complex” ore. However, if the ore is containing free gold, it can be efficiently enriched by gravity, flotation and direct leaching methods. The refractory gold ores, which cannot be easily leached by the cyanide leaching method, are difficult to enrich by cyanide leaching and its recovery is very low (Fraser, K. S., R. H. Walton, and J. A. Wells, 1991).

The selection of gold recovery method depends on size of gold particles and the physical and chemical properties of other minerals in the ore. The most common method of gold recovery is to take gold into solution by cyanide method. This method is preferred in many parts of the world. Except for this method, flotation and gravitation methods are also use to recovery of gold (John O. Marsden-C.Iain House, 2006).

On the other hand, placer ores are ones these do not require pre-treatment. These ores are enriched by physical methods such as gravity separation. Knelson and Falcon concentrators, which are modern gravity separation devices, utilize centrifugal force in addition to gravity force to obtain fine gold particles. Another process is the coal-oil agglomeration. This method is recommended for alivual ores and other free gold ores. (Bellamy, House and Veal, 1989; La Brooy and al., 1994).

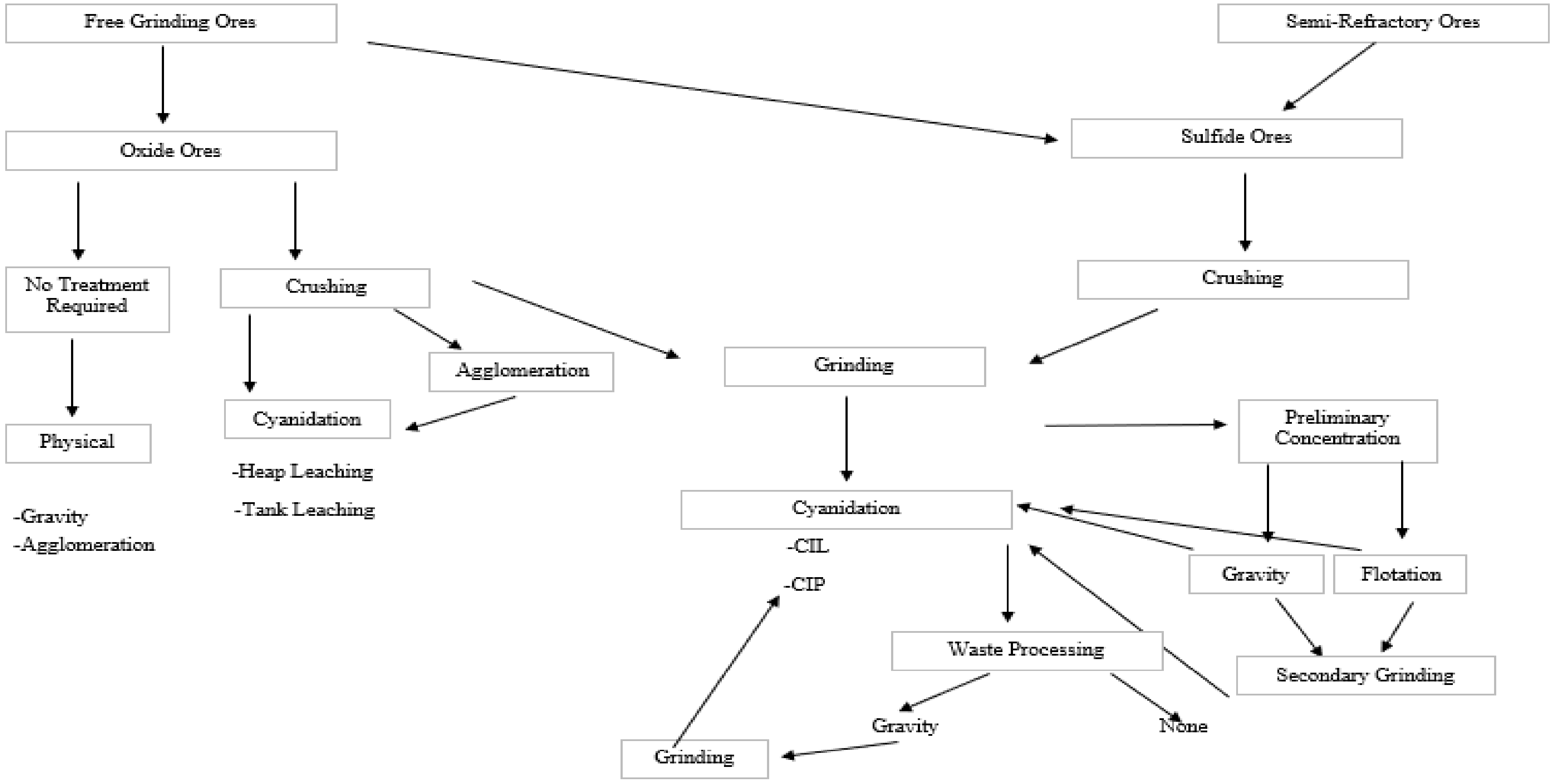


Figure 1.6. General Flow Diagram of Free Grinding and Semi-Refractory Ores (La Brooy et al., 1994)

Gold enrichment methods are;

i. Flotation
ii. Gravity Concentration
iii. Amalgamation
iv. Leaching Method

The leaching method is attractive as metal recovery from the solution is an easy and inexpensive process. The precious metals are recovered directly from the solution, as a result, wastes that require a lower cost of elimination are produced.

1.5.1 Flotation

Flotation is a processing applied to minerals which can be separated in -0.2+0.01 mm particle size. At the same time, the advantage of the flotation process for the gold particles that are released at large sizes is that there is no need to grind very fine particles. In this method, chemicals are used to take as a concentrated froth. Also, the desired mineral is floated, the unwanted mineral is suppressed with the appropriate chemicals.

This process is not applicable to free gold particle. Gold ore is recovered by flotation where gold is present with copper, lead and zinc. In addition to this, if the gold ore is associated with gold ores like pyrite, the flotation process is effective. The oxidized ores are not suitable for the flotation process. The flotation concentrate is usually sent to off-site plants to be melted for the production of gold and other metals (Allan, G. C., and J. T. Woodcock, 2001).

By flotation,

i. 1% Lead ores,
ii. 0.5% grade Copper ores,
iii. 0.3% grade Molybdenum ores,
iv. If gold minerals should contain 5 g / t Au, this can be economically enriched by this method.

The requirements for flotation are;

One or some of the ore minerals must have a small surface tension to the air.

The flotation water should contain substances that activate the surface tension. One of these substances should reduce the water-air surface tension, the other should make the foam and the amount should be sufficient to create a stable foam.

For flotation to be continuous;

A Sufficient grinding of ore to separate one mineral from another (liberation)

The continuity of the froth

The medium that adheres to the air bubbles must be continuous for the mineral particles.

1.5.2 The Flotation Reagents

The classification of these substances, called flotation reagents, is as follows:

i. Collectors and Promoters
ii. pH Regulators
iii. Activators
iv. Frothers
v. Protectors

1.5.2.1. Collectors and Promoters

The first basic condition is to cut off its contact with water. After the contact of a particle with water has ceased, ie the water film covering its surface is partially or completely disposed, air bubbles may adhere to that particle to allow it to float. The collectors must be dissolved in water to be effective. The collectors are designed to be enriched as quickly as possible and to prevent corrosion.

In order to float a solid, it is usually sufficient to stop 15-20% of its surface from contact with water. From this, it becomes clear that the use of more than necessary reagents (collectors) is of little use.

The collectors are mainly divided into two groups:

I. Anion active group
II. Cation active group

The anion active group is used in sulfur flotation and the cation active group is used in oxide flotation.

The anionic collectors are composed of organic acids and their salts. In these compounds, hydrogen or alkali metal is attached to the hydrocarbon chain via a polar group. This polar group may be an oxygen atom and a sulfur atom, a carbon atom, or a variety of linkages thereof. Depending on the type of polar group, various organic acids and salts thereof can be mentioned.

The cationic collectors are collectors formed by the hydrocarbon chain attached to the nitrogen. Different types of amines are formed by coupling various hydrocarbon chains from one to four to nitrogen. The fifth link is always the hydroxyl ion.

1.5.2.2. pH Regulators

pH Regulators: In selective flotation, the minerals in the ore are recovered respectively by changing the pH of the medium. The number of free H ions is changed in the medium and the character of the medium is provided to be acidic or basic.

1.5.2.3. Activators

Essentially it is not much different from the activation event. The mechanism, in this case, is slightly different. If a passivated portion of a mineral is desired to be recovered for a certain period of time, reinforcing reagents are added to the medium. This makes weaker the the use of their aggregation properties than the activators. The most important activator is sodium sulphide. Additionally, copper sulphate and various acids are also used for this purpose.

1.5.2.4. Frothers

The main task of the foaming agents is to reduce the surface tension of the flotation water. When the surface tension is not reduced, it appears as if a particle on the surface hits a hard surface. Thus, foam explodes and leaves the particles.

As the concentration increases, surface tension decreases. As soon as the medium is saturated with the foaming, the foaming stops and the surface tension is zero. Although the low surface tension is desired, the most suitable condition for flotation is the maximum foaming moment. It is necessary to keep the foamer dose constant at this point.

1.5.2.5. Protectors

The elements that prevent the flotation poisons either are activated by the ore or naturally present in the water used. Flotation waters usually contain humic acid, which is a substance produced by the roots and prevents the flotation event. Since the mud particles cover the face of the mineral particles, the collectors cannot function. The ferric and aluminum salts have a stopping effect on flotation. To sum up, the protectors are to prevent each of these situations according to their place. The alcalines are used against these problems.

1.5.3 Amalgamation Process with Mercury

This process, which was applied in ancient times and which is used in the recovery of relatively coarse gold particles, is based on contact of metallic mercury with pure gold to make amalgam. Between gold and mercury surface tension is much lower than the surface tension between gold and water, the gold surfaces are preferably coated with mercury in the pulp. Thus, gold can make a compound with mercury. Amalgamation method is used generally coarse-grained native gold which enriched gravimetric concantrated. Today it is applied to concentrates enriched by the gravity method of ores containing native gold (Veiga, Marcello M., et al.,2009).

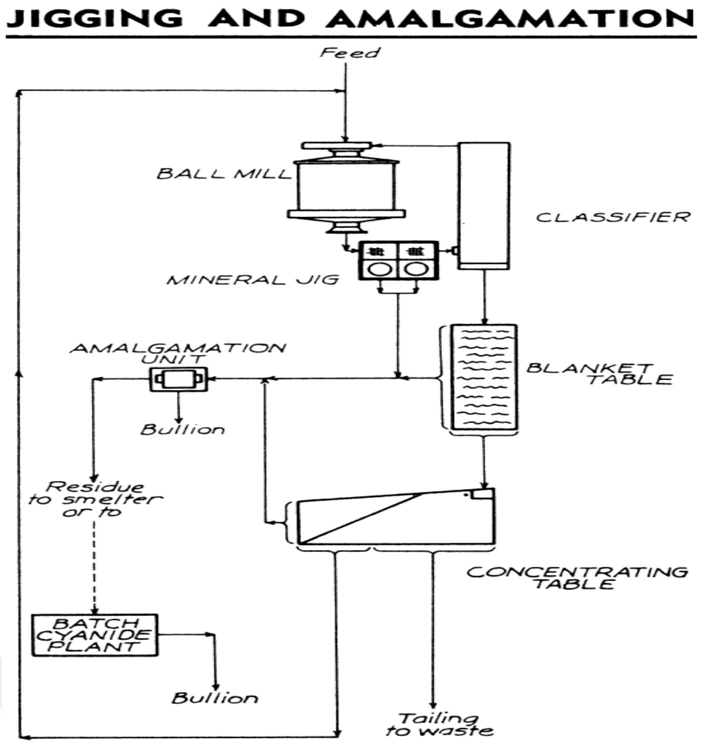


Figure 1.7. Process of Amalgamation

1.5.4 Gravitation Process for Gold Recovery

It is an enrichment method which is applied by using the specific gravity difference of mineral particles. In this enrichment method, while water is mostly used, heavier fluid or air can occasionally be used. The most commonly used tools for gravimetric enrichment are gutters, flat or slatted tables, cylindrical Johnson separator, endless belt conveyor, jig and Reichert cones.

Gravity enrichment method is applied to obtain free and coarse gold particles from ores, while hydrometallurgical processes are applied to the enrichment of fine-grained gold ores. The use of gravity enrichment method recover coarse gold particles prior to flotation and/or cyanidation increases total gold recovery. It takes a long time for the coarse gold particles to be completely dissolved in cyanide solutions. Pre-separation of coarse particles by gravimetric methods reduces leaching time relatively. Furthermore, the surfaces of the gold particles can be naturally coated with dirty, iron-hydroxide or organic compounds to prevent dissolution with cyanide. Using this type of ores will increase leaching efficiency (Harris, D. C., 1990).

The most important parameter is to reduce the liberation size of ore and the range must be more than 80%. With these methods, coarse-sized gold can be recovered in a range of 40-70%, while the recovery of fine-sized gold ocur in very high losses. The important ones are:

i. Coarse gold particles completely dissolve take a long time in the cyanide solution. When this method is applied, the leaching process will be shortened as the coarse particles will be separated from each other.
ii. Because the specific gravity of natural gold is too high, ore must be subjected to gravimetric enrichment in closed grinding circuits.

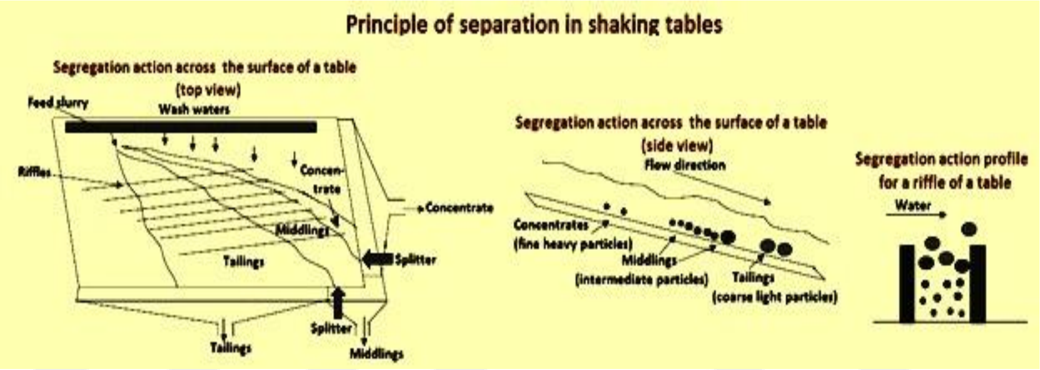


Figure 1.8. Working Principle of Shaking Table

It is a low capacity machine used in the final stage of gold production. The rocking table is a thin layer and shear flow process equipment that separates the fed ore by density, shape, core and size of that ore. Metals are classified according to their net weight, from light to heavy and from fine to thick.

1.5.4.1. Knelson Gravity Concentrator

Gold ores are classified as “free-grained”, “complex” and “refractory’’. 90% gold recovery is obtained from free-grained ores by conventional cyanide leaching for 24-72 hours. Gravity enrichment of fine particles are difficult. Jigs, tables and spirals are enriched up to 100 microns and lose their effectiveness under this size. As an alternative, new centrifugal gravity separators have been developed (F. Vincent,1997).

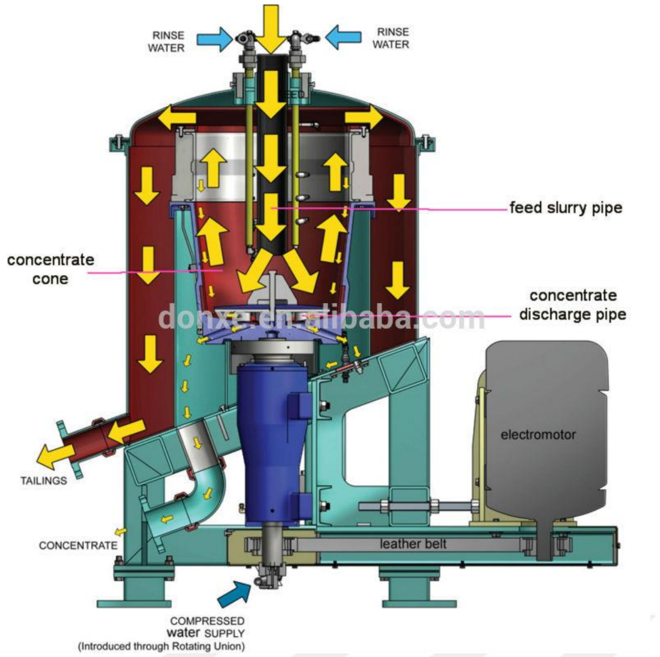


Figure 1.9. Knelson Gravity Concentrator

The Knelson Concentrator is highly effective in obtaining fine gold when compared to other methods. It works with 0-70% solid ore. The waste is continuously discharged from the outlet of the device with water, while the concentrate accumulates in the conical part until cleaning. The Knelson Concentrators are subject to the classification of disintegration in a centrifugal force field. When the perforated conical portion including horizontal grooves rotates along the inner wall, it forms a force of 50-60 times the force of gravity. Concentration is carried out after stopping the device in standard type Knelson Concentrators (Laplante, Andre R., Liming Huang, and G. B. Harris, 1996).

1.5.4.2. Falcon Gravity Concentrator

The Falcon Concentrator separates minerals with specific gravity differences in a high-intensity centrifuge environment. The Falcon Concentrator is intended to concentrate on the recovery of fine-grained, precious minerals that go wasting as a slime in other density separating devices (where no centrifugal force is used). The applied force can be up to 400G. -1 mm material can be fed to the device. It can be used in the recovery of gold as well as other heavy metals (Yıldız, 2010).

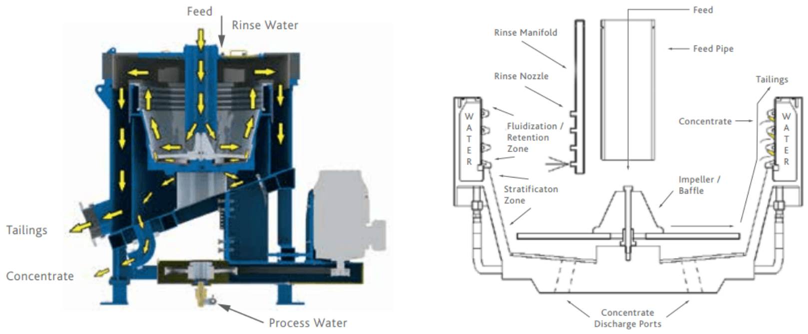


Figure 1.10. Falcon Gravity Concentrator

1.6. Alternative methods for gold recovery

Alternative methods include pyrometallurgical methods if gold mineralization is present with non-ferrous sulphides. The ores and concentrates that require oxidizing roasting generally contain one or more of arsenopyrite, stibnite, pyrrhotite and pyrite minerals.

The ore and concentrates generally contain, which one requires oxidizing roasting, arsenopyrite, stibnite, pyrrhotite and pyrite one or more of the minerals. Gold is dispersed in these minerals as very fine particles, even fine grinding may not be sufficient to obtain enough gold by cyanidation. Oxidizing roasting is one of the solutions for rich refractory ores. In this roasting process, the temperature is generally below 600°C. High oxygen concentration leads to the formation of compounds such as As_2O_5, Sb_2O_5. Their main drawback is that they form compounds which reduce recoveries by increasing cyanide expenditure. As a result of roasting, iron sulphide minerals turn into hematite. The specific surface area of the formed hematite is high due to the porosity of this material. However, a temperature higher than 500°C results in crystal growth and sintering, reducing the specific surface area. As a result, cyanide is difficult to reach gold surfaces and recovery is reduced (Bayraktar İ. ve Yarar B., 2010).

1.7. The Cyanide and Gold

The cyanidation process has been used for nearly 100 years in the production of gold and silver from ores. In recent years, as a result of the development of new processes that make the function more economical and efficient, this method has become the only option in today's mining, especially in the evaluation of low-grade deposits containing small particle size gold. In general, the process infiltrates anionic cyanide complexes to dissolve the gold and silver in the ore into the aqueous phase (Logsdon, Mark J., Karen Hagelstein, and Terry Mudder, 1999).

It is an anion having the chemical formula CN^-. It has advantages such as being easy and inexpensive to recover metal from the solution, the process does not require high temperature and is widely applied to low-grade ores. Besides, in the leaching process, precious metals are directly recovered from the solution, and by preventing the emission of harmful gases such as SO_2, less harmful to the environment and less costly elimination occurs. Besides these advantages, hydrometallurgical processes have some disadvantages. These are the low metal recovery in solution, the difficulties in solid-liquid separation, the need for more time due to low temperature application, and the negative effects of impurities during purification (Fleming, C. A., 2010).

Cyanidation of gold ores has been carried out for almost a century. Today, cyanidation is the main process of almost any gold operation. The majority of chemical enrichments benefit from the difference in chemical solubility. According to this method called leaching, the precious metal or metals are selectively introduced into the solution using a suitable solvent, which can be recovered selectively from the charged solution in the next step.

The gold deposits, which is close to the surface, are oxidized by the effects of water, oxygen, sunlight and bacteria. After that, these deposits can easily be processed with cyanide leaching. This type of ore is very easy to enrich and therefore quickly exhausting. For this reason, gold is produced by deep refractory gold deposits. The gold particles in the refractory gold deposits are locked in the sulfur minerals, especially pyrite and arsenopyrite, which cannot be dissolved by cyanide. The evaluation of this type of gold resources is made possible by the pre-treatment of cyanide leaching, which will break down the sulfur minerals and contact the gold with cyanide. (Hilson, Gavin, and A. J. Monhemius, 2006).

The enrichment process by changing the chemical structure of precious metals or minerals contained in an ore by using suitable chemical reagents is called chemical processing.

The reasons for the low response of ores to conventional cyanide leaching are summarized as follows;

i. Gold can remain locked in the mineral matrix in highly refractory ores and therefore may not reach under leaching reagents.
ii. If complex ores contain high amounts of cyanide and oxygenconsuming components such as metal sulfides in the ore, the gold may not be absorbed into the solution due to insufficient cyanide and / or oxygen.
iii. It may have acid components that cause lime consumption.
iv. Gold may be present as insoluble alloys or compounds such as; maldonite, tellurides.
v. Gold may be composed of clay material to prevent cyanide from acting (McDonald, W., R., Jhonson, J., L., ve Sandberg, R., G.,1990).

Some ores may contain more than one of these factors, which may affect the strategy of the process. Therefore, ore mineralogy determines the degree of refractoriness. Complex ores can be classified as high cyanide, oxygen depleting or gold adsorbing ores.

1.7.1 Reaction of Gold with Cyanide

This method is used today as a technology for producing gold from ore.In gold production with cyanide, gold is dissolved by treating with sodium cyanide at pH=10.5 and in an oxygenated environment. Although gold acts linearly against oxidation, it is reduced to Au+ cation by forming a cyanocomplex ($Au(CN)_2$) with the help of a complex-forming reagent such as CN (Kondos, P.D.,Wesley, F.Griffith, 1996). This behavior of gold in cyanide solutions was known to 18th century chemists and in 1783, Scheele and in 1843, Bagraton found that alkaline cyanide solutions solved gold (Yannopoulos, J.C., Van Nostrand Reinhold, 1991). MacArthur and the Forrest brothers became explorers of the process of dissolving gold from gold ores with the help of a weak cyanide solution (Wang, X., Forssberg, K.S.E., 1990). However, L. Elsner was the first to understand the importance of oxygen in the dissolution of gold (Arslan, F., Yüce, E., 2003). Elsner's equation, written below, shows the role that oxygen plays in the cyanidation of gold.

$$4Au + 8KCN + O_2 + 2H_2O \rightarrow 4KAu(CN)_2 + 4KOH$$

The behavior and reactions of gold in cyanide solutions have been extensively studied and it is known that the dissolution reaction of gold has an electrochemical nature (Wang, X., Forssberg, K.S.E., 1990). MacArthur and Forrest brothers also developed the ongoing process by precipitating gold in solution with the help of zinc. The industrial application of the cyanide reaction and the precipitation of the gold in solution with zinc became one of the milestones in gold metallurgy, and after that date, the world's gold production increased incredibly (Yeğin, M.,1997). The main reasons for this are the fact that cyanide is a very strong complex maker, the high yield of gaining metal, the ease of obtaining gold from solution and, most importantly, its applicability on an industrial scale. Given all these advantages, no alternative other than gold production methods are used in spite of the big problems in their environmental impact (Yeğin, M.,1997). Cyanide also plays a role in the recovery of industrial metal wastes as an alternative.

1.7.2 Theories on the Cyanidation of Gold

Different theories have been proposed to explain the dissolution mechanism of gold in aqueous cyanide solution. The first of these is the oxygen theory put forward by Elsner in 1846 and is based on the assumption that oxygen is an important element for dissolving gold in cyanide solutions (Yeğin, M.,1997)

$$4Au + 8NaCN + O_2 + 2H_2O \rightarrow 4NaAu(CN)_2 + 4NaOH \qquad (1)$$

The second theory is hydrogen theory. L. Janin (Yannopoulos, J.C., Van Nostrand Reinhold, 1991) showed that hydrogen gas was released during the gold cyanurization process by the following reaction:

$$2Au + 4NaCN + 2H_2O \rightarrow 2NaAu(CN)_2 + 2NaOH + H_2 \qquad (2)$$

The third theory is hydrogen peroxide theory. G. Bodloender revealed that there are two steps in dissolving gold with cyanide:

$$2Au + 4NaCN + O_2 + 2H_2O \rightarrow 2NaAu(CN)_2 + 2NaOH + H_2O_2 \qquad (3)$$

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

The total reaction of the decolonization that occurs following the intermediate reactions specified in 3 and 4 is as follows.

$$4Au + 8NaCN + O_2 + 2H_2O \rightarrow 4NaAu(CN)_2 + 4NaOH \qquad (5)$$

Hydrogen peroxide decays here as an intermediate product. Bodloender has shown that H_2O_2 is formed and can reach about 70% of H_2O_2, which should be formed according to this reaction, experimentally.

Experiments have shown that gold and silver have a slow dissolution process in NaCN and H_2O_2, in the absence of oxygen. So, the second reaction of Bodloender is;

$$2Au + 4NaCN + H_2O_2 \rightarrow 2NaAu(CN)_2 + 2NaOH \qquad (6)$$

and it is a reduction step.

$$H_2O_2 + 2e^- \rightarrow 2OH^- \qquad (7)$$

is less common. Indeed, if large amounts of H_2O_2 resulting from the oxidation (ionizing) of cyanide ion to cyanate ion is present in the system, the dissolution of gold is restricted. Cyanate ion does not exist on the dissolution yield on the metal:

$$CN^- + H_2O_2 \rightarrow CNO^- + H_2O \qquad (8)$$

Habashi (Yeğin, M.,1997) supported Bodloender's first equation when studying the cyanurization theory,

$$2Au + 4KCN + 3/2O_2 + 2H_2O \rightarrow 2KAu(CN)_2 + 2KOH + H_2O_2 \qquad (9)$$

The fourth theory is the cyanogen formation theory. S.B.Christy believes that oxygen is the active reactant in the dissolution of gold according to the following reactions and argues that it is necessary for the formation of cyanogen gas.

$$O_2 + 4NaCN + 2H_2O \rightarrow 2(CN)_2 + 4NaOH \qquad (10)$$

$$4Au + 4NaCN + 2(CN)_2 \rightarrow 4NaAu(CN)_2 \qquad (11)$$

Christy's two-step process complements Elsner's equation.

The fifth theory iscorrosion theory. B. Boonstra emphasizes that the dissolution of gold in cyanide solution is similar to a metal corrosion process in which oxygen dissociates into hydrogen peroxide and hydroxide ion, and Bodloender's reactions can be partitioned as follows.

$$O_2 + 2H_2O + 2e^- \rightarrow H_2O_2 + 2OH^- \qquad (12)$$

$$Au \rightarrow Au^+ + e^- \qquad (13)$$

$$Au^+ + CN^- \rightarrow AuCN^- \qquad (14)$$

$$AuCN + CN^- \rightarrow Au(CN)^-_2 \qquad (15)$$

The total reaction is as follows.

$$Au + O_2 + 2CN^- + 2H_2O + e \rightarrow Au(CN)^-_2 + 2OH^- + H_2O_2 \qquad (16)$$

Janin's misunderstood hydrogen theory is the only theory that says that there is no need for oxygen during the cyanidation of gold and silver. The opposite is that the thermodynamic development of oxygen theory shows a very high equilibrium constant. The reaction will continue until all cyanide is depleted or all gold is dissolved.

1.7.3 Mechanism of gold with a cyanide

The technique of obtaining gold using cyanide was first used in the United States in 1867. However, the use of the cyanidation technique has increased since it was determined that gold can be easily adsorbed with activated carbon in cyanide solution in the early 1950s. This method; sodium cyanide solution is based on the selective dissolution of gold and recovery of gold from the solution. The extraction of gold from solution is carried out by metallic precipitation or carbon absorption. Gold is recovered from the electrolysis by stripping. Finally, the process is completed by refining.

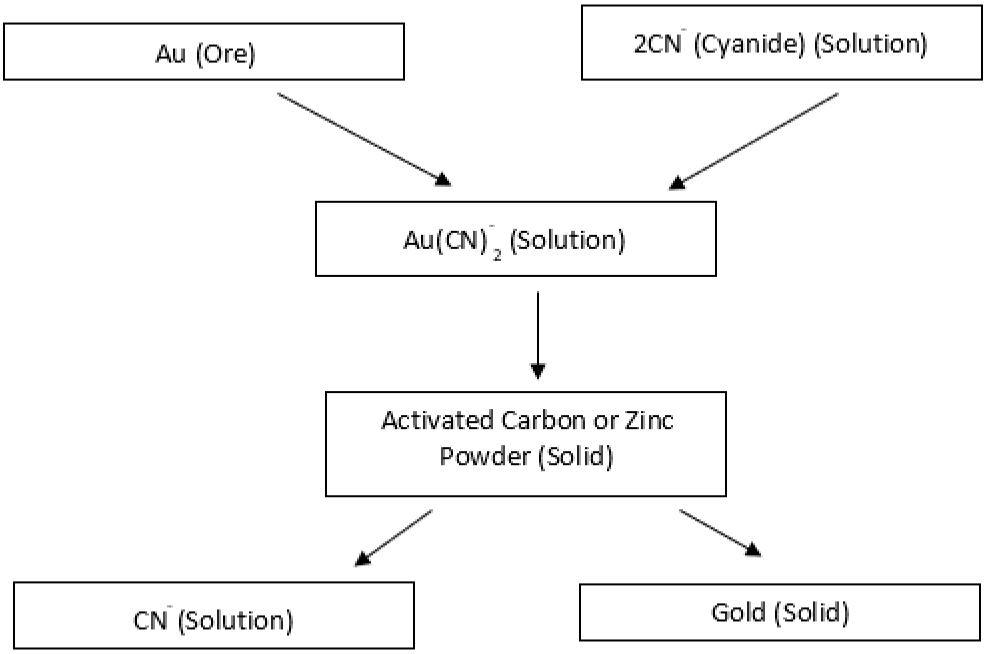


Figure 1.11. Cyanidation Process (Acarkan, 1997)

The term free cyanide includes cyanide ion (CN^-) and hydrogen cyanide (HCN), also known as hydrocyanic or prussic acid. In obtaining free cyanide, inorganic salt components such as NaCN, KCN, $Ca(CN)_2$ are used. The dissociation reactions of these compounds in various solutions are related to the pH of the solution. When the pH increases, there is a decrease in the presence of HCN (Habashi, 1970; Logsdon, Hagelstein, Mudder, 1999).

pH control should be done very well in the cyanide leaching process. To prevent the release of hydrogen cyanide gas, quenched lime is added to the solution to supply the pH balance (Logsdon, M., Hagelstein, J. ve Mudder, K. T., 2001).

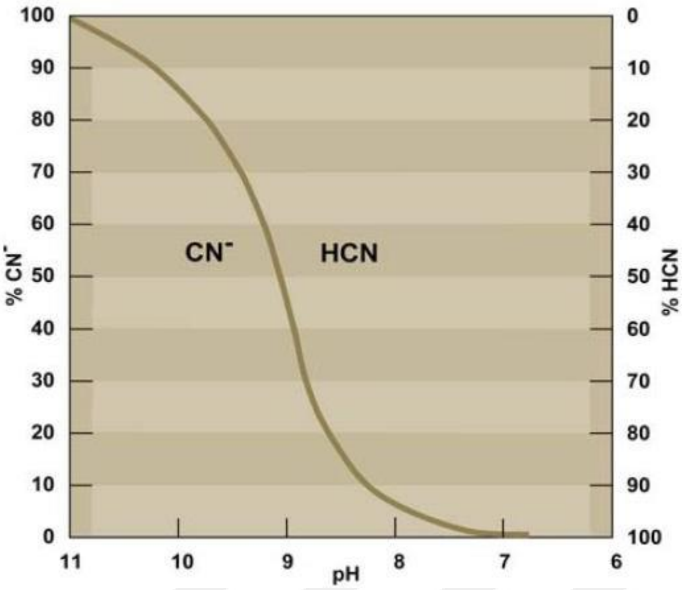


Figure 1.12. CN⁻ / HCN Balance Depending on pH (Logsdon et al., 1999)

Gold is usually found naturally in nature. Along with gold, it is present in sulfurous minerals such as pyrite, galena, sphalerite, arsenopyrite, stibnite, pyrrhotite and chalcopyrite.

Gangue minerals are quartz, feldspar, mica, garnet and calcite. While gangue minerals are generally insoluble in a cyanide solution, some metallic minerals are slightly soluble (Karadeniz, 1996).

1.8. Leaching Process for Gold

The recovery of gold with cyanide leaching is more prominent both in terms of economics and applicability than the other methods. At the same time, this method can be applied to very fine-grained ores with high efficiency. In the early 1950s, the use of the cyanidation technique was increased after the determination of gold in cyanide solution easily adsorbed with activated carbon.

In cases where gold recovery methods are insufficient except for complex ores, the cyanide recovery method is used.

Gold, which is transferred to the liquid phase by cyanidation, is obtained by hydrometallurgical processes consisting of direct precipitation or adsorption to activated carbon, stripping and electrolysis.

Table 1. Solubility of Metal-Containing Minerals in Cyanide Solution (Karadeniz, 1996)

Metal	**Mineral**	**Chemical Formula**	**Solubility(%) (24 hours)**
Gold	Kalaverit	$AuTe_2$	Easily Soluble
Silver	Arjantit	Ag_2S	Easily Soluble
Copper	Azurit Malakit Tetraedrit Kalkopirit	$2CuCO_3.Cu(OH)_2$ $CuCO_3.Cu(OH)_2$ $4Cu_2S.Sb_2S_3$ $CuFeS_2$	94,5 90,2 21,9 5,6
Zinc	Sfalerit	ZnS	18,4
Iron	Pirotin Pirit	FeS FeS2	Easily Soluble Hardly Soluble
Arsenic	Orpiment Realgar Arsenopirit	As_2S_3 As_2S_2 FeAsS	73,0 9,4 0,9
Antimony	Stibnit	Sb_2S_3	21,1
Lead	Galen	PbS	Soluble in alcali medium

Minerals are substances such as copper, arsenic, antimony and pyrrhotite in the ore that make it difficult for gold and silver, which consume large amounts of cyanide, to pass into the solution. Carbon dioxide and acid-forming minerals cause cyanide to break down. Their effects are reduced by adding lime or caustic so that the pH of the solution remains between 10 and 11.

Copper minerals dissolve together in cyanide solution and cause large amounts of cyanide and oxygen consumption. On the other hand, arsenic minerals consume cyanide and prevent gold from dissolving. The Heavy metals rapidly deplete cyanide and delay the recovery of gold. Besides. if the concentrate subjected to copper sulphate and iron sulphate, the ore is leached with dilute sulfuric acid and the copper and iron are separated and the ore is washed with alkali solution (Çilingir, 1990).

In terms of chemical reaction, gold is a metal that is resistant to all of the chemicals present, except for a few chemicals, such as cyanide and thiourea.

Today, gold is produced using cyanide of the world's approximately 84% (Roshan, 1990).

The basic principle of the cyanidation method is to selectively decompose the gold in the rock into a relatively selective solution as a cyanide complex. According to the way, the cyanidation method changes that the gold is taken from the rock and divided into two groups: tank and heap leaching.

In the tank leaching, the ore coming from the open pit is first reduced the size. Subsequently, the crushed material is milled in the mill and both the particle size is reduced and the material becomes sludge. Cyanide solution is passed through the tank to dissolve gold and silver. It may take a few days for all of the gold to go into solution. This gold-bearing solution is continuously passed through the activated carbon columns.

In the heap leaching, the ore, which is brought from the open pit, is conveyed using belt conveyors on an impermeable base after the particle size is reduced. Cyanide solution is applied to this heap by the spraying method. The flow rate of cyanide solution is given to the heap by calculating per ton according to the test studies before the leaching process starts. The first solution spray reach on the batch takes 3 or 5 days to the bottom. Gold, which is separated from the ore within a certain time, is filtered to the bottom in solution and collected in a pool.

1.9. Absorption to activated carbon

Activated carbon can be defined as a highly improved adsorbent material with an internal surface area and pore volume through the activation process applied to materials with high carbon content. (McDougall, G. J., et al., 1980). The expanded surface area of activated carbon is microporous structure, high adsorption capacity and surface reactivity of make it a unique adsorbent.

The volume and surface area of the pores of the activated carbons are generally greater than 0.2 ml/g and 400 ml/g . Activated carbon is made of wood, petroleum coke, nuts and coconut shells (McDougall, G. J., et al., 1980).

The loading capacity of the activated carbon determines gold and cyanide concentration of the solution, pH level, ionic strength and temperature duration the most suitable conditions of the adsorption process. Furthermore, the loading capacity of carbon also change depending on the effective surface area activity and ash content.

In practice, the active carbon adsorption method can be carried out in 3 different processes.

These are:

i. Carbon in Column (CIC) and Carbon in Pulp (CIP)
ii. Carbon in Leach (CIL)

With the carbon-in-column operation, solution flows through a series of fluidized bed columns in an upflow direction. Columns are the most frequently open top but closed top pressurized columns are occasionally used.

1.9.1 Carbon in Column (CIC) and Carbon in Pulp (CIP)

CIC method is not suitable for the solid-liquid separation method. Carbon columns use to recover gold and silver from heap leach solutions. The major advantage of this method is that it can process solutions containing 2 to 3 %wt solids.

Figure 1.13. Group of CIC Tank

CIP (carbon-in-pulp) operation is a variation of the conventional cyanidation process. Ore is crushed, finely grounded, and cyanide leached in a series of agitated tanks to solubilize the gold values. After the fine grinding process, the sludge is supplied to the first mixing leach tank to combine with cyanide and oxygen and proceeds to the last leaching tank. The pulp overflows from the last leaching tank at which dissolution is completed and passes to the first adsorption tank and carbon is advanced from the last adsorption tank to the first tank. Thus, the pulp and carbon proceed in the opposite direction and the gold is adsorbed to the carbon in the solution. Carbon transfers are carried out between the tanks by means of pumps (Jones, W. G., C. Klauber, and H. G. Linge, 1989).

This method is economically advantageous for mining companies because solid-liquid separation is not required. The ore passed through the grinding process is taken to the mixing tanks for cyanidation. The cyanide is kept below 0.1% and pH is between 10-11 to use sodium hydroxide or lime (Fraser, K. S., R. H. Walton, and J. A. Wells, 1991).

The loaded carbon is transferred to the stripping column by means of a carbon transfer pump. Finally, the gold-free cyanide pulp passes from the last adsorption tank to the cyanide detoxification tank. The cyanide is detoxified and pulp sent to the waste dam by pump. The solid material precipitates in the dam and, if appropriate, the process water is pumped for re-use in the plant.

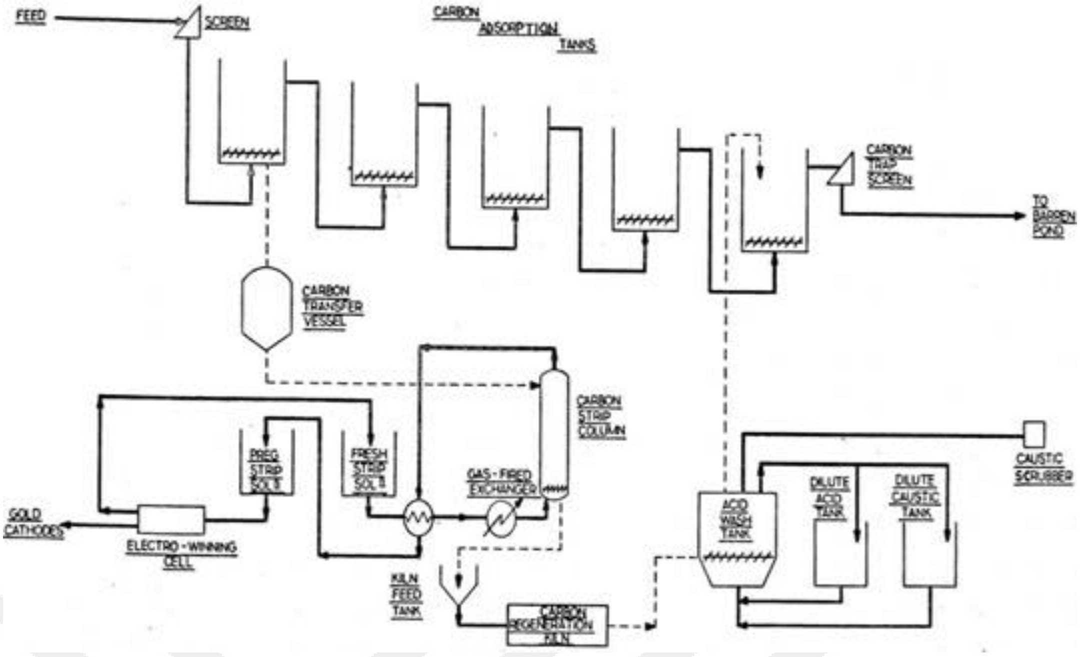


Figure 1.14. Carbon in Column

1.9.2 Carbon in Leach (CIL)

In the leaching carbon (CIL) method, the activated carbon is introduced into the system during the cyanidation step and dissolution and absorption into the carbon are carried out simultaneously. This method is applied for the pre-robbing ores which adsorb gold back from solution after dissolution due to its organic carbon content. The high gold recovery is achieved by adsorbing soluble gold to activated carbon (Fleming, C. A., et al., 2011).

Mixed tank leaching, widely applied to a wide variety of ore types, has been used for over 200 years. Leaching is typically performed in steel tanks and the solids are suspended by air or mechanical stirring. Air in conical bottom leaching tanks (Browns and Pachuca tanks) mixing with cyanide was widely applied in the early years of the curing process; however, mechanical mixing, which requires less energy and provides better mixing efficiency, has largely replaced this application.

The material to be leached to the tank is typically milled to a particle size of 80% <53-150 µm. Tank leaching with agitator, more than approximately 150µm is rarely applied due to its coarse size , because it is difficult to suspend the particles and the wear rate of the equipment increases.

The sludge density changes between 35% to 50% depending on the specific gravity of the solid, particle size and the presence of minerals affecting the viscosity of sludge. The reactive consumption decreases with increasing sludge density and optimum concentrations are obtained at lower doses as works per unit mass of material decreases.

In order to prevent hydrogen cyanide gas from being released in leaching systems, the pH range is maintained between 10 and 11. These values should be kept under control throughout the leaching period in order to maintain the desired working pH, especially during the processing of ores containing pH reducing components.

If there are no minerals consuming cyanide in the ore or concentrate to be leached; the cyanide concentrations used in practice range from 0.05 to 0.5 g / L NaCN and typically work at 0.15-0.30 g / L NaCN. Typical cyanide consumption in mixed tank leaching systems for easily processable ores ranges from about 0.25 to 0.75 kg / t. The cyanide consumption is calculated and optimized when necessary in laboratory test works.

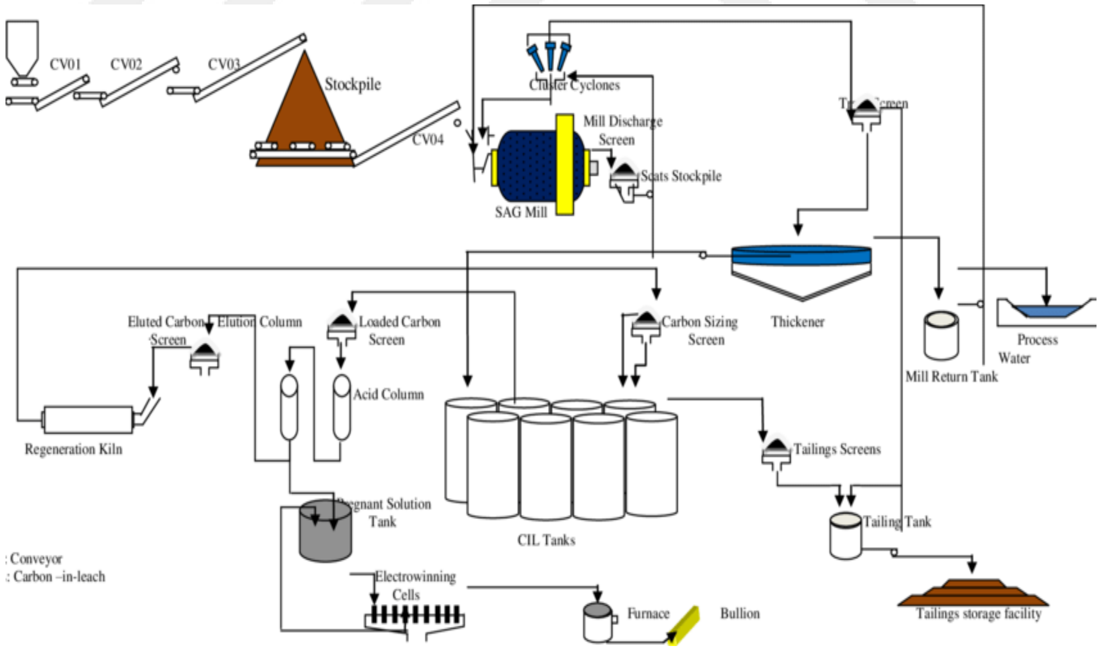


Figure 1.15. Carbon in Leach Process

1.10. Heap Leaching Method

Heap leaching method is the method of making low-grade metals into a heap and obtaining ore by cyanide on this heap. For the application of the process, the ore extracted from the quarry is crushed and stacked on an impermeable ground and cyanide is introduced into this heap. In the heap various materials such as clay, asphalt, cement and plastic sheeting are used to ensure impermeability base (D.W.Kappes, 2005).

The high recovery in the heap leach depends on the cleanliness of the ore placed in the heap, the absence of a compact material in the heap, and the absence of acid-forming components which consume high amounts of cyanide and lime.

The gold cyanide solution is taken over by the heap through the collecting and collecting system and is passed through activated carbon columns to absorb gold as carbon. The metalized cyanide solution is pooled for re-use. Gold loaded carbon is stripped off by carbon desorption. Gold is then recovered from this solution by electrochemical methods (Trexler, Dennis T., et al.,1990).

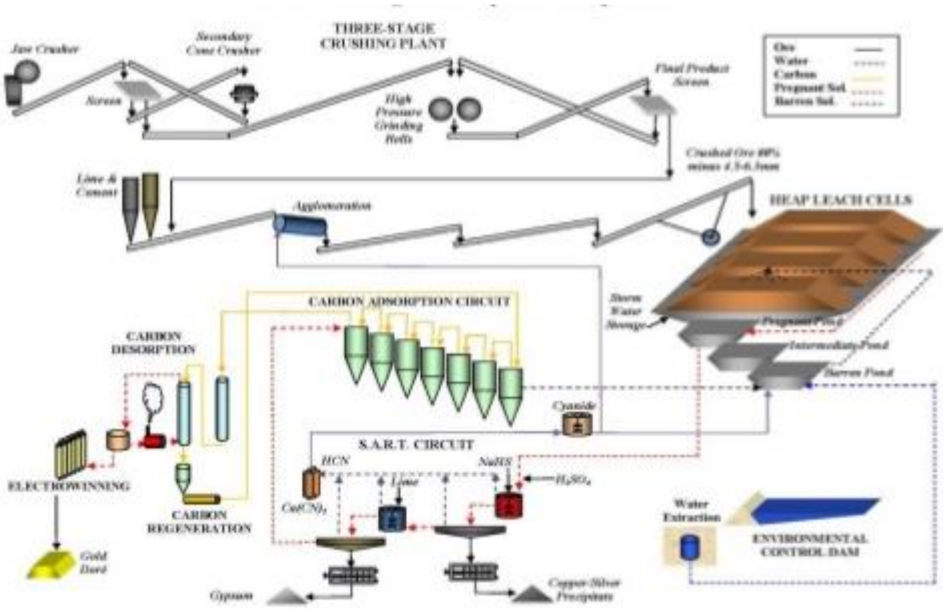


Figure 1.16. Heap Leach Flowsheet

1.10.1 Factors affecting heap leaching

In order to obtain good results from heap leaching applications, it is necessary to consider a wide variety of factors, especially the material to be treated, the solvent and other reagents to be used, the preparation of the heap.

1.10.2 Types of the ore for heap leaching

Mineralogical structure is very important in the heap leaching process and it is preferred that the mineral to be taken into solution needs to be readily soluble. To facilitate dissolution, it may be necessary to reduce the particle size of the material prior to batch preparation. The particle size usually varies depending on the structure of the ore. Since the clay will adversely affect the batch process, the clay material must be removed by a pretreatment to ensure that the solution flow is uniform. In case the material itself is fine-grained, the pelletization process makes it possible to make the particles coarsely sized. (Chamberlin, 1986)

Typical examples of low-sulfur acid volcanic or intrusive rock-type ores are found around Round Mountain, Nevada and Peru. Sulfur content is 2-3% pyrite and in some cases gold pyrite is inclusions. Oxidized ore leaching recovery generally vary between 65-85%. However in order to reach these values, the ore should be broken below 12mm (Emre Erkan, 2015).

Oxidized sulphide ores contain gold and silver in iron oxide and breaking below 75 mm does not increase leaching efficiency. Since these ores are soft and thin, agglomeration is carried out by using cement.

Carlin ores contain very fine (submicroscopic) gold in swollen and limestone. This type of ores are generally broken by rough crushing to 75 mm, about 70% gold recovery or can be taken from quarry material.

Figure 1.17. Heap Leach Plant in Australia

1.10.3 Heap Preparation

Depending on the nature of the solvent, regional factors and cost, the most commonly used impermeable floor materials are asphalt, concrete, various types of plastics and clay. Stacks of 4-5 meters in height can easily be prepared without the need of truck travel over the heap, and a slope of at least 3° is required to ensure that the saturated solution is collected in ponds (Heinen, Harold J., et al., 1978).

Depends on the situation; stainless steel, fiber-reinforced plastics and rubber-lined materials are used for pumping systems, pipes and pools. Depending on the application, the formation of the soil and the pile and the regular flow of solution in the pile are among the factors that will ensure the high leaching efficiency.

In order to achieve high recovery in the heap leaching, the permeability must be high. A pond or stratification that may occur in the heap adversely affects productivity and makes it difficult to obtain precious metals. In order to avoid this situation, heap height should be determined according to ore type. The highest floor application to date is 120 m (in hard ores) heap consisting of a total of 10 floors. This heap height varies depending on the ore structure. This heap height is with cement agglomeration, up to 30 meters in soft ores.

1.10.4 Heap Leaching Ore Transportation Methods

According to the ore structure, the transport process can be applied in two ways to the region to be stacked; by truck and conveyor belt.

The crushed ore, with high solubility, can be transported directly to the heap area by truck without the aid of a belt. In the truck laying method, coarse particles remain at the bottom of the heap, while fine particles can remain at the top of the heap. Due to the deposition of the fine material on the surface, stratification can be seen in the heap. This may lead to a reduction in the rate of dissolution and leachability, at the same time it can lead to a decrease in recovery.

To overcome this problem, after the laying process, the heap is loosened by the backhoe loader.

Figure 1.18. Laying Process by Truck

In the spreading process with a conveyor belt, the crushed ore conveyed to the leaching zone by a belt of a certain length. There are some varieties of these belts and their specific names and properties. For example:

1. The ore is sent from the fixed bands to the heap by means of mobile bands called grasshoppers.
2. The mobile belt, called the horizontal belt, feeds to the spreading belt.

iii. The radial bands scan the heap and move back and forth to perform the paving. (Emre Erkan, 2015).

1.10.5 Solution Application for the Heap

In the main lines are commonly used wobbler irrigation systems to deliver solution to the heap. Solution management is an important aspect of operating the heap leach facility. Ponds should always be managed at the various solution design elevations where possible i.e. at the elevation for the combined operating clarified and live solution volumes. The pools are very important for heap leaching and are used to collect solutions. Stormwater ponds should be keep empty and not used as make-up water storage facilities (Wan, Rong-Yu, et al., 2003). The pool designs and the existing ponds, located downslope of the heap:

i. Pregnant Leach Solution (PLS) pond (for collection of the leachate from the heap),
ii. Barren solution pond (for temporary containment of processed pregnant solutions),
iii. Intermediate solution pond (for recycling of leachates coming from the previously leached ore or older heaps to the fresh ore or newer heaps in order to build up the solution metal grade),
iv. Overflow/Stormwater Pond (standby for emergencies).
v. Detoxification pond (standby for emergencies).

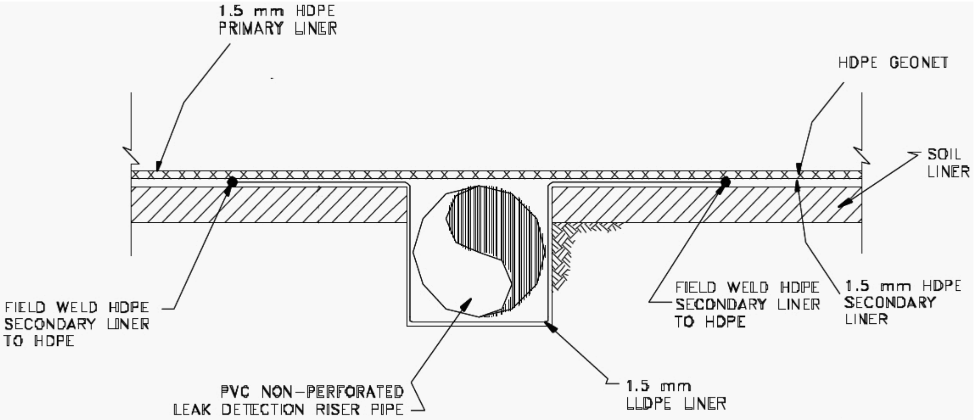


Figure 1.19. Leak Detection System Section (Kisladag Gold Mine, Turkey, Tüprag Gold Co.)

When designing the ponds in the area where heap leaching method will be made, it is important that their capacity is sufficient and that the precautionary measures against a natural disaster (flood, etc.) that can occur are at an acceptable level. Each new design should have a site-specific solution management system design based on the region and climate.

1.10.6 Solution Speed and Leaching Time

The determination of parameters such as solution flow rate or delivery rate and cyanidation is related to the following parameters from laboratory experiments:

a. Dissolution time of the gold
b. The effect of chemical reactions that can disrupt the cyanide solution
c. Rate of cyanide percolating through cracks in the ore and dissolving gold particles.

2. MATERIAL AND METHODS

The material of thesis is which collected from Balıköy is located approximately 90 km from Kütahya. Access to this project site within the borders of Kütahya province can be provided both Balıkesir Dursunbey and Simav road. For this study, samples were taken from the field belonging to Zenit Mining Company and test experiments were conducted.

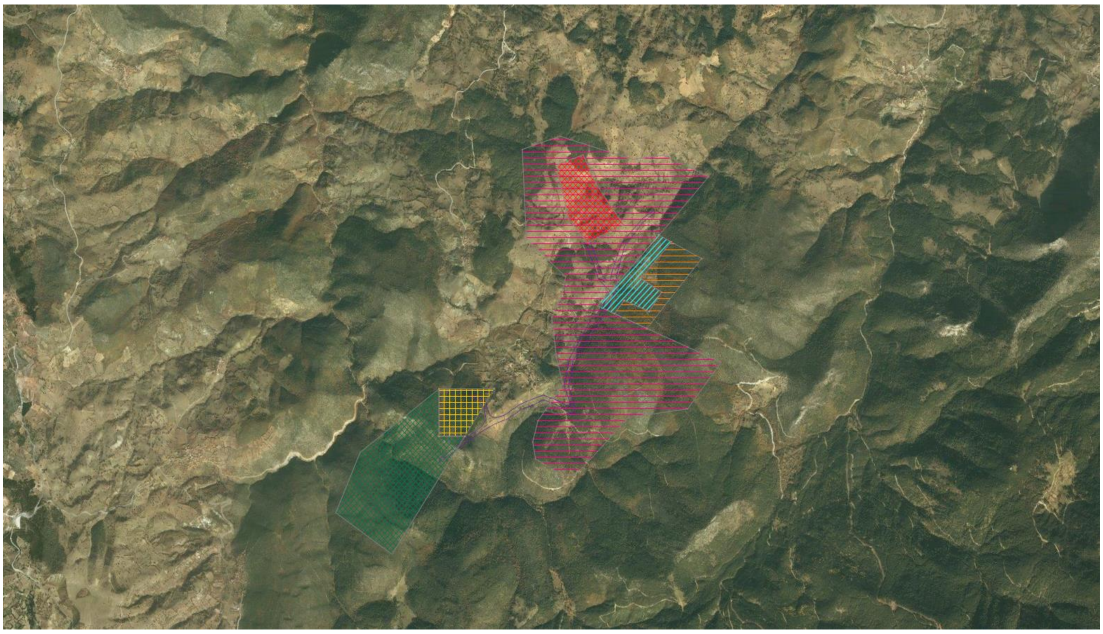


Figure 2.1. Tavşanlı-Balıköy Mining Area

The Tavşanlı gold deposit is jasperoid hosted, and occurs along a thrust fault surface between a hanging wall of ophiolite and a foot wall of limestone. The jasperoid presents two phases of silicification: an early pervasive fine-grained event, and a later clear quartz veinlet with local open space filling event.

The Tavşanlı property includes an upper thrust plate of Late Cretaceous ophiolitic rocks (Dagardi Melange), jasperoid gold-bearing silicification along the thrust surface, and a foot wall of Jurassic Budagan Formation massive micritic limestone. Ophiolitic rocks include serpentinites, mudstones, limestone pods, wackes, and local diabase. The ophiolitic assemblage is typically fresh to propyllitically altered, and displays a yellow iron oxide rich clay locally along the thrust contact with jasperoid. This ophiolitic sequence can be considered an aquitard to silicic fluids using the thrust as a conduit. No “leakage” was noted where thrust-controlled fluids escaped through the ophiolite.

No epithermal feeder structures were found in the foot wall limestones, and no epithermal leakage was found in the hanging wall ophiolites. Gold grades do not appear to be zoned, nor do gold/silver ratios immediately appear to be zoned. Neither those with experience on the property nor a short term consultant see visual controls to grade.

2.1. Mineral Liberation and Mineralogical Petrographic Analysis Method

In order to obtain information about the mineral contents of the samples before starting the MLA analysis, ore microscopy, mineralogical-petrographic analysis and XRD analysis are performed.

The powdered sample, which is separated into certain fractions, is first quartered by using a rotary riffler. The quartered representative samples are mixed with cold molding resin and poured into 30-millimeter molds and allowed to solidify at room temperature. Fully solidified samples are removed from the molds and their surfaces are polished in approximately 8 steps using an automatic etching-polishing machine. Before being placed in the MLA, the surfaces of the polished samples are coated with carbon to suit the analysis.

After the samples are placed in the instrument and necessary controls and calibrations are performed, the samples are analyzed in the instrument. At this stage, BSE (Back Scattered Electron) images are automatically taken from the particles (minerals) in all fractions and the chemical analysis (EDS) data of the points having different composition on each particle are collected. After the data is collected automatically by the device, in all fractions; mineral identification, grouping, classification, identification of unidentified particles (Unknown, Low-Counts, No-Xray), separation of artificial clamped particles. In the last stage, mineral reference, modal mineralogy, calculated theoretical elemental content (calculated assay), elemental distribution (elemental distribution), mineral release (mineral locking) and other data are produced.

2.2. Sample Preparation Stage

In the mineralogical analyzes, the frequency of the mineral fractions were investigated and reported. This particle size distributions are; +500, +300, +212, +106, +53 and +38 microns.

Sieved sample milled to 500-micron particle size was quartered in automatic rotary riffler device and 1.8 gram representative sample was prepared.

Sieved sample milled to 300-micron particle size was quartered in automatic rotary riffler device and 1.8 gram representative sample was prepared.

Sieved sample milled to 212-micron particle size was quartered in automatic rotary riffler device and 1.8 gram representative sample was prepared.

Sieved sample milled to 106-micron particle size was quartered in automatic rotary riffler device and 1.7 gram representative sample was prepared.

Sieved sample milled to 53-micron particle size was quartered in automatic rotary riffler device and 1.6 gram representative sample was prepared.

Sieved sample milled to 38-micron particle size was quartered in automatic rotary riffler device and 1.6 gram representative sample was prepared.

2.3. ICP Analysis Method

ICP (Inductively Coupled Plasma) Spectroscopy is an analytical method used to detect and measure elements to analyze chemical samples. The process is based on the ionization of a sample by an extremely hot plasma, usually made from argon gas.

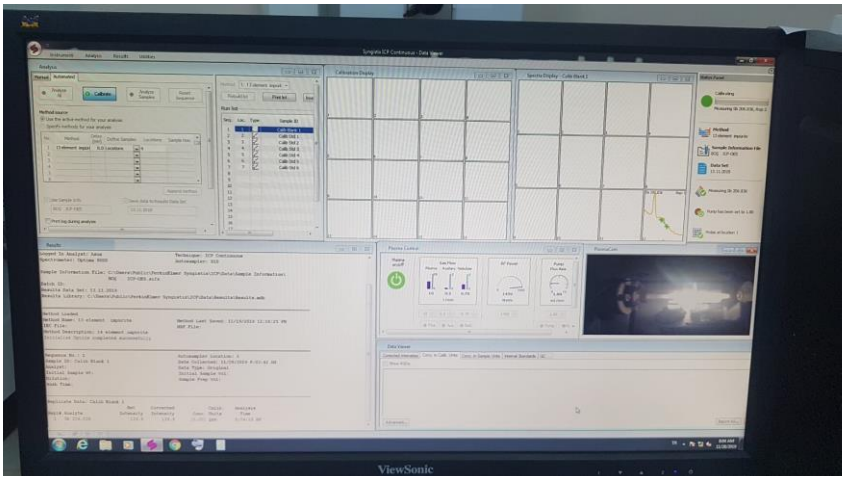


Figure 2.2. ICP Test Equipment

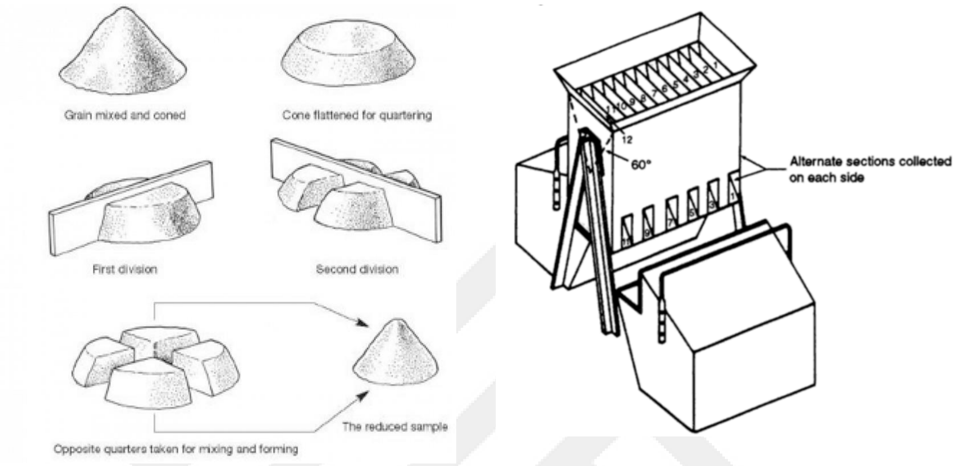
In an ICP spectroscopy unit, argon gas streaming is charged with an electromagnetic coil through a torch apparatus and lit with a Tesla unit that produces a quick discharge arc through the argon flow to trigger the ionization sequence. As soon as the plasma is “lit,” the ICP spectroscopy unit is switched off the Tesla device.

Inductively Coupled Plasma Atomic Emission Spectroscopy (ICP-AES) or ICP Atomic Emission Spectroscopy is a technique that can determine concentrations of a trace for major elements and can detect most elements in the periodic table. Reliable results can be obtained from nearly 70 elements with detection limits in the parts per billion range.

2.4. Sample Preparation for Laboratory Tests

The 600 kg sample from the Balıköy region, the largest particle size of approximately 300-500 mm, was first broken with the help of a sledgehammer and then reduced to -25 mm which is the particle size to be fed to the laboratory-scale jaw crusher. After this material was sieved in a controlled manner, the material on the sieve was fed to the jaw crusher and reduced to below -20 mm particle size. After the coning and quartering process, the material which was reduced to 100% of the particle size was -20 mm was subjected to a sample splitting process with riffle type sample divider.

-20mm sample was adjusted for the first column test, then the sample was broken in a controlled manner to -12,5 mm for the second group of column test. The material to be used for this test was also separated and the material to be used for tank leaching test was prepared by grinding below -0,075mm.



Coning and Quartering Method Riffle Sampling Method

Figure 2.3. Sampling Method for the Tests

In order to determine which type of enrichment to be carried out in a region, the ore in the region needs to be subjected to many different laboratory tests. If the process to be applied is a heap leaching, parameters such as cyanidation and cyanide consumption, amount of cement required for pelleting, crushing size for heap leaching, heap stability, permeability to height, leaching time, gold recovery, step height and laying technique are important.

If the process to be applied is tank leaching, many parameters such as mill particle outlet size, leaching time on average, how many CIL tanks are needed, cyanide and lime consumption and so on are determined according to the results of the tests to be performed in the laboratory.

During the experimental studies, 2 different types of tests were applied and the leaching kinetics of low-grade ore were examined by conducting column and mixing leaching tests. Each test was based on a specific concentration of pH and cyanide, and when there was a decrease, it was added. The test conditions were kept according to the determined principles. During these tests, liquid samples were taken from leached ore at certain time intervals, gold contents were determined for the liquid in these samples and the effect of free cyanide concentration on leaching was investigated.

2.5. Column Leaching Test

One of the most important tests examining the efficiency of low-grade ores is column tests. Therefore, a total of 10 column tests were performed in 2 different particle size distributions. These particle sizes are -20mm and -12.5mm. Column diameters were selected according to the maximum particle size obtained at the end of the crushing process. The diameter and height of these columns are important for the stability of the sample to be placed for testing. The solution caused by gaps in the coarse particles in the ore causes short cuts and prevents permeate from flowing faster than normal. The ore was placed in 2 m long columns.

Figure 2.4. Column Test Assembly (1st group column tests)

The first test was initiated by setting up 5 columns at the same time and the samples are broken to -20mm and were placed in the columns. The flow to the columns was provided by peristaltic pumps and the leaching time was 72 days for -20mm and continued for 86 days for -12.5mm.

Table 2. 1st Group of Column Test Parameters

Column Leach No	**1**	**2**	**3**	**4**	**5**
Date	6.11.2018	6.11.2018	6.11.2018	6.11.2018	6.11.2018
Sample Type	Rock	Rock	Rock	Rock	Rock
Particle Size d(100)	-20 mm	-20 mm	-20 mm	-20 mm	-20 mm
Leaching Time	72 days	72 days	72 days	72 days	72 days
Column Height (m)	2,0	2,0	2,0	2,0	2,0
Inner diameter of column (mm)	140	140	140	140	140
Solid sample quantity (g)	38394	38237,9	40810	40901	43221
NaCN Concentration (mg/L)	1000	1000	1000	1000	1000
pH	10,5-11	10,5-11	10,5-11	10,5-11	10,5-11
Dripping Rate (L/m2/h)	13	13	13	13	13
Dripping Flow Rate (L/h)	0,20	0,20	0,20	0,20	0,20
Dripping Flow Rate (L/day)	4,80	4,80	4,80	4,80	4,80

Table 2 gives information about the column tests for -20mm. The pH of 20 liters of clean water was adjusted to be in the range of 10.5-11 to prevent the release of hydrogen cyanide gas. NaCN concentration is 1000ppm is prepared as an unloaded solution. The dosing pumps of all columns start at the same time and the test is started. In order to determine leaching kinetics at the beginning, solution samples are taken from barrels every day and then once a week and twice if necessary. In addition, samples are taken from barrels once a week and samples are taken from column drains at the same time. When samples taken from drums are delivered to the laboratory weekly, other metals are analyzed by ICP as well as gold analysis.

Figure 2.5. Column Test Assembly (2nd group column tests)

The second group of column leaching test was initiated by setting up 5 columns at the same time and the material with a particle size of -12.5 mm was used and the same set-up as the first one was re-established and a 86 days test period was initiated (Table 3).

Table 3. 2nd Group of Column Test Parameters

Column Leach No	**6**	**7**	**8**	**9**	**10**
Date	20.01.2019	20.01.2019	20.01.2019	20.01.2019	20.01.2019
Sample Type	Rock	Rock	Rock	Rock	Rock
Particle Size d(100)	-12,5 mm	-12,5 mm	-12,5 mm	-12,5 mm	-12,5 mm
Leaching Time	86 days	86 days	86 days	86 days	86 days
Column Height (m)	2	2	2	2	2
Inner diameter of column (mm)	140	140	140	140	140
Solid sample quantity (g)	16801	19711	20130	21711	21911
NaCN Concentration (mg/L)	1000	1000	1000	1000	1000
pH	10,5-11	10,5-11	10,5-11	10,5-11	10,5-11
Dripping Rate (L/m2/h)	13	13	13	13	13
Dripping Flow Rate (L/h)	0,2	0,2	0,2	0,2	0,2
Dripping Flow Rate (L/day)	4,8	4,8	4,8	4,8	4,8

Table 3 gives information about the column tests for -12,5mm. In this column test group, the pH of 20 liters of clean water was first increased to the range of 10.5-11. After, cyanide was added and prepared 1000 ppm free cyanide concentration for the test. The columns were started to be delivered with the help of dosing pumps and drip-free solution.

2.5.1 Sampling for Column Leaching Test

Before sampling samples from barrels containing 20 liters of a solution on a weekly or daily basis, it is calculated by weighing the known tare barrels, how much solution is returned to the barrel and how much is in the column. In addition, knowing the weight of the drums help us to calculate how many grams of gold in the ore in the column have passed into the solution and to calculate the efficiency.

After mixing the solution in the weighed barrel with the aid of an apparatus which will not react with the solution, about 40 ml of the liquid sample is taken and filtered. After this process is completed, gold, pH and NaCN levels are checked in the sample taken. According to the results of the analysis, if the pH has dropped below 10.5 lime is added. If the NaCN has dropped below 1000ppm, the cyanide concentration of the solution in the barrel is completed to 1000ppm. Finally, after these analyses are performed, the remaining solution from the analysis is added to the drums to maintain the volume in the barrel.

2.6. Tank (Agitation) Leaching Test

In laboratory tests, the sample is ground to -75µm (micron). Additionally, in the field applications, ore is grind longitudinally to 80% of material 45 and 150 microns for the tank leaching enrichment method. It is difficult to hold the ore in larger particle sizes.

Table 4. Tank Leach Test Parameter

Tank Leach No	%Solid	Weight (gr)	NaCN (ppm)	d100 (µm)	pH	Liquid (ml)	LeachingTime (hours)
1	50	1000	2000	-75 µm	10,5-11	1000	72
2	50	1000	2000	-75 µm	10,5-11	1000	72
3	50	1000	2000	-75 µm	10,5-11	1000	72
4	50	1000	2000	-75 µm	10,5-11	1000	72

In table 4 shows tank leaching test parameters. The leaching was carried out at a sludge density of 50% solids for this test, depending on the specific gravity of the solids, the particle size and the presence of minerals (eg clay minerals) affecting the viscosity of the sludge. Tap water is added at the same rate as the amount of solid to form a 50% solids ratio and the mixer is started. The reactive consumption decreases with the increasing sludge density and optimum concentrations are obtained at lower doses as the volume per unit mass of material decreases.

Figure 2.6. Tank (Agitation) Leach Test Assembly

After starting to mix, powder lime is added to adjust the pH range of the slurry and the pH is stabilized before the cyanide to be added for testing. To avoid the release of hydrogen cyanide gas, the pH should be between 10.5 and 11.

The test start time is recorded after insertion into the beaker. Compared to other laboratory tests, the frequency of mixing leaching test analysis is higher. This agitation method apply generally high-grade ores on the field application as the purpose of this method is to provide faster recovery of metal.

The total number of hours determined for this test is 72. The initial pH, cyanide, oxygen and solid + liquid weight are recorded in the layer prepared for the test. After the test has started, the frequency of analyzes is higher than other tests. The mixers are stopped and weighed for each sample first. Then, the dissolved oxygen and pH are measured. If the pH is below 10.5, quenched lime is added as needed. For free cyanide measurement and gold analysis, the slurry sample from each beaker is filtered on filter paper. Analysis of cyanide and gold is performed from the filtered clear solution. Free cyanide was measured by silver nitrate titration method with 5 ml solution sample. If there was a decrease in the initial cyanide concentration, sodium cyanide was weighed to the specified free cyanide concentration and added to the beaker. The gold is analyzed in the liquid with the remaining solution sample. The gold efficiency is determined for each analysis hour.

2.6.1 Sampling for Tank Leaching Test

During the 72-hour test, the mixers are stopped within the specified hours and each sample is weighed. After weighing, a short time is allowed for the material to settle. After filtering the slurry sample, the solid remaining on the filter paper is added to the beaker again. The best results of the tests depend on the low solid loss. The amount of initial solids should be close to the leaching final solids. Leaching efficiency graphs and their results are relevant.

Figure 2.7. Slurry Sample

For the analysis of gold, NaCN and pH, samples are filtered on filter paper with approximately 30 ml of solution from each beaker. The sludge remaining in the filter paper is then added to the beakers to which they belong.

If the cyanide and pH levels of the analyzed samples decrease, free cyanide is added by weighing the powder cyanide at 1000ppm. If the pH is decreased, powder lime is added and the mixers are re-commissioned.

2.7. Measurement of Free Cyanide

Measurements of free cyanide in the solution were carried out by the silver nitrate titration method. In this method, first, sludge is filtered in an amount sufficient to obtain 10 ml of solution needed for analysis. Take 5 ml of liquid and transfer to a 100 ml beaker and add 2-3 drops of rhodanine as an indicator. Burette faucet is opened and silver nitrate is dropped into the beaker. The color change is observed by mixing the beaker during the process. When the color change occurs, the process is stopped and the amount of silver nitrate added is measured. The amount of silver nitrate consumed is multiplied by 1.96 in ml and the amount of free cyanide is calculated as a ppm.

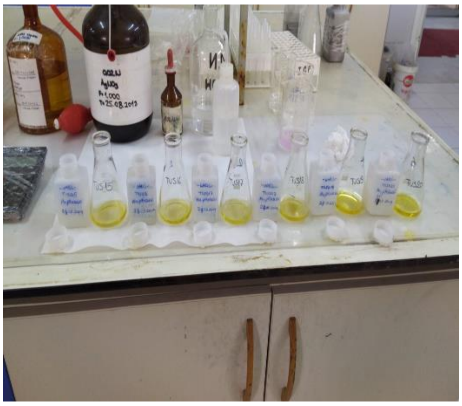


Figure 2.8. Solution prepared with rhodanine for free cyanide concentration analysis

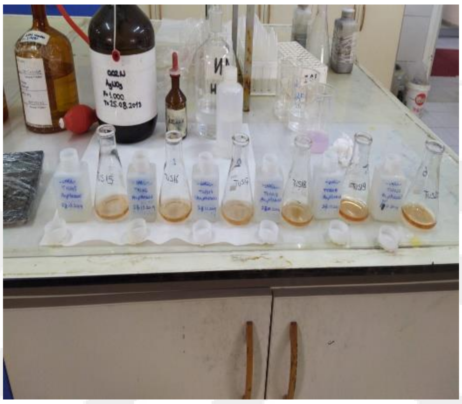


Figure 2.9. During free cyanide analysis the color change after dropping silver nitrate prepared solution

3. RESULTS AND DISCUSSIONS

In addition to increasing operating costs, low-grade gold ore has gained further value due to the price of $ 1,500 an ounce of gold. The operation of low-grade gold is possible with the heap leaching process. The process for which ore is suitable is generally determined by laboratory tests.

When the analyzes carried out during the tests and the analysis of the starting samples are considered, the amount of silver metal in the ore is negligible. Since the amount of silver contained in the ore was trace, no silver was found in the solution samples taken during the tests. However, the microscopic investigations showed silver in some fractions by MTA and detailed results and images of the analyzes are included in the thesis.

The ore used in this thesis was obtained by blending the samples obtained from the field where the mine will be operated to represent the region. Metallurgical studies were conducted on this blended ore.

3.1. MLA Analysis Results and Evaluations

When evaluating the results of MLA analysis, modal mineralogical composition, calculated theoretical elemental content, elemental distribution and mineral release degree tables for the minerals of interest (including silver and gold elements) were taken into consideration.

MLA measurements of 500 microns labeled sample; total of 2167 particles and 14.432 different mineral particles were processed.

MLA measurements of the sample marked 300 microns; the counts were made on a total of 3235 particles and a total of 17,953 different mineral particles were processed.

MLA measurements of the sample labeled 212 microns; a total of 7343 particles were counted and 30,352 different mineral particles were processed.

MLA measurements of the 106 microns marked sample; total of 19,359 particles were counted and a total of 56,836 different mineral particles were processed.

MLA measurements of 53 microns labeled sample; total 29,301 particles were counted and 51,738 different mineral particles were processed.

MLA measurements of the 38 microns labeled sample; total of 28,550 particles were counted and a total of 43.468 different mineral particles were processed.

When the modal mineralogical analysis tables are examined, it is observed that Quartz, Calcite, Muscovite and K-Feldspar minerals are dominant minerals in all fractions. Other minerals are composed of other mineral ores and gangue minerals in varying amounts according to the samples.

The following table shows the simplified model mineralogical composition in table 5 (% w by weight) :

Table 5. Mineralogical Composition Table

	+500µm	**+300µm**	**+212µm**	**+106µm**	**+53µm**	**+38µm**
Quartz	82.98	79,11	73,2	73,19	70,84	65,85
Calcite	11,35	13,67	17,14	16,97	20,05	21,74
Muscovite	1,49	2,73	3,10	3,85	3,45	4,46
K-Feldspar	2,6	2,59	3,46	3,21	2,83	3,22
Acantite	0,0004	-	0,000017	0,00004	0,000015	-
Gold	0,00016	0,000041	0,00	0,00003	-	-
Silver	-	0,00	0,00015	-	-	-
Other Minerals	1,57945	1,899959	3,10983	2,77993	2,82999	4,73
Total	100	100	100	100	100	100

According to mineralogical analyses, native gold was detected in 1000+500 µm, -500+300 µm and -212+106 µm fractions as in table 5. The total elemental Ag value was observed to be due to acanthite and Native Silver minerals in varying proportions according to fractions.

All of its fractions except gold have varying proportions of quartz and calcite. The quartz content, which is 83% in coarse fractions, falls to 66% at a fraction of -53+38 µm. In contrast, calcite content is up to 11% in coarse fractions and up to 22% in fine fractions.

The following were BSE and MLA images and EDS spectrum images of minerals listed in Table-5.

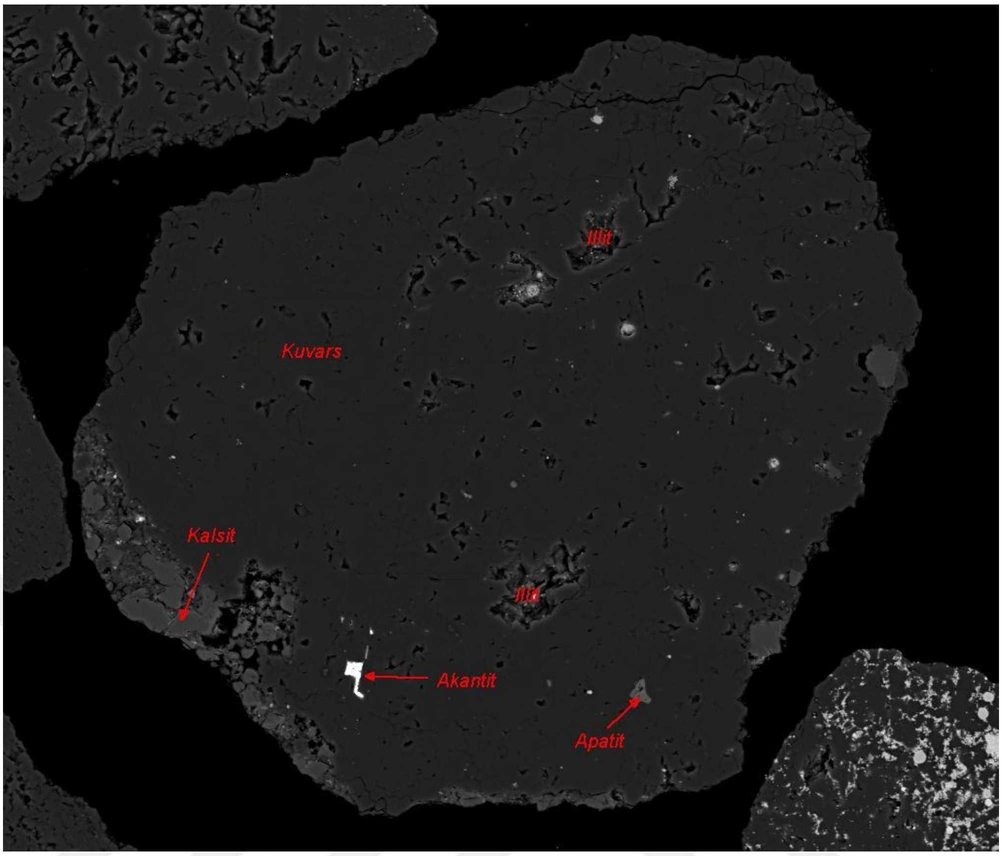


Figure 3.1. Image of Acanthite Mineral in Quartz (+500micron)

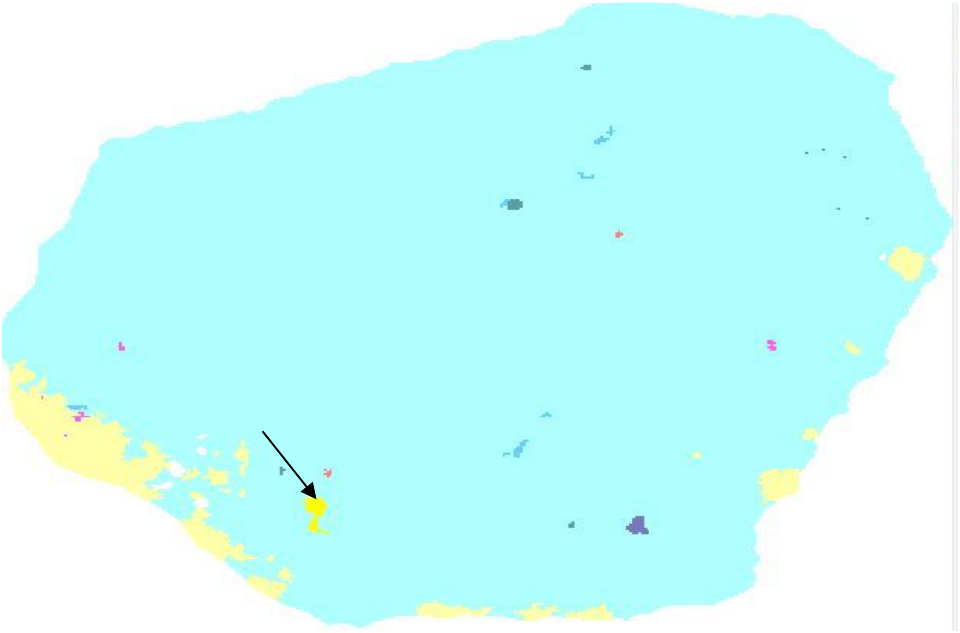


Figure 3.2. Image of Acanthite from MLA (+500 micron)

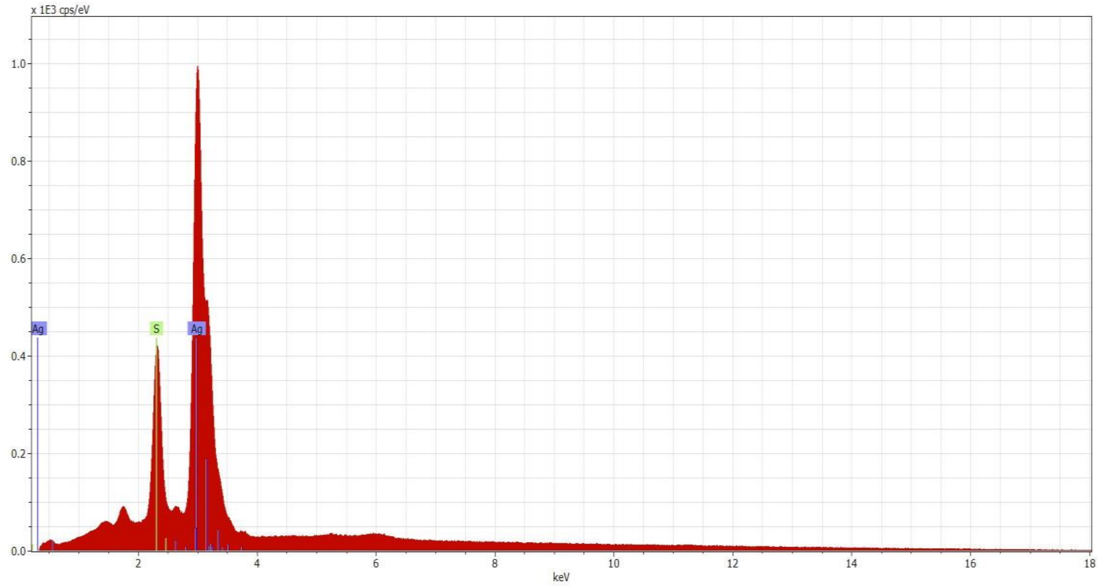


Figure 3.3. BSE-MLA Spectrum of Acantite (+500 micron)

When figure 3-1, 3-2 and 3-3 were examined, BSE, MLA image and EDS spectrum images for the +500 micron acanthite mineral were included. As a result of the BSE images, the acanthite mineral showed a distribution mostly in Quartz.

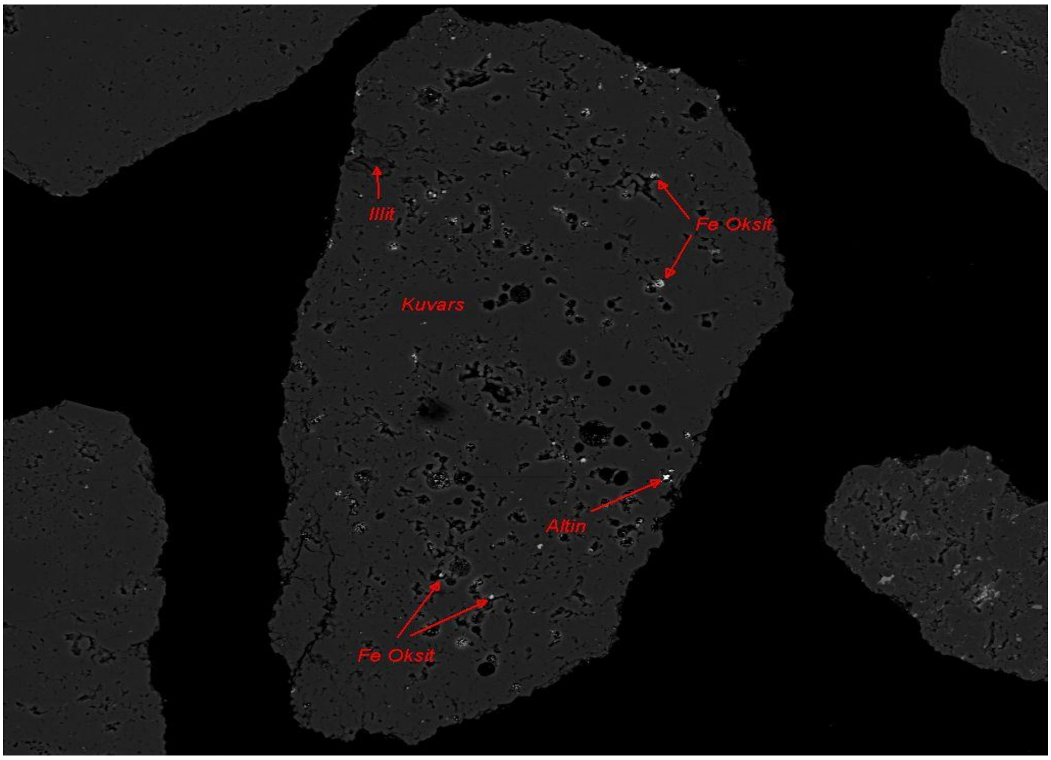


Figure 3.4. Image of Gold Mineral in Quartz (+500micron)

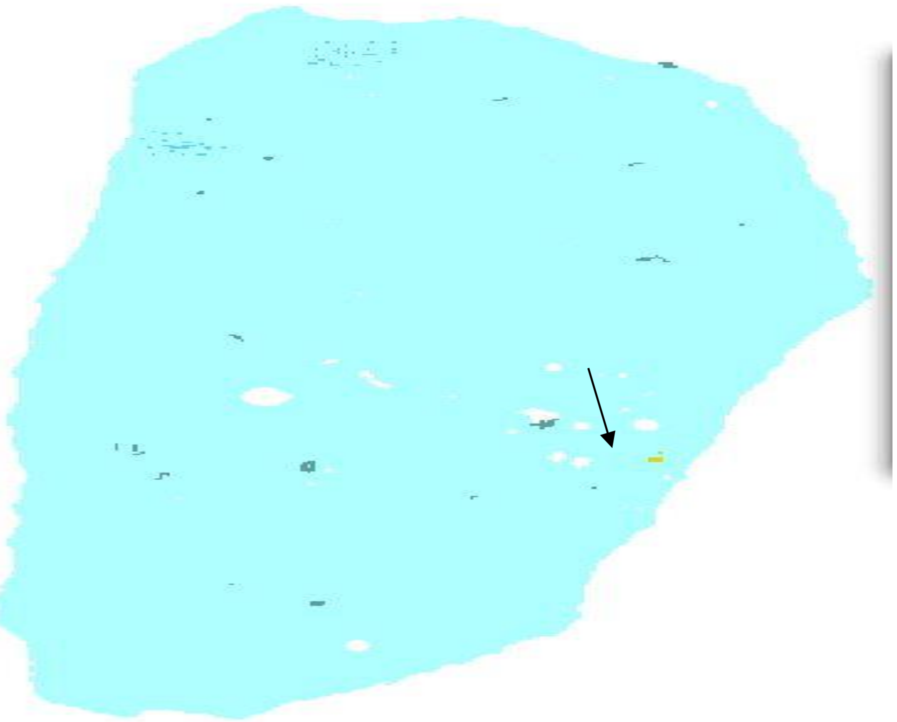


Figure 3.5. Image of Gold from MLA (+500 micron)

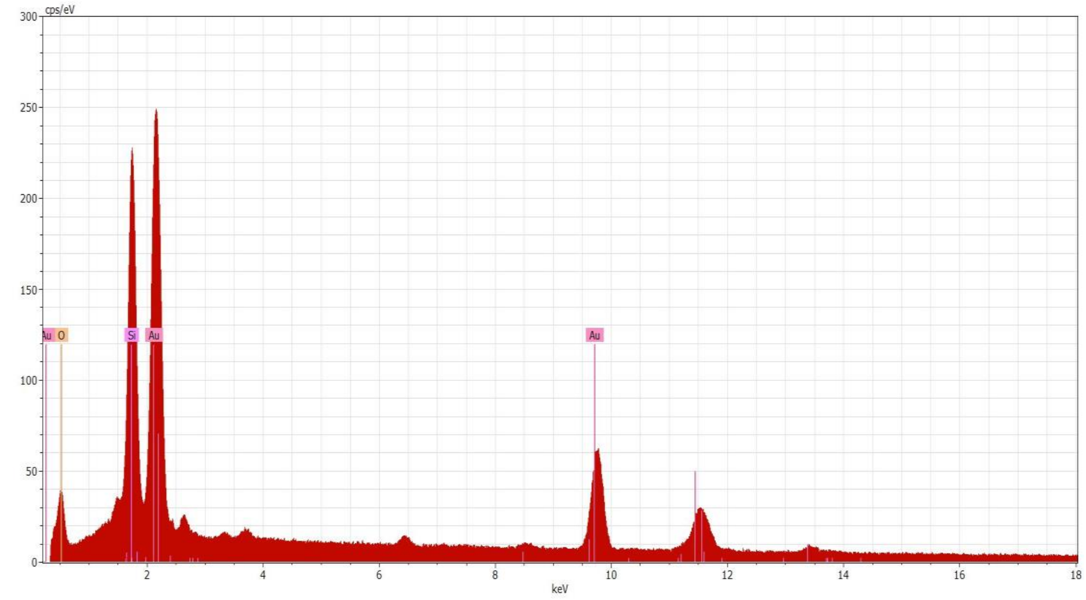


Figure 3.6. BSE-MLA Spectrum of Gold (+500 micron)

When figure 3-4, 3-5 and 3-6 were examined, BSE, MLA image and EDS spectrum images were available for the +500 micron gold mineral. As a result of the BSE images, the gold mineral for +500 µm was distributed in quartz.

When the spectrum belonging to the golden element for figure 3-6 was examined, it was observed that the electron volt value is 0-0,5, 2-2,50 and 9,50-10,0 keV. This means that the kinetic energy gained by one e- passing the difference in electrostatic potential changes to + 500 micron.

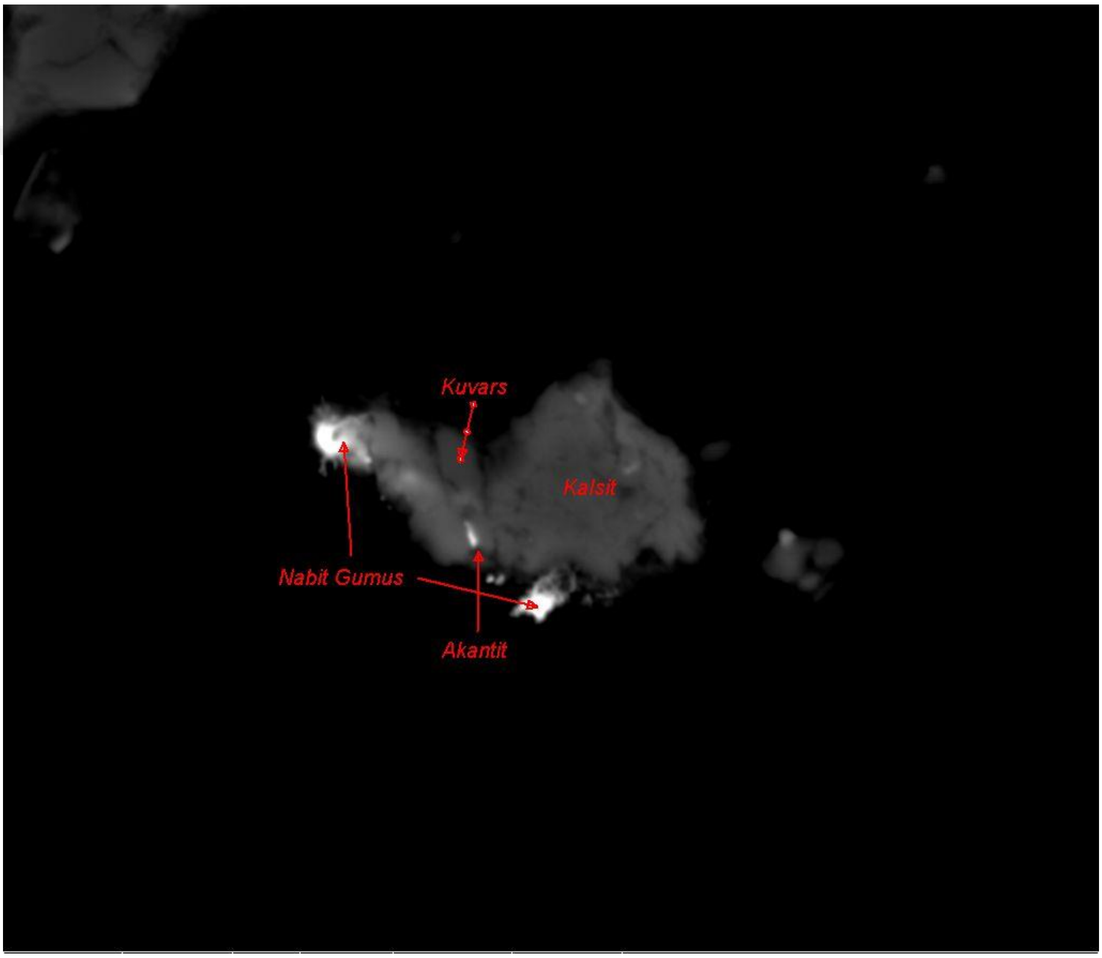


Figure 3.7. Image of Native Silver Mineral in Calcite (+212micron)

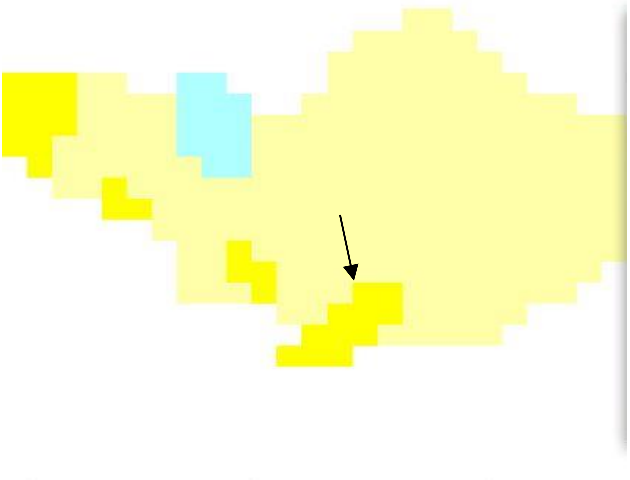


Figure 3.8. Image of Native Silver from MLA (+212 micron)

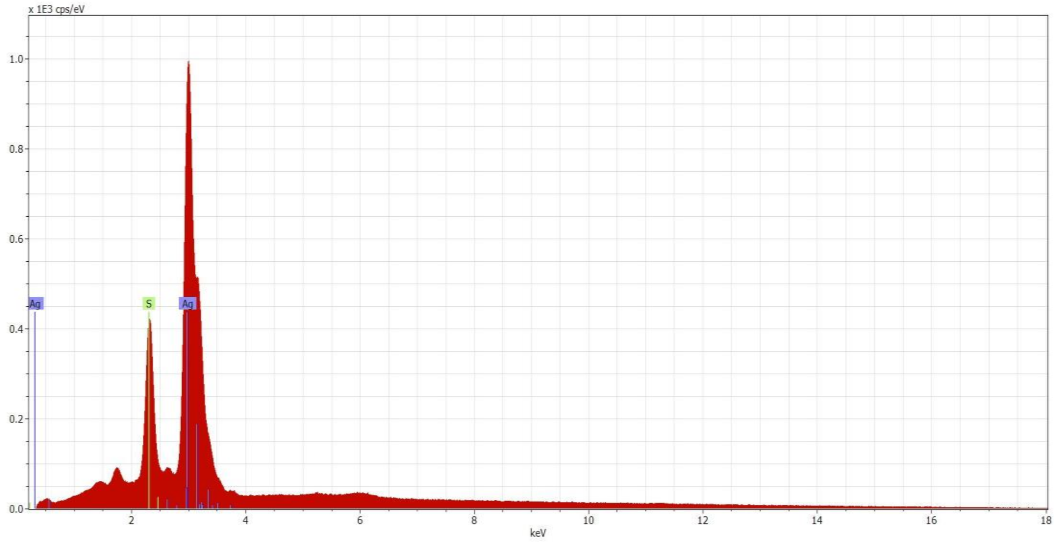


Figure 3.9. BSE-MLA Spectrum of Native Silver (+212 micron)

When the figure 3-7, 3-8 and 3-9 were examined, the BSE, MLA image and EDS spectrum images for native Silver of +212 Micron were included. As a result of the BSE images, native silver showed dispersion in quartz for +212 µm.

When the spectrum belonging to the silver element was examined for figure 3-9, it was observed that the electron volt value was 0-0,5 and 2.50-3.00 keV. This means that the kinetic energy gained by one electron passing the difference in electrostatic potential changes to + 212 micron.

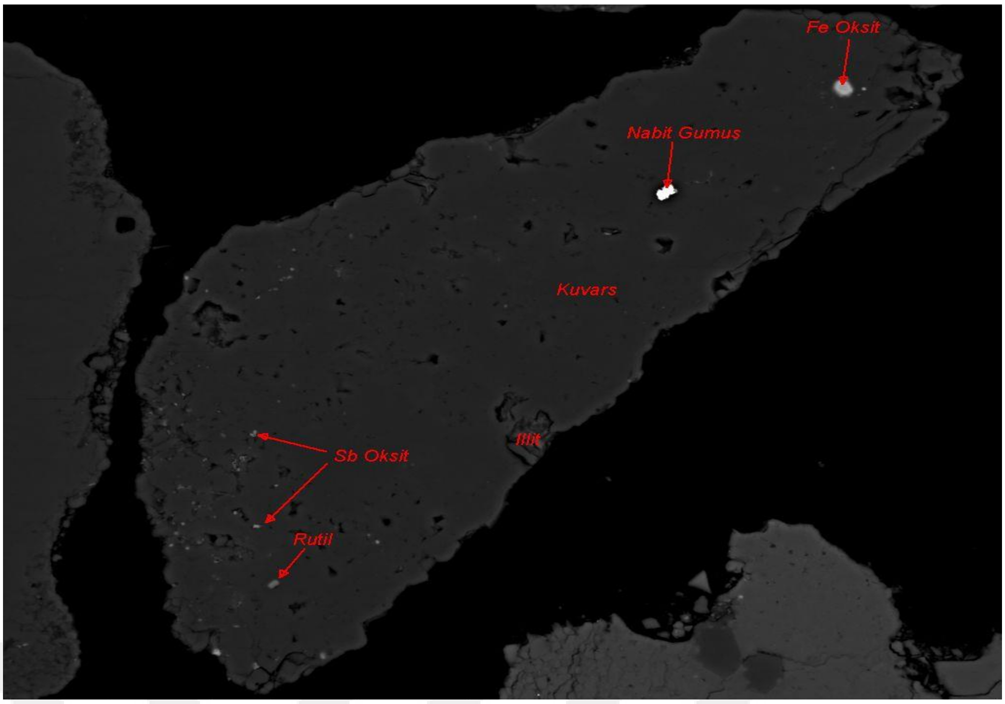
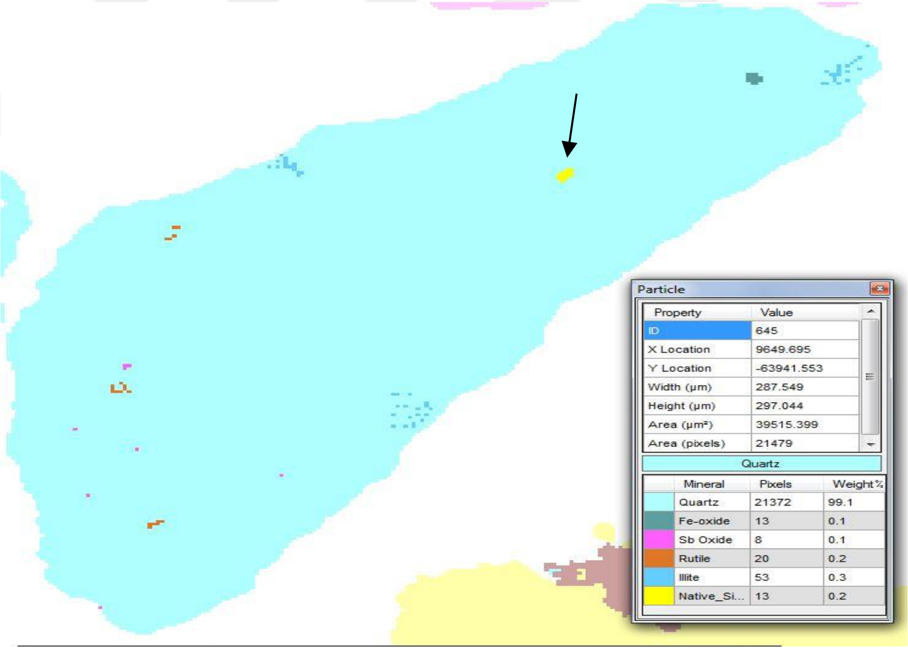


Figure 3.10. Image of Native Silver Mineral in Calcite (+212micron)



Property	Value
ID	645
X Location	9649.695
Y Location	-63941.553
Width (µm)	287.549
Height (µm)	297.044
Area (µm²)	39515.399
Area (pixels)	21479

Quartz

	Mineral	Pixels	Weight%
🟦	Quartz	21372	99.1
🟩	Fe-oxide	13	0.1
🟪	Sb Oxide	8	0.1
🟧	Rutile	20	0.2
🟦	Illite	53	0.3
🟨	Native_Si...	13	0.2

Figure 3.11. Image of Native Silver from MLA (+212 micron)

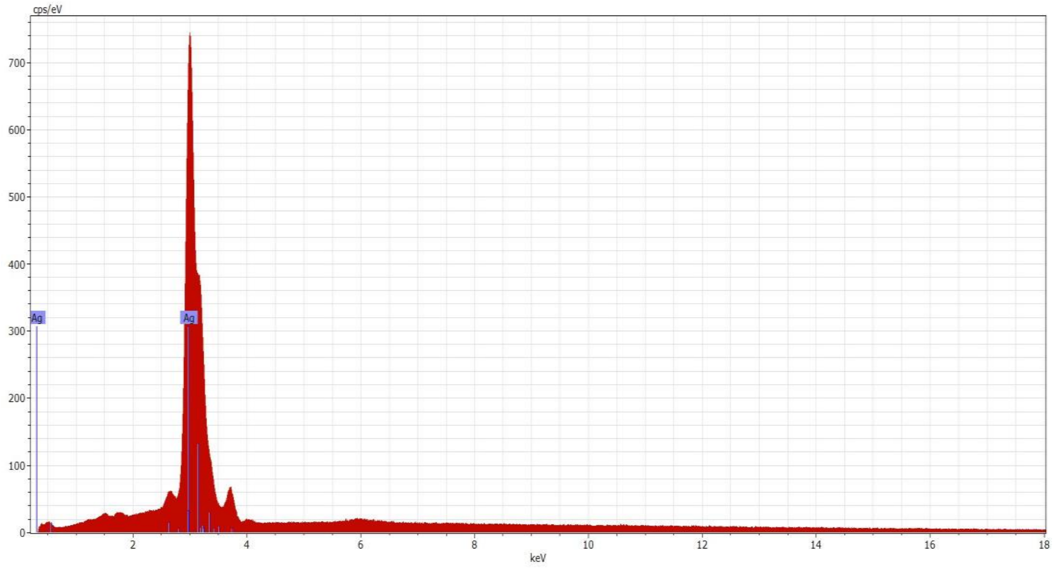


Figure 3.12. BSE-MLA Spectrum of Native Silver (+212 micron)

When the figure 3-10, 3-11 and 3-12 were examined, native silver was observed in calcite for a particle size of +212 micron.

When the native silver spectrum found in figure 3-12 was examined, it was determined that the electron volt value was between 2.00 and 4.00 kEV.

3.2. Result of Column Tests

These tests, which were carried out in 20mm and -12.5mm particle sizes, were performed in different time periods and detailed results are given below.

3.2.1 Column Test Results (Coarse Particle Size -20mm)

The particle size of the samples belonging to the first column tests to be performed in the crusher was broken to -20mm and sampled using coning, quaternary and riffle methods.

Sieve analysis was performed to observe the particle size distribution of the broken material and the results are given in Table 6.

Table 6. Sieve Analysis for -20mm Particle Size

Size Fraction (mm)	Weight (gr)	Dist. (%)	Cum. Oversize %	Cum. Undersize %
+20mm	0	0	0	0
-20mm+16mm	1250,1	27,92	27,92	100
-16mm+12,5mm	995,1	22,22	50,14	72,08
-12,5mm+8mm	842,5	18,81	68,95	49,86
-8mm+6,3mm	750,3	16,75	85,70	31,05
-6,3mm	640,2	14,30	100,00	14,30
TOTAL	**4478,2**	**100**		

In the table 7 shows the cyanide and lime consumption calculated per ton as a result of column tests performed in a particle size of-20mm.:

Table 7. NaCN and Lime Consumption per ton

Sample No	1	2	3	4	5
Lime Addition(g)	50	60	60	60	60
NACN Addition(mg)	34900	37200	37200	37300	40600
Remaining NaCN (mg)	864	922	960	882	1274
Lime Consumption (kg/t)	1,30	1,57	1,47	1,47	1,39
NaCN Consumption(kg/t)	0,89	0,95	0,89	0,89	0,91

When the table 7 was examined, it is seen that cyanide consumption of each column test is close to each other. Test results show that the average cyanide consumption will be about 0.90 kg / t when the leaching process is applied in the field.

During the column tests, samples were taken weekly from each column barrel for impurity analysis and element analyses were performed on the ICP device.

In the table 8, the results of the weekly impurity analysis of gold for the-20 mm column test samples are as follows:

Table 8. ICP Analysis Results for Au

Date	SAMPLE 1 (Au ppm)	SAMPLE 2 (Au ppm)	SAMPLE 3 (Au ppm)	SAMPLE 4 (Au ppm)	SAMPLE 5 (Au ppm)
14.11.2018	1,95	0,93	0,94	0,99	0,98
21.11.2018	2,20	1,1	1,1	1,13	1,15
28.11.2018	2,30	1,19	1,18	1,21	1,23
5.12.2018	2,32	1,15	1,1	1,11	1,13
13.12.2018	2,38	1,16	1,15	1,15	1,2
21.12.2018	2,47	1,16	1,16	1,18	1,21
26.12.2018	2,46	1,19	1,18	1,19	1,23
2.01.2019	2,48	1,2	1,18	1,20	1,23
14.01.2019	2,48	1,2	1,19	1,19	1,24

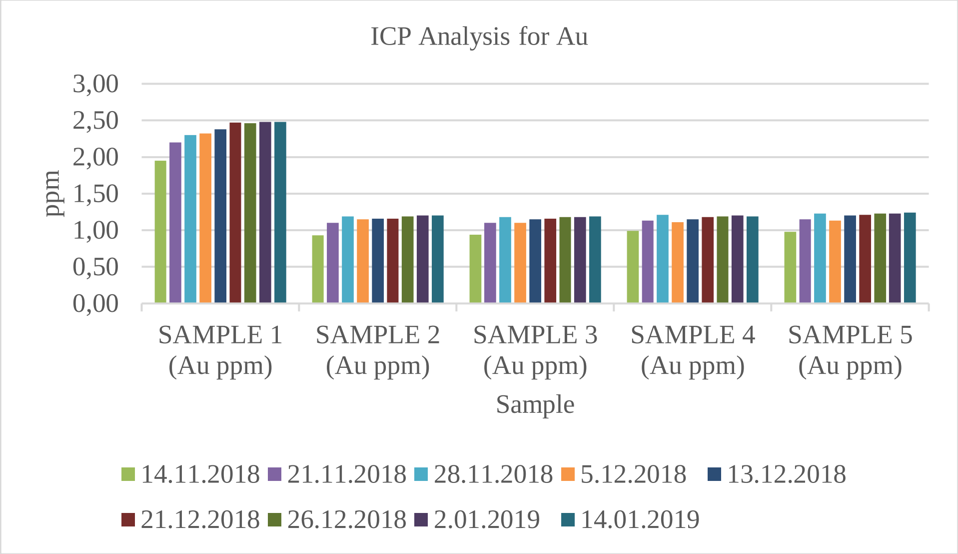


Figure 3.13. -20mm ICP Gold Analysis Results

When the figure 3-13 were examined, gold efficiency for the 1st column sample was almost fixed in the 3rd and 4th weeks, but gold efficiency continued to increase after the 4th week. Leaching that continued at the same tendecy in the 6th and 7th weeks increased its dissolution in the 8th week and gold recovery was completed by being fixed in the 9th week.

In the figure 3-13, after 2 weeks when leaching began for 2,3,4 and 5th column samples, gold efficiency was almost fixed. Gold efficiency remained constant throughout the leaching period.

The iron, iron-oxide and sulfur compounds have also been observed in the lodes where gold is also heavily present for this field. Cyanide can easily react with various iron minerals. For example, pyrite mineral is an iron mineral, and if it is found in the fields where gold is present, it can increase cyanide consumption. Iron and sulfur are the elements prone to bond with cyanide.

In the table 9, weekly iron impurity analysis results for-20mm column test samples are as follows:

Table 9. ICP Analysis Results for Fe

Date	SAMPLE 1 (Fe ppm)	SAMPLE 2 (Fe ppm)	SAMPLE 3 (Fe ppm)	SAMPLE 4 (Fe ppm)	SAMPLE 5 (Fe ppm)
14.11.2018	2,0	3,4	4,4	6,7	6,7
21.11.2018	2,1	3,8	4,5	7,0	7,0
28.11.2018	2,2	3,9	4,1	7,0	7,0
5.12.2018	2,6	4,2	5,1	6,8	7,1
13.12.2018	2,6	4,0	5,0	6,0	6,0
21.12.2018	2,7	4,6	5,5	6,7	6,7
26.12.2018	2,9	4,7	5,6	6,6	7,3
2.01.2019	3,9	6,7	6,3	6,2	7,8
14.01.2019	3,9	5,4	5,2	5,2	9,0

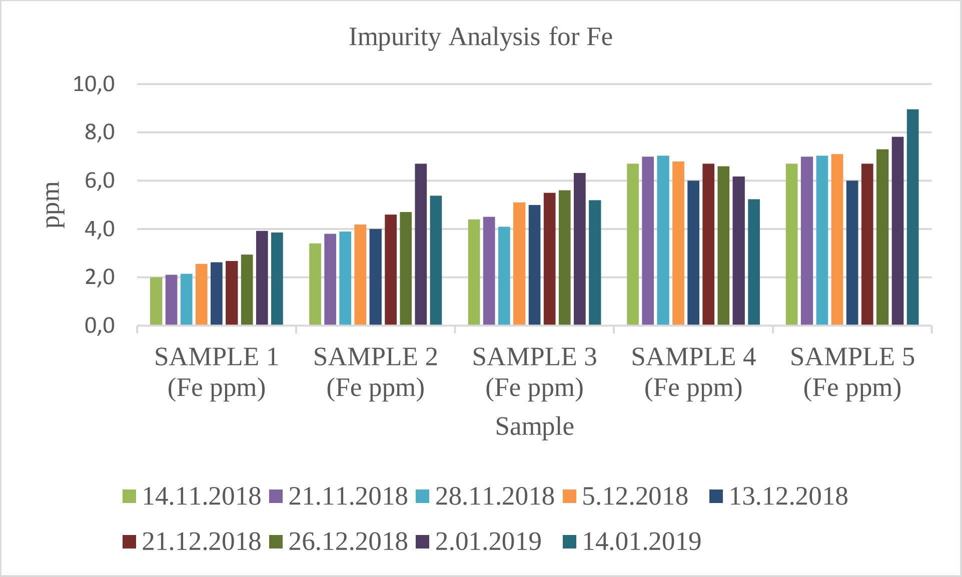


Figure 3.14. -20mm ICP Fe Analysis Results

The figure 3-14 examined, iron recovery of sample belonging to 5th column is observed to be higher compared to other samples. Iron efficiency fell in the 5th week, then increased again.

When the other column samples were examined, iron efficiency for column test-1 sample was fixed for the last weeks, while iron efficiency for samples 2, 3 and 4 increased in week 8.

In the table 10, the results of the nickel's weekly impurity analysis for the-20mm column test samples are as follows:

Table 10. ICP Analysis Results for Ni

Date	SAMPLE 1 (Ni ppm)	SAMPLE 2 (Ni ppm)	SAMPLE 3 (Ni ppm)	SAMPLE 4 (Ni ppm)	SAMPLE 5 (Ni ppm)
14.11.2018	0,68	1,7	3	5	6
21.11.2018	0,78	2,4	3,1	5,2	6,8
28.11.2018	0,85	2,1	3,1	5,7	6,9
5.12.2018	0,89	2,5	3,2	5,1	6,2
13.12.2018	0,93	2,9	3,01	4,47	4,67
21.12.2018	0,97	2,49	3,43	4,63	5,28
26.12.2018	1,05	2,58	3,06	4,72	6,88
2.01.2019	1,15	3,24	3,52	4,89	6,03
14.01.2019	1,17	2,96	3,41	4,51	6,04

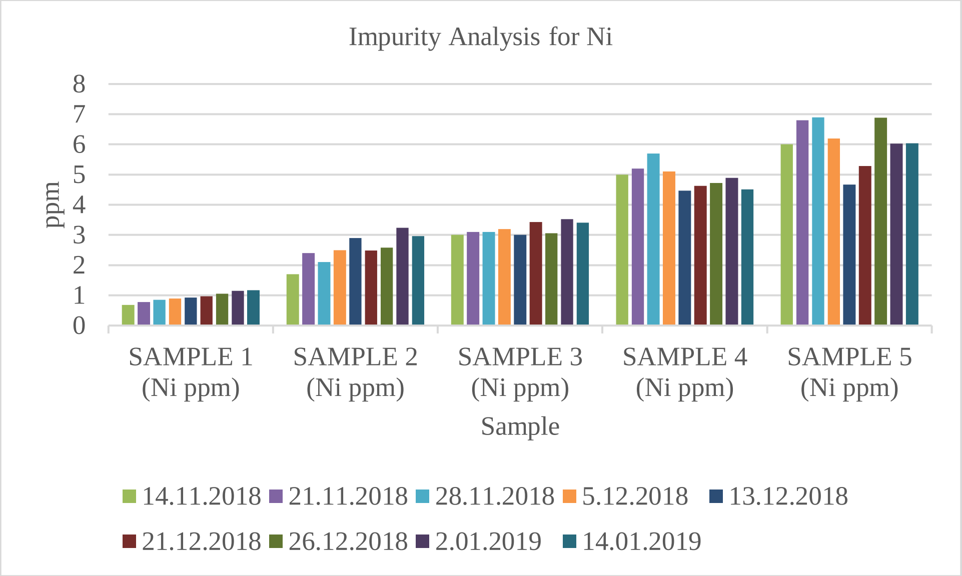


Figure 3.15. -20mm ICP Ni Analysis Results

When the figure 3-15 was examined, it was observed that the nickel content of the 1st column sample was lower than that of the other 4 column samples. This is an indication that the nickel metal is more collected in the contents of the samples in columns 2, 3, 4 and 5. The sample belonging to Column 5 had a higher efficiency in nickel metal as well as in iron metal.

Other than iron and iron minerals, another element prone to bond with cyanide is antimony. Antimony, depending on the region where the ore is found and its geographical structure, can be found extensively in the regions where the gold ore deposits are present. Antimony and antimony minerals are prone to bond with cyanide, increasing the consumption of cyanide in the region where it is found. If the antimony is not very fine scattering in the ore, it may not adversely affect cyanide leaching.

In the table 11, weekly impurity analysis results of antimony for-20mm column test samples were as follows:

Table 11. ICP Analysis Results for Sb

Date	SAMPLE 1 (Sb ppm)	SAMPLE 2 (Sb ppm)	SAMPLE 3 (Sb ppm)	SAMPLE 4 (Sb ppm)	SAMPLE 5 (Sb ppm)
14.11.2018	0,20	0,10	0,30	0,10	0,10
21.11.2018	0,20	0,10	0,50	0,20	0,20
28.11.2018	0,20	0,18	0,52	0,31	0,38
5.12.2018	0,28	0,22	0,61	0,40	0,60
13.12.2018	0,24	0,79	0,83	0,51	0,70
21.12.2018	0,26	0,84	0,87	0,60	0,76
26.12.2018	0,28	0,80	0,86	0,60	0,82
2.01.2019	0,30	0,80	0,77	0,60	0,98
14.01.2019	0,30	1,71	1,93	0,86	1,47

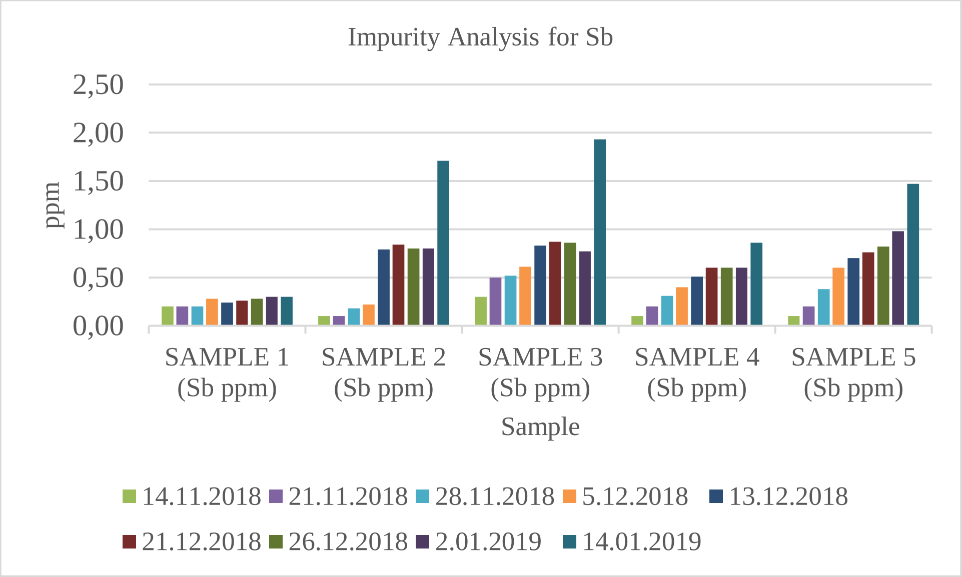


Figure 3.16. -20mm ICP Sb Analysis Results

When the figure 3-16 was examined, antimony efficiency was lower for the 1st column sample when compared to other columns. In column tests, the antimony efficiency of the other 4 columns was lower in the first weeks, although it underwent a rapid increase in the last week of leaching testing. Once the maximum gold efficiency is complete for these samples, the chart has shown a trend towards increasing antimony efficiency.

The most common zinc mineral found in various zinc minerals along with gold is sphalerite (ZnS). In table 12, the results of the weekly impurity analysis of zinc for the-20mm column test samples were as follows:

Table 12. ICP Analysis Results for Zn

Date	**SAMPLE 1 (Zn ppm)**	**SAMPLE 2 (Zn ppm)**	**SAMPLE 3 (Zn ppm)**	**SAMPLE 4 (Zn ppm)**	**SAMPLE 5 (Zn ppm)**
14.11.2018	2,7	1,9	2,2	3	4
21.11.2018	3	2,4	2,5	3,5	5
28.11.2018	3,6	2,5	2,7	3,6	5,6
5.12.2018	3,7	2,6	3,01	3,7	5,2
13.12.2018	3,4	2,37	3,01	3,2	4,06
21.12.2018	3,6	2,31	2,92	3,73	4,62
26.12.2018	3,5	2,4	2,84	3,4	4,89
2.01.2019	3,6	2,86	2,96	3,28	3,93
14.01.2019	4,27	2,63	2,7	4,7	3,8

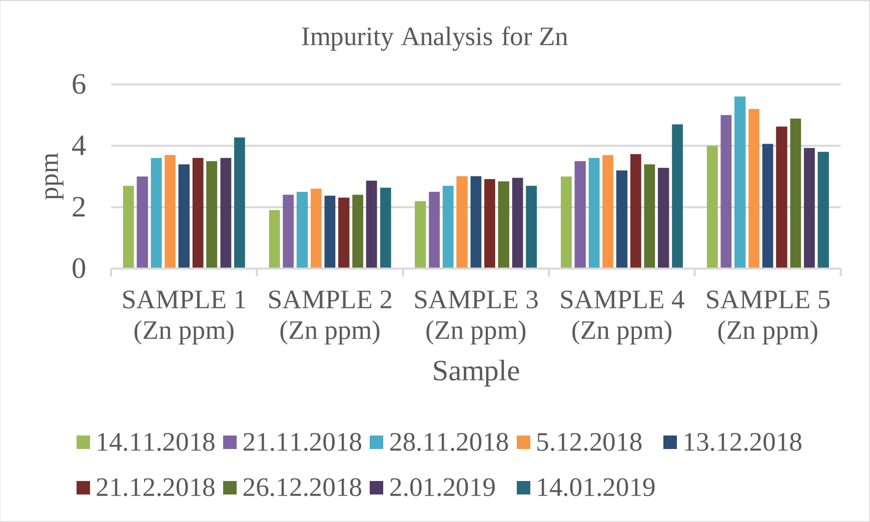


Figure 3.17. -20mm ICP Zn Analysis Results

When the figure 3-17 was examined, the zinc gain was lower for the 2nd and 3rd column tests, while the highest zinc gain was realized in the 5th sample. According to the analysis results of samples taken each week from the barrels of all columns, zinc recovery continued throughout the leaching period.

Table 13. Column Tests Results

	Sample-1	Sample -2	Sample -3	Sample -4	Sample -5
Solid Weight(g):	38394	38237,9	40810	40901	43221
Solution Weight(l) :	20	20	20	20	20
Leach Sample Au(g/t)	1,62	1,53	1,55	1,65	1,58
Leach Residue Au (g/t)	0,60	0,65	0,65	0,64	0,66
Recovery (%)	63%	58%	58%	61%	58%

When the table 13 was examined, the gold efficiency of the column tests applied as-20mm at the end of leaching was between 58% and 63%. The difference in gold efficiencies between these samples representing the same region was due to differences in particle size distribution and the placement of these samples in the column.

When the results of the tests carried out in a particle size of -20 mm were examined, it was found that gold leaching efficiency was highest in the 1st sample and therefore the best dissolution occurred in this sample. During the leaching period, the solution supplied to the column with the aid of the dosing pump penetrated the entire sample equally.

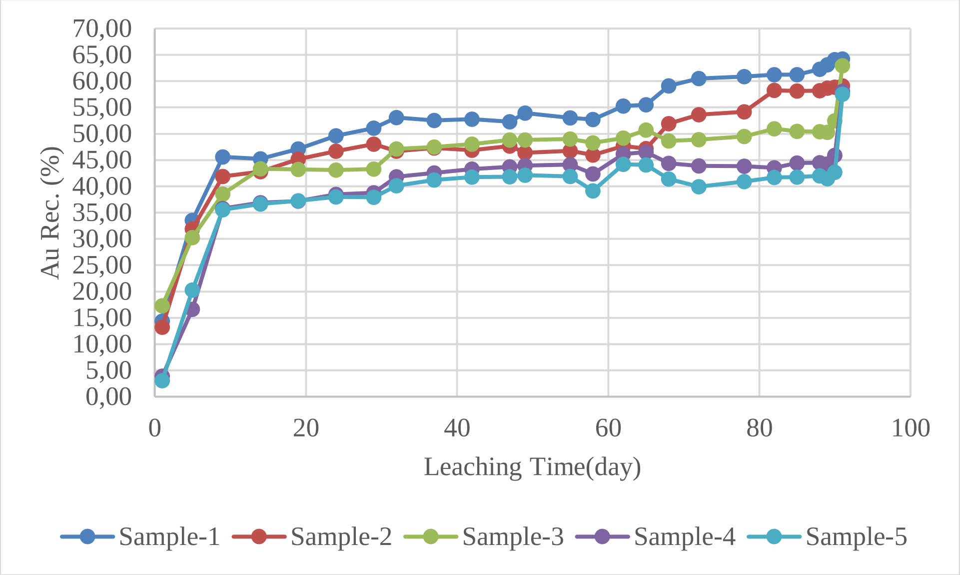


Figure 3.18. Column Tests Leach Kinetics

When the figure 3-18 leach kinetic graph of samples (crushed as -20mm) was examined, it was observed that the gold recovery rates of the samples have similar tendencies.

The gold recovery (%) increased very fast within the first 10 days. However, after the 10th day, the tendency of the increase become much slower. The gold recovery rate obtained as a result of this test tends to range between 58% and 63%.

For all samples, the overall trend is similar. As a result, sample 1 can be used to determine the upper limit, and sample 5 can be used to determine the lower limit for the determination of the leaching behaviour.

Table 14. The solution analysis results from samples during test:

Sample-1	Au(g/t)	Sample-2	Au(g/t)	Sample-3	Au(g/t)	Sample-4	Au(g/t)	Sample-5	Au(g/t)
Leach Sample	1,62	Leach Sample	1,53	Leach Sample	1,55	Leach Sample	1,65	Leach Sample	1,58
Leach Residue	0,6	Leach Residue	0,65	Leach Residue	0,65	Leach Residue	0,64	Leach Residue	0,66
Recovery(%)	63%	Recovery(%)	58%	Recovery(%)	58%	Recovery(%)	61%	Recovery(%)	58%
Leach Liquor Au (mg/lt)	**Au Recovery (%)**	**Leach Liquor Au (mg/lt)**	**Au Recovery (%)**	**Leach Liquor Au (mg/lt)**	**Au Recovery (%)**	**Leach Liquor Au (mg/lt)**	**Au Recovery (%)**	**Leach Liquor Au (mg/lt)**	**Au Recovery (%)**
0,17	14,37%	0,16	13,23%	0,23	17,31%	0,05	3,91%	0,04	3,08%
0,39	33,54%	0,40	31,89%	0,40	30,28%	0,22	16,63%	0,29	20,28%
0,53	45,62%	0,52	41,88%	0,51	38,56%	0,47	35,82%	0,51	35,56%
0,52	45,22%	0,53	42,77%	0,58	43,36%	0,48	36,92%	0,52	36,63%
0,54	47,13%	0,56	45,17%	0,57	43,20%	0,49	37,18%	0,53	37,24%
0,57	49,61%	0,58	46,68%	0,57	43,12%	0,50	38,48%	0,54	38,01%
0,59	51,05%	0,60	48,00%	0,57	43,26%	0,51	38,78%	0,54	37,91%
0,61	53,09%	0,58	46,71%	0,63	47,13%	0,55	41,84%	0,57	40,11%
0,61	52,53%	0,59	47,27%	0,63	47,46%	0,56	42,55%	0,59	41,20%
0,61	52,76%	0,58	46,90%	0,64	48,00%	0,57	43,26%	0,60	41,78%
0,60	52,30%	0,59	47,64%	0,65	48,81%	0,57	43,70%	0,60	41,83%
0,62	53,91%	0,58	46,40%	0,65	48,82%	0,58	43,98%	0,60	42,11%
0,61	53,03%	0,58	46,77%	0,65	49,01%	0,58	44,12%	0,60	41,89%
0,61	52,73%	0,57	45,97%	0,64	48,27%	0,55	42,40%	0,56	39,13%
0,64	55,25%	0,59	47,66%	0,65	49,19%	0,60	46,14%	0,63	44,19%
0,64	55,51%	0,59	47,20%	0,67	50,73%	0,61	46,51%	0,63	44,09%
0,68	59,10%	0,65	51,93%	0,64	48,62%	0,58	44,39%	0,59	41,40%
0,70	60,50%	0,67	53,63%	0,65	48,86%	0,57	43,90%	0,57	39,90%
0,70	60,87%	0,67	54,18%	0,66	49,51%	0,57	43,80%	0,58	40,88%
0,71	61,21%	0,72	58,23%	0,68	50,93%	0,57	43,55%	0,59	41,69%
0,71	61,25%	0,72	58,15%	0,67	50,46%	0,58	44,45%	0,60	41,79%
0,72	62,25%	0,72	58,22%	0,67	50,42%	0,58	44,50%	0,60	42,00%
0,73	63,12%	0,73	58,68%	0,67	50,26%	0,58	44,27%	0,59	41,48%
0,74	64,11%	0,73	58,85%	0,70	52,48%	0,60	45,91%	0,61	42,70%
0,74	64,20%	0,73	59,08%	0,84	62,96%	0,76	58,06%	0,82	57,52%

In the table 14 shows the results of gold analysis and weekly efficiency of the solution samples taken from barrels on a weekly basis. When the solid results and solution results of the column tests performed in -20 mm particle size were compared, it was observed that there was no significant difference between them. In the results of the solution obtained from the barrels, the instantaneous golden tenor was determined by the analysis performed in the ICP device and entered into the table. In order to calculate the gold efficiencies correctly, it is important that the volume of barrels belonging to each column and containing the solution loaded in it is not changed. For this reason, after the analysis of the samples taken from the barrels, each sample remaining in the sample container was re-added to the barrels of the columns to which they belong.

The analysis of these samples taken every day and after the leaching test is important for determining leaching kinetics.

3.2.2 Column Test Results (Fine Particle Size -12,5mm)

The results of the leaching test, which is the second of the column tests, carried out in a particle size of -12.5 mm, are below and the sieve analysis results of the samples belonging to this particle size were given in table 15.

Table 15. Sieve Analysis for -12,5mm Particle Size

Size Fraction (mm)		**Weight (gr)**	**Dist. (%)**	**Cum. Oversize %**	**Cum. Undersize %**
+12,5		0	0	0,0%	100,0%
-12,5	+8	2025	31,945102	31,9%	100,0%
-8	+6,3	869,7	13,71983	13,7%	68,1%
-6,3	+4,75	657,4	10,370721	10,4%	54,3%
-4,75	+2,36	1070,9	16,893832	16,9%	44,0%
-2,36	+1	772,5	12,186465	12,2%	27,1%
-1	+0,5	349,3	5,5103329	5,5%	14,9%
-0,5	+0,3	134,1	2,1154756	2,1%	9,4%
-0,3	+0,212	94,4	1,4891939	1,5%	7,3%
-0,212	+0,106	99,6	1,5712257	1,6%	5,8%
-0,106		266,1	4,197823	4,2%	4,2%
TOTAL		**6339**	**100**	**100,0%**	**100,0%**

Table 16. NaCN and Lime Consumption per ton

Sample No	6	7	8	9	10
Lime Addition(g)	22,5	43,8	28,2	35,2	25,3
NACN Addition(mg)	31810	35820	31540	34500	35820
Remaining NaCN (mg)	8920	9010	7500	8000	9010
Lime Consumption (kg/t)	1,34	2,22	0,99	1,15	1,46
NaCN Consumption(kg/t)	1,36	1,36	1,19	1,22	1,22

When the table 16 was examined, it was observed that cyanide consumption amounts for 5 column tests were close together. Compared to -20 mm, cyanide consumption is high because the particle size has been reduced to -12.5.

During the column tests, samples were taken weekly from each column barrel for impurity analysis and element analyses were performed on the ICP device.

In the table 17, the results of the weekly impurity analysis of gold for the -12,5 mm column test samples were as follows:

Table 17. ICP Analysis Results for Au

Date	SAMPLE 6 (Au ppm)	SAMPLE 7 (Au ppm)	SAMPLE 8 (Au ppm)	SAMPLE 9 (Au ppm)	SAMPLE 10 (Au ppm)
28.03.2019	0,87	0,75	0,72	0,68	0,71
10.04.2019	0,82	0,77	0,75	0,72	0,73
17.04.2019	0,87	0,81	0,83	0,75	0,78
25.04.2019	0,95	0,82	0,85	0,77	0,80
3.05.2019	0,94	0,82	0,88	0,82	0,82
9.05.2019	0,97	0,83	0,91	0,85	0,83
14.05.2019	0,96	0,85	0,91	0,86	0,85
22.05.2019	0,97	0,88	0,96	0,86	0,86
30.05.2019	0,99	0,91	0,95	0,88	0,88
12.06.2019	0,99	0,93	0,96	0,90	0,92

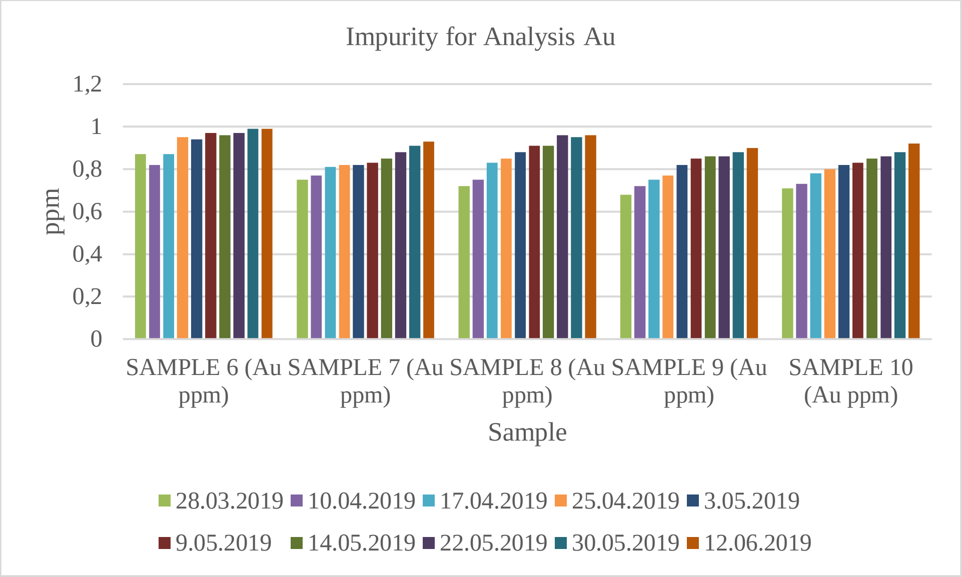


Figure 3.19. -12,5mm ICP Au Analysis Results

When the results of the -12.5 mm weekly impurity analysis in figure 3-19 were examined, gold efficiency showed an increase over the leaching period.

In the table 18, the results of the weekly impurity analysis of aluminum for the -12,5 mm column test samples were as follows:

Table 18. ICP Analysis Results for Al

Date	**SAMPLE 6 (Al ppm)**	**SAMPLE 7 (Al ppm)**	**SAMPLE 8 (Al ppm)**	**SAMPLE 9 (Al ppm)**	**SAMPLE 10 (Al ppm)**
28.03.2019	0,36	0,54	0,45	0,50	0,38
10.04.2019	0,36	0,6	0,48	0,53	0,42
17.04.2019	0,38	0,42	0,40	0,44	0,43
25.04.2019	0,40	0,65	0,53	0,58	0,45
3.05.2019	0,60	0,72	0,66	0,73	0,55
9.05.2019	0,69	0,75	0,72	0,79	0,58
14.05.2019	0,69	0,82	0,76	0,83	0,63
22.05.2019	0,64	0,93	0,79	0,86	0,67
30.05.2019	0,60	0,54	0,57	0,63	0,68
12.06.2019	0,47	0,56	0,52	0,57	0,67

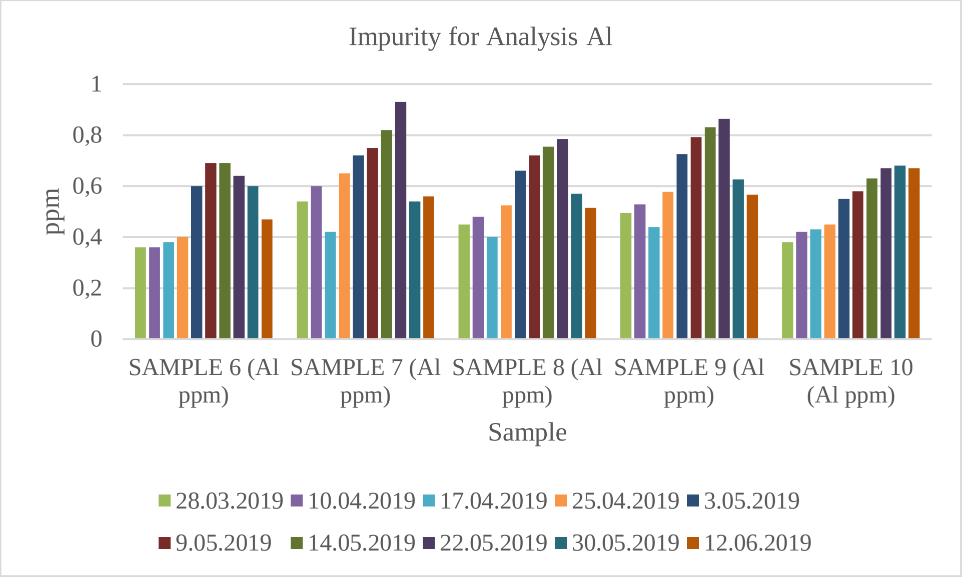


Figure 3.20. -12,5mm ICP Al Analysis Results

When the figure 3-20 was examined, aluminum, one of the transition metals, is one of the metals prone to bonding with cyanide. After the 5th week, Al dissolution began to increase and the reaction was completed towards the end of leaching. Aluminum efficiency during leaching period was high for column 7 and 9 samples compared to column 6, 8 and 10 samples, and dissolution was better.

In the table 19, the results of the weekly impurity analysis of iron for the -12,5 mm column test samples were as follows:

Table 19. ICP Analysis Results for Fe

Date	SAMPLE 6 (Fe ppm)	SAMPLE 7 (Fe ppm)	SAMPLE 8 (Fe ppm)	SAMPLE 9 (Fe ppm)	SAMPLE 10 (Fe ppm)
28.03.2019	0,59	1,47	0,65	0,78	1,10
10.04.2019	0,76	1,51	0,84	1,00	1,13
17.04.2019	0,79	1,62	0,87	1,04	1,22
25.04.2019	1,61	1,86	1,77	2,13	1,40
3.05.2019	1,91	2,22	2,1	2,52	1,67
9.05.2019	1,80	2,38	1,98	2,38	1,79
14.05.2019	1,42	2,44	1,56	1,87	1,83
22.05.2019	1,85	2,86	2,04	2,44	2,15
30.05.2019	1,33	2,83	1,46	1,76	2,12
12.06.2019	1,53	3,18	1,68	2,02	2,39

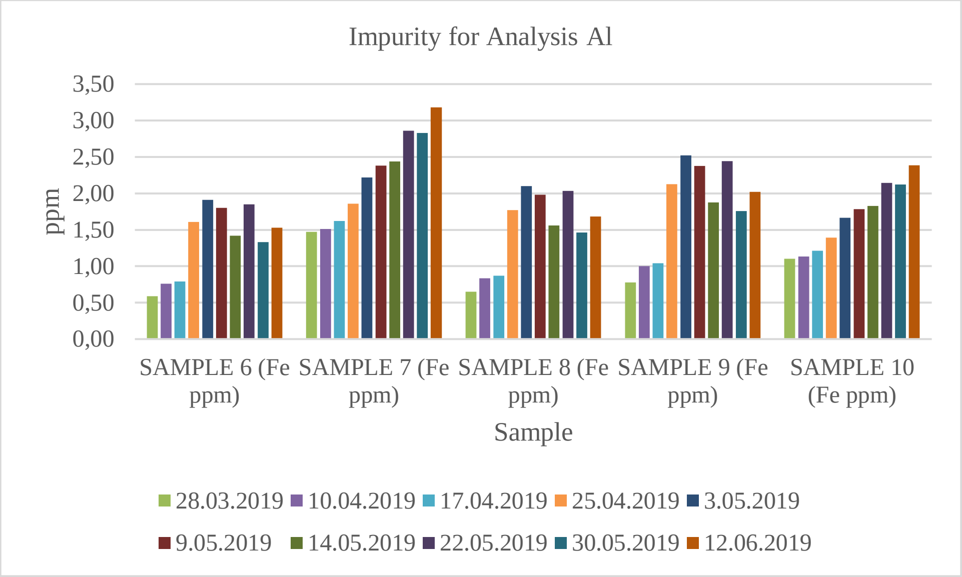


Figure 3.21. -12,5mm ICP Fe Analysis Results

When the figure 3-21 was examined, it was observed that the Iron efficiency of the 7th column sample was high during leaching. Once gold reaches maximum efficiency, cyanide began to bond in other metals. During the leaching period, iron efficiency of 6,8,9 and 10th column samples were examined depending on the weeks, and iron recovery rates were observed to be close to each other. As iron content increases in an ore, cyanide leaching test shows an increase in cyanide consumption.

In the table 20, weekly impurity analysis results of antimony for -12.5 mm column test samples were as follows:

Table 20. ICP Analysis Results for Sb

Date	**SAMPLE 6 (Sb ppm)**	**SAMPLE 7 (Sb ppm)**	**SAMPLE 8 (Sb ppm)**	**SAMPLE 9 (Sb ppm)**	**SAMPLE 10 (Sb ppm)**
28.03.2019	0,37	0,98	0,41	0,83	0,48
10.04.2019	0,3	1,71	0,33	1,45	0,39
17.04.2019	0,38	1,99	0,42	1,69	0,49
25.04.2019	0,3	2,35	0,33	2	0,39
3.05.2019	0,39	2,98	0,43	2,53	0,51
9.05.2019	0,4	3,3	0,44	2,81	0,52
14.05.2019	0,4	3,81	0,44	3,24	0,52
22.05.2019	0,4	4,65	0,44	3,95	0,52
30.05.2019	0,6	5,8	0,66	4,93	0,78
12.06.2019	0,7	6,9	0,77	5,87	0,91

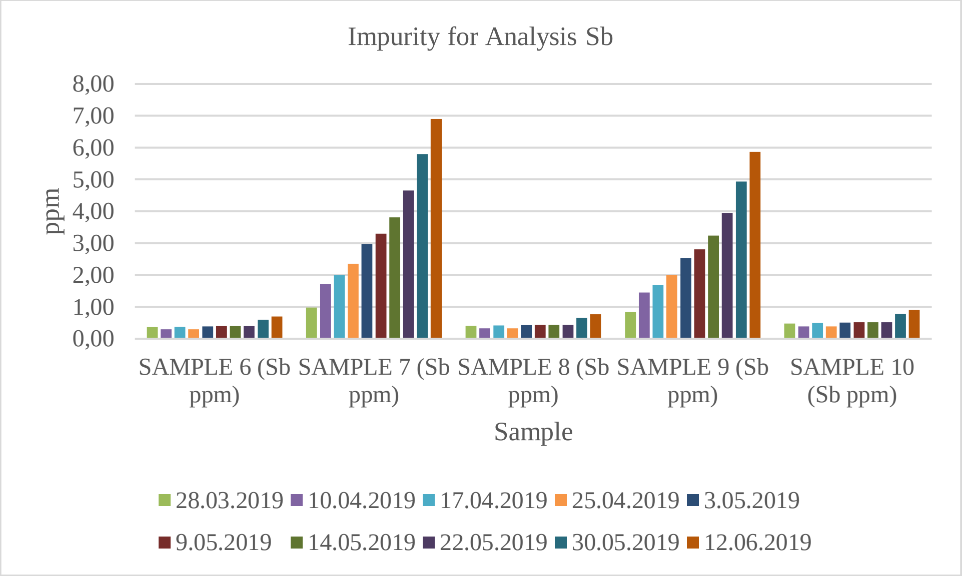


Figure 3.22. -12,5mm ICP Sb analysis results

When the figure 3-22 was examined, antimony was observed in all column tests. For the 7th and 9th column samples, antimony efficiency increased linearly during the test period, and for the 6th, 8th and 10th column samples, antimony dissolution was very low. When the higher the antimony content found in the ore, cyanide consumption increases.

In the table 21, the results of the weekly impurity analysis of zinc for the -12,5 mm column test samples were as follows:

Table 21. ICP Analysis Results for Zn

Date	SAMPLE 6 (Zn ppm)	SAMPLE 7 (Zn ppm)	SAMPLE 8 (Zn ppm)	SAMPLE 9 (Zn ppm)	SAMPLE 10 (Zn ppm)
28.03.2019	1,35	1,13	1,24	1,11	1,06
10.04.2019	1,68	1,21	1,33	1,19	1,13
17.04.2019	1,81	1,42	1,56	1,39	1,33
25.04.2019	1,6	1,29	1,42	1,26	1,21
3.05.2019	1,55	1,96	2,16	1,92	1,83
9.05.2019	2,3	1,98	2,18	1,94	1,85
14.05.2019	2,52	2,02	2,22	1,98	1,89
22.05.2019	2,61	2,15	2,37	2,11	2,01
30.05.2019	2,9	2,78	3,06	2,72	2,60
12.06.2019	3,01	2,99	3,29	2,93	2,80

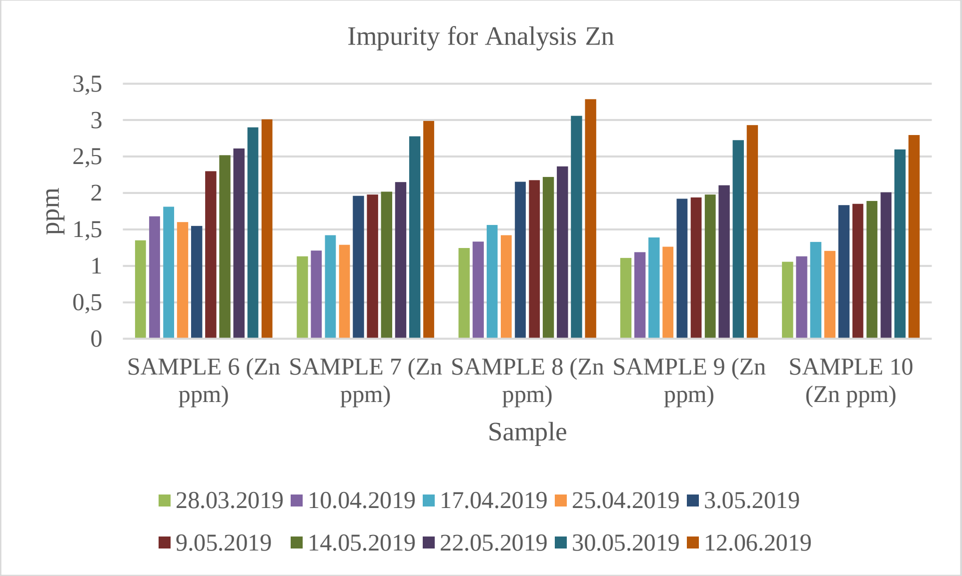


Figure 3.23. -12,5mm ICP Zn analysis results

When the figure 3-23 was examined, zinc efficiency increased in all column samples after the 4th week, and the highest zinc efficiency at the end of leaching was realized in the 8th column sample.

Table 22. Column Tests Results

	Sample-6	Sample -7	Sample -8	Sample -9	Sample -10
Solid Weight(g):	16801	19711	20130	21711	21911
Solution Weight(l) :	20	20	20	20	20
Leach Sample Au(g/t)	1,67	1,55	1,62	1,58	1,63
Leach Residue Au (g/t)	0,50	0,35	0,45	0,40	0,45
Recovery (%)	70%	77%	72%	75%	72%

When the table 22 was examined, gold efficiency at the end of leaching of column tests as -12.5 mm was observed between 70% and 77%. The difference in gold efficiencies between these samples representing the same region was due to differences in particle size distribution and the placement of these samples in the column.

When the results of the tests carried out in a particle size of -12,5 mm were examined, it was observed that gold leaching efficiency was highest in the 7th sample and therefore the best dissolution occurred in this sample. During the leaching period, the solution supplied to the column with the aid of the dosing pump penetrated the samples of all the columns equally.

In another project, 8 different column tests were performed and each test utilized material crushed to the size of 100% passing 9.5 millimeters. During testing, the material was leached for 143 days with a sodium cyanide solution. Periodic samples of the column solutions were taken throughout the leaching period and laboratory analyses were performed (Eagle Gold Project, December 2013).

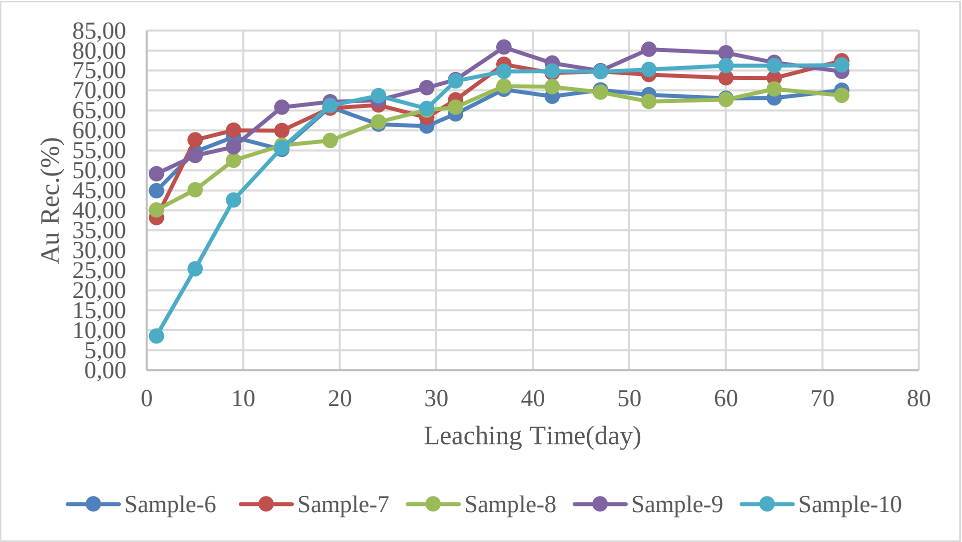


Figure 3.24. Column Tests Leach Kinetics

When the figure 3-24 leach kinetic graph of samples (crushed as -12,5 mm) was examined, it was observed that the gold recovery tendencies of the leaching tests were close to each other.

The gold recovery rates for -12.5mm particles were significantly higher at the end of the 1st day comparing to the previous leaching test carried out for -20mm particles.

The gold recovery rate obtained as a result of this test was tent to range between 70% and 77%.

For all samples, the overall leaching performance has a similar tendencies. As a result, sample 9 can be used to determine the upper limit, and sample 10 can be used to determine the lower limit for the determination of the leaching behaviour.

On the other hand, trials were performed on 3 different composites and one of the samples leached faster than the others. After 10 days, almost 80% of the gold passed to the solution. After the first 10 days, leaching kinetics slowed down. At the end of 54 days, the gold recovery realized as 86.06%, 89.51% and 87.72% for samples 1-2 and 3, respectively. The highest gold recovery was achieved in Example 2. (Erkan, 2015).

Table 23. The solution analysis results from samples during test

Sample-6	Au(g/t)	**Sample-7**	Au(g/t)	**Sample-8**	Au(g/t)	**Sample-9**	Au(g/t)	**Sample-10**	Au(g/t)
Leach Sample	1,67	**Leach Sample**	1,55	**Leach Sample**	1,62	**Leach Sample**	1,58	**Leach Sample**	1,63
Leach Residue	0,50	**Leach Residue**	0,35	**Leach Residue**	0,45	**Leach Residue**	0,4	**Leach Residue**	0,45
Recovery(%)	70%	**Recovery(%)**	77%	**Recovery(%)**	72%	**Recovery(%)**	75%	**Recovery(%)**	72%
Leach Liquor Au (mg/lt)	**Au Recovery (%)**	**Leach Liquor Au (mg/lt)**	**Au Recovery (%)**	**Leach Liquor Au (mg/lt)**	**Au Recovery (%)**	**Leach Liquor Au (mg/lt)**	**Au Recovery (%)**	**Leach Liquor Au (mg/lt)**	**Au Recovery (%)**
0,63	44,88%	0,58	38,14%	0,65	40,11%	0,84	49,15%	0,15	8,55%
0,77	54,63%	0,88	57,61%	0,74	45,10%	0,92	53,73%	0,45	25,35%
0,82	58,35%	0,92	60,04%	0,86	52,47%	0,96	55,88%	0,76	42,55%
0,77	55,19%	0,92	59,94%	0,92	56,23%	1,13	65,82%	0,99	55,66%
0,92	65,88%	1,00	65,55%	0,94	57,45%	1,15	67,19%	1,18	66,23%
0,86	61,59%	1,01	66,39%	1,01	62,11%	1,16	67,54%	1,23	68,70%
0,86	61,05%	0,97	63,20%	1,06	65,07%	1,21	70,71%	1,17	65,45%
0,90	64,14%	1,03	67,61%	1,07	65,83%	1,25	72,65%	1,29	72,39%
0,99	70,26%	1,17	76,51%	1,16	71,08%	1,39	80,82%	1,34	74,78%
0,96	68,53%	1,14	74,37%	1,16	70,90%	1,32	76,86%	1,34	74,85%
0,98	70,14%	1,14	74,73%	1,13	69,59%	1,29	74,93%	1,33	74,68%
0,97	68,91%	1,13	73,96%	1,10	67,24%	1,38	80,26%	1,34	75,24%
0,95	68,00%	1,12	73,16%	1,10	67,75%	1,36	79,45%	1,36	76,23%
0,96	68,09%	1,12	73,12%	1,15	70,38%	1,32	77,01%	1,36	76,23%
0,98	70,06%	1,18	77,42%	1,12	68,75%	1,28	74,78%	1,36	76,35%

In table 23 shows the results of gold analysis and weekly efficiency of the solution samples taken from barrels on a weekly basis. When the solid results and solution results of the column tests performed in 12,5 mm particle size were compared, it was observed that there was no significant difference between them. In the results of the solution obtained from the barrels, the instantaneous golden tenor was determined by the analysis performed in the ICP device and entered into the table. In order to calculate the gold efficiencies correctly, it is important that the volume of barrels belonging to each column and containing the solution loaded in it is not changed. For this reason, after the analysis of the samples taken from the barrels, each sample remaining in the sample container was re-added to the barrels of the columns to which they belong.

The analysis of these samples taken every day and after the leaching test is important for determining leaching kinetics.

3.3. Tank (Agitation) Leach Test Results

Mixing test leaching mechanism was created for 4 different samples with a particle size of -0.075 mm. Leaching kinetics, cyanide consumption and time-dependent gold efficiency of each sample were examined and shown in the tables and graphs below.

The total cyanide and lime consumption according to the mixing leaching test results performed in a particle size of -0.075 mm were given in Table 24:

Table 24. NaCN and Lime Consumption per ton

Sample No	1	2	3	4
Lime Addition(g)	0,90	1,50	1,52	0,80
NACN Addition(mg)	2910	3010	3100	3020
Remaining NaCN (mg)	1333	1392	1372	1352
Lime Consumption (kg/t)	0,90	1,50	1,52	0,80
NaCN Consumption(kg/t)	1,58	1,62	1,73	1,67

When the table 24 was examined, cyanide consumption was observed between 1.58 kg / t and 1.67 kg / t. The cyanide consumption of the sample found in the 1st beaker was calculated to be less than the sample found in the other 3 beakers. The cyanide consumption varies according to the content of the elements found in the ore.

In another leaching test with a particle size of -0,075 mm, cyanide is added depending on the gold recovery rate and the gold recovery increases to 95% at the end of the leaching. As the particle size decreased, the increase in cyanide concentration decreased the recovery (Sayın, Haziran 2010).

Before starting the tank leaching test, samples were given to the laboratory for analysis of all elements from the sample milled to -0.075 mm.

Mixing leaching tests lasted 72 hours, unlike column tests. In order to accurately calculate leaching efficiency, it is also important for this test not to decrease the volume of prepared samples. In a short test period of 72 hours, only the pH, NaCN and gold analyses were performed at the specified time intervals and the samples increased after the analysis were added to the beakers.

In the table 25 gave the initial impurity results of the sample used for the test:

Table 25. Initial Impurity Results of Tank Leach Tests

Element (ppm)	SAMPLE-1 LEACH INITIAL SOLID	SAMPLE-2 LEACH INITIAL SOLID	SAMPLE-3 LEACH INITIAL SOLID	SAMPLE-4 LEACH INITIAL SOLID
As(ppm)	500	470	655	452
Ni(ppm)	65	65	70	40
Sb(ppm)	740	550	550	340
Zn(ppm)	105	95	95	115

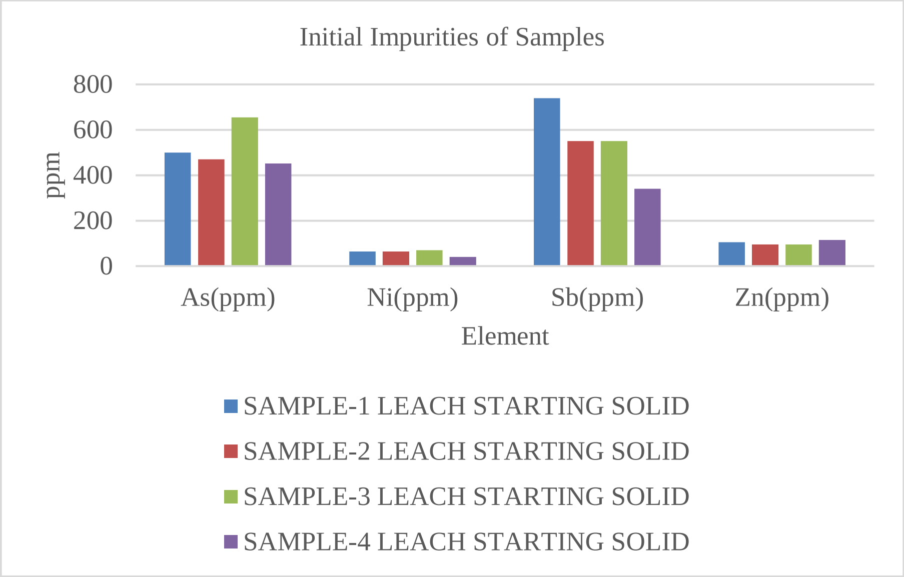


Figure 3.25. -0,075mm Initial ICP Results

When the figure 3-25 was examined, according to the results of the impurity analysis performed at the beginning of the leaching test, it was found that the samples contain nickel and zinc metal in a small amount and the arsenic and antimony were found much higher than the other metals. High arsenic and antimony mean that the amount of cyanide consumption will increase, which is a sign that the analysis of the sample gives us. Furthermore, antimony is often found in combination with bismuth and arsenic minerals. For this reason, it is highly likely that antimony is present at the site of arsenic.

Table 26. Residue Impurity Results of Tank Leach Tests

Element (ppm)	SAMPLE-1 LEACH END SOLID	SAMPLE-2 LEACH END SOLID	SAMPLE-3 LEACH END SOLID	SAMPLE-4 LEACH END SOLID
As(ppm)	450	444	570	400
Ni(ppm)	60	60	64	37
Sb(ppm)	650	430	500	280
Zn(ppm)	81	62	61	105

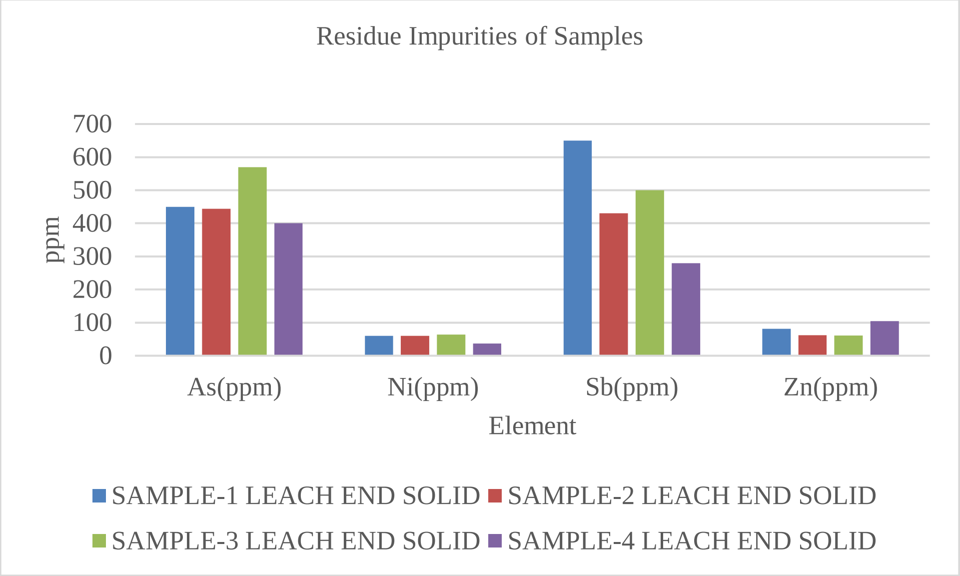


Figure 3.26. -0,075mm Residue ICP Results

When the (Figure 3-26) analysis results of the samples taken at the end of the leaching and at the beginning of the leaching were compared, it was observed that there was not much change in the efficiency of the other metals in the ore for 72 hours.

Table 27. Recovery of Initial and Final Impurities

	SAMPLE				
		As(ppm)	Ni(ppm)	Sb(ppm)	Zn(ppm)
		ICP	ICP	ICP	ICP
Initial	Sample 1	500	65	740	105
Final	Sample 1	450	60	650	81
Recovery(%)		10%	8%	12%	23%
Initial	Sample 2	470	65	550	95
Final	Sample 2	444	60	430	62
Recovery(%)		6%	8%	22%	35%
Initial	Sample 3	655	70	550	95
Final	Sample 3	570	64	500	61
Recovery(%)		13%	9%	9%	36%
Initial	Sample 4	452	40	340	115
Final	Sample 4	400	37	280	105
Recovery(%)		12%	8%	18%	9%

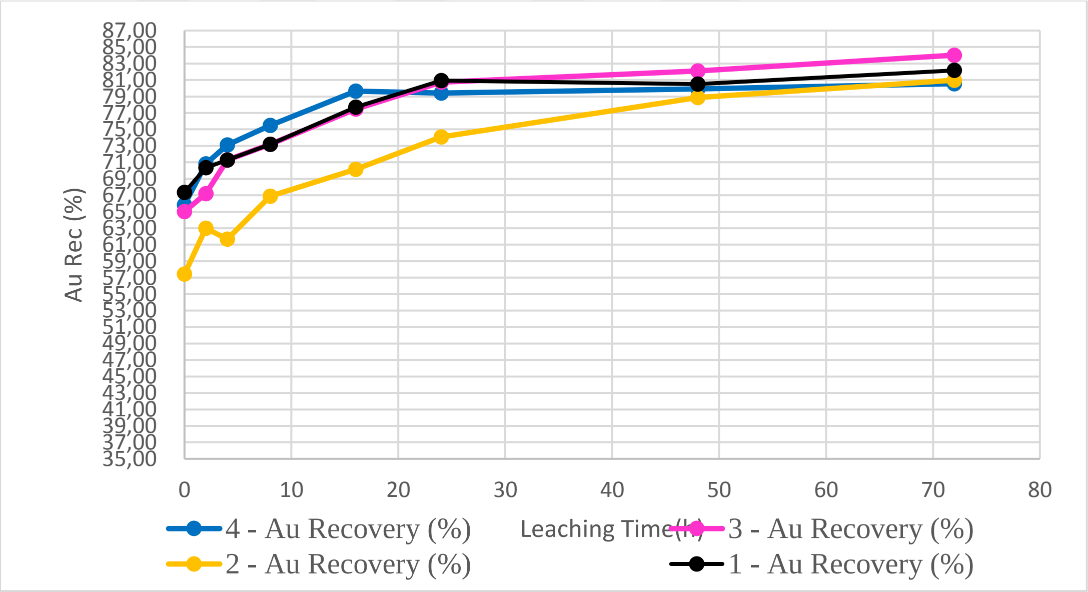


Figure 3.27. Tank Leach Test Kinetics

When the figure 3-27 leach kinetic graph of samples ground as -0,075 mm was examined, it was observed that the gold recovery rates of the samples have similar increasing tendencies.

The gold recovery rates were relatively faster within the first day. When the first day of the leaching is completed, the leaching rate of the gold become slower as shown in above figure.

The gold recovery rate obtained as a result of this test tends to range between 78% and 82%.

For all samples, the overall trend is similar. As a result, sample 3 can be used to determine the upper limit, and sample 2 can be used to determine the lower limit.

In a different study, similar results were obtained at -0,075 mm and -0,053 mm particle sizes and coarser particle sizes were found lower in this concentration ratio. For this ore, it would be appropriate to select -0.075 mm of working size considering the grinding costs (Sayın, Haziran 2010).

Table 28. The Solution Analysis Results from Samples During Test

Leaching Time (h)	Sample-1		Au(g/t)	Sample-2		Au(g/t)	Sample-3		Au(g/t)	Sample-4		Au(g/t)
	Leach Sample		1,70	Leach Sample		1,17	Leach Sample		1,10	Leach Sample		0,93
	Leach Residue		0,30	Leach Residue		0,22	Leach Residue		0,20	Leach Residue		0,20
	Recovery(%)		82%	Recovery(%)		81%	Recovery(%)		82%	Recovery(%)		78%
	Leach Liquor Au (mg/lt)	**Pulp Weight(g)**	**Au Recovery (%)**	**Leach Liquor Au (mg/lt)**	**Pulp Weight(g)**	**Au Recovery (%)**	**Leach Liquor Au (mg/lt)**	**Pulp Weight(g)**	**Au Recovery (%)**	**Leach Liquor Au (mg/lt)**	**Pulp Weight(g)**	**Au Recovery (%)**
		2021			2018,40			2007			2007,2	
0	1,10	2040,76	67,34%	0,65	2033,50	57,42%	0,70	2021,8	65,02%	0,60	2020,6	65,85%
2	1,15	2039,7	70,33%	0,72	2023,90	63,01%	0,75	1985,7	67,21%	0,65	2013,1	70,81%
4	1,19	2018,3	71,28%	0,75	1962,40	61,69%	0,75	2045,6	71,29%	0,68	1999,9	73,11%
8	1,22	2020,0	73,20%	0,80	1978,50	66,91%	0,78	2032,3	73,20%	0,70	2002,8	75,48%
16	1,27	2040,0	77,69%	0,82	2001,10	70,16%	0,80	2065,4	77,48%	0,69	2073,6	79,65%
24	1,31	2050,0	80,91%	0,85	2019,60	74,07%	0,82	2082,9	80,73%	0,69	2070,5	79,42%
48	1,35	2013,7	80,50%	0,89	2036,90	78,88%	0,85	2062,3	82,09%	0,73	2018,0	79,91%
72	1,40	1997,7	82,16%	0,93	2018,70	80,97%	0,87	2062,3	84,02%	0,75	1998,8	80,55%

Table 28 showed the gold analysis results and hourly efficiency of the solution samples taken from barrels on a weekly basis. When the solid results and solution results of mixing leaching tests performed at 0.075mm particle size were compared, there was no significant difference between them. In the solution results obtained from each beaker, the instantaneous gold tenor was determined by the analysis performed on the

ICP device and entered into the table. In order to calculate the gold efficiencies correctly, it is important that the volume of beakers belonging to each beaker with a loaded solution in it does not change.

Table 29. Tank Leach Tests Results

Sample No	1	2	3	4
Solid Weight(g):	1000	1000	1000	1000
Solution Weight(ml) :	1000	1000	1000	1000
Leach Sample Au(g/t)	1,70	1,17	1,10	0,93
Leach Residue Au (g/t)	0,30	0,22	0,20	0,20
Recovery (%)	82%	81%	82%	78%

When the table 29 was examined, the gold efficiency at the end of leaching was observed between 78% and 82% of the mixing leaching tests applied as -0.075 mm. For these homogenously prepared samples, gold efficiency was very close to each other, and compared to column tests, gold efficiency was higher at the end of leaching.

When the results of the tests carried out with a particle size of -0.075 mm were examined, it was found that gold leaching efficiency was higher in 1st and 3rd samples, and dissolution was best realized in these two samples.

3.4. Comparison of Cyanide Consumption

In addition to gold efficiency, lime and cyanide consumption were examined for all these tests, and in column and mixing leaching tests, it was observed that cyanide consumption differed depending on particle size. Lime consumption is not related to particle size, but to the pH value of the tests and the properties of the metal they contain.

In the table 30 shows the consumption chart of cyanide consumed per ton for column tests carried out in a particle size of -20 mm.

Table 30. Coarse Size NaCN Consumption

Column (coarse size -20mm)	NaCN Consumption (kg/t)
Column Leach 1	0,89
Column Leach 2	0,95
Column Leach 3	0,89
Column Leach 4	0,89
Column Leach 5	0,91
Average NaCN Consumption	0,91

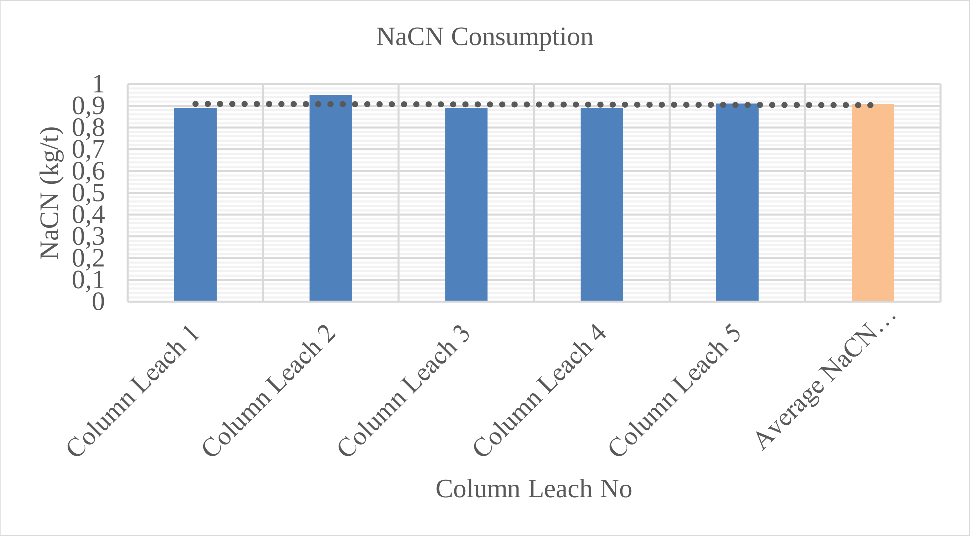


Figure 3.28. NaCN Consumption for Coarse Particle Size

When the figure 3-28 was examined, it was observed that cyanide consumption was close to each other in column tests performed in a particle size of -20 mm. Cyanide consumption of all samples are between 0.89 kg / t and 0.95 kg / t and the average consumption was 0.91 kg / t. Cyanide consumption did not show any difference between sample types.

In the table 31 shows the consumption chart of cyanide consumed per ton for column tests carried out in particle size -12.5 mm.

Table 31. Fine size NaCN consumption

Column (fine size -12,5mm)	**NaCN Consumption (kg/t)**
Column Leach 6	1,36
Column Leach 7	1,36
Column Leach 8	1,19
Column Leach 9	1,22
Column Leach 10	1,22
Average NaCN Consumption	1,27

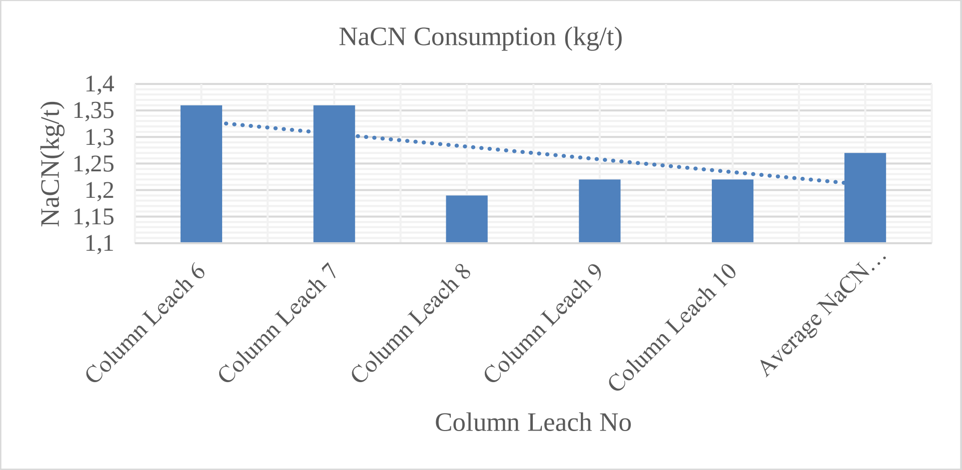


Figure 3.29. NaCN Consumption for Fine Particle Size

When the figure 3-29 was examined, it was observed that cyanide consumption was the same in column tests 6-7 and column tests 9-10 in column tests conducted in particle size -12.5 mm. The cyanide consumption of the No 8 column test was examined to be lower than the cyanide consumption compared to other column tests. Cyanide consumption for these samples is approximately between 1.19 kg / t and 1.36 kg / t and the average consumption was 1.27 kg / t. Cyanide consumption did not show any difference between sample types.

When the cyanide consumption in column leaching and mixing leaching tests was examined, it was observed that cyanide consumption differed depending on particle size and dissolution.

In the table 32 contains cyanide consumption per ton for mixing leaching tests carried out in a particle size of -0.075 mm.

Table 32. -0,075mm NaCN consumption

Tank Leach Test (0,075mm)	**NaCN Consumption (kg/t)**
Tank Leach 1	1,58
Tank Leach 2	1,62
Tank Leach 3	1,73
Tank Leach 4	1,67
Average NaCN Consumption	1,65

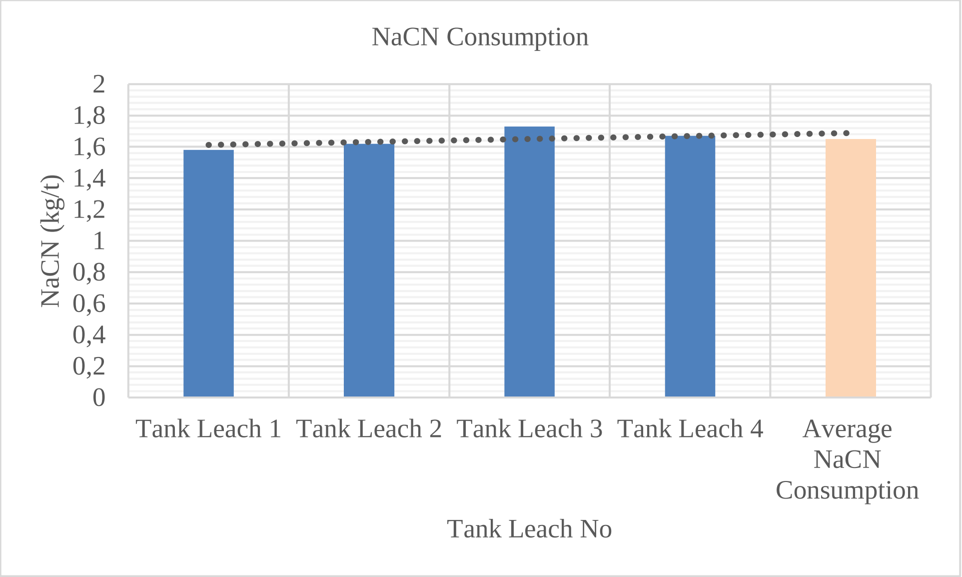


Figure 3.30. NaCN Consumption for Tank Leach Test

When the figure 3-30 was examined, mixing cyanide leaching tests of ore about the consumption were among 1,58 kg/t and 1.73 kg/t and silver content of the sample outside of gold, antimony, and other sulfide minerals in cyanide consumption were found to be effective. For this test, it was observed that the average cyanide consumption was 1.65 kg/t per ton.

Average cyanide consumption for all tests was compared in Table 33.

Table 33. Comparison of NaCN consumption

Test	Average Consumption for NaCN (kg/t)
Column Leach Test (coarse size -20mm)	0,91
Column Leach Test (fine size -12,5mm)	1,27
Tank Leach Test (-0,075mm)	1,65

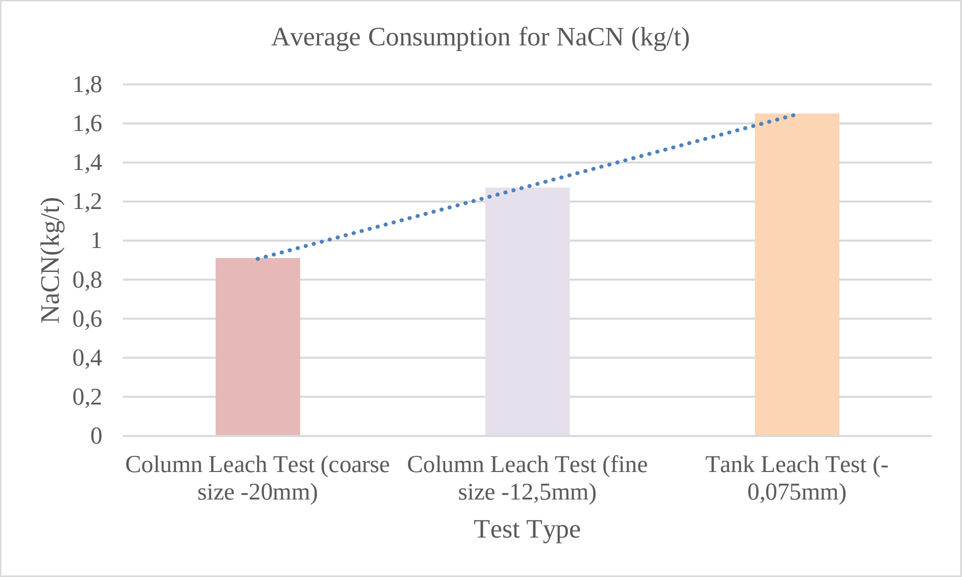


Figure 3.31. Average NaCN consumption for each test

When the figure 3-31 was examined, if cyanide consumption of all tests is evaluated, it is observed that cyanide consumption amounts per ton in general are close to each other in large particle sizes, while this consumption increases visibly in thin particle sizes. As a result of the drop in particle size, cyanide consumption has shown an increase in mixing leaching tests

3.5. Comparison of Lime Consumption

The use of cyanide in laboratory tests and in all leaching processes in field applications requires a chemical to raise the pH value to prevent the release of hydrogen cyanide gas. This chemical is often chosen as extinct industrial lime (powdered lime). It is more advantageous compared to other chemicals in terms of cost and ease of use.

In the table 34 contains lime consumption per ton for column leaching tests carried out in a particle size of -20 mm.

Table 34. Coarse size lime consumption

Column (coarse size -20mm)	**Lime Consumption (kg/t)**
Column Leach 1	1,30
Column Leach 2	1,57
Column Leach 3	1,47
Column Leach 4	1,47
Column Leach 5	1,39
Average Lime Consumption	1,44

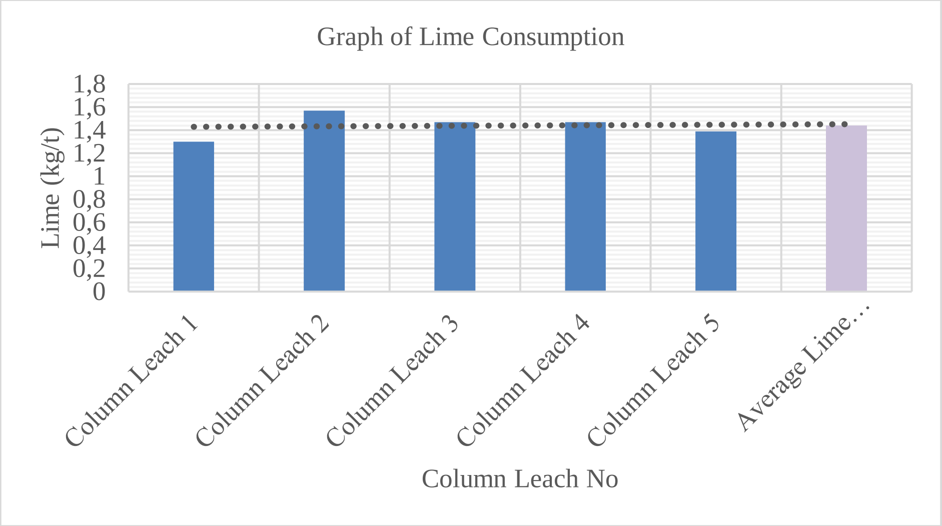


Figure 3.32. Lime Consumption for Column Coarse Particle Size

In the figure 3-32, lime was added between 1.30 kg / t and 1.57 kg / t in order to adjust the pH in the range of 10.5-11 in column leaching tests performed by a particle size of -20 mm. Average lime consumption was calculated 1.44 kg/t. When looking at all columns in general, there were not very large differences between lime consumption. The amount of consumed lime varies depending on the pH value of the ore and the water to be used in the region. Apart from these, when we look at the results of impurity analysis of the ore taken from the region, there is no metal in its content that will cause a high pH drop. This is an indication that lime consumption will not be high for the process to be implemented.

In table 35 contains lime consumption per ton for column leaching tests carried out in a particle size of -12,5 mm.

Table 35. Fine Size Lime Consumption

Column (fine size -12,5mm)	**Lime Consumption (kg/t)**
Column Leach 6	1,34
Column Leach 7	2,22
Column Leach 8	0,99
Column Leach 9	1,15
Column Leach 10	1,46
Average Lime Consumption	1,43

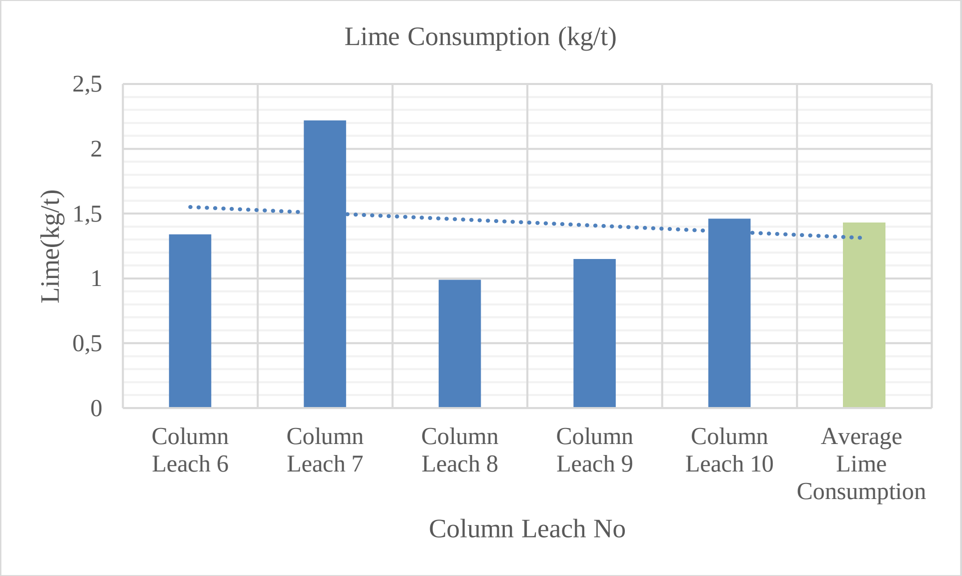


Figure 3.33. Lime Consumption for column fine particle size

In the figure 3-33, in column leaching tests performed by a particle size of -12.5 mm, lime was added between 0.99 kg / t and 2.22 kg / t to adjust the pH in the range of 10.5-11. Average lime consumption was 1.18 kg / t.

Table 36 contains lime consumption per ton for mixing leaching tests carried out in a particle size of -0.075 mm.

Table 36. -0,075mm lime consumption

Tank Leach Test (0,075mm)	Lime Consumption (kg/t)
Tank Leach 1	0,90
Tank Leach 2	1,50
Tank Leach 3	1,52
Tank Leach 4	0,80
Average Consumption for Lime	1,18

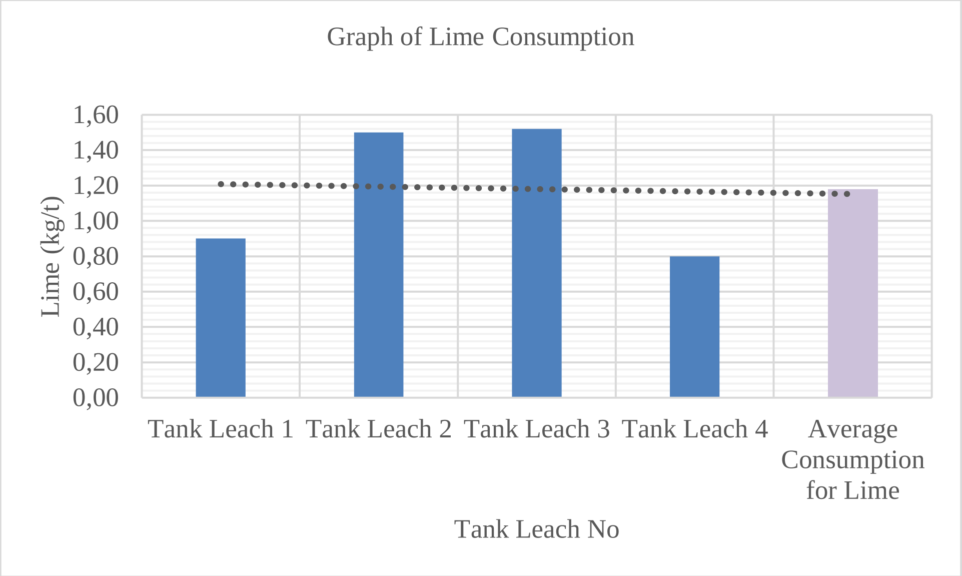


Figure 3.34. Lime Consumption for Tank Leach Tests

When the figure 3-34 was examined, lime was added between 0.80 kg/t and 1.52 kg/t to adjust the pH in the range 10.5-11 in the mixing leaching tests. The average lime addition was calculated as 1.18 kg / t for these tests.

In table 37, lime consumption amounts per ton are given.

Table 37. Comparison of lime consumption

Test	Average Consumption for Lime (kg/t)
Column Leach Test (coarse size - 20mm)	1,44
Column Leach Test (fine size - 12,5mm)	1,43
Tank Leach Test (-0,075mm)	1,18

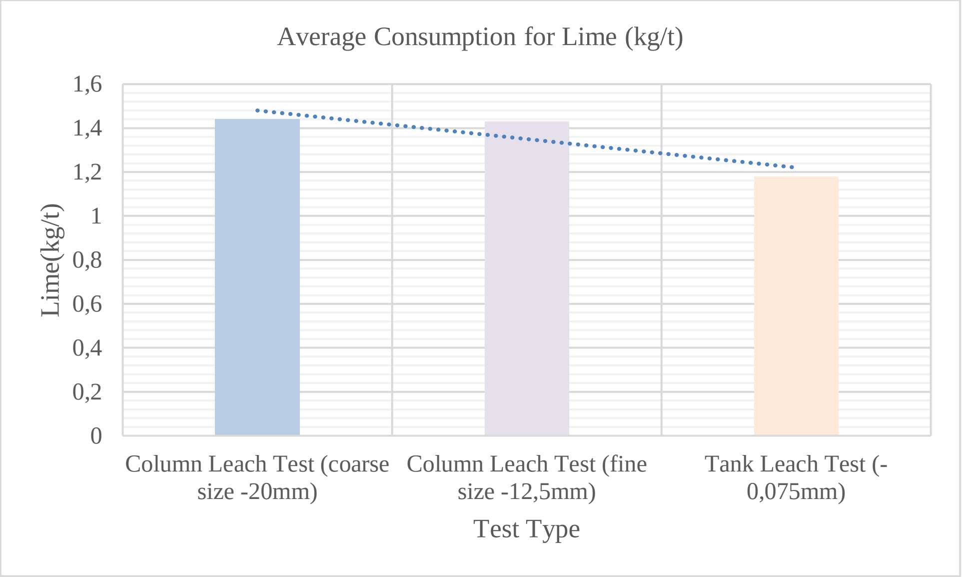


Figure 3.35. Average Lime Consumption for Each Test

When the figure 3-35 was examined, it was observed that if lime consumption of all tests is evaluated, the lime consumption amounts per ton in general are close to each other in column tests. Lime consumption does not depend on particle size, but varies depending on the pH value of the ore and the region where the process will be done.

4. CONCLUSIONS AND RECOMMENDATIONS

Three different particle sizes were tested in cyanide leaching tests with ore belonging to the region. The aim of the studies on three different dimensions is to examine the time-dependent kinetic behavior of tank leaching and stack leaching methods, which are enrichment methods. In this context, the test studies and their results are summarized below:

In the tests using the mixing leaching method, the ore particle size was reduced to D100 - 0.075 mm. D100-0,075 mm milled ore, homogenously sampled, 4 pcs of 1 kg samples were prepared and samples were placed in 4 separate beakers. Water has been added to beakers with a solid ratio of 50%. In order to prevent the formation of HCN gas, the pH was kept in the range of 10.5-11 by adding lime into the pulp.

NaCN was used as a solvent chemical in the leaching process and the concentration of NaCN was set to 1000 mg/L in the test studies. During leaching, a solution sample was taken from each beaker and the time-dependent leaching kinetics were investigated. Mixing leaching continued for 72 hours, after which the remaining gold content was found in solid samples. Gold winning efficiency was calculated between 78% Au and 82% Au according to the results of solid analysis obtained as leaching waste.

D100-0.075 mm particle size of the ore, surface area is more than, particle liberalization is more free gold particles in the ore. Therefore, the interaction of cyanide with gold is increasing. Because of the high liberalization in fine particle sizes, gold compounded with cyanide to pass into solution because the rate is high and fast, gold winning efficiency is also increased.

Due to the fact that grinding costs did not increase and the efficiency of earning gold economically was sufficient, the leaching tests were not conducted in finer particle sizes and the mixing leaching tests were conducted in D100-0.075 mm. During this research, no work has been done to account for the cost, and when the current industrial applications of the gold sector are examined, it is anticipated that the method to be applied in finer particle sizes may not be economical for the ore belonging to this region. At the start of the feasibility study for the ore used in the tests, the efficiency of gaining 82% Au by the mixing leaching method was deemed economically sufficient.

Within the scope of the thesis, experiments were made with cyanide and bulk leaching method, which is another method for comparison. The application of stack leaching at laboratory scale is carried out by using the column leaching method. Two different particle sizes were examined in the tests using the column leaching method and controlled crushing was performed using sieves for both particle sizes, P100 -20 mm and -12.5 mm. D100-20 mm and -12.5 mm of ore prepared in both particle sizes, homogenously sampled, approximately 40 kg samples were prepared and 10 column tests were conducted for each particle size in total.

NaCN was used as a solvent chemical in the leaching process and the concentration of NaCN was set to 1000 mg/L in the test studies. The pH of the solution dripped by the pump on the broken ore in the column was set to 10.5-11 and no HCN gas output was allowed during leaching. During leaching, solution samples were taken from each column and barrel of the column and the leaching kinetics were investigated depending on the duration. Column leaching tests continued for 72 and 86 days, after which gold content was detected in solid samples that had been leached in the column. Gold winning efficiency was calculated according to the results of solid analysis obtained as leaching waste. Accordingly, gold winning yield is between 58% Au and 63% Au for P100-20mm and between 70% Au and 77% Au for D100-12.5 mm.

Different gold-winning efficiencies were obtained in the column test studies based on particle size differences. In tests conducted in two different particle sizes, the interaction of cyanide with gold increased as the ore broken to -12.5 mm was greater than the particle liberalization-20mm, and the yield of gold casings increased by about 15%.

Column and tank leaching tests and results are summarized below. When gold digging efficiencies were compared, the highest gold yield was 77% in column leaching tests, while the highest was 82% in mixing leaching tests. The mixing leaching test took 72 hours, while the column leaching tests were completed in as long as 10 weeks.

When the test results were evaluated, gold gain increased as particle size decreased and the increase in gold gain increased cyanide consumption at the correct rate. The duration of gold leaching with cyanide in mixing leaching tests is significantly shorter compared to column leaching tests. When the impurity results are examined, the metal gain efficiency obtained in the impurity analysis results in mixing leaching tests performed in D100-0.075 mm particle size is higher than in Column leaching tests. This leads to the contamination of gold, which is the final product obtained at the end of leaching.

As a result, in this experimental study, the effect of particle size, leaching time, cyanide and lime consumption were compared by applying two different leaching methods for low-grade gold ore.

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CURRICULUM VITAE

Personal Information

Name-Family Name : Göksel KAYGISIZ ŞENDUR
Nationality : T.C.
Date of Birth : 16/05/1989
Marital Status : Married
Tel : 0 541 947 75 10
E-posta : gokselkaygisiz1605@hotmail.com

Education

	University- Departmant	**Mezuniyet Yılı**
Bachelor's Degree	Muğla Sıtkı Koçman University/ Mining Engineering	2016

Working Company

Year	**Company**	**Position**
Jan 2017-March 2017	Alimoğlu Marble and Granite Factory Kemalpaşa İZMİR	Quality and Control Engineer
April 2017-…	Zenit Gold and Silver Mine Sındırgı BALIKESİR	Process Metallurgist

Language Competence

English	**Elementary**	**Intermediate**	**Advanced**
Writing Skills		x	
Speaking Skills		x	

Germany	**Elementary**	**Intermediate**	**Advanced**
Writing Skills	x		
Speaking Skills	x		

Spanish	**Elementary**	**Intermediate**	**Advanced**
Writing Skills	x		
Speaking Skills	x		

Certificates

Türkiye Dağcılık Federasyonu Education Certificate (2015)

Daimi Nezaretçi Education Certificate (19.02.2018 - 23.02.2018)

Mineral Processing Symposium Participation Certificate (23.10.2018 – 25.10.2018)

Filtrasyon ve Akışkan Transferi Seminar Participation Certificate (04.10.2018 – 05.10.2018)

Leaching and Recovery of Gold Ores (10.12.2018-11.12.2018)

Eğiticinin Eğitimi (21.05.2018-24.05.2018)

Autocad Certificate (2019)