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2 Abstract

The purpose of the research is to optimize parameters for cyanidation on sulfidic gold rich ore from Olon-Ovoot mine. Characteristic of ore has changed due to depth displacement. Therefore processing plant parameters had to be updated. Ore samples were grinded to 65%, 75% and 80% of -74μ and leached in 200ppm and 250ppm sodium cyanide solution which are the most suitable concentration range for sulfidic ore. As a result of the experiment, leaching 80% of -74μ ore in 250ppm NaCN solution for 32 hours was the optimal parameters for highest gold recovery which is 92.1%.

3 Introduction

Gold is commonly occurred as fine grain rather than coarse grain in gold ore. Host mineral of gold is commonly calverite, montbroyite and sylvanite and occurs in association with minerals such as sulfide. It is rare metal, because its concentration in earth crust is just 0.005g/t while silver and copper concentrations are 0.07g/t and 50g/t.

During amalgam and gravitation process, gold recovery is not sufficiently high for fine gold grain in ore. Therefore cyanidation is main process which is worldwide used and give higher recovery for this type of ore. The method conducts alkaline cyanide, for instance KCN, NaCN, $Ca(CN)_2$ to dissolve gold in solution with facility of oxygen in air. If 10% of gold particle is sized less than 20micrometer, the most efficient way to extract gold is cyanidation. Albeit ore must be roasted before cyanidation, in case of gold granule is covered by iron oxide or other mineral. Cyanidation process have been main extraction method of gold for more than a century (de Andrade Lima, 2006).

Since invention of cyanidation process by John Steward MacArthur in 1887, it has been studied worldwide from different aspects such as its chemistry, kinetics, application and affecting factors for its efficiency. Affecting factors consist of pH of cyanide solution, cyanide concentration, oxygen concentration, retention time, particle size of ore, temperature, pressure, slurry density, mixing rate and presence of other minerals. Even though the variables have already investigated, the most beneficial optimization of the variables differs for every ore in some range. Because non of ore is exactly same, every variables needs to be optimized by doing experiments on exactly the ore from desired mine site. Thus this research conducts Olon-Ovoot ore for finding optimal size of feed, retention time and cyanide concentration.

Sodium cyanide, potassium cyanide and calcium cyanide are used in cyanidation plants. The substance dissolves easily in water and form metal cation and cyanide ion.

$$NaCN = Na^{+} + CN^{-} \quad (1)$$

Undesirably hydrogen cyanide,toxic gas, could be formed as well when its pH is not sufficient high. At pH 9.3, half of cyanide forms hydrogen cyanide and half of cyanide forms free cyanide ion. Also free cyanide ion could be oxidized to form cyanate ion which is not able to dissolve gold.

$$CN^{-} + H_2O = HCN + OH^{-} \quad (2)$$

$$4HCN + 3O_2 = 4CNO^{-} + 2H_2O \quad (3)$$

$$3CN^{-} + 2O_2 + H_2O = 3CNO^{-} + 2OH^{-} \quad (4)$$

Gold is oxidized and forms gold cyanide ion $[Au(CN)_2^{-}]$.

$$Au + 2CN^{-} = Au(CN)_2^{-} + e^{-} \quad (5)$$

Also, oxygen is reduced to hydrogen peroxide as an intermediate product. Then it could form hydroxide ions or oxygen and water.

$$O_2 + 2H_2O + 2e^{-} = H_2O_2 \quad (6)$$

$$O_2 + O_2 + 2H_2O + 2e^{-} = H_2O_2 \quad (7)$$

$$2H_2O + 4e^{-} = 4OH^{-} \quad \text{(Srithammavut, 2008)} \quad (8)$$

Generally this reaction occurs in the solution

$$2Au + 4CN^- + O_2 + 2H_2O \leftrightarrow 2Au(CN)_2^- + H_2O_2 + 2OH^- \qquad (9)$$

3.1 Background of thesis topic

Olon-ovoot mine is located in Mandal-Ovoo sum of Omnogovi province. From surface to 40m deep ore is oxidized and as it goes deeper, ore becomes sulfidized. The sample ore which is used for this experiment was taken from 1140th level of east ore body which means under 80m from surface. It is gold rich and sulfide ore.

Property of the ore changed due to level of extraction. Thusparameters for processing plant needed to be updated with combination of theory andpractical work to enhance efficiency of the plant.

3.2 Purpose of the research

Purpose of the thesis is to estimate optimum technology parameters for the ore under laboratory condition.

To reach the purpose, those targets were set:

1. Definition of chemical composition of ore

2. Research about gold cyanidation

3. Definition of optimal size of ore grain by grinding for different time duration

4. Comparison of recovery of gold in most optimal cyanide concentration of 200ppm and 250ppm cyanide in solution.

5. Definition of optimal leaching time duration

4 About the mine

Olon-Ovoot mine is located in Mandal-Ovoosoum, Omno-Gobi province. The mine site is 550 km far from Ulaanbaatar, 100km from Dalanzadgad city and

40km from center of Mandal-Ovoosoum. Climate of the area is dry and hot in summer and cold in winter. Annual average temperature of the environment ranges from +5-6 C, highest temperature is +31C in July, lowest temperature is -23C in January, annual precipitation amount is 8.24 mm, deepest depth of soil freeze is 1.73m,annual average wind velocity is 2.6 m/s, highest velocity of wind is 30m/s, it mostly winds from west and average humidity is 50-60% (Olon-ovoot LLC, 2008).

Altitude located 1100-1300m above sea level. Surface water in the area is included in non outflow zone and does not have permanent water-net. Ulaanlake lies in north-west 60kms away from the mine. Ongiinriver which used to flow into the lake had shrank in last years. Mine site located in level 5-6 earthquake zone in Richter magnitude scale. In other words, it located in marge of seismic dangerous zone. Desertification occurs rapidly in the area and soil layer is so thin and plants are very rare (Radnaa, 2008) .

4.1 Geology research

Geology research of Olonovoot mine started in end of 19th century by Russian researcher Prijevaliskii (1885), M.B.Pevtsov (1883) etc. In 1940 A.Delitov searched for petroleum, 1946-1949 Russia-Mongolian expedition research was done, 1968 A.P.Zinkow did geology-hydrogeology research. In the A.P.Zinkow's research, there was findings of gold, silver, cobalt, chrome, copper and other rare metals in the area.Olon-Ovoot mine was discovered by research in 1989-1991 andits preliminaryreserve was determined by Batchuluun and his team in 1992-1993. In 1994 mine was registered (Radnaa, 2008).

4.2 Geology reserve

On February 2005, according to "mineral reserve classification form" mine reserve was evaluated. In the evaluation, silver content was not considered because its contents was <2g/t and its market value was equal to 1.3% of gold price.

The mine contains 100.1 tn gold reserve, out of it, 18.54 tn which is 19% of total reserve is proved and possible reserve.6% of the total ore is oxidized ore. Area used for soil stripping, dumping the tailing and road covers 3-4 times more (228755m2) than area of total ore which is 71486 m2. From that, efficient area occupies 80% which is 228755m2.

4.3 Ore characteristics

Ore district consist of silur, Devon, carbon-perm and Mesozoic sedimentary, volcanic rocks, down-Devon, carbon intrusive rocks.The ore consist of 4 ore bodies. Mostly the ore bodies consist of altered diorite around quartz vein,apofiz of quartz vein and altered diorite in quartz vein. Olon-Ovoot ore body occupies 280m*20m area. Ore body intervened in white quartz and surrounded the quartz vein. Edge or other part of quartz, which was intensively catalyzed and diffused with ferrous hydroxide , contains gold. Sometimes white quartz contains high amount of gold because of its uniform dispersion. Core of the ore body, quartz vein, ramifies as it goes deeper (Olon-ovoot LLC, 2006).

4.4 Ore mineral composition

Ore body contains carbonate, sulfide, diorite around milky quartz. Quartz grains are relatively big 4.509 mm. Hydrated iron(III) oxide-hydroxide and calcite was filled in few crevices of quartz. Rarely trace grains of chalcopyrite was found in quartz. According to mineralogy analysis, ore contains gold, pyrite, chalcopyrite, arsenopyrite galena, martitized magnetite, hydrated iron(III) oxide-hydroxide, martite, scheelite, ilmenite. Free gold grains are covered in hydrated iron(III) oxide-hydroxide in quartz vien and altered diorite and typically shaped like dendrite. Some of gold grains were grown with quartz and hydrated iron(III) oxide-hydroxide. Oxidized part of ore was analyzed that contained 882 carat gold by Geological Central laboratory and 930-950 carat gold by Japanese laboratory. (Radnaa, 2008)

5 Olon-ovoot ore processing plant

Processing plant of “Olon-Ovoot Gold” LLC has capacity of 1800 ton of ore to process daily. The plant comprise of crushing section, milling section, leaching section, desorption section and tailing section.

5.1 Crushing section

Crushing section is the first stage of the processing plant.

Ore ,came from open pit, is classified by its gold content and characteristics and dumped. Then ore is loaded to receiving part and fed to jaw crusher. The Jaw crusher crushes up to 500mm sized ore into -130mm and has capacity of 250 tn/hr. Crushed ore is conveyed into Semi-Autogenous Grinding mill.

Figure 1 Dumped ore

Figure 2 Jaw crusher

5.2 Milling section

Ore is fed into conveyor of Semi-Autogenous Grinding /SAG/ mill by board feeder from ore store and maintain sufficient amount of ore. There are 3 SAG mill installed in the plant. Each of them is 4.2 m in diameter, 1.4m in length and has 280kW engine and 30 tn of ore feed/hr capacity. Rate of feed is monitored by automatic control. Water is added to feed and product of SAG mill and the product is passed through sieved tube as feed of ball mill. Ball mill grinds up to 3.5mm sized ore into 80% passes 74 micrometer grain. Then product of ball mill is transferred into sump through sieved tube and pumped to cyclone.

Figure 3 Conveyor

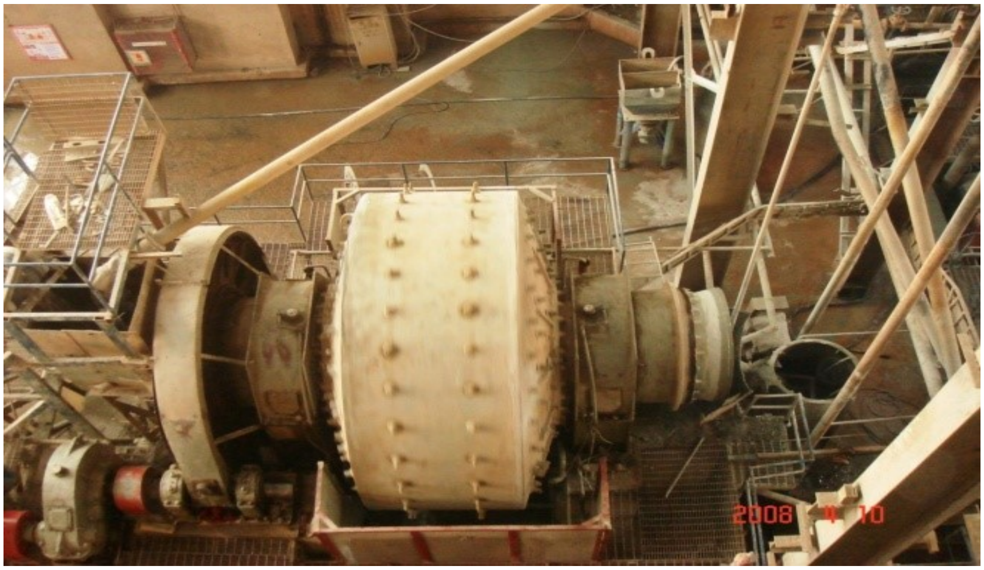


Figure 4 SAG mill

Figure 5 Ball mill

There are 2 ball mills are used in the processing plant. Their size is 2.1 m in diameter, 4.2m in length and has 280kW engine and 25 tn of ore per hour capacity.

5.3 Leaching and adsorption section

In this section there are 2 leaching tanks and 8 adsorption and leaching combined tanks and those tanks hold the slurry for 28 hours. There are 1 tank, volume of 800m^3 and 9 tanks, volume of 400m^3 and sized 8:8.5 ratio.

At first, ore is mixed in alkaline cyanide solution and liberated gold grains in ore are dissolved mainly in leaching tanks. Then the dissolved gold is adsorbed by activated coal in the adsorption and leaching combined tanks and between every tank, sieves were installed to prevent transferring the coal to next tank. Coal is transferred by airlifting and flow direction of coal and slurry are reverse. The slurry flows into further tanks and outflow of last tank flows into filter press section after controlling sieve which prevents coal loss to tailing.

Figure 6 Leaching and adsoption tanks

1.1 Desorption section

At 0.5MPa desorption is proceed by 1-5% NaOH solution in 120-150C, then gold which has been adsorbed in the coal transforms into solution as dissolved gold cyanide complex and be separated from coal.

The solution containing dissolved gold passes through electrolysis. In result of the electro-chemical process, the gold is collected on steel cathode. Then gold content of solution will be decreased and it is fed to electrolysis again after some heating to extract all of gold. Cathode is taken after significant time duration, dried in electric dryer and melted with some additional ingredients to cast gold bars.

Figure 7 Electrolysis

Figure 8 Gold casting

5.4 Filter press section

Cyanide slurry which has already passed the controlling sieve in adsorption section is pumped to filter press and filtered liquid flows to pool of plant's technology water and recycled. Another product of filter press, solid part, contains 15-20% humidity and conveyed to detoxifying section.

Figure 9 Filter press

5.5 Detoxifying section

Calcium hypochlorite is used for detoxifying cyanide. Chlorite ion in calcium hypochlorite reacts with normal and complex cyanide and forms cyanate salt.

Oxidation reactions for the conversion of cyanide to cyanate.

$$2CN^{-} + Ca(OCl)_2 + 2H_2O = 2CNCl + Ca(OH)_2 + 2(OH)^{-} \quad (10)$$

$$CNCl + 2(OH)^{-} = CNO^{-} + Cl^{-} + H_2O \quad (11)$$

$$CNO^{-} + 4H_2O = (NH_3)_2CO_3 + CO_3^{2-} + 2H^{-} \quad (12)$$

Then cyanate is hydrolyzed and decomposes into carbonate and ammonia which can be used as nitrogen fertilizer. Also, organics are oxidized during this process and detoxified.

$$CNO^{-} + 2H_2O = NH_4^{+} + CO_3^{2-} \quad (13)$$

Finally WAD and Free cyanide in detoxified cake is analysed and dumped in tailing. (Olon-Ovoot LLC)

6 Gold ore processing technology

It is considered that minimum content of gold in ore which can be economical efficient is 3-10g/tin refractory gold ore, 0.3-1g/t for non-refractory ore.

Type of processing technology depends on mineralogy of ore. If gold ore is placer, it can be concentrated by gravity concentration and flotation. Refractory ore requires more delicate or complex processing method such as gravity-flotation and gravity-leaching.

6.1 Gravity concentration

Gravity concentration is a method for separate valuable metals from tailing by their gravity difference. It uses fluid, less dense than valuable metal, so that gangue will be floated or at least not precipitate. In practice, water is used for concentration of placer because of gold density. Main equipments are shaking table, sluice and pan. It is traditional, very simple and the cheapest method. On the other hand, the weakness of this method is it can not collect all of the gold. Fine grain or small flakes of gold will be swept away from concentrate. Thus tailing of gravity concentration is usually sent to further concentration method, for instance leaching.

6.2 Amalgamation

Amalgamation is reaction of ore with mercury for extract valuable metal. Only valuable metals will be wetted by mercury and dissolved easily in it. Thus scattered gold and silver can be separated. The process is consisted of 2 stage, at 1^{st} diffusion of gold surface, then reaction of gold with mercury. Gravity concentration and amalgamation process is used for collecting relatively coarse gold grain.

6.3 Flotation

Gold can be recovered by flotation because it is naturally floatable in most industrial systems which means that it can be recovered without collector addition. (Dagel, 1989)

Disadvantage of flotation for gold is its concentrate usually contains sulfide minerals beside gold. If ore doesn't contain sulfide mineral, flotation can hardly separate gold as well because of its amount. Concentrate would be trace and froth would be unstable which causes lower recovery.

Flotation is mainly for placers which contain low concentration of gold and its gold grains are infinitesimal. Therefore gravity concentration could not recover gold particles effectively. (Harris, 2000)

For the gold ore with valuable sulfide gangue, flotation condition and reagent amount is optimized to most economical outcome whilst ore with non-valuable sulfide gangue, reagents should be gold-selective.

On the other hand, before gold extraction, gangue material can be floated to minimize unnecessary material or clean the ore from material which can affect gold recovery. Also reagent consumption could be reduced through this pre-flotation.

6.4 Cyanidation

Nature gold particles are usually very fine, <50micrometer, to be concentrated by gravity method leached in cyanide, they are extracted by cyanidation.

Cyanidation is a process that dissolutes gold in alkaline metal cyanide solution with a help of oxygen in order to separate gold from its ore.

Cyanide is commonly used in gold extraction as a ligand which forms stable complex with gold due to its low cost and effectiveness of dissolution of gold and silver. Gold particle needs to be oxidized to gold cation in order to react with cyanide. Oxidant for cyanide leaching is usually oxygen, can be provided from air. In some processing plant, lead nitrate is added to the leach circuit and has the effect of accelerating the gold dissolution rate. (K. Kongolo, 1998)

6.4.1 The chemistry of cyanide solution

Sources of cyanide leaching are cyanide salts including potassium, sodium and calcium cyanide. They can dissolve and ionize in water to form their metal ion and free cyanide.

$$KCN \leftrightarrow K^{+} + CN^{-} \quad (14)$$

These 3 salts have different solubility and cyanide contents. Sodium and potassium cyanide are more soluble than calcium cyanide and are mostly easily found pure substance.

Table 1 Solubility of cyanide compounds (John O.Marsden, 2009)

Compound	Available Cyanide (%)	Solubility in Water at 25°C (g/100 cc)
NaCN	53.1	48
KCN	40.0	50
$Ca(CN)_2$	56.5	Decomposes

Cyanide ion hydrolyzes in water to form molecular hydrogen cyanide HCN and hydroxyl OH^- ion. Here, hydrogen cyanide is weak acid and incompletely dissociates in water.

$$CN^{-} + H_2O \leftrightarrow HCN + OH^{-} \quad (15)$$

$$HCN \leftrightarrow H^{-} + CN^{+} \quad (16)$$

The extent of this dissociation reaction at equilibrium as function of pH and speciation of free cyanide and hydrogen cyanide in aqueous solution is shown in diagram. At pH 9.3 half of the total cyanide exists as hydrogen cyanide and half of them exist as free cyanide. At pH 10.2, >90% of the total cyanide is present as free cyanide, (CN^-), while at pH 8.4, >90% exists as hydrogen cyanide. The hydrogen cyanide is highly evaporates has relatively high vapor pressure and may volatilize at ambient temperature. Therefore most cyanide leaching is commonly performed at pH values over 9.4 to prevent excessive loss of cyanide by hydrolysis.

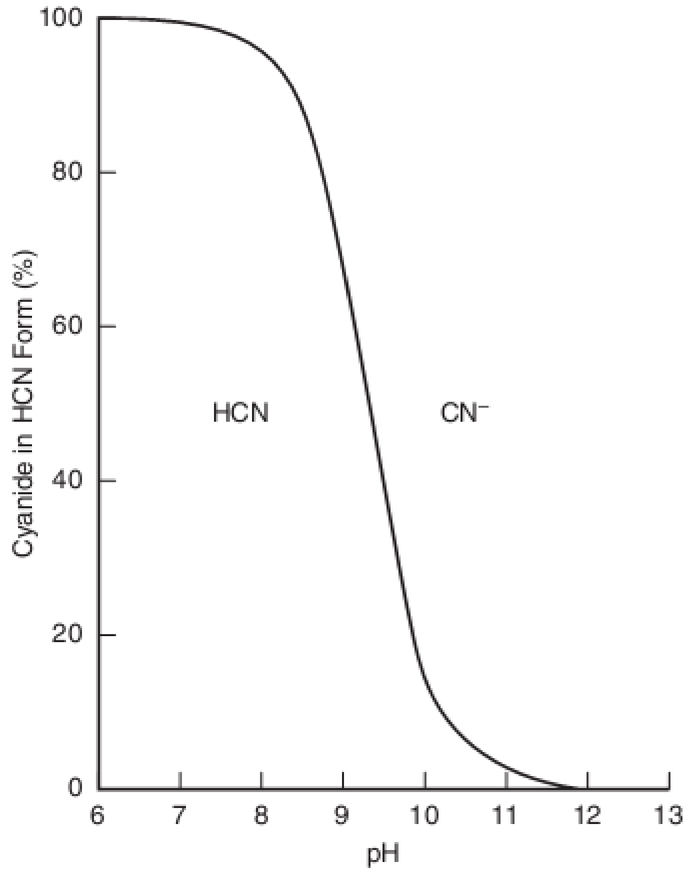


Figure 10 HCN formation dependence of pH (John O.Marsden, 2009)

In the presence of oxygen, both hydrogen cyanide and free cyanide can be oxidized to cyanate under suitably oxidizing conditions. The reactions are as follows:

$$4HCN + 3O_2 \leftrightarrow 4CNO^- + 2H_2O \quad (17)$$

$$3CN^- + 2O_2 + H_2O \leftrightarrow 3CNO^- + 2OH^- \quad (18)$$

The reactions reduce free cyanide concentration and the formed cyanate doesn’t dissolve gold. But the reaction is very slow and some strong oxidizing agents such as ozone, peroxide help to get higher oxidizing rate.

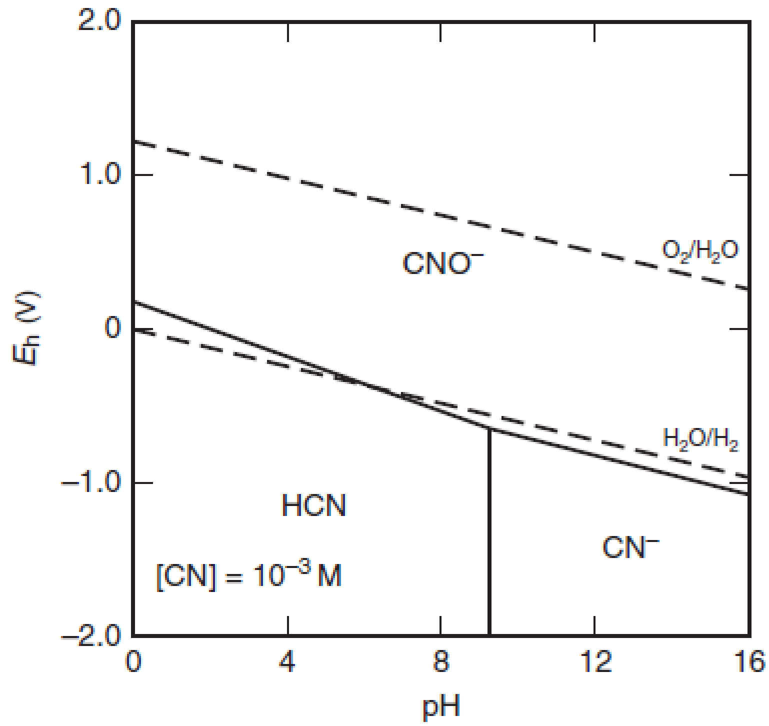


Figure 11 Cyanide potential dependence of pH (John O.Marsden, 2009)

According to the graph, cyanide forms cyanate when oxygen is present in the solution, but the reaction occurs very slow in normal condition. Only strong oxidants such as hydrogen peroxide or oxygen with catalysts can form it in significant rate.

Cyanidation could occur by 2 reactions that indicates gold react with cyanide and forms cyanide complex compound and hydrogen peroxide. The hydrogen peroxide is intermediate product and consumed by gold. According to Elsner:

$$2Au + 4\mathrm{NaCN} + O_2 + 2H_2O \leftrightarrow 2\mathrm{Na[Au(CN)_2]} + H_2O_2 + 2\mathrm{NaOH} \quad (19)$$

$$H_2O_2 + 2\mathrm{Au} + 4\mathrm{NaCN} \leftrightarrow 2\mathrm{Na[Au(CN)_2]} + 2\mathrm{NaOH} \quad (20)$$

Second reaction occurs, but does not add significant value because hydrogen peroxide diffuses away from the gold surface and reduces to hydroxide ion. (Rees, 2000)

6.4.1.1 *Anodic reaction*

Gold dissolves in alkaline cyanide solution to form Au(I) cyanide complex, $Au(CN)_2^-$ and the reaction occuring on anode is presented:

$$Au + CN^- \leftrightarrow Au(CN)_2^- + e \qquad (21)$$

The oxidation process occurs in 3 stages which is shown in below graph.

- First peak indicates intermediate product AuCN was formed.It causes temporary passivation of gold surface.

$$AuCN + e \leftrightarrow Au + CN^- \qquad (22)$$

- Second peak of the graph indicates intermediate product AuCN connected to cyanide ion and forming Au(I) cyanide

$$AuCN + CN^- \leftrightarrow Au(CN)_2^- \qquad (23)$$

- Third peak indicates passivation of gold,Au (III)oxide, occured because of some heavy metals. Also it requires very high positive potential, thus it usually does not happen in practice.

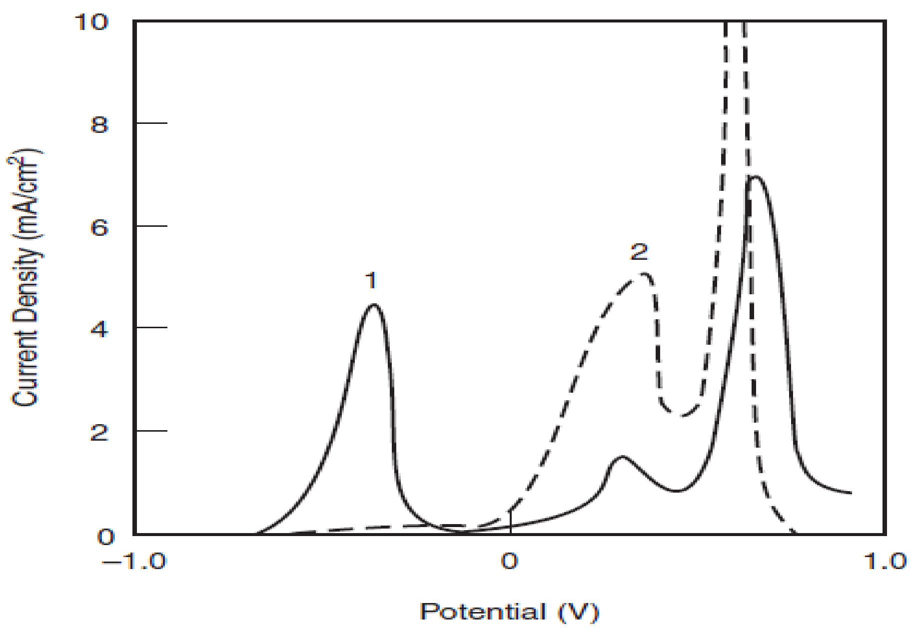


Figure 12 3-Oxidation steps (John O.Marsden, 2009)

6.4.1.2 *Cathodic Reaction*

Anodic reaction is accompanied with cathodic reaction of oxygen. Practically, the reaction involves many cathodic reactions but the major reaction to be as follows,

$$O_2 + 2H^+ + 2e \leftrightarrow H_2O_2 \quad (24)$$

The hydrogen peroxide is strong oxidizing agent and it can further cause passivation of gold surface by oxide layer formation.

Hydrogen peroxide in the solution can affect the rate of dissolution with different consequences.

$$H_2O_2 + 2e \leftrightarrow 2OH^- \quad (25)$$

According to this reaction, hydrogen peroxide reduces to hydroxyle ion means oxygen in the solution is consumed to form hydrogen peroxide, not regenerated. Thus it could affect adversely for dissolution of gold. On the other hand, hydrogen peroxide decomposes into water and oxygen and form dissolved oxygen in solution.

$$2H_2O_2 \leftrightarrow O_2 + 2H_2O \quad (26)$$

About 85% of H_2O_2 diffuses away from the reaction site and only tiny amount of it reduces directly to hydroxyl ion. [21 Chem.ge].It means hydrogen peroxide almost does not play a role in gold dissolution. Albeit in some significant condition, for instance 0.015M H_2O_2 in solution of 0.01M NaCN at ph10 can increase gold dissolution rate.

Finally oxygen might form to hydroxyl ions rather than to H_2O_2.

$$O_2 + 2H_2O + 4e = 4OH^- \quad (27)$$

The reaction rate is extremely slow and requires high potential.

6.4.2 Cyanidation process

Activated carbon is used for adsorption of gold from the solution which contains dissolved gold. The continuous processes for gold leaching and adsorption are Carbon-in-leach and Carbon-in-pulp. The processes have been widely used worldwide, about 44% of world production uses this two methods. About 30% is produced by solid-liquid separation and zinc separation and 19% for flotation and gravity concentration. (John O.Marsden, 2009)

6.4.2.1 Carbon in pulp CIP process

Finely milled ore is leached in alkaline solution and followed by adsorption of activated carbon.

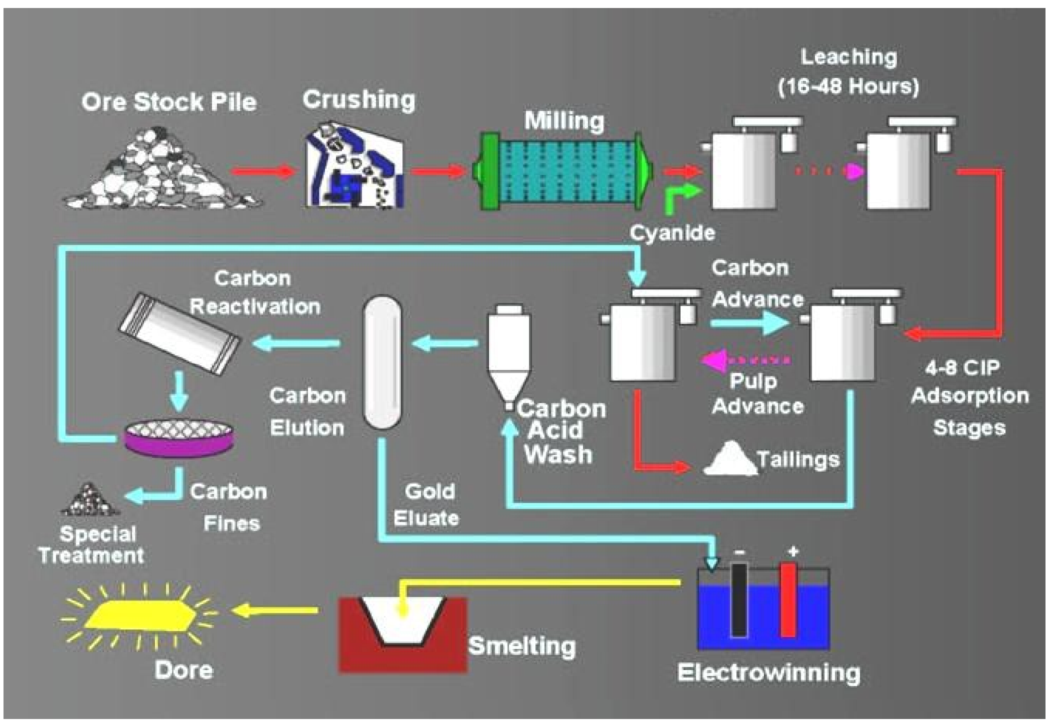


Figure 13 Flowsheet of a modern CIP plant (Fleming, 2004)

CIP plant usually consist of 6series of steel vessels which are agitated with blade and shaft, also each vessel has approximately 4 hours retention time. The vessels are covered by floating or stationary cover in order to prevent oxygen dissipation from the solution into air. In this type of leaching reactor, gold is dissolved from the pulp. Either pure oxygen or air could be used for aeration, but price would be different.

Gold recovery from cyanide solution is shown in the equation. (McDoudall, 1985) (John O.Marsden, 2009):

$$Me^{+} + nAu(CN)_2^{-} \rightarrow Me^{+} [Au(CN)_2]_n^{-} \qquad (28)$$

$$4Au(s)+8NaCN(aq)+O_2(g)+2H_2O(l)\rightarrow 4Na[Au(CN)_2](aq)+ 4\ NaOH(aq) \qquad (29)$$

Where Me+ [Au(CN) 2 -] n, is the adsorbed gold species which can be hydrogen, sodium, potassium and calcium.

CIP consists of 3 stages:

(a) Adsorption: the dissolved gold in the pulp is loaded onto activated carbon

(b) Elution: gold is removed from carbon into an alkaline cyanide solution

(c) Electrowinning: gold is removed by an electrical process from the alkaline

cyanide solution and deposited on electrodes.

Each vessel contain significant amount of activated carbon.

Activated carbon in first vessel adsorb highest amount of gold, because gold solution flows in series by gravity from one to another by overflow. Size of the carbon, usually 1.7*3.4 mm (Marsden and House, 1992), is bigger than apertures of screen which is placed between stages, thus only solution can enter next stages. After adsorption process, gold is removed from the activated carbon by desorption in elution process and then it is treated by electrowinning to recover metallic gold.

6.4.2.2 Carbon in leach process

Carbon in leach process is similar to Carbon in pulp, only difference is in CIL process leaching and adsorption process take place simultaneously. Thus gold would be adsorbed immediately after leached in solution. This process is used when ore is containing organic material or any material which can adsorb gold anions, for example clayey minerals. Advantage of CIL is it economizes the capital cost because of not distinguishing leaching and adsorption vessels. Disadvantage is CIL consumes large amount of activated carbon, also gold dissipation is higher.

As presented in Figure 11, cyanide solution flows down to next tanks while activated carbon is loaded from down to up stream. In the first tank, leaching is dominantly occurred rather than adsorption and further tanks are dominant in adsorption. The slurry passes through the screen between the tanks. Carbon is then collected and thermally treated to reuse it. After elution process, gold dissolved solution is sent to electrowinning part where the gold collected on steel wool cathodes.

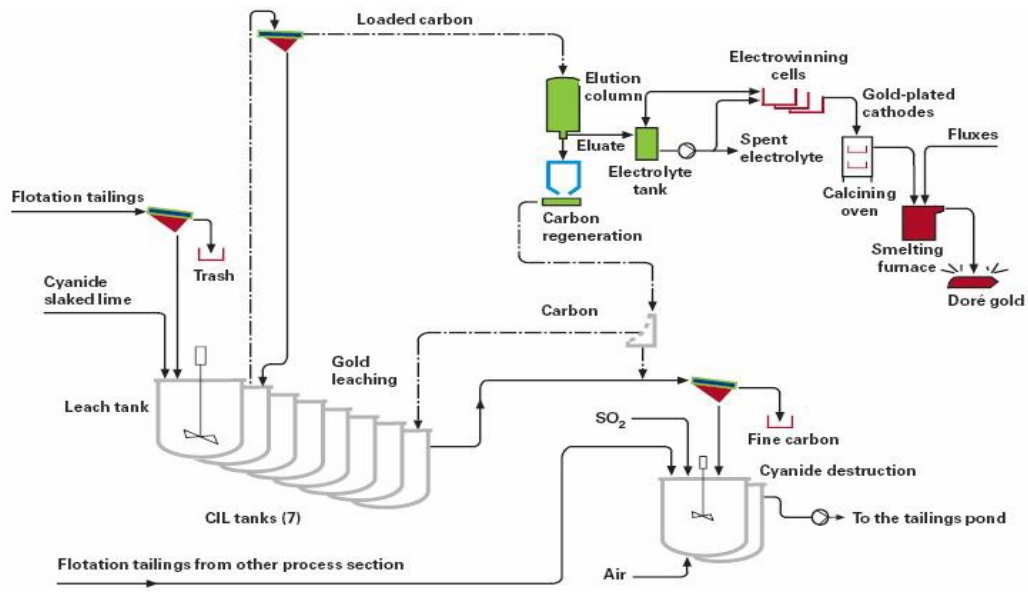


Figure 14 The carbon-in-leach process (Outokumpu, 2004)

7 Olon-ovoot gold mine leaching experiment

Research object

Gold ore of Olon-ovoot mine and significant concentration of cyanide solution are used as research object of cyanide bottle roll leach test.

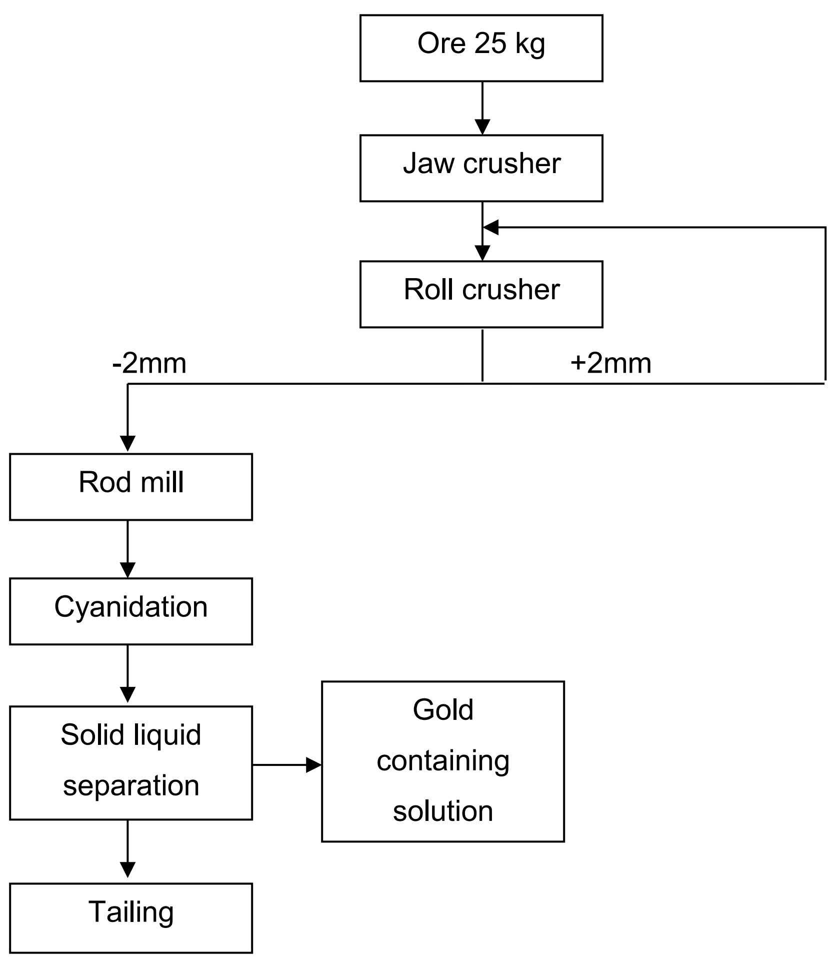
About the experiment

Ore preparation and cyanidation process took place in laboratory of Khanlab LLC and all of the analysis such as mineralogy, gold content definition analysis were done by Khanlab LLC.

For cyanidation experiment, ore characteristics, shape and size of gold, pulverizing level, leaching time duration, host rock, cyanide concentration, lime consumption and temperature of environment influences a lot for cyanidation efficiency.I did experiment for changing main affecting factors- pulverizing level, leaching time duration and cyanide concentration. To optimize, ore was grinded for different retention time and other factors remained constant. Also, it is compared with same grinded ore in differently concentrated cyanide solution. All of the experiments held for 32 hours.

In plant process, leaching slurry comprises of 45% dry ore and 55% liquid. Albeit leaching slurry in the experiment comprises of 33% dry ore and 66% cyanide solution because every 2-4 hours approximately 5 ml of solution was taken out as sample. Each experiment was done with parallel experiment in order to get more reliable result.

Experiment follows the diagram:



7.1 Experiment condition

- Solid liquid ratio of the slurry is 1:2. In other words, slurry was comprised of 33.3% solid, 66.7% liquid by mass. (500g of ore was used)
- Sodium cyanide concentration of the slurries were 200ppm and 250ppm.
- pH condition of the slurry was 10-11
- Rotation rate 6-8rpm
- Temperature 20C
- Amount of passing 74micrometer ore were 80%, 75%, 65% by mass
- Leaching took place in 2 l volume bottle on roller for 32 hours.

Reagents and devices used in the experiment:

- Laboratory rod mill
- Gold ore
- Laboratory rolls
- 2 l volume bottles
- pH paper
- Solid NaCN
- Solid CaO

7.2 Sample preparation

About 100 kg of sample was taken from the ore bodyand out of 100kg ore, 25 kg of representative sample was used in the experiment. Then it was crushed in jaw crusher and roll crusher and it was sieved by 2mm sieve. +2mm partcrushed in roll crusher again. Finally all of the grain in ore sample was crushed until they became smaller than 2mm in size. Crushed ore was mixed by cone and circle method until it became uniform. Then 200g of ore sample was taken for each of mineralogy analysis, 28 element definition and gold content definition analysis which is fire assay by atomic absorption spectrophotometer(the analysis is further explained)

Table 2 Technical data of the used jaw crusher

	Feed opening size mm	100*60
	Maximum feed size mm	<50
	Output size mm	0.1-15.1
	Capacity kg/hr	45-550
	Motor power kW	2.2
	Weight kg	260

7.3 Definition of time duration for grinding to obtain 65%, 75%, 80% of sample become less than 74micrometer in grain size

In order to define the time duration, -2mm ore was milled in rod mill and amount of ore for each milling is 1kg ore with 1kg water for different time durations.

Table 3 Grinding time interpolation

Time duration [minutes]	-74 micrometer sieved part [g/1kg ore]
15	914.5
X1	800
X2	750
10	746.7
X3	650
5	411.5

Milling for 15 minutes made 914.5g of -74micrometer part out of 1kg ore. Milling for 10 and 5 minutes made 746.7g and 411.5g each. Each sample was washed by 74 micrometer sieve and dried in drying oven at 105^{o}C for a day.

Then I calculated time durations corresponding to each purposed amount of -74 micrometer sized grain by using interpolation.

Figure 15 Washing grinded ore with 74 micrometer sieve

Table 4 Technical data of rod mill

	Model	XMB 240*200 lab rod mill
	Size of body mm	200*240
	Grinding capacity g	500-1000
	Feed size mm	<2
	Product size mm	< 0.074
	Rotation rate rpm	110

7.4 Preparation of cyanide solution

In water which has pH less than 9.3, cyanide forms hydrogen cyanide, HCN, acute toxic gas and which causes reduced content of free cyanide in the solution. In order to prevent forming it, water pH increased due to additional 50ml of 10% CaO solution to 950ml of water. Then pH of the water should be adjusted around 10-11.

Then each 0.2g and 0.25g of pure sodium cyanide is water-soluble white solid. 0.2g of solid NaCN was weighed in electric weigh precisely and added to 1 l of water. It was mixed until it dissolves completely. Cyanide concentrations were analyzed by titration of $AgNO_3$.

At the end each 500g of 65%, 75%, 80% -74micrometer grinded samples were added in the 200 ppm sodium cyanide solution. (In case of 250ppm sodium cyanide solution, 0.25g of solid NaCN was added)

Figure 16 Sodium cyanide

7.5 Leaching

Variously grinded 1 kg of ore samples were divided into 2 equal parts after mixing thoroughly. Each part was added to 1 l of cyanide solution.

Leaching took place in cylindrical plastic bottle which is 2 l in volume. Slurries in the bottle were placed on a set of laboratory rolls which is 6-8 rpm. Rolling throughout the duration of the test mixed the slurry. Samples were taken after 0.5, 1, 2, 4, 8, 12, 16, 20, 24 and 32 hours. For every sample taking, pH of the solution was monitored and maintained between 10-11. Those liquid samples were analyzed by AAS to define gold dissolution rate.

Figure 17 Bottle roll test

After 32 hour cyanidation process, liquid and solid particles were separated by filtered funnel. Then washed 2-3 times in 1 l plastic bottle by mixing with water and changing fresh water 2-3 times after ore was completely precipitated. After it is separated from water and dried approximately 100g amount of sample was taken from the solid part and analyzed by fire assay to define gold concentration.

Figure 18 Taking sample

Figure 19 Taking samples

7.6 Detoxifying

After the experiment, remained cyanide solution and samples which are given to further analysis were neutralized by potassium permanganate and collected in jerry can which addressed “cyanide tailing”. Also glasses, bottles and other tools which might contaminated with cyanide were washed with potassium permanganate solution.

$$2KMnO_4 + NaCN + 2KOH = 2K_2MnO_4 + NaCNO + H_2O \quad (30)$$

$$2MnO_4^- + CN^- + 2OH^- = 2MnO_4^{2-} + CNO^- + H_2O \quad (31)$$

Potassium permanganate detoxifies cyanide free ion due to oxidation of cyanide to cyaniate ion.

Figure 20 Neutralized cyanide tailing

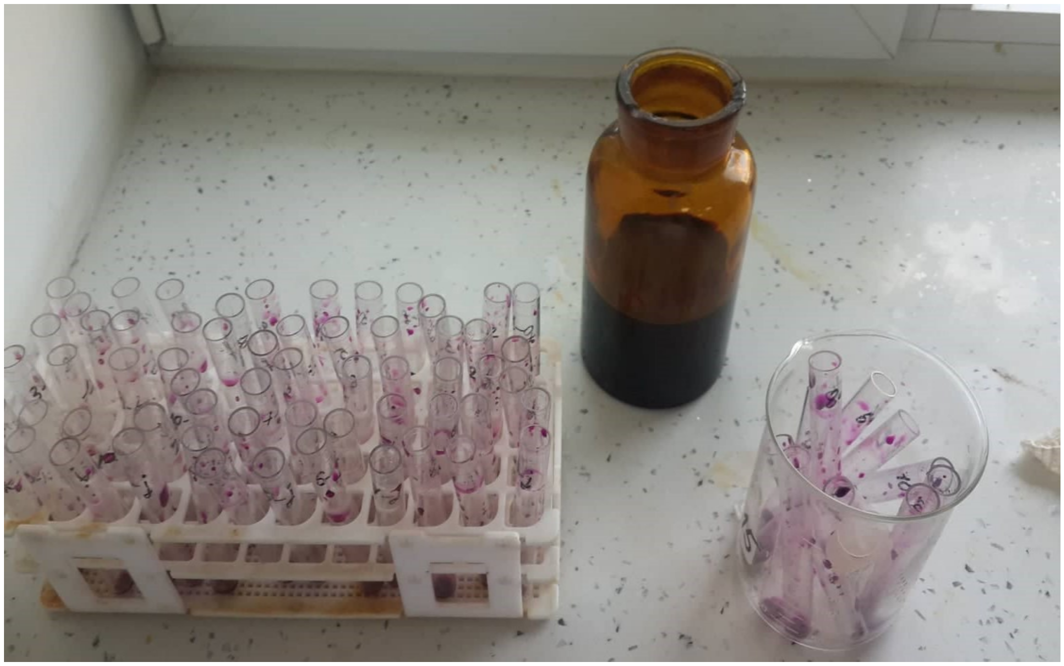


Figure 21 Washing with KMnO4 solution

7.7 Sample calculation for gold recovery

For 65% 250ppm solution:

Rich leachate concentration of first parallel – 3.11 mg/l

Rich leachate concentration of second parallel- 3.00 mg/l

Average concentration: 3.05mg/l

Concentration of tailing of first parallel- 1.05g/t

Concentration of tailing of second parallel-1.13g/t

Average concentration of tailing-1.09 g/t

Total gold content in the ore= $C_{tailing}$+(100-33)/33*$C_{leachate}$= 1.09+ (100-33)/33*3.05 =7.28g/t

8 Methodology of analysis

8.1 Fire assay

Fire assay test is used to determine gold grade of ore or concentrate by melting and dissolving sample with flux. Depending on the mass of the sample and dilution of the solution, fire assay test is cooperated with Atomic-Absorption-Spectroscopy test to define gold content ranging from 0.01 gram/ton to 20 gram/ton while with gravimetric method for defining higher gold content, ranging from 1.0 gram/ton to 100gram/ton.

The gold fire assay method is based on high solubility of gold and some precious metals in lead when melting the ore and it is reduced with lead. When ore melted in high temperature with the help of flux which is comprised of borax, sodium carbonate and lead (III) oxide, there are two phases are formed, which are different from each other by their viscosity. Non-oxidized precious metals are collected and concentrated in crude lead bullion while easily oxidized elements turned into slag. This principle is the core of fire assay to separate the metal from its surrounding parts. After melting process, coroleck which contains precious metal is picked up from lead button with helping of cupellation process. From solution of dissolved coroleck, content of precious metal could be determined by classic method of gravity, AAS or ICP.

1. Sample preparation

The important part of fire assay method deals with non-uniform gold distribution of ore sample. Gold containing ore constitution cannot be uniform because gold is very malleable metal, thus it won't be smashed and only changes its shape during crushing process.

To prepare most suitable sample for represent the whole ore, the following guidance should be complied.

- Before crushing and breaking down, the ore sample must be fractured into 2mm-3mm. the particle size should be minimized in 0.074mm or 200 mesh.
- Required weight of the sample is 200g-500g. During crushing process, big sized parts have to be re-crushed by machine and sieving must be done after each

crushing cycle to prevent form inaccuracy of result. If the big parts of gold sample remains over sieve, it can be taken to extra analysis to determine correct result.

- During melting Iron-rich sample, the sample must be well sieved several times with 0.074mm /200mesh/ sieve.

2. Procedure

A. Melting

- Stir well sieved sample and put it on smooth paper. Make crosspieces /approximately 20/, take 50mg sample from each section and put it into crucible. During weighting, prevent the sample to pollute from surrounding.
- Weight up and put fluxes (quartz, oxidizer, reducer etc) and reagents which are suitable for the sample.
- Drop 0.1 ml of Ag solution (10% concentration) to crucible using micropipette. 0.1 ml solution contains 10mg Ag.
- Put the crucible into furnace which pre-heated to 800°C-900°C, melting will be done at 900°C for 20 minutes then continued at 1000°C for another 20 minutes. Open the door slightly and check if molten sample is spilled, boil over and loss-making.
- After 40 minutes, adjust the temperature to 1080°C and wait for 20 minutes. With increasing temperature, the sample molten for 1 hour. Then pour it into pre-heated iron crucible. When molten sample cools down, remove lead button from slag and shape it like cube with hammer.

B. Cupellation

Figure 22 Cupels

- After numbered cupels which absorb liquid lead were put in heating furnace for 20 minutes at 200°C-250°C, locate lead buttons in the cupels. After 5 minutes from starting cupellation, lead melted and its surface turned into red then process of set lead apart from precious metal will begin.
- When cupellation began, leave the furnace door open with 1-2cm, let the air flow in then oxidize lead. The temperature dropped into 820°C and remained. Sometimes the process named ‘frozen’ will cause because of leaving the door open for long time. In that case close the door and heat furnace up to 950°C, melt the lead once more and begin the cupellation again.
- When the remained lead on the cupel becomes 0.5 cm in diameter, adjust temperature to 880°C to finish cupellation.

C. Dissolution of coroleck

- After cooling down gold and silver globule, coroleck, transfer it in volumetric flask of 10ml with helping of crucible clamp.
- Add 0.5ml $HN0_3$ (con) to flask, heat slightly and dissolve silver. After brownish steam of nitrate exhaled, dissolution can be stopped.
- Afterwards dissolving globule silver, add 1.5 ml of HCl (con) to flask with nitrate solution, make aqua regia and remained other precious metals dissolved completely. Then heat solution slowly and dissolve gold globule. In the end of dissolution process, be aware of no black precipitation at the bottom of flask. (If there are white gold elements in the sample black precipitation will remain.)
- After dissolution of globule, cool down the solution and fill the flask with distilled water. Shake the solution and leave it over 4 hours till silver chloride precipitate. Then measurement can start.

Figure 23 Dissolution of coroleck in aqua regia

D. Atomic absorption spectrophotometer (AAS)

- AAS can determine gold grade in dissolved solution.

  Determine gold grade

Table 5 AAS

1	Wave length	242.8nm, 267.7 nm
2	Width of absorption line	0.5nm
3	Deuterium lamps	Turned on
4	Width of flame	10cm
5	Type of flames	Air-acetylene
6	Absorption	0.8/10 mg/ml Au
7	Standard solution for comparing	0.02-10 mg/ml

Figure 24 Atomic absorption spectrophotometer

3. *Preparing solution for calibration curve*

- In order to make calibration curve, prepare solution with 100mg/ml concentration from solution of 1000mg/ml concentration.
- To prepare 100mg/ml standard gold solution, take 10 ml of 1000mg/ml standard solution with micropipette and pour it into 100ml volumetric flask. Add 5ml of HNO_3 (con) and 15ml of HCl (con) then fill it with distilled water and shake it.
- To prepare standard solution to compare, put 5 ml of HNO_3(con), 15 ml HCl (con) to 5 volumetric flasks then prepare standard solutions with concentrations of 0.0mg/ml, 1.0mg/ml, 3.0mg/ml, 5mg/ml, 10mg/ml to compare.

Table 6 To prepate standard solution

100mg/ml standard gold solution	Concentration of gold solution
0	0.0
1	1.0
3	3.0
5	5.0
10	10.0

- To simplify analyze gold grade in solution, use factors of the prepared standard solution as calibration. So that take same sample weight and dissolution volume for each sample. Sample weight= 50 gr

4. *Result estimation*

Gold grade in sample defined as gram/ton (gram in per ton ore) is calculated as follows:

$$C_{Au} = \frac{V * C_1}{n} \quad (32)$$

- C_{Au}- gold content in sample (gram/ton)
- C_l- concentration of gold solution mg/ml
- V- volume of solution
- n- weight of sample

8.2 Definition of total cyanide in solution- chemical test

Cyanide is analyzed by 3 terms: total cyanide, free cyanide and weak acid dissociable (WAD). Total cyanide contains all cyanide complexes and free cyanide. Free cyanide is hydrogen cyanide and cyanide ion in solution pH higher than 6. WAD or CATD (Cyanide Amenable to Chlorination) is weak to moderate strong metal-cyanide complexes such as which $Ca(CN)_2$ $(Zn(CN)_4^{2-}$, $Cd(CN)_3^-$, $Cu(CN)_2$ dissociates in 3-6pH and forms hydrogen cyanide. Remained cyanide complexes are strong Metal-Cyanide complexes which include $Fe(CN)_6^{2-}$, $Fe(CN)_6^{4-}$, $Co(CN)_6^{4-}$, and $Au(CN)_2^-$ and hydrogen cyanide is formed as well. This type of cyanide complexes dissociates in very low pH <2.

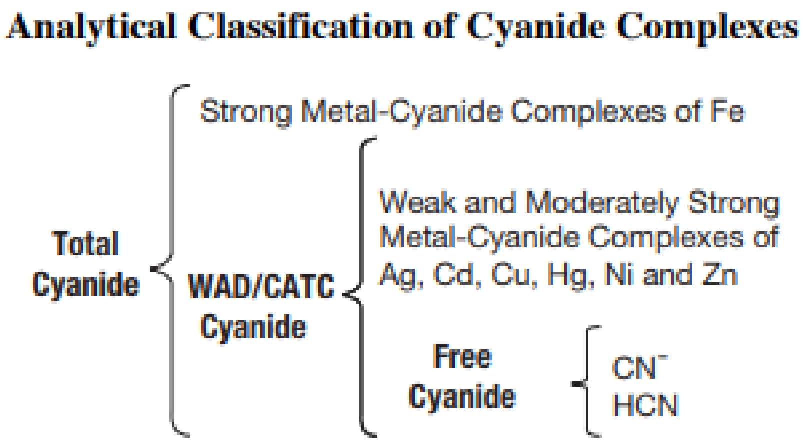


Figure 25 Analytical classification of cyanide complexes

8.2.1 Definition of total cyanide analysis

Cyanide complexes react with chemicals which are further added and cyanide liberates as hydrogen cyanide gas. Then it is collected by air flow in NaOH absorber.

1. Reagents:

   Reagents have to be pure (99.9%) and accomplish with measurement requirements. To make sure of reagents purity, blank sample measurement can be done with other measurement.

   - Distilled water
   - Hydrogen chloride, 1mole/L
   - Tin(II) chloride,$SnCl_2$, 50%- Dissolve 50g tin(II) ($SnCl_2.H_2O$) in 40ml of HCl then fill distilled water up to 100ml.
   - Sodium hydroxide NaOH, 1mole/L
   - Zinc cadmium sulfide solution- 100g zinc sulfate ($ZnSO_47H_2O$) and 100g cadmium sulfate ($3CdSO_48H_2O$) filled with distilled water to 100ml.
   - Copper(II) sulfate, $CuSO_4$, 20%- 200g $CuSO_4.5H_2O$ within 1000ml distilled water.

2. Procedure of measurement

   - Turn on biosafety cabinet
   - Put 10 ml NaOH in absorber.

- Put reagents in order with funnel to inlet tube. 30ml distilled water, 2ml $SnCl_2$, 10ml $CuSO_4$, 100ml sample, 10ml HCl. Close inlet tube.
- Connect absorber with distilling flask.
- Turn on compressor and adjust air flow at 20L/h
- Connect absorber, previously filled 10ml NaOH to condenser.
- Start magnetic stirrer.
- Heat the solution until it boils.
- Cooling water should drop 1-2 times in a second.
- Control flow in-out of cooling water with monitor.
- Check biosafety cabinet.
- After 1 hour, stop air flow and heat.
- Open inlet tube.
- If distilled solution is transparent, pour it into 25ml flask. Washout distillation flask with distilled water several times.
- If distilled solution is not transparent, do distillation again.
- Finally titrate the distilled product with AgNO3 solution in a same way as Free cyanide titration.

8.2.2 Definition of free cyanide analysis.

Silver complex ion forms due to the following reaction:

$$2\ CN^- + Ag^+ = [Ag(CN)_2]^- \qquad (33)$$

Silver cyanide precipitates when silver ion is excessive.

$$[Ag(CN)_2]^- + Ag^+ \rightarrow 2AgCN \qquad (34)$$

In order to define last point of titration precisely, potassium iodide is added because silver iodide is formed less reactive than silver cyanide. Colloid formation of silver iodide is recognized by Tindal effect.

1. Reactants used in the analysis:
    - All of the reactants are purified and water must be distilled.
    - Silver nitrate solution, C(AgNO3)=0,01mol/l, is stored in dark place.
    - In order to prepare potassium iodide indicator, 20g of potassium iodide is dissolved in water and add water until the solution volume reaches 100ml.
2. Procedure

- 10-25ml sample solution is taken into flask.
- 4-5 drops of 5-10% concentration of potassium iodide are added.
- Titrate with silver nitrate solution until yellow precipitate forms. Silver iodide formation is relatively slow, thus titration should be proceeded slowly.

Figure 26 Titration

3. Result estimation

Volume of silver nitrate which is consumed is defined.

1ml $AgNO_3$=0.01% CN

It depends on concentration of silver nitrate.

8.2.3 Definition of WAD analysis

Cyanide ion in weak to moderate strong metal-cyanide complex reacts with picric acid and forms isopurpuric acid. Picric acid changes its yellow colour to orange which is colour of isopurpuric acid. Intensity of colour or absorbance depends directly on concentration of solution according to Beer Lambert law. In this method pH is kept 9-9.5 and the solution must be heated in order to occur this reaction.

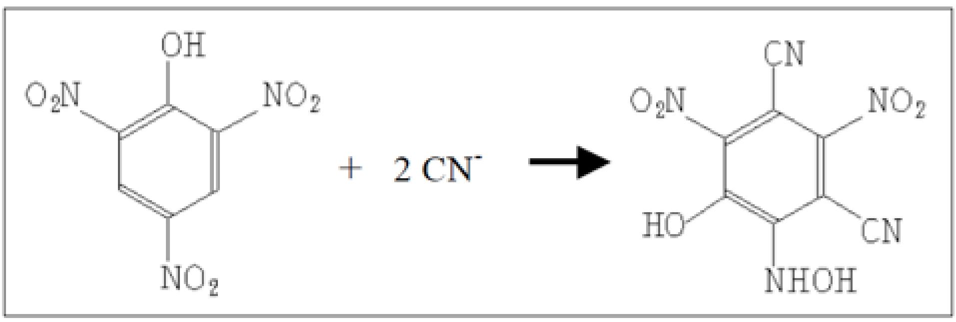


Figure 27 Isopurpuric acid formation

1. Reactants used in the analysis
    - Cyanide solution to measure which is distilled before in Total cyanide analysis.
    - 4% concentration of Ni solution
    - 25 ml of picric acid
    - Standard solution, its concentration is known.
2. Procedure
    - 10ml of distilled cyanide solution is taken into 100ml flask.
    - 1ml of Ni solution, 30 ml of distilled water and 25ml of picric acid are added in the flask
    - Mix the solutions and take it in boiling water pools for 20 min
    - Cool it down until it reaches room temperature

- Dilute the mixed solution until it becomes 100ml.
- Measure absorbance of the solution by spectrometer in λ=520nm
- Make calibration curve with a plot of absorbance vs concentration by standard solutions
- Determine matching concentration by using absorbance from calibration curve

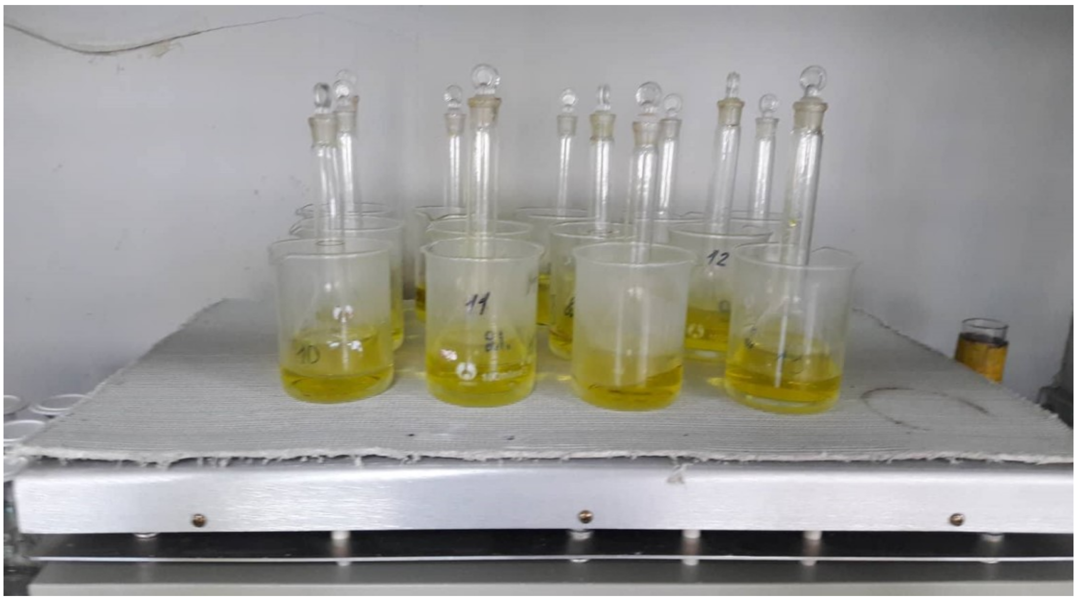


Figure 28 Heating the mixed solution in boiling water pool

Figure 29 Spectrometer

Figure 30 After reaction occured

3. Result estimation

   Concentration CN_{WAD} [mg/l]= Concentration CN measured / $V_{sample\ solution}$ [ml]*1000

9 Result and discussion

9.1 Definition of chemical composition of ore

28 element definition was done by Khan-Lab LLC.

Table 7 28-Element definition

Element %					Element mg/kg								
Al	Ca	Mg	K	Fe	As	Ba	Be	Bi	Cd	Co	Cr	Cu	La
7.25	4.98	1.83	0.8	6.26	15.9	128.3	<5.0	<10.0	<1.0	32.2	48.85	13.25	<5.0

Table 8 28-Element definition

Element mg/kg													
Li	Mn	Mo	Ni	P	Pb	Sb	Se	Sr	Ti	V	Y	Zn	Zr
18.3	1631	<5.0	20.4	683	7.42	<10	6.98	249.2	1086	173	12.29	87.2	26.54

From the analysis, the ore used in the experiment contains relatively low As and other heavy metals that inhibit the leaching process. Thus it is not necessary to have prior process for cleaning the elements.

9.2 Definition of optimal size of ore grain by grinding for different time duration

In the experiment, ore was grinded for 3 different time duration to define the optimal size of ore grain which gives highest gold recovery. Sample ore was grinded each of 65%, 75%, 80% passing 200 mesh sieves. According to the experimental data, after first half an hour of the cyanidation, sample containing 65% of -74 μ was dissolved slowest with approximately 20% gold recovery whilst 75% and 80% started with approximately 29% gold recovery in general. Sample ore containing 80% of it passing 74 micrometer sieve gave highest gold recovery in both of 200ppm and 250ppm of cyanide solution. At this finest ore sample, surface area is largest and gold particles are liberated well, therefore it influenced gold dissolution rate and gold recovery. Thus 80% -74 μ is the optimal size of ore for cyanidation.

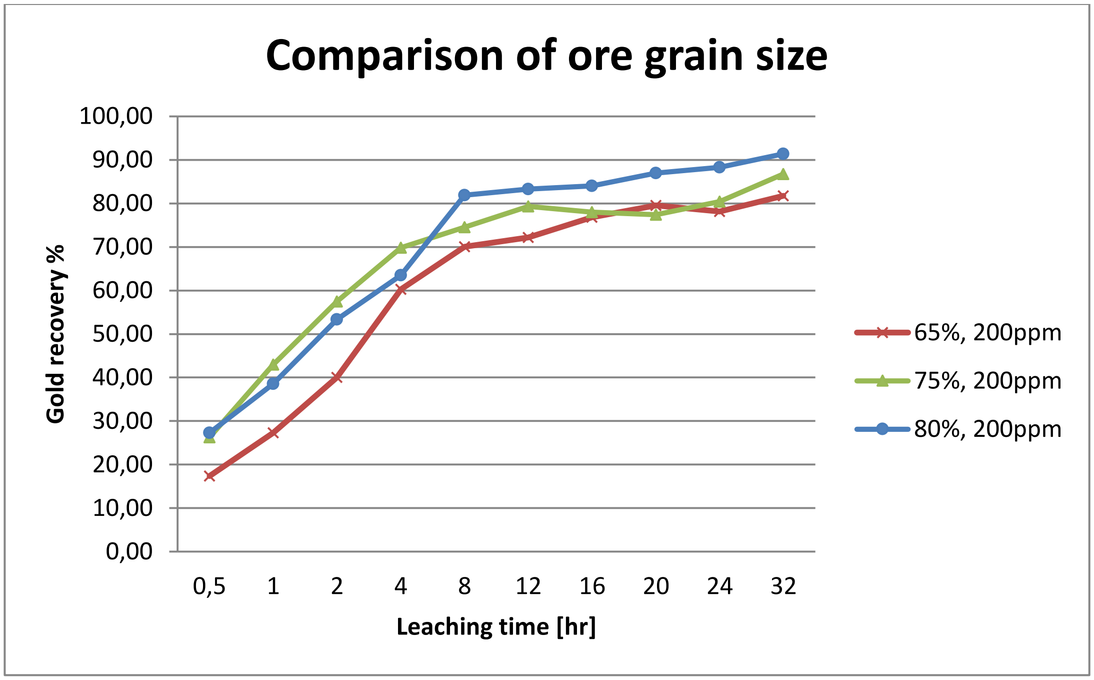


Figure 31 Comparison of ore grain size distribution

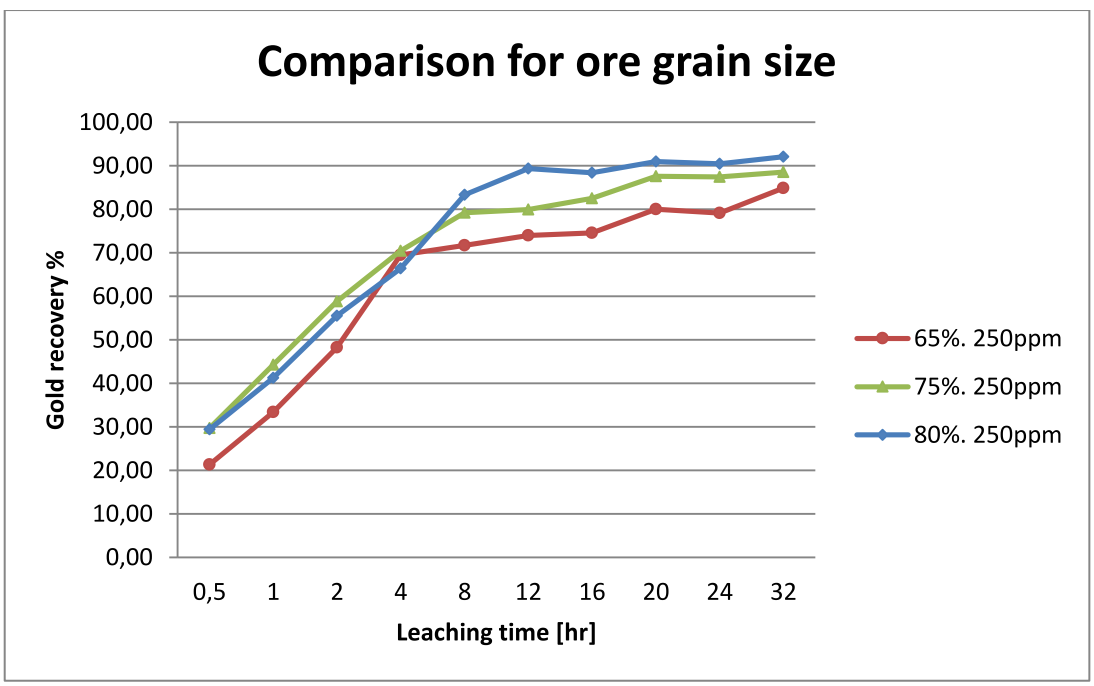


Figure 32 Comparison of ore grain size distribution

9.3 Comparison of recovery of gold in most optimal cyanide concentration of 200ppm and 250ppm cyanide in solution.

Cyanidation took place in 200ppm of cyanide solution and 250 ppm of cyanide solution because of its characteristics. For combination of oxidized and sulfide ore, 150-200ppm solution was used in Olon-Ovoot gold ore processing plant. Sulfide ore consumed higher amount of cyanide solution, according to cyanidation research on Olon-Ovoot mine (Radnaa, 2008). In this experiment, only sulfide ore is tested. Therefore it is assumed that 200-250ppm is the optimal cyanide concentration.

Table 9 Comparison of cyanide concentrations for 65% of -74µ

		Leaching time [hr]										Gold content in Tailing g/t
	Leaching time [hr]	0.5	1	2	4	8	12	16	20	24	32	
65% 200 ppm	Gold content g/t	7.45	7.45	7.45	7.45	7.45	7.45	7.45	7.45	7.45	7.45	1.37
	Dissolved gold in solution g/t	0.65	1.02	1.49	2.24	2.61	2.69	2.86	2.96	2.91	3.04	
	Gold recovery %	17.41	27.30	40.05	60.24	70.00	72.20	76.80	79.51	78.15	81.73	
65% 250 ppm	Gold content g/t	7.20	7.20	7.20	7.20	7.20	7.20	7.20	7.20	7.20	7.20	0.9
	Dissolved gold in solution g/t	0.77	1.20	1.74	2.50	2.58	2.66	2.68	2.88	2.85	3.05	
	Gold recovery %	21.31	33.38	48.26	69.53	71.66	73.99	74.54	79.99	79.10	84.86	

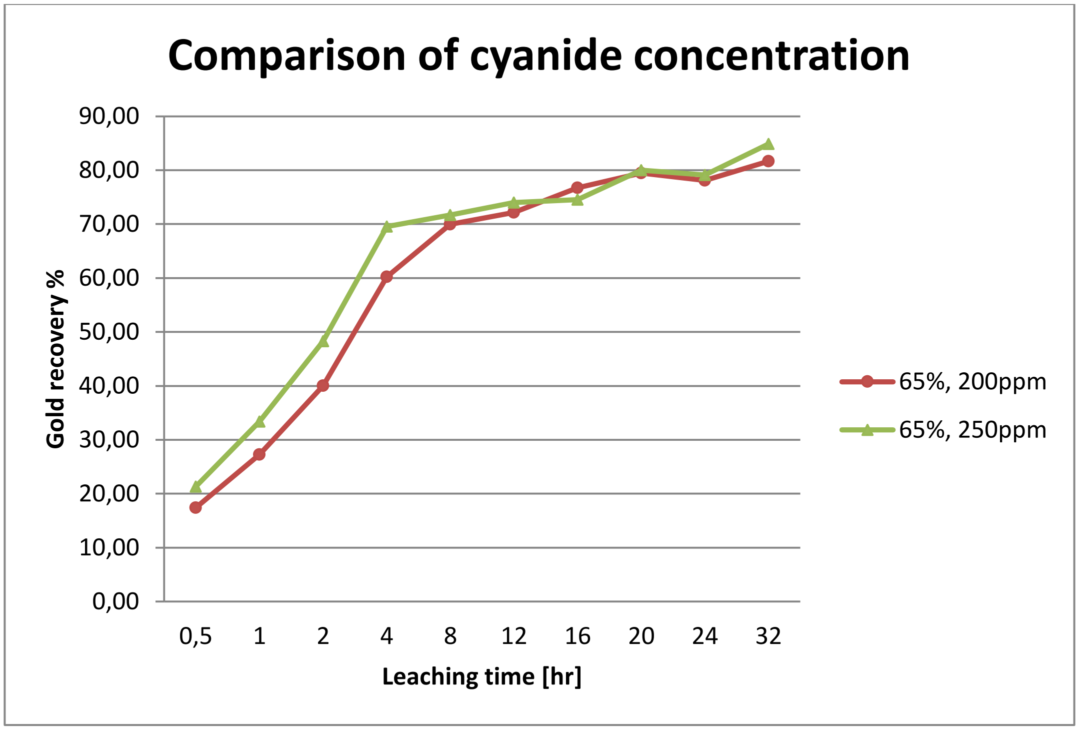


Figure 33 Comparison of cyanide concentrations for 65% of -74µ

As presented in Table 8 and Figure 33, at the beginning cyanidation 250ppm solution gave higher gold recovery. Between 8 and 24 hours of leaching, gold recovery was approximately equal in both solutions. Then 250ppm solution ended with higher gold recovery which is 84,86% than in 200ppm solution which is 81,73%.

Table 10 Comparison of cyanide concentrations for 75% of -74µ

		Leaching time [hr]										
	Leaching time [hr]	0.5	1	2	4	8	12	16	20	24	32	Gold content in Tailing g/t
75% 200 ppm	Gold content g/t	7.21	7.21	7.21	7.21	7.21	7.21	7.21	7.21	7.21	7.21	
	Dissolved gold in solution g/t	0.95	1.55	2.07	2.52	2.69	2.86	2.81	2.79	2.90	3.13	0.96

	Gold recovery %	26.29	42.99	57.46	69.89	74.56	79.34	77.99	77.46	80.46	86.75	
75% 250 ppm	Gold content g/t	7.40	7.40	7.40	7.40	7.40	7.40	7.40	7.40	7.40	7.40	0.85
	Dissolved gold in solution g/t	1.10	1.64	2.18	2.61	2.93	2.96	3.05	3.24	3.23	3.27	
	Gold recovery %	29.72	44.25	58.85	70.43	79.23	79.94	82.50	87.56	87.43	88.51	

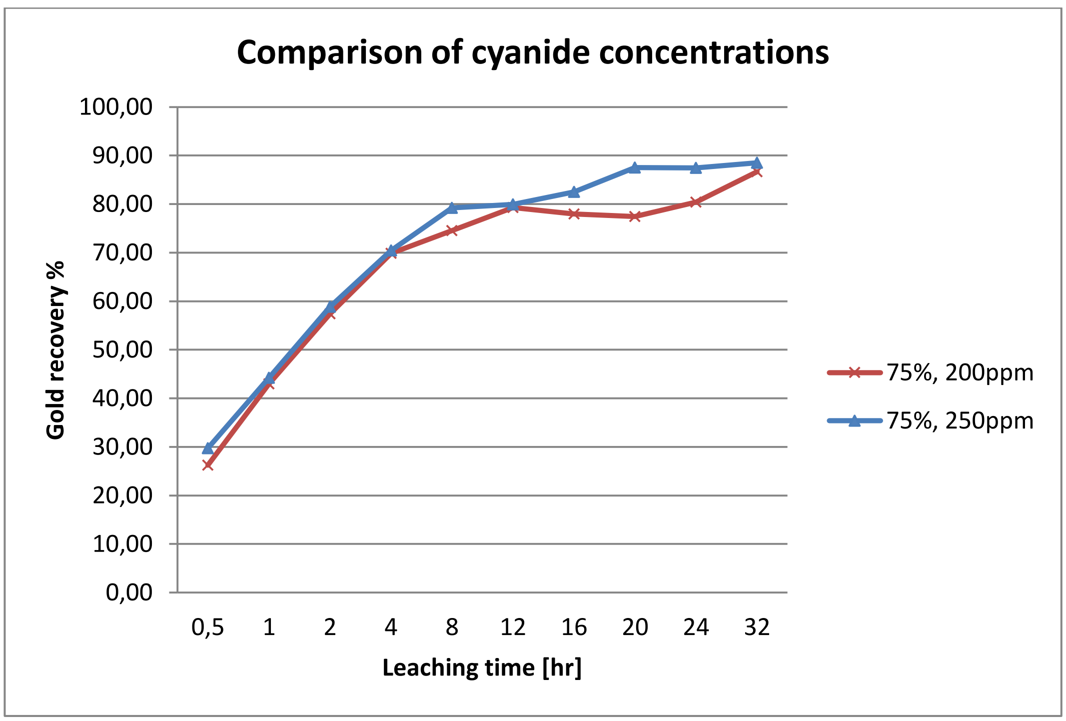


Figure 34 Comparison of cyanide concentrations for 75% of -74µ

For 75% of -74 µ sample, both 200ppm and 250ppm solution dissolves gold at the almost same rate until 4 hours of leaching. From 8 hours to 32 hours, 250ppm solution speeded up . Final gold recovery for 250ppm was 88,51% and for 200ppm solution was 86.75%

Table 11 Comparison of cyanide concentrations for 80% of -74µ

	Leaching time [hr]	Leaching time [hr]										Gold content in Tailing g/t
		0.5	1	2	4	8	12	16	20	24	32	
80% 200ppm	Gold content g/t	7.38	7.38	7.38	7.38	7.38	7.38	7.38	7.38	7.38	7.38	
	Dissolved gold in solution g/t	1.01	1.42	1.97	2.34	3.02	3.07	3.10	3.21	3.26	3.37	
	Gold recovery %	27.32	38.56	53.34	63.52	81.89	83.24	84.01	86.92	88.26	91.4	0.64
80% 250ppm	Gold content g/t	7.62	7.62	7.62	7.62	7.62	7.62	7.62	7.62	7.62	7.62	
	Dissolved gold in solution g/t	1.12	1.57	2.12	2.53	3.17	3.40	3.37	3.47	3.45	3.51	
	Gold recovery %	29.39	41.23	55.54	66.37	83.31	89.31	88.39	90.96	90.47	92.06	0.61

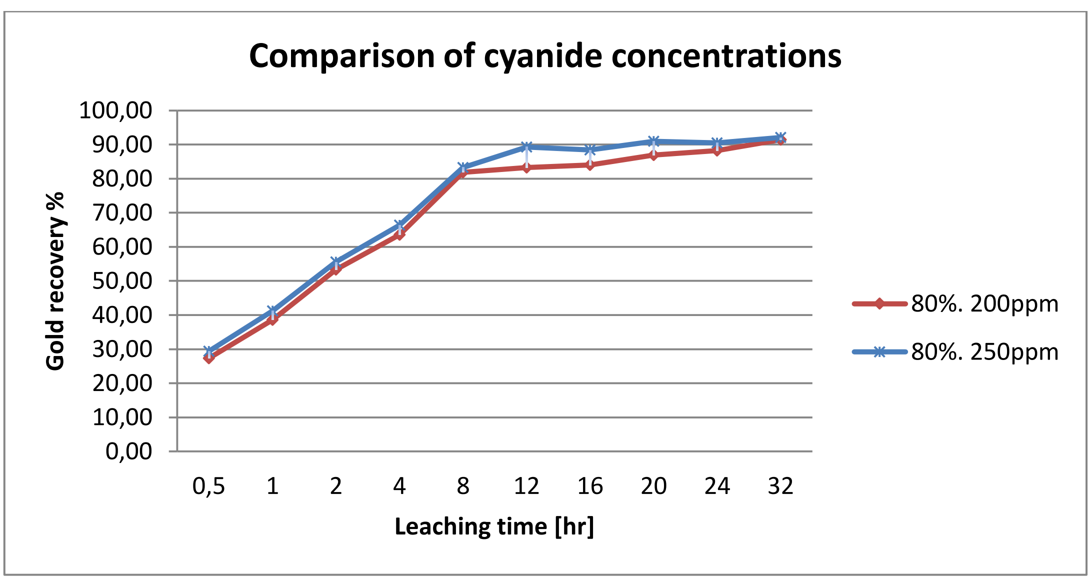


Figure 35 Comparison of cyanide concentrations for 80% of -74µ

For 80% of -74μ ore, 250ppm solution gave higher gold recovery during all of the leaching time and final point was 92,1 gold recovery whilst in 200ppm solution final point was 91,4. Thus optimum cyanide concentration would be 250ppm.

9.4 Definition of optimal leaching time duration

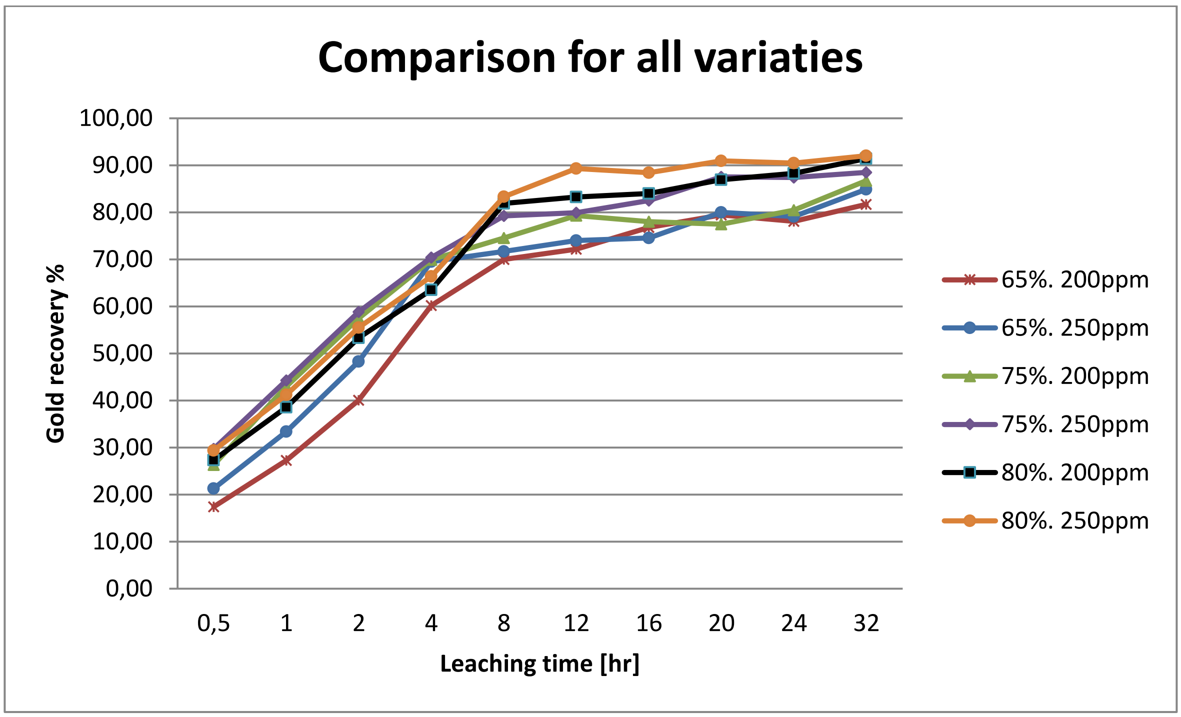


Figure 31 Comparison for all varieties

A presented in Figure 36, gold dissolution occurred actively in first 8 hours. Between 8 to 32 hours leaching proceeded gradually and approximately 10% of gold is recovered within the time frame. For instance the ore containing 80% of -74 μ was leached for 8 hours resulting 81,89% gold recovery and after 32 hours of leaching gold recovery elevated into 91.4% . From the graph presented above, all of the lines are slightly increasing until 32 hours. Therefore 32 hours of leaching is the optimal time duration.

10 Conclusion and recommendation for further research

The experiment conducted on gold rich sulfidic ore of Olon-ovoot mine and the ore has 7,24g/t. gold grade. Each cyanidation proceeded with 500g of ore sample and 1000ml of cyanide solution. Cyanide solution varies 200ppm and 250ppm. Grain size of ore varies 65%, 75% and 80% passing 74µ sieve. Leaching continued for 32 hours and liquid samples were taken every 2 to 4 hours. Those liquid samples were analyzed by AAS and solid tailings were analyzed by fire assay method to define its gold content.

The highest point of gold recovery was 92,1% for ore which grinded 80% of 74µ in 250ppm cyanide solution for 32 hours.

As concluding:

- Optimal size for leaching is 80% of -74 µ
- Optimal cyanide concentration 250 ppm
- Leaching time 20-32 hours

Recommended researches are research more deeply in nature of gold particles and doing the experiment depending on its nature. Also could continue the experiment with higher concentration of cyanide with aeration pump for next level of researchers.

11 Acknowledgement

I would like to express my deep gratitude to my research supervisor Mrs.Oyunbat for her valuable guidance and advice with professional knowledge. I would like to thank the staff of Khanlab LLC for providing laboratory to accomplish my research and enabling me to apprentice the analysis which were done to my samples. Assistance provided by Ms.Azjargal was greatly appreciated.

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