

An effective option to reduce environmental impact of reprocessing mercury contaminated tailing in Artisanal Gold Mining

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Abstract

Artisanal small-scale gold mining (ASGM) is characterized by the use of rudimentary process to extract gold. Worldwide, mercury loss to the environment from ASGM sources can reach 1,000 tonnes annually. The Brazilian Amazon is one of the most important ASGM regions in the world, with more than 200,000 miners. In a location called Garimpo Ouro Roxo, miners use amalgamation and cyanidation in vat-leaching. Each cycle usually recovers 50% of the gold and takes 20 days per tank, consuming around 3,300 kg NaCN/month. A new process has been developed and implemented in a pilot plant in this area, involving gravity concentration followed by cyanidation in a ball mill. Concentrate leaching is conducted with a PVC capsule filled with activated carbon inserted in the cyanide solution in the mill. The cycle takes less than 24h and recovers up to 98% of the gold in the concentrate. The main advantages of the new process include reducing the gold recovery cycle and reducing cyanide consumption by up to 22 times. This process has proved to reduce the annual cyanide consumption from 22,000 kg to 980 kg. In addition, the process may be suitable to eliminate the use of amalgamation at the site.

Keywords: artisanal mining; mercury contamination; environmental education; cyanidation; gravity concentration.

1 Introduction

1.1 Artisanal small-scale gold mining

Artisanal small-scale gold mining (ASGM) is characterized by the use of rudimentary techniques to extract gold from any type of ore, employing low level of mechanization and very low safety standards [1]. ASGM typically employs an extensive contingent of unskilled people, causing a large influx of migrant workers and often the results are low productivity and high environmental impacts. Other characteristics of ASGM also include illegality due to mining without concession rights and work by trial and errors as miners do not know their geological reserves [1].

The Brazilian Amazon is one of the most important ASGM regions in the world, considering the number of miners, gold production, mercury consumption and the overall environmental impacts derived from ASGM activities. The peak of the cycle of gold in the Amazon happened in the

1980s, when the population of artisanal miners was estimated in 1 million people [2]. This population is currently believed to be around 200,000 artisanal miners [3]. The loss of mercury to the environment is only the most classical example of the problems associated to ASGM, but the indiscriminate use of cyanide is becoming an increasing trend among miners, posing serious risks to their health and the environment.

Tapajos River Basin, in the Brazilian Amazon, was one of the pilot sites chosen by the United Nations GMP - Global Mercury Project [4] in 2002. This project gave important contribution by helping miners improve their practices, and allowed the implementation of a pilot plant to demonstrate improved techniques for concentration and cyanidation. Results of the tests in this Brazilian location can be easily adapted to other similar ASGM sites.

ASGM in Brazil is known as “garimpo” and the artisanal miners are called “garimpeiros”. In 2008, Brazil officially produced 54 tonnes of gold, and around 6 tonnes were produced by artisanal miners [5], of which around 70% is believed to come from the Tapajós Region.

The main objective of this study is to describe an effective alternative to replace traditional artisanal gold mining activities using amalgamation or vat-leaching, with gravity pre-concentration followed by cyanidation of the concentrate. The proposed alternative is a combination of steps of existing processes with new steps adapted to make it simple and accessible to artisanal miners. A pilot plant implemented in a pilot site in the Tapajos region has shown positive results not only by improving gold recovery, but also by contributing to replace mercury amalgamation and reduce cyanide consumption.

1.2 The use of mercury in artisanal gold mining

Mercury amalgamation has been a traditional method for gold recovery for centuries. In artisanal mining, due to the low efficiency of the gravity concentration and amalgamation, the gold recovery is generally not higher than 30 to 40% and the tailings contain residual mercury [6].

Amalgamation is more efficient with gold liberated from the gangue material, generally coarser than 200mesh (0.074 mm) [6]. This also depends on the shape of the gold particles. Grinding is a crucial step in this process, permitting the exposure of gold particles to mercury.

The main concern regarding the use of mercury amalgamation is the high toxicity and potential to pollute the environment. Mercury accumulates in vital organs, affecting the human's nervous system and causing serious neurological diseases [7].

Once the amalgam has been formed, in order to separate mercury from gold, miners generally burn the amalgam and evaporate the mercury. Ideally, retorts should be used to condense the evaporated mercury, but this practice is not sufficiently disseminated among artisanal miners worldwide [8, 9, 10].

Slowey et al. [11], studying placer gold mine, have found that the metallic mercury used by the miners can form different compounds in the ore processing steps, undergoing volatilization, globule relocation, dissolution, oxidation, precipitation and amalgamation and sorption onto sediments. A significant part of the mercury, in form of colloid, may stay in solution or adhere to sediment components such as iron and aluminum (hydr) oxides, which facilitate mercury transportation in aqueous mean [12]. Slowey et al. [11] found that more than 50% of the elemental mercury added to the sediment during amalgamation can be transformed into soluble and particulate species, including mercury organically bound, mercury absorbed to sediment minerals, and forming HgS.

Depending on the mercury species, it may become more, or less, amenable to gravity concentration.

Telmer et al. [3], studying mercury in the Tapajos River, found that Hg bound to suspended sediment has around 600 times the concentration of dissolved Hg per litre of water in impacted areas, and 200 times in unpolluted areas. This allows mercury transportation up to 200-300km downstream in the Tapajos River.

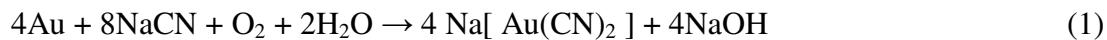
Velasquez-Lopez et al. [13], working with ASGM in Ecuador, found that from the total metallic mercury in the tailings submitted to cyanidation, 24% is trapped at the bottom of agitated tanks, 33% is lost as tailings associated with solid material, 12% is lost in solution and 31% of Hg in solution is reported to zinc precipitation cells. From the latter, 28% is precipitated on the zinc shavings and evaporated when the shavings are burnt, and 3% stays in the barren solution.

1.3 *The use of cyanide in artisanal gold mining*

According to Akcil [14], cyanidation responds for nearly 90% of the gold processed in the world. The technique is efficient to dissolve fine gold, usually smaller than 0.2mm. A solution is prepared with the ground ore, the pulp pH is adjusted to be alkaline and a diluted solution of cyanide is added to the pulp. Typically sodium cyanide crystals are used.

The control of pH is critical to avoid loss of cyanide as gaseous hydrogen cyanide, which is highly toxic. In low pH, hydrogen cyanide may be formed, as follows: $\text{CN}^-(\text{aq}) + \text{H}^+(\text{aq}) \rightarrow \text{HCN}(\text{g})$. Therefore the free proton concentration is kept low by the addition of alkali such as lime (calcium hydroxide) or sodium hydroxide to ensure that the pH during cyanidation is maintained over 10 [15, 16].

The chemical reaction proposed by Elsner (1846), cited by Marsden & House [15], is as follows:



Cyanidation requires, therefore, the presence of free or exposed gold, a cyanide solution and dissolved oxygen in an aqueous system.

As per Equation 1 dissolved oxygen is an integral component of cyanidation, and deficiency in solution can slow or even halt the leaching reaction. Oxygen (O_2) can be supplied through natural or forced air, or pure oxygen bubbled into the pulp to increase the dissolved oxygen concentration. Oxygen can also be added into the system through hydrogen peroxide solution.

The effectiveness of the leaching process depends on the interaction of every component in the system gold-air-cyanide, but generally gold recoveries situate above 90% when the pulp is agitated.

Cyanide is highly toxic to human and to the environment, but it is safe if handled accordingly. Free cyanide is readily absorbed through inhalation, ingestion or skin contact and is distributed throughout the body via blood [6]. It induces tissue anoxia so that oxygen cannot be utilized by the cells and death can result from the depression of the central nervous system. In solution, only 3 to 5 mg of cyanide/kg body weight can be lethal, and in respiratory exposure to hydrocyanic acid gas (HCN), only 0.1 to 0.3 g/m^3 can cause death. Fish are even more sensitive to cyanide, and concentration of free cyanide between 0.05 and 0.2 mg/L are enough to cause mortality [6].

The use of cyanide in ASGM can be safe provided that miners implement safe procedures to monitor the use in all stages of the process and to destroy residual cyanide in the tailing. Processes requiring the cyanidation of the whole ore must handle large amount of cyanide, whereas processes based on pre-concentration of gold followed by cyanidation can reduce the amount of cyanide some hundreds times.

Vat-leaching with cyanide is relatively new to most miners in Brazil. Millions of tonnes of tailings accumulated for decades in streams and riverfronts from old operations are now being treated to extract the residual gold. This is particularly true in the cases where alluvial and colluvial deposits are exhausted and miners have difficulties to find new deposits. Vat-leaching has been used with poor adaptations and it is very inefficient. This may pose risks of cyanide spills and human and fauna contamination. Miners work in highly environmental sensitive areas located along the rivers and other water bodies, creating impacts to these habitats.

Cyanide dissolves not only gold but also any remaining mercury left behind from the amalgamation process. Mercury cyanide can be formed and it is more bioavailable or easier to be methylated than metallic Hg when discharged into the water streams [17]. Mercury cyanide released by ASGM can cause serious environmental damages, which may include higher levels of Hg in fish in regions where amalgamation and cyanidation have been used combined and the tailings have not been properly treated and disposed [7].

Due to its properties, effectiveness and low cost, cyanide has shown to be the most promising reagent to replace mercury. Artisanal mining tends to follow the trend of large operations, where cyanidation has been the main reagent for leaching gold.

1.3.1 Intensive cyanidation

Conventional cyanidation generally uses cyanide concentrations around 0.2 to 1g/L and natural oxygen concentration. However the kinetics can be accelerated with the use of high concentrations of cyanide and forced oxygen into the system, referred to as intensive cyanidation [18, 19]. Intensive cyanidation is commonly used to leach concentrates obtained by gravity separation or flotation. An intensive cyanidation procedure has been used by Gekko Systems (manufacturer of a intensive cyanidation unit ILR - InLine Leach Reactor) and it has shown recoveries above 95.0% [20]. In intensive cyanidation processes, conventional activated carbon circuits can be removed by using direct electro winning of gold bearing solutions [18].

Atmospheric oxygen acts as the oxidant agent in gold cyanidation but its solubility in the leaching solution can be limited depending on the leaching process. Cyanidation rate is determined by the oxygen concentration in solution. Oxygen concentration decreases as the reactions proceed and it is not replaced rapidly enough due to the viscosity of the medium and inadequate aeration. This decreases the gold leaching rate [19].

According to Guzman et al. [19] and Longley et al. [21], several authors have reported that the addition of small amounts of hydrogen peroxide (0.012g/L) in the gold leaching has no effect on the cyanidation rate. Kameda [22] reported that larger concentrations of H₂O₂ (0.20g/L) increase the gold dissolution rate, but concentrations higher than 0.40g/L start decreasing the dissolution due to cyanide oxidation by peroxide. Day [23] found that there is no direct correlation between the gold dissolution rate and the cyanide or peroxide concentrations. Guzman et al. [19] concluded that although some authors described that hydrogen peroxide is as important as cyanide, at concentrations lower than 0.34g/L, pH 11.5 and 25°C, cyanide is not oxidized, and therefore any decrease in the cyanidation rate cannot be attributed to cyanide oxidation by H₂O₂. Nevertheless, cyanidation rate can be doubled by adding H₂O₂ at concentrations close to 0.50g/L, varying from 0.17g/L to 0.68g/L, compared to conventional cyanidation at pH 10.

Gray et al. [18] and Longley et al. [24], describing Gekko Systems, recommends the use of 2% (equivalent to 20,000ppm or 20g/L) of NaCN and 20mg/L (equivalent to 20ppm or 0.02g/L) of H₂O₂. According to Deschenes et al. [25], the minimum oxygen concentration [O₂] inside a ball mill has to be at least 6 times larger than the concentration of cyanide [CN]. The free space in the mill is sufficient to assure O₂ supply, and over the course of the leaching process the mill is frequently stopped for analysis of pH and [CN], which allows air replenishing.

When adding lime to the solution, pH above 12 has to be avoided as it slows the cyanide reaction due to formation of lime crusts on the gold surface [19].

2 Material and methods

2.1 Site description

The study site is called Garimpo Ouro Roxo (GOR) and is located in the municipal district of Jacareacanga, state of Para, in the village of Sao Jose. GOR is located in the Pacu River Basin, a

tributary of the Tapajos River, as shown in Figure 1. The geographic coordinates are lat 04°16,' lon 55°35', and altitude 45.0m.



Figure 1 – Garimpo Ouro Roxo location

The GOR mining claim covers 150 ha, larger than most typical “garimpos” in the region. However, GOR’s use of mercury and cyanide are representative of most locations in the Amazon. The first mining activities in this area were initiated in the 1960s with manual exploitation of alluvial gold and later on with the use of hydraulic monitors and carpeted sluice boxes for gold concentration from colluvial deposits. Mercury amalgamation has been largely employed over this period of time. In a second phase, in the 1990s, miners discovered the primary deposits and started to exploit gold veins in shafts and drifts. Miners extract the auriferous ore and grind the chunks of rocks in hammer mills to recover gold in copper plates amalgamated with mercury [26].

Tailings have accumulated over the past 3 decades in low lying land close to the operations which the miners have nicknamed “baixao”, These tailings are the source of material submitted to vat-leaching (Figure 2).

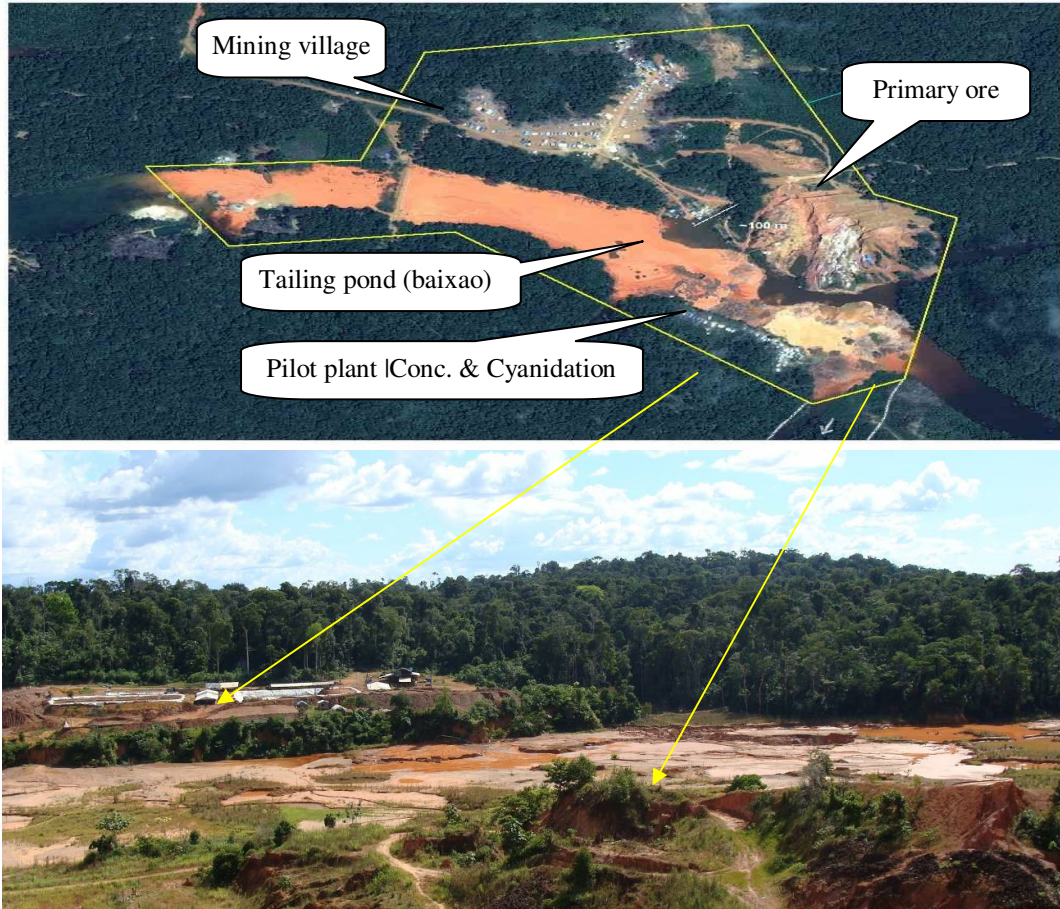


Figure 2 – Panoramic view of the “baixao” (“improvised” tailing pond) at GOR

2.2 The current processing procedure

According to GOR miners, 80kg of gold have been produced since the introduction of vat-leaching in 2007. The miners practice no method to estimate gold recovery from the vats. The current process flow is shown in Figure 3.

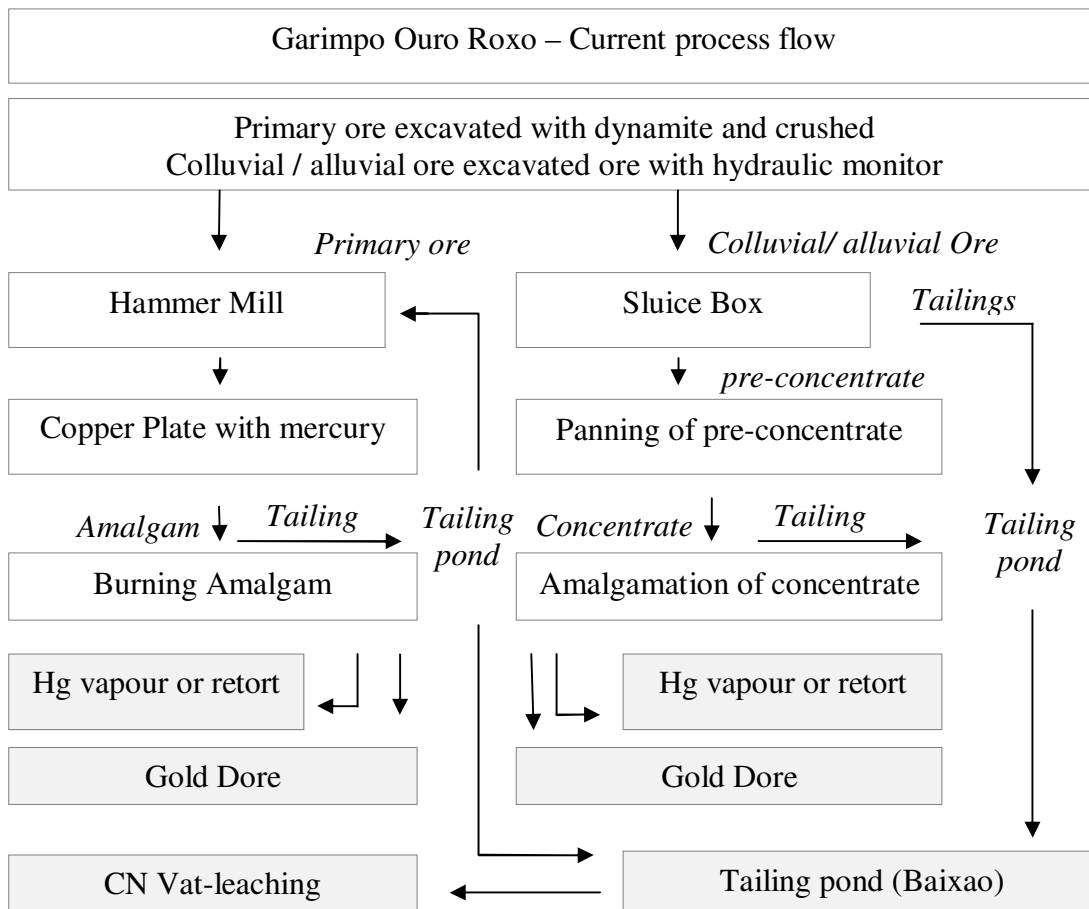


Figure 3 – Overview of Garimpo Ouro Roxo process flow

Since 1990, more than 40 shafts and drifts were constructed at GOR, of which 6 are currently in operation. The ore is crushed with hammer mills, and then passed over mercury coated copper plates. The resulting amalgam is scratched off the plates and then burned with a propane torch. Each shaft generally employs about 10 miners and 2 supervisors/ shaft and requires 1 or 2 hammer mills. One or two operators manually shovel ore into the hammer mill while operating the copper plate.

Average daily ore production is 3.6t/day and the average gold production is 60g/day per shaft, which corresponds to 17g Au/tonne (recovered gold by amalgamation alone). Miners believe the overall grade ranges between 5 to 55g/t, while recoveries can vary from 30 to 60%.

All amalgamation tailings are discharged into the “baixao” (Tailing pond). The GOR’s partners decided to restrict the use of amalgamation to primary ore: they do not reprocess tailings with copper places. Amalgam is collected at the end of every day and gold production is around 1.5kg Au/month.

GOR implemented the first vat-leaching tank in September 2007 in order to reprocess the tailings. By February 2009, 10 tanks with the capacity of processing 3,120tonnes/tank per cycle had consumed a total of 33,000kg of NaCN (22,000kg/year), representing an average of 1.0kg NaCN / tonne ore. Each tank has area of 960m², with dimensions of 40m x 24m, subdivided in 24 sections aiming at better percolation control of the cyanide solution. Each section has 8m x 5m x 2.5m receiving about 100 m³ of ore, with the apparent density estimated in 1,300kg/m³.

As the tank is filled with tailings from the “baixao”, a cyanide solution is prepared in a 60m³ pool, with the dimensions of 8m x 5m x 1.5m (depth). The cyanide solution is pumped into the system through irrigation of the sections. The pH of the system is adjusted to 10 with addition of lime. A second pool of 60m³ receives the gold-pregnant solution. The total amount of water in the system is affected by rain but this variable is not properly controlled. The CN⁻ concentration in solution is verified by operator using titration with AgNO₃, as described at Cyanide Management Code [27], and is usually kept around 1.0g/L. Figure 4 gives the general view of the vat-leaching process.



Figure 4 – General view of the vat-leaching process

The miners take around 8 days to fill the vat-leaching tank with tailings, using a front end shovel and a team of 8 men per 12h working shift. Once the vat is filled, the cyanide solution passes through activated carbon and is recycled to the vats. After 12 days 300kg of loaded carbon is removed for elution and replaced with 150kg of fresh carbon, which is removed after 8 days. Thus in theory, the production cycle is 28 days. However, the real cycle is often over 50 days due to operating problems, such as broken machines, excessive rain, delays in transportation, or the lack of material to leach.

At the end of each cycle, the spent CN^- solution is recomposed by increasing the CN^- concentration to at least 1g/L and adjusting the pH to between 10 to 11 with lime. If the used solution needs to be discharged the CN is first destroyed with the addition of sodium hypochlorite and by exposure to sunlight. Despite the fact that miners have oversimplified the process and have also an inadequate control of CN concentration and pH, this process is yet more efficient than previous processes relying on amalgamation. Past production figures indicates that the gold recovery in this vat-leaching process is around 50%.

2.3 Centrifugal concentration

Through this study, the most recent innovation implemented at GOR was a pilot plant for concentration and cyanidation of concentrate. This pilot plant was implemented in February/2009, and since then it has been frequently adapted and improved, with the goal of gradually replacing the existing process based on vat-leaching and mercury amalgamation. Gravity pre-concentration by was selected as it dramatically reduces the chemicals required for leaching while also reducing the amount of mercury contaminated tailings exposed to cyanide. Gravity pre-concentration also allows for the removal of sulphides from reprocessed tailings reducing the potential mobilization of heavy metals and acid generation [20]. The reduced use of cyanide also lowers the amount of cyanide which needs to be destroyed.

In the start up phase, the pilot plant equipment consisted of 2 hammer mills, a vibratory 2mm sieve, one Falcon iCon® centrifugal concentrator and a small ball mill. Figure 5 shows the general process flow sheet.

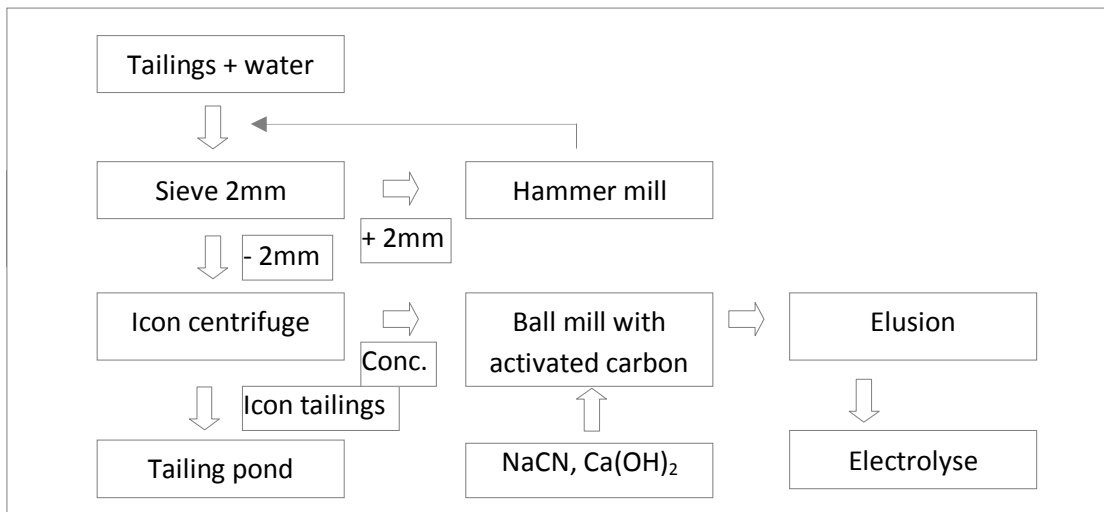


Figure 5 – Flowchart of the pilot plant process

The tailings from “baixao” are fed into the pilot plant manually at about with 30% of solids by weight, at a rate of approximately 1.4 t/h. As the feed is handled by one miner with a shovel, the flow and pulp density are not constant. Another operator cleans the centrifuge and a third operates the cyanidation of the concentrate in the ball mill.

The iCon® centrifuge was developed by Falcon Concentrators to make modern gravity concentration equipment affordable for some ASGM sites. The current cost of an iCon® in Brazil is around US\$7,000.

Equipment specifications include:

- Capacity: pulp of 30% solids, up to 2t/h.
- Transportation: The centrifuge assembled in a 200L drum. The base is concreted to avoid bouncing in operation.
- Process water: Water pressure 10 PSI, eliminates the lightest material in the bowl, by opposing to the centrifugal force. (how much per hour, cleanliness)
- Rinse hose is used to wash the bowl after discharge.
- Concentrate is retained in the bowl and flushed periodically.

An iCon is shown in Figure 6.

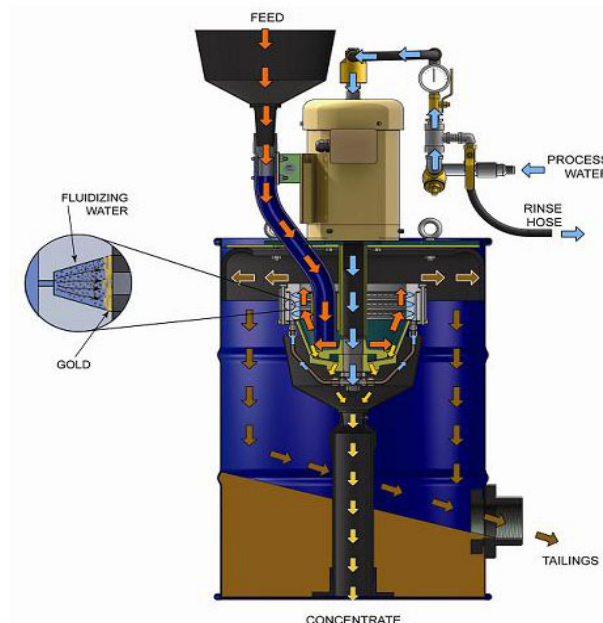


Figure 6 – iCon® scheme (source: Falcon concentrators' website [28])

The concentration flush cycle was tested at several different times. Tests performed previously indicated that longer concentration times yielded richer concentrates but the final recovery was compromised.

A combination of variables, including particle size and shape, specific gravity, and mineral liberation size influences the optimum retention time. Generally, recovery rates decrease when the flush cycle time increases and the grade of gold usually has the opposite behaviour [29, 30, 31].

At GOR, a typical flush cycle time was 20 to 30 minutes, producing around 1kg of concentrate per discharge. Field tests were undertaken with flush cycle times of 15, 20, 30, 45 and 60 minutes.

2.4 Cyanidation in a ball mill

Traditional cyanidation plants make this process in distinct steps. In a first step the ore is crushed and ground in a pulp of 70% solids for gold liberation or exposure to cyanide. Resident times in ball mill varying from 15 minutes to 2 hours are common, and the final product may present grain size (P80) varying from 48meshes (0.419 μ m) to 200 meshes (0.105 μ m) [6]. In a second step the ground pulp is diluted to 30% to 40% of solids and submitted to cyanide leaching. This solution generally circulates through activated carbon columns, where carbon traps the gold in solution.

At the GOR pilot plant the gravity concentrates were treated with intensive cyanidation. Traditionally grinding, cyanidation, and carbon absorption stages take place in series, but at GOR all three stages were combined in a small ball mill with capacity of 100kg/batch. The concentrates were leached with 3g of NaCN per 1kg of ore, in a solution with 2g NaCN/L. Thus 150g of NaCN would be added to 50kg of concentrate and 75L of water. According to Marsden and House [15], the typical gold concentration on activated carbon is around 10,000ppm, which represents 100g of carbon for each gram of gold to be leached. However, historical data of the activated carbon used by GOR has shown 60g of carbon per gram of recovered gold. Using this ratio, 3.3kg of activated carbon is needed to adsorb 55g of gold per batch, and this was the amount used in the tests.

During grinding, a perforated PVC capsule containing a nylon screen bag with activated carbon was introduced into the mill. The capsule has diameter of 12.7cm and 56.0cm long, holding 3.3kg of activated carbon. The nylon bag accommodates the carbon firmly into the PVC capsule, assuring that the carbon does not have excessive attrition with the turbulence of the grinding process. The screen size was 1mm, much smaller than the carbon grain size (15 x 18mm).

After removing and draining the PVC capsule, the carbon moisture content was found to be around 25%. Figure 7 shows a detail of the PVC capsule with the activated carbon in the ball mill.



Figure 7 – PVC capsule containing the nylon screen with the activated carbon

The capsule of activated carbon is introduced in the ball mill at the beginning of the grinding, or after the first 8 hours of grinding and leaching, to avoid stress of the capsule and loss of carbon. The pH was adjusted with lime to stay around 10 to 11. After each batch the capsule of carbon was removed, washed, identified and stocked for subsequent elution and electrolysis.

Several tests were performed to validate gravity pre-concentration followed by cyanidation in the ball mill, as shown in Table 1.

Table 1 – Summary of 6 tests performed

Test	Description	Main objective	Characteristics of sample/test
1	No grinding and conventional leaching of head sample	Intended to be the closest comparison to vat-leaching, although it is it agitated	Head sample, [CN] of 1.00g/L, pH of 10-11, 72h
2	Simultaneous grinding and conventional leaching of concentrate	Test of resistance of the activated carbon in the mill and effects on gold recovery	Concentrate, [CN] of 2.00g/L, pH of 10-11, 72h
3	No grinding and conventional leaching of concentrate	Emulation of agitated tank (ball mill without balls) and cyanidation without grinding	Concentrate, [CN] of 2.00g/L, pH of 10-11, 72h
4	Previous grinding and conventional leaching of concentrate	Emulation of agitated tank (ball mill without balls) and cyanidation after grinding	Concentrate, [CN] of 2.00g/L, pH of 10-11, 72h
5	No grinding and intensive cyanidation of concentrate	Test intensive cyanidation without previous grinding (emulate Gekko system process)	Concentrate, [CN] of 20.00g/L, pH of 10-11, H ₂ O ₂ , 24h
6	Grinding and intensive cyanidation of concentrate	Test intensive cyanidation after grinding extensively for maximum gold liberation	Concentrate, [CN] of 20.00g/L, pH of 10-11, H ₂ O ₂ , 24h

The basic steps in these tests involved:

1 - Sample preparation: The pulp was prepared with 30% of solids. Before cyanidation, sub-samples of the material were analyzed by Fire Assay.

2 - Preparation of the PVC capsule with activated carbon: Coconut shell activated carbon with grain size 6 to 12 mesh was utilized. The carbon was sifted and washed to eliminate fine grains to avoid loss of gold with carbon during the grinding-leaching process. The mesh of the fabric used to accommodate the carbon generally defines how much fine particles have to be eliminated in the washing process. The carbon was later dried in open air until constant humidity, weighted and firmly accommodated in the capsule.

3 - pH control and CN concentration: The natural pH of the process water was 6.0 and the pH of the gravity pre-concentrate slurry was 6.5. The pH of the solution was adjusted to 10.5 with the addition of lime. The [CN] was controlled by titration of the samples with AgNO₃.

4 - *Au in solution*: Samples of the solution were collected every 12h. Previous test confirmed that when the activated carbon is present, gold in solution is promptly adsorbed, and the concentration in solution remains very low.

5 - *Au in the activated carbon*: The gold in the carbon was estimated based on metallurgical balance, considering the original grade of the sample and the final tailings and was confirmed with the balance between gold in the carbon and gold in the tailings.

3 Results and discussion

3.1 Centrifugal pre-concentration

The average gold grade of the feed of 2.13g/t was obtained by fire assay and was relatively stable throughout the test program. The inflow of 1,408kg/h was assumed to be constant for all cases. Tailings grades were obtained by fire assay and concentrate grades were calculated by difference through mass balance of the gold concentration in the feed and centrifuge tailings. The average found for 30 minutes was 205g/t. Gold recovery with time was calculated and presented in Table 2.

Table 2 – Gravity pre-concentration gold recovery with time

Time minute	Feed		Concentrate		Tailing		Recovery
	Weight Kg	Au g	Weight Kg	Au g	Weight Kg	Au G	%
15	352.1	0.75	1.0	0.37	351.1	0.38	49.44
20	469.5	1.00	1.0	0.47	468.4	0.53	46.60
30	704.2	1.50	1.0	0.72	703.2	0.78	47.96
45	1056.3	2.25	1.0	0.97	1055.3	1.28	43.11
60	1408.4	3.00	1.0	1.21	1407.4	1.79	40.42

As shown in Table 2, gold recoveries ranged from 40 to 50%. Recovery decreased significantly after 45 minutes. Historical data shows that the average grade of tailings from “baixao” is around 3.0g/tonne. The mass of the concentrate at the centrifuge discharge was variable, in general varying from 0.8 to 1.5kg of wet material, with a mean of 1.4kg/discharge, which corresponds to 1.0kg of dry material. Assuming for instance discharges every 30 minutes, the daily concentrate

production of one centrifuge, after 40 discharges, can reach around 54kg (wet mass). The gold recovery rates obtained through these tests were found to be statistically significant (P = 0.000157).

The concentration ratio must also be considered when deciding the discharging time. Table 3 shows a summary of the concentrate production for different discharging times in a regular day of operation in the pilot plant (20h/day).

Table 3 – Production of concentrate and concentration ratio

	Concentrate / day	Concentration ratio	Au balance / day (20h/day)
Time	Dry mass	Feed mass / Concentrate mass	Au in the Concentrate
Minute	Kg		g/day
10	122.40	230.13	
15	81.26	345.19	29.66
20	60.94	460.25	27.96
30	40.63	690.38	28.78
45	30.60	920.49	25.86
60	20.40	1380.73	24.25

Although a discharge time of 15 minutes results in better recovery, it generates a larger amount of mass to be leached. Conversely, the longer time of 60 minutes generates richer concentrates but a smaller amount of mass, resulting in lower gold production. The combination that results in best gold production is situated between 20 and 30 minutes.

The operating availability is 20h/day due to frequent interruptions. In a regular day, the pilot plant processed 28 tonnes of tailings. A flush cycle of every 30 minutes resulted in a production of 41kg/day of dry material, resulting in a gold recovery of 50% in 0.14% of the mass.

A flotation plant has been developed to be tested as an alternative method of concentration, but it has not been put in operation yet.

3.2 Cyanidation of concentrate

The result of the tests described in Table 1 are shown in Figure 8

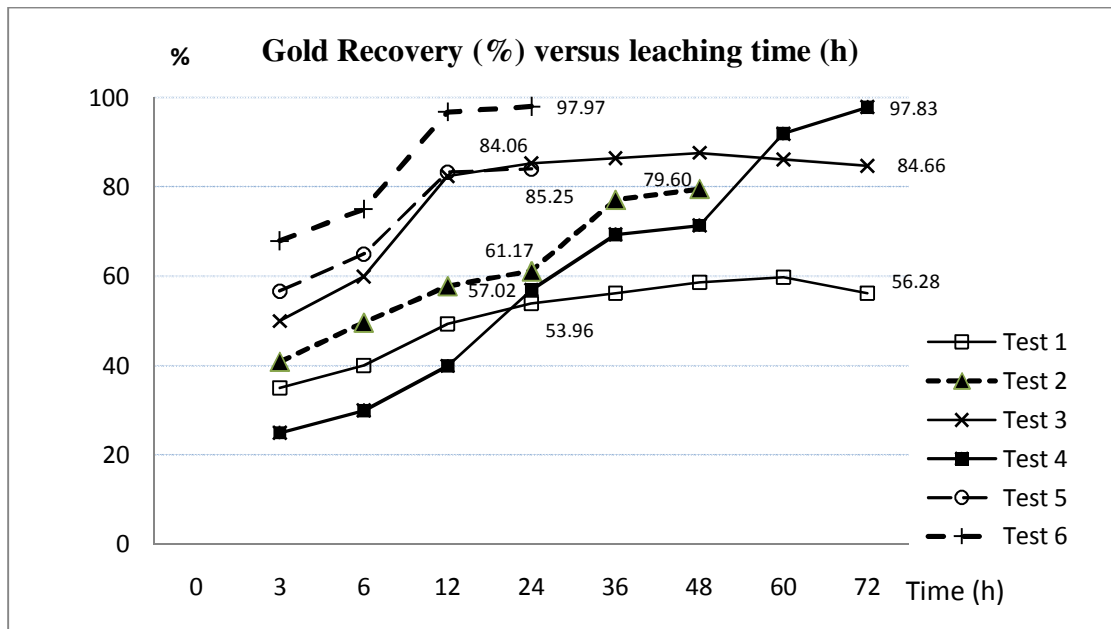


Figure 8 – Gold recoveries – Tests 1 to 6

3.2.1 Test 1 – Conventional cyanidation of head sample

The objective of this test was to observe the effect of conventional cyanidation of a head sample without grinding. In this case the [CN] was 1g/L, as used in the vat-leaching process at GOR. No grinding was performed and no H₂O₂ was used. This process simulates an agitated tank for a period of 72h.

Gold recovery resulted in 56.3%. The carbon adsorbed 91.8% of the gold in solution, as determined by comparing the gold in solution before and after the introduction of the capsule..

3.2.2 Test 2 – Cyanidation of concentrate – the resilience of the capsule of activated carbon

This test aimed to verify the results of simultaneous grinding and leaching of the concentrate in the ball mill with a capsule of activated carbon introduced at the beginning of the process. The initial [CN] was 2.00g/L. The processing time was 72h, to verify the influence of such long grinding time on the activated carbon. The pH was controlled between 10 and 11 with lime.

Although there was no direct contact between the grinding balls and the activated carbon, the carbon was partially ground inside the capsule by friction. This was evidenced by the presence of a black smudge similar to coffee powder left behind after the capsule was removed. This loss was not noticed in the first 24h but after 36h it became evident. Considering the rotation of the mill, of 31RPM, after 48h the PVC capsule had suffered 89,280 internal tumbles, and this stress caused partial disintegration of the carbon. At the end of the process, the net carbon loss was estimated in 11.2%, resulting in an unnecessary loss of gold to the tailing. Subsequent tests did not leave the PVC capsule in the ball mill for longer than 24h. For longer leaching tests, the capsule was added during the last 24 hours of the test. Gold recovery was 79.6%.

3.2.3 Test 3 – Conventional cyanidation of concentrate

The objective of this test was to observe the effect of conventional cyanidation of the concentrate without grinding, simulating an agitated tank. The [CN] concentration was 1g/L, and the leaching time was 72h. This test simulates an agitated tank.

Gold recovery stabilized after the first 12h (82.4%), with peak at 48h (87.7%). The small decrease after 48h (84.7%) may be a case of re-adsorption of gold or simply a problem with the sampling. The carbon capsule adsorbed 93.0% of the gold in solution.

3.2.4 Test 4 – Simulation of agitated tank with previous grinding, leaching of concentrate

Test 4 used conventional [CN] as described in Test 3, however, the concentrate was ground previously for 2h without cyanide. After grinding, the balls were removed, and NaCN and the capsule with carbon were inserted, simulating an agitated tank.

The concentrate was ground to 80% of the material passing through a 0.074mm screen (a P80 of 200 mesh) at a pulp density of 70% of solids by mass. After grinding the pulp was diluted to 30% of solids, the balls were removed, the pH was adjusted to 10.5 and the leaching was initiated.

The initial [CN] was 2.0g/L, using 4.7g of NaCN per kg of gravity concentrate. The preparation of the PVC capsule with activated carbon followed the same procedure explained for Test 3. Samples of solution were collected every 12h. The [CN] dropped drastically in the first 12h, requiring addition of NaCN and lime to adjust the process. The effective consumption of CN was estimated as 3.1g/L.

Gold concentrations in solution were as low as 0.03mg/L, indicating rapid adsorption on the carbon. Gold recovery reached 97.8% after 72h. As there were no balls in the mill, no significant carbon loss was observed. After 72h the mill had rotated 133,920 times (31RPM) and the capsule remained positioned at the bottom part of the mill.

The leaching time was longer than expected. The gold particle size is one of the most important factors affecting leaching time. Study by Veiga et al. [17] working with ore from ASGM in Indonesia has shown that gold recovery after 2h of grinding and 8h of leaching reached 93.0%, and artisanal miners in Ecuador using agitated tanks took 31h to obtain the same results. This provides an indication on how grinding can accelerate the leaching process. In Test 4 an equivalent recovery was obtained only after 60h. As both pH and [CN] were controlled along the process, the most probable cause of such higher time is the presence of coarse gold in this particular sample. An alternative to handle this problem is by removing the coarse gold of the concentrate prior to cyanidation.

3.2.5 Test 5 – Intensive cyanidation without grinding

The objective of this test was to observe the effect of the intensive cyanidation on the gravity concentrate without grinding. The concentrate was submitted to leaching with cyanide concentration of 20g/L NaCN, using H₂O₂ and processing for 24h. A solution of H₂O₂ 0.5g/L was added to the pulp at the beginning of the process. The balls were removed from the mill, and a clean solution was collected every 12h. The capsule of activated carbon was introduced only after the first 24h, and stayed in solution for 3 hours. This whole cycle took 27h, with 24h for leaching and 3h for carbon adsorption.

Gold recovery based on gold in solution was 84.1% after 24h and based on difference of grades before and after cyanidation was 85.8%. The efficiency of the carbon adsorption was 95.7%, comparing the gold in solution before and after the introduction of the capsule with carbon.

3.2.6 Test 6 – Intensive cyanidation after grinding

The objective of this test was to observe the effect of the intensive cyanidation on the concentrate after grinding aiming to liberate the gold particles. The concentrate was previously ground for 2h (P80 < 200mesh) with high concentration of cyanide before the capsule of activated carbon was introduced. All other conditions were kept as in previous tests.

Gold recovery based on gold in solution was 98.0% after 24h and based on difference of grades before and after cyanidation was 97.0%. The efficiency of the carbon absorption was 97.4%, comparing the gold in solution before and after the introduction of the capsule with carbon.

The difference in gold recovery between 12h (96.8%) and 24h (98.0%) was not significant. This test confirms the importance of grinding for gold liberation and exposure of gold particles to cyanide. As both tests 5 and 6 were performed in the presence of H₂O₂, the difference in gold recovery can be attributed to previous grinding. Table 4 summarizes the results of Tests 1 to 6.

Table 4 – Summary of gold recoveries of Tests 1 to 6

Test	Description	Au Rec. (%) in 24h	Au Rec. (%) Total time (h)	Main finding
1	No grinding and conventional leaching of head sample	53.96	56.28 (72h)	Low recovery after long leaching time
2	Simultaneous grinding and conventional leaching concentrate	61.17	79.60 (48h)	Capsule of carbon leaks if left more than 24h, losing Au to the tailings
3	No grinding and conventional leaching of concentrate	85.25	84.66 (24h)	Reasonable gold recovery, low leaching time (stable after 12h)
4	Previous grinding and conventional leaching concentrate	57.02	97.83 (72h)	High recovery, but long leaching time
5	No grinding and intensive cyanidation of concentrate	84.06	84.06 (24h)	Reasonable gold recovery, low leaching time (stable after 12h)
6	Grinding and intensive cyanidation of concentrate	97.97	97.97 (24h)	High recovery, low leaching time (stable after 12h)

These results show that process 6 is promising for the GOR site. The intensive cyanidation results in high recovery in 12h (96.8%). The capsule of carbon should not be introduced at the beginning of the process to avoid carbon destruction by friction.

At Garimpo Ouro Roxo, mercury in the tailings is not uniformly distributed as miners used a mix of traditional concentration with carpeted sluice boxes and whole ore amalgamation using copper plates, but the average concentration was found to be 0.1g of Hg/t. The pilot plant was preliminary developed to recover residual gold from tailings, and the residual mercury in the tailings has always been a concern, as it may react with cyanide. The gravity tests have shown that only 17.1% of mercury was retained with the gravity concentrate in the centrifuge. This low

concentration may have different reasons, but it is most likely that mercury is associated with very fine suspended particles and it is carried away with the centrifuge tailings.

The analyses of the tailings of the ball mill after cyanidation of concentrates have shown that 62.5% of mercury in the concentrate (10.7% of the total mercury entering the centrifuge) was adsorbed into the charcoal and the 37.5% residual mercury (6.4% of the total mercury entering the centrifuge) was disposed with the final tailings. In the event of mercury cyanide is formed in the leaching process, the amount of mercury exposed to cyanide was reduced more than 5 times after gravity concentration, comparatively to the current vat-leaching method.

4 Conclusion

The results have shown that the intensive cyanidation ($\text{NaCN } 20 \text{ g/L} + \text{H}_2\text{O}_2$) in a ball mill reduces the leaching time considerably. While the current vat-leaching process at GOR takes over 20 days per cycle, the new process requires less than 24h. Intensive cyanidation has proven to be effective resulting in gold recoveries from concentrates of up to 98%. However, gravity concentration recovered only around half of the gold, indicating room for improvement, either by using centrifuges in series or by implementing a separate operation, such as flotation. Although the final gold recovery (97.97% of 50.0%) is equivalent to the existing gold recovery of vat-leaching (50.0%), the main advantages of the new process are the significant reduction of time, operating cost, chemicals and environmental impacts.

With a concentration ratio of 1/700, the cyanide consumption is reduced around 22 times, and the amount of mercury-contaminated tailing in contact with cyanide is reduced up to 700 times. Whereas vat-leaching uses 1.0 kg NaCN per tonne of material processed, the pilot plant of concentration followed by cyanidation uses 47.0 g of NaCN per tonne of material. Assuming the full replacement of the current process (vat-leaching) by the proposed concentration and cyanidation method, the annual consumption of 22,000 kg NaCN could drop to 980 kg. Furthermore, as mercury concentration is low (17%), the amount of mercury exposed to cyanide was reduced more than 5 times after gravity concentration, comparatively to the current vat-leaching method. The economical and environmental components of such cyanide reduction are of great incentive for the implementation of the new process. More detailed economical studies currently in progress have shown that diesel consumption and labour are also reduced with the

new process. The proposed method can be adapted for different artisanal mining sites according to mineral characteristics, location, and investment capacity of the owners. The pilot plant is transportable, which allows its reinstallation in new sites where tailings may be available for reprocessing.

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