MYTHS AND REALITIES IN ARTISANAL GOLD MINING MERCURY CONTAMINATION

by

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ABSTRACT

The world faces a major gold rush currently. Worldwide 15 million people work directly in artisanal gold mining in more than 55 countries. The usual method of extraction is amalgamation and cyanidation, or very commonly a combination of both. Significant amounts of contaminants are released to rivers and soil, generating environmental and health concerns. The majority of artisanal miners are involved in micro-mining, but it is believed that the majority of contamination comes from small to large artisanal operations. Larger operations involve less people, but process much more material, releasing much more contaminants as well.

The present work explains how contamination is generated, through a comprehensive analysis of the labor division in small-scale mining sites around the world, and also analyses aspects of different intervention approaches.

Different approaches are taken by different groups working on the problem around the world. This work evaluates the most common kinds of intervention, with special attention to technical and legal aspects in trying to eliminate mercury use in gold extraction. The study is conducted by comparing different interventions strategies to evaluate the myths and realities from an engineering perspective. Special attention is paid to the evaluation of alternatives to amalgamation process for gold such as intensive cyanidation, borax direct smelting and chlorination. Accordingly to the location in which the technology is introduced, different methods can be applied, but only cyanide remains a proven method to deal with complex ores.

Important aspects such as education, training, financial aid and understanding of a community before intervention are also shown to be essential for success.

The association of Small-scale Gold Mining and potential gold concentrate consumers is pointed to as a possible solution, applicable where concentrates can be shipped to a smelter. Association with large companies is cited as an alternative means to reduce contaminants generated by gold extraction as well.

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LIST OF ACRONYMS

AGM – Artisanal Gold Mining
ASGM – Artisanal and Small-scale Miners (or Mining)
CERM3 – Centre for Environmental Research in Minerals, Metals and Materials
COMTRADE – United Nations Commodity Trade Statistics Database
GEF – Global Environment Facility
GMP – Global Mercury Project
GRG – Gravity Recoverable Gold
ITCAM – International Training Center for Artisanal Miners
ITDG – Intermediate Technology Development Group
LVG – Lake Victoria Goldfields
MIBC – Methyl Isobutyl Carbinol
MM – Micro Miner (or Mining)
NaCN – Sodium Cyanide
NGO – Non Governmental Organization
PPM – Parts Per Million
PVC – Polyvinyl Chloride
SGM – Small-scale Gold Mining
SRB – Sulfate Reducing Bacteria
TPD – Tonnes Per Day
TPH – Tonnes Per Hour
UK – United Kingdom
UN – United Nations
UNECA – United Nations Economic Commission for Africa
UNEP – United Nations Environmental Programme
USD – United States Dollar
WHO – World Health Organization
XRD – X-Ray Diffraction

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1. INTRODUCTION

Eliminating mercury use in artisanal mining is a challenge. The metal is used to amalgamate gold and silver from concentrates or with the whole ore. Amalgamation is very simple and easy. When compared to cyanide leaching, amalgamation is faster, easier and more direct; it can be perfectly conducted by a person with no technical knowledge.

On the other hand, although being easy to use, the contamination generated by mercury to the worker and to the environment, is notably high. Therefore, mercury needs to be abolished from artisanal mining in order to avoid the contamination generated.

Different approaches when intervening in mining communities are taken by different groups, governments, and organizations around the world. While some believe legislation is the key, others believe that introducing new technologies are the best solution.

Definitions of size and technology of the main mercury polluters are proposed in this work as a means to differentiate where funds should be invested in order to maximize their impact on mercury release reduction.

The objective of this work is to identify the myths and realities behind each type of approach. This is accomplished by identifying the main sources of pollution within the artisanal mining sector and how labour divisions contribute to the level of contamination caused by an operation.

Interventions should aim at reducing contamination while providing the miners with increased profit. As artisanal mining is a business, usually a subsistence business, there will be little reception for a substitute method that

does not yield more or equal profit to previous methods. This should be the first thing to consider when preparing an intervention. It is unusual to observe successful interventions that paid no attention to this important fact.

When applicable, concentration in conventional mining is a standard process. However, it is not always observed in artisanal mining. Introducing this process, where possible, is highlighted as a means to reduce the emission of contaminants while increasing gold recovery to the miners.

A description and analysis of several alternative gold extraction methods is also conducted. Their suitability to each kind of artisanal mining operation is analysed according to the concentration method (and concentration ratio) needed.

A flowsheet investigation of an ore sample from Costa Rica is conducted as an example to show that technical assistance is needed for an artisanal mining site to evolve. A simple characterization study can show to the mine owner ways to increase recovery by as much as 66%, while eliminating mercury use.

The primary conclusion of this work will be that presence of technical interveners and education of miners are fundamental steps in successful introduction of a new technology. Moreover, introduction of a new technology is not a panacea. Rather, helping the miners form strong cooperatives such that the innovations are affordable and assist the miners in avoiding their exploitation by the richest (and smartest) people is of extreme importance.

2. MERCURY IN ARTISANAL MINING

In virtually all developing countries, approximately 30 million individuals extract more than 30 different minerals using rudimentary techniques. Gold is the preferred product since the metal price has increased by 700% in the last ten years. An estimated 10 to 15 million people are directly involved in this activity, resulting in high levels of river siltation, mercury pollution and other environmental problems (Velasquez et al, 2010).

More than 55 developing countries located in South and Central Americas, Asia and Africa have ASGM[1] (Artisanal and Small Scale Gold Mining) as an important economic activity (Spiegel and Veiga, 2005) usually conducted by those living in poverty and providing means of living for people who drove themselves away from their regular activities such as cattle, fishing or farming mainly because of the low income they are able to obtain from such jobs or because the instability inherent to these activities (Eisler, 2004, Li et al., 2009, Spiegel and Veiga, 2010). Worldwide, around 15 million people work directly with ASGM (Cordy et al., 2011, Spiegel and Veiga, 2010, Velásquez-López et al., 2010).

Mercury has been used for many centuries as a means for extracting gold by amalgamation. Although mercury has many other applications in the modern world such as dental amalgam, thermometers, Chinese medicine, chlorine and caustic soda production, fluorescent lamp, as thiomersal used as a vaccine preservative and in skin care products, the main pathway for environ-

[1] Artisanal and Small Scale Gold Mining – An in depth discussion on differentiation between Artisanal and Small Scale Mining is found on page 35. Where differentiation is not needed, this work uses the term ASGM.

mental contamination is the gold processing application. It is important to note that mercury has been replaced by alternative methods or products in the developed world. However, it remains intensively used in the developing countries (McKelvey and Oken, 2012).

It was estimated in 1998 that the annual ASGM gold production constituted 20 to 30% of the global production, ranging from 500 to 800 tonnes (Loh, 2002). A recent study (Seccatore, 2012) estimated that ASGM produced between 337 and 447 tonnes of gold annually, (average = 392 t) or 12.2% of the total gold produced in 2012 (2827.7 from organized mines (Reuters, 2013) + 392 tonnes from ASGM = 3219.7 tonnes).

In ASGM, the ratio between $Mercury_{lost}$:$Au_{produced}$ observed around the world varies greatly. Veiga et al., (2004b) and Veiga et al., (2006), witnessed values ranging from 1 to 90, so it is difficult to be assertive about an uniform global number (Veiga and Baker, 2004). However, the estimated amount of mercury released from ASGM obtained by UNEP, (2013) and from Swain et al., (2007) was 1400 tonnes/a and 1000 tonnes/a respectively, indicating a global average of Hg_{lost}:$Au_{produced}$ around 3.5:1, taking into consideration the gold production by ASGM suggested by Seccatore, (2012).

The European Union stopped exporting mercury in 2011 and a treaty signed with UNEP (United Nations Environmental Programme) in 2008 established that the United States must ban its commercialization in 2013. On the other hand, cinnabar (mercuric sulfide), a mineral that has been the main source of mercury as a commodity for thousands of years, is still mined in China, a country producing 1300-1400 tonnes of the metal annually. China remains the largest source of elemental mercury and its production is in-

creasing in this country to meet internal demand for batteries, chlor-alkali plants, vinyl chloride production, etc. (Swain et al., 2007). In 2008, Peters, (2011) indicated that Afghanistan, had 33 mercury ore deposits. The incentive for Afghanistan to become a mercury producer will only increase as the price of metallic mercury rises. In fact, the international price of mercury has increased sharply from USD 4.93/kg in 2003, USD 55.2/kg in 2011 to USD 95.7/kg in 2013 (RecycleInMe, 2013; USGS, 2009)

Artisanal and small-scale mining contribute to environment contamination by the emission of metallic mercury vapour to the atmosphere and by the release of metallic mercury to water streams. Swain et al., (2007) estimated that in 2005, some 1000 tonnes of mercury were discharged per annum into the environment by ASGM alone; this amount is equal to one third of all anthropogenic mercury emissions. Based on this estimate, Siegel and Veiga, (2010), based on field observations, estimate that 30% of these emissions are volatilized directly into the atmosphere and the remaining 700 tonnes are discharged with tailings, contaminating soil, river sediments and water streams. The latter type of contamination often spreads throughout the watershed and can reach downstream communities and wildlife. Out of the 1960 tonnes of mercury emitted into the atmosphere annually by human activities assessed in 2013, 35% or 727 tonnes are attributed to the evaporation of mercury from artisanal miner's amalgams. This is twice the value of obtained in 2005 (UNEP, 2013). Recently Koekkoek (2013) estimated the amount of mercury released by ASGM in 70 countries at 1607.8 tonnes/a. This represents 93% of the countries using mercury in ASGM and it was obtained based on assumption made by Telmer and Veiga, (2009), using data from UN COMTRADE. This database uses the total reported imports of

mercury by country, excluding legitimate industrial uses such as batteries, chlor-alkali, dental, lamps, etc. This method assumes that the unreported mercury is in fact that used by ASGM; however, there are some cases in which mercury is legally imported to be used for dental applications, like in Brazil, but is diverted for use by ASGM (Veiga et al. 2006) The map from the NGO MercuryWatch shown in Figure 2-1 shows the amount of mercury released to the environment by ASGM operations in 2010.

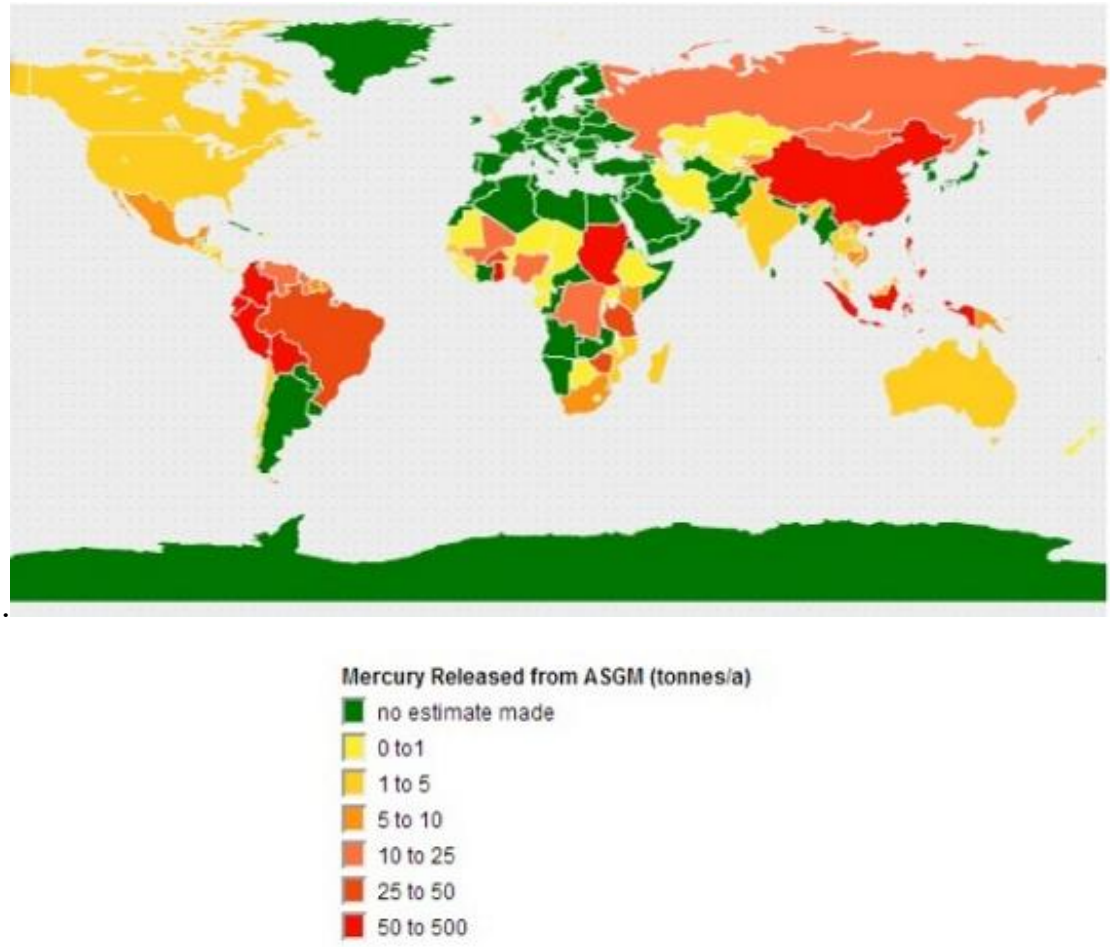

FIGURE 2-1. MERCURY RELEASED BY ASGM (2010 - MERCURYWATCH.COM)

2.2. HOW MERCURY IS USED IN ASGM

Gold amalgamation is one of the oldest gold extraction methods; mercury added to gold and silver ores creates an amalgam by contact with such metals. An amalgam is an extremely malleable metallic state that is not an alloy and the dissolved metals can be separated from the mercury by evaporating the amalgam.

Usually in ASGM operations that amalgamate only concentrates, mercury is added to a bowl, pan or a small barrel together with the gold concentrate where the amalgamation process takes place by agitation. After this step the amalgam is squeezed in a piece of fabric to eliminate the excess liquid mercury not bonded to gold. The amalgam has between 40 and 50% mercury and the main process to separate mercury from gold is by decomposing the amalgam at a temperature above 460 °C, at which mercury compounds evaporate. This process produces a gold *doré* that can have residual mercury at levels between 0.2 and 5% depending on the evaporation process, (Veiga and Hinton, 2002, Veiga et al., 2004a). In rare cases, like those observed in some African countries, when amalgams are burned in low-temperature bonfires, the *doré* contains up to 20% of mercury, (Veiga et al., 2006a)

Evaporation is usually conducted in open-air on a bonfire or using a propane or gasoline torch. As stated in Section 2.3 the vapours generated are highly toxic and simple solutions such as retorts (shown in Section 6.2) are technological approaches to reduce air emissions.

Whole ore amalgamation is the oldest technique applied to gold processing. Even for large-scale mining it was the most popular method until the end of the nineteenth century when cyanide leaching was introduced (Eissler, 1888).

The flowsheet for this process is simple and, when applied by ASGM, is usually employed in the following ways:

- Grinding of ore in a stamp or hammer mill that discharges the ground material into copper plates covered with mercury. The gold contained in the ground material is amalgamated when in contact with the plates. Plates are then cleaned by scraping the gold/silver/mercury amalgam that is later burned in order to separate the valuable metals from mercury.
- Grinding of ore in small ball mills for a few hours where mercury is added and amalgam is formed to be burned later using the previous process.

These processes are the main ones for whole ore amalgamation still in use today; the first is popular in Brazil, China, Ghana, Venezuela, Zimbabwe, etc. while the second is used in Bolivia, Colombia, Ecuador, Indonesia, Peru, etc. (Veiga et al., 2013).

There are two main drawbacks to amalgamating the whole ore. The first one is the low recovery such methods achieve as there is no control of the degree of liberation of gold particles. Sometimes the ore is ground too fine while other times it is not ground enough. When grinding is excessive, particles become too small and the contact between them and mercury is more difficult, because finer particles of gold tend to float on the mercury not amalgamating. When grinding is too coarse, unliberated gold is not accessible to mercury to be amalgamated. This problem is not observed in those ASGM using cyanide, since overgrinding is actually beneficial for this technique, as fine grinding gives more opportunities to fine gold particles to be exposed.

The other – and most important – drawback in amalgamating the whole ore is the contamination generated. The mercury in contact with all the material being processed ends up being dragged to the tailings that will eventually end up in the environment or being leached by cyanide creating a major environmental concern as described in Section 2.3. Mercury, when introduced in ball mills, loses coalescence and becomes fine drops that are discharged with the tailings, contributing also to loss of gold.

In some places these rudimentary extraction methods are employed because of the miner's lack of knowledge or access to more modern techniques. The amalgamation of the whole ore is a consequence of ignorance and/or tradition related to the way of working. If one has access to cyanide and the knowledge to employ this technique, there is no rational justification for using an amalgamation step ahead of leaching. However, this type of flowsheet is used to remove relatively coarse gold from the lixiviation circuit feed. In the past there was no awareness about mercury cyanide contamination but nowadays, in ASGM, the amalgamation process is operated by the miners with the cyanide process operated by the processing center owner. The center rents the gravity equipment to the miner who pays the rental fee by leaving the tailings to be reprocessed by the owners.

Although this scenario is changing mainly because of tighter cyanide regulations, most large-scale mining companies in the world, while not using mercury at all, still apply a leaching process to the whole ore. This involves large pieces of equipment units and agitators as well as high costs due to cyanide consumption and the need for cyanide destruction. Depending on the financial analysis and the operation size, it can be even more profitable to concen-

trate and leach an ore, than only leach the whole ore. This rationale can also be applied to ASGM.

2.2.1. GOLD ORE PROCESSING CENTERS AND MERCURY

After four decades of the intensification of artisanal and small-scale gold mining, the environmental, social, and health situation of ASGM workers has barely improved. Efforts from various international groups and governments in order to organize and educate the workers have achieved little result. One example is the idea of communal processing centers, which began to be implemented a decade ago in Africa. This ended up causing more social and pollution problems than it resolved.

In Africa, the Shamva Center in Zimbabwe, built in 1989, was supposed to process ores from 40 gold mines in the region. This project was a collaboration of the National Miners Association of Zimbabwe with the UK Non-Government Organization ITDG (Intermediate Technology Development Group, nowadays known as Practical Action). The main environmental benefits were to move tailings away from rivers and discourage miners from using mercury. The center was active for many years but the main hurdles were: 1) the distance of the center from the mines and 2) the lack of an efficient process to extract fine gold (Simpson, 2012).

The original idea of the Shamva Center was applauded worldwide. The merit of the idea was clear; the processing centers provide, for "free" or for a symbolic fee, the whole service to generate, in short time, a bar of gold in the miners' hands avoiding the discharge of mercury into rivers. As a payment, miners leave their tailings in the centers' facilities. The "catch" in this system is the fact that the centers never actually used clean techniques. They

barely grind the ore enough to liberate fine gold particles before processing, which drops substantially the possibility of extracting gold efficiently. They also use rudimentary methods to concentrate the gold associated with amalgamation to trap the gold particles eventually liberated in a gravity concentrates or in the whole ore. With this inefficient processing technique, the gold recovery is usually below 30% (Cordy et al., 2011). The owners of these centers use then techniques such as cyanide leaching to extract the residual gold from the tailings left behind after amalgamation. The process is unfair to the miners and extremely polluting since the material being re-processed is contaminated with mercury, which is then complexed by cyanide. If miners want to take the tailings with them, the processing centers charge for the value of their work. In Portovelo, Ecuador, this costs USD 20 per 50-kg bag of tailings (Cordy et al., 2011), which can be more than the value of the contained gold. Veiga and Beinhoff, (1997) investigated the UNECA processing centers in Venezuela, an example that also turned out to be a bad practice in the long run. Nowadays these problems persist. As the simple and inefficient amalgamation techniques usually extract less than 30% of the gold from the ore, some individuals are learning to copy and adapt processing technologies from large industrial mining companies. There are problems in this technology transition that goes beyond simple demonstration of new techniques and this has led to large environmental and health problems. This deals with the fact that the "smart individuals" with more knowledge and resources are taking advantage of the less educated miners. In other words, information and capital about more efficient techniques does not flow democratically among the artisanal mining sector. This is the case in the propagation of processing centers.

In Zimbabwe in 2004, it was estimated that there were between 300,000 and 400,000 artisanal gold miners sustaining the livelihood of at least 2 million people (Veiga, 2004b). There are about 1000 small-scale gold mining sites and almost 250 gold processing centers in the country in which 117 are registered in the Kadoma-Chakari region (Metcalf, 2008). Miners take low gold grade material (3 to 15 g Au/tonne) to the custom processing centers to be ground in stamp mills (1 tonne/h) to below 1 mm. Concentration is done in a locally manufactured Knudsen-type[2] centrifuge or amalgamation the whole ore with copper-amalgamating plates is used (Figure 2-2). The copper plates of 2 to 3 m^2 are covered with a layer of mercury. Miners typically pay USD 3 to 5 per hour of grinding and concentration, depending on the hardness of the ore. Using stamp mills, hard rocks take 5 hours/tonne to be ground and concentrated, whereas soft ores take 1.8 hours/tonne (Veiga, 2004b). This definitely does not cover the operating costs. When low speed and inefficient centrifuges are used, the concentrate is constantly discharged and manually amalgamated. As miners do not conduct assays to evaluate their flowsheet, they just estimate the adequate concentrate discharge time. They also add 50 to 150 g of mercury into the centrifuges hoping to trap more gold. This disperses mercury droplets to the tailings.

In the case of copper-amalgamating plates, due to the attrition of the material with the mercury layer, about 20 g of mercury is lost per tonne of material processed (Boese-O'Reilly et al., 2004). Nowadays, mercury losses in Zimbabwe are above 25 tonnes/a and it is estimated that around 70% of this

[2] Knudsen is a centrifugal concentrator that uses no pressure water to fluidise the concentrate bed. It achieves a 10 G centrifugal force, while more developed models achieve 150 G.

mercury is lost to tailings that eventually undergo cyanidation in vats with 20 to 70 tonnes of material (Figure 2-3) (Metcalf and Spiegel, 2008).

FIGURE 2-2. PROCESSING CENTER IN ZIMBABWE WITH STAMP MILLS AND A) KNUDSEN-TYPE CENTRIFUGE B) COPPER-AMALGAMATING PLATE (PHOTO: M. VEIGA)

FIGURE 2-3. PERCOLATION TANKS IN KADOMA, ZIMBABWE, LEACHING MERCURY-CONTAMINATED TAILINGS WITH CYANIDE (PHOTO: M. VEIGA)

In Tanzania, artisanal and small-scale mining (ASM) activities are spread all over the country but those miners in the LVG (Lake Victoria Goldfields) region produce more than 95% of all the gold in the country. The current number of artisanal gold miners in the country is estimated to be between 500,000 and 1,500,000 (UNEP, 2012). Clearly, no accurate estimates are available. In the Geita District, with 7,825 km², 185 villages, and a population around 712,000), there are around 150,000 artisanal gold miners (UNEP, 2012). A typical extraction production by a team of 4-5 miners is 800 kg of ore/week of hard rock and/or 8000 kg/week of weathered (soft) ore. The material is transported in 40-kg bags to the processing centers where operators are hired to process the ore (Veiga, 2004a). The material is dry-ground in Ø60 x 90 cm ball mills. As the operators are paid by bags of ground material, it is easier to fill bags when the ore is dry. The ground ore is carried to the sluice boxes, lined with sisal clothes and set up at the riverbanks. The dry ore is discharged into to a feed box and water is carefully added using buckets. Many miners all over Geita District have been reprocessing old tailings using this system. After processing a customer's bags, the sisal clothes are washed in metallic trays and mercury is added for manual amalgamation in cement tanks. Tailings from the sluice boxes and from amalgamation are sold to people with cyanidation plants (vat leaching, mainly). This system is widespread throughout other regions with processing centres.

In Piura, Peru, mercury is introduced in *chanchas* (small ball mills) or in *quimbaletes* (large pieces of stone rocked back and forth, by one or more people to grind the ore with mercury) to amalgamate the whole ore without previous gravity concentration. Mercury losses range from 30 to 40% depending on

the type of ore and the type of whole ore amalgamation process used. *Chanchas* lose more mercury than *quimbaletes* because of the mercury "flouring" (loss of coalescence).

The processing centers in the North of Colombia, Antioquia are similar to those in Southern Ecuador. They also use *chanchas* (locally known as *cocos*) to amalgamate the whole ore without previous concentration. About 60 to 70 kg of ore is added to each *coco* together with about 100 to 120g of mercury. This procedure can cause a mercury loss of up to 82% (average of 15 centers was 46.3%) of the mercury introduced in the *cocos*. Tailings with up to 5000 mg/kg of mercury were observed in Colombia. All processing is conducted in the urban areas and the level of mercury in air is one of the highest in the world (Cordy et al., 2011). Mercury-contaminated tailings are leached in percolation vats for up to 30 days.

The usual method for gold processing in more than 320 processing centers of Antioquia is the amalgamation of the whole ore in small ball mills (Figure 2-4). In this poor practice, an average of 46.3% of the mercury entering the process is lost to the tailings. Tailings, left as payment, are subsequently leached producing mercury cyanide, probably a pollutant bioaccumulated by aquatic biota. Around 3.8% of the initial mercury added to the ball mill is lost to the air during the amalgam burning step (Cordy et al., 2011). Amalgamation-aid reagents such as molasses and lime juice are added in this case; these are believed to increase amalgamation efficiency.

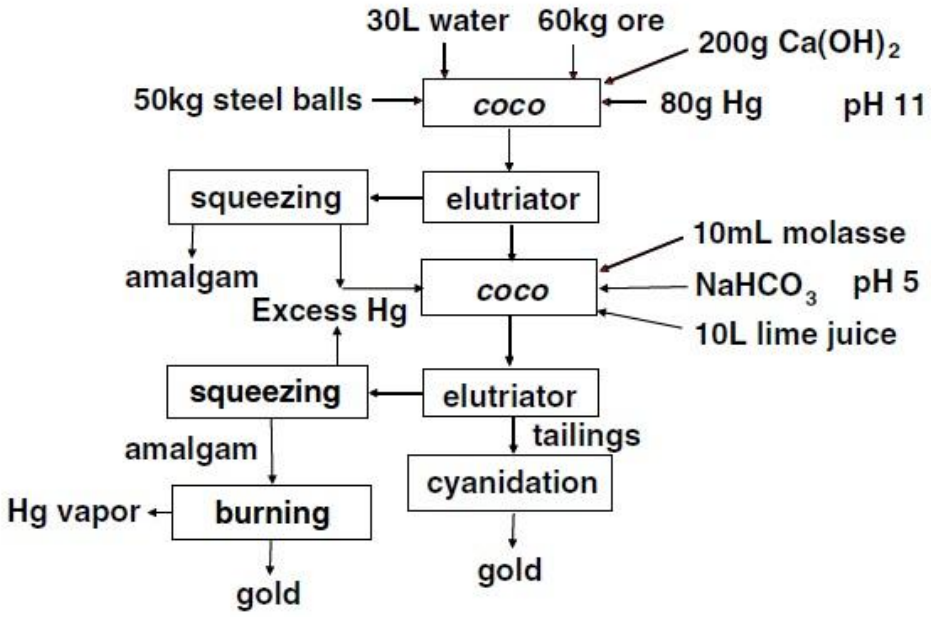

FIGURE 2-4. FLOW SHEET OF A TYPICAL WHOLE ORE AMALGAMATION IN ANTIOQUIA CORDY ET AL., 2011)

The inefficient process of whole ore amalgamation in ball mills was also observed in North Sulawesi, Indonesia (Castilhos et al., 2006). About 10,000 miners were located in a site called Talawaan near the Manado international airport. Ayhuan et al, 2003, studied the work in this locality. Miners extract from 0.5 to 1 tonne/day of partially weathered ore from shafts 17 m deep. The material is crushed to –1/2" by stamp mills to be delivered to the processing centers. The processing centers charge a nominal fee to the miners to process the ore in small ball mills named *tromols* (same as a *chancha*). Each *tromol* receives 40 kg of ore, 1 kg of mercury and 40 river cobbles as the grinding media. The grinding step lasts 5 hours, the ground product is discharged in plastic bowls, and the amalgam is separated by panning to be burned with a torch in the open air. A processing plant with 12 mills recovers 4 to 6 grams of gold per cycle (0.3 to 0.5 g Au/mill). Eventually there are two cycles per day. The mills operate 8 hours/day, 6 days a week. About

30% of the mercury is "floured" and lost with tailings. The total mercury loss at this site may be as high as 90 tonnes/y and the gold production can be possibly as high as 10 tonnes/a (Ayhuan et al., 2003).

In Portovelo, Southern Ecuador, miners bring their ores to one of the 87 processing centers in the region. The system is the same as described above. Miners take their ores to be processed for free or for a small fee, but they must leave all tailings in the centers. The material left behind is stored for eventual leaching with cyanide by the owner, who then recovers about 90% of the gold remaining in the tailings. The larger centers process 40 to 90 tonnes of low-to-medium grade (3-6 g Au/t) material per batch in Chilean mills that consist of two or three heavy cement wheels with steel rims connected to a 20 HP electric motor. The wheels rotate over a 25 cm wide, 2-inch thick steel plate to crush and grind the material below 0.2 mm (Figure 2-5). The ground material is then concentrated in sluice boxes with wool carpets. The produced concentrate, around 200 kg, is either further panned to reduce the mass to 15 to 20 kg before being amalgamated or directly leached with cyanide. In the first case, amalgamation of small amount of panned concentrate represents mercury losses (to tailings) of only 1.4% of the mercury introduced in the process. However, it is not known how much gold is lost when operators pan the sluice box concentrates to reduce to 10% of its original volume. It was estimated that the amount of gold recovered fluctuates between 40 and 50% (Velásquez-López et al., 2010) but there is evidence that the losses may be higher. It was observed in 2012 that some miners are no longer panning the sluice box concentrates, but rather, amalgamating the whole concentrate in *chanchas*.

The sluice concentrate weighs 200 – 300 kg, representing 3 to 4% of the mass of the original ore. Cyanidation of gravity concentrates is a mercury-free option also observed in Portovelo since 2011. Miners can rent a 14 m^3 cyanidation tank for USD 200 per batch and leach the concentrates, with the gold precipitated from solution with zinc which is then burned in open-gas furnaces emitting zinc into the atmosphere. This procedure, in spite of polluting the urban air with zinc, has reduced the mercury being released by almost 560 kg/a of (Velásquez-López et al., 2010). But use of mercury is not completely eliminated. It is still used in *chanchas* to amalgamate the relatively coarse gold retained inside the Chilean mills or to process high-grade ores or gravity concentrates. All mercury-contaminated tailings from *chanchas* are sold to processing centers to be leached with cyanide.

FIGURE 2-5. CHILEAN MILL WITH SLUICE BOXES COVERED WITH WOOL CARPETS IN ECUADOR

Mercury is the main contaminant released by artisanal miners. It is more persistent than cyanide in the environment and bioaccumulation of mercuric compounds is possible. Mercury contamination effects are further discussed in the next section.

2.3. MERCURY RELEASES AND EMISSIONS

According to UNEP, (2013), mercury release is defined by the metal being discharged into water or land while emission is when the element is being radiated to the atmosphere by evaporation.

Mercury has historically been used for gold amalgamation. While large companies completely replace this method with cyanidation (Eissler, 1888), there are still old mercury-contaminated tailings all over the world. Unfortunately, AGM still relies heavily on mercury amalgamation use because of its simplicity and rapid gold in the miner's hand (Metcalf and Veiga, 2012).

Mercury is a reactive metal that combines with other elements, organic and inorganic molecules to form soluble compounds that can increase bioavailability. All forms of mercury may affect human health depending on the species and the exposure pathway. Different mercury compounds have different impacts on aquatic and terrestrial life.

2.3.1. METALLIC MERCURY

As mercury vapour pressure at ambient temperatures is relatively high, this metal volatilizes easily and contaminates the atmosphere to be dispersed around the planet. In fact the atmospheric concentration of mercury is today three times higher than it was in pre-industrial times (Lindberg et al., 2007, Nriagu and Becker, 2003, Pyle and Mather, 2003, Swain et al., 2007).

Although AGM plays an important role in atmospheric contamination there are other activities emitting mercury to the atmosphere as for example: combustion of coal and fossil fuels – which may contain traces of mercury – (Carpi, 1997, Hassett et al., 2004, Joensuu, 1971), chlorine-alkali production (Southworth et al., 2004) and refining of some metals (Streets et al., 2005, Swain et al., 2007). Volcanic eruptions have also been stressed as an important vector of atmospheric mercury pollution with emissions of approximately 110 tonnes/a of mercury (Krabbenhoft and Schuster, 2002, Nriagu and Becker, 2003, Pyle and Mather, 2003).

Krabbenhoft and Schuster (2002) claim that most of the atmospheric pollution by mercury is caused by anthropogenic activities, such as mining and other industrial activities. Using ice cores in polar caps – that trap deposited atmospheric mercury through the years – this work provides a graph, with major peaks of mercury emissions: in the 1860s and 1870s related to the world gold rushes, Tambora (1815), Krakatau (1883), and Mt. St. Helens (1980) volcanic activity (Figure 2-6) (courtesy of the U.S. Geological Survey). The green is background values for atmospheric mercury today.

Air contamination by mercury at sites where miners amalgamate the whole gold ore, but not necessarily burning it, can reach values as high as 1 million ng/m³ (Cordy et al., 2011) whereas 100 ng/m³ is the proposed recommended value for long periods of exposure (Beate et al., 2010) and 1,000 ng/m³ is the guideline for public exposure by WHO, (2007). In humans, approximately 80% of the inhaled mercury is absorbed by the respiratory tract where it reaches the circulatory system and is distributed throughout the whole body

(Cherian et al., 1978), accumulating particularly in the central nervous system (Langford and Ferner, 1999).

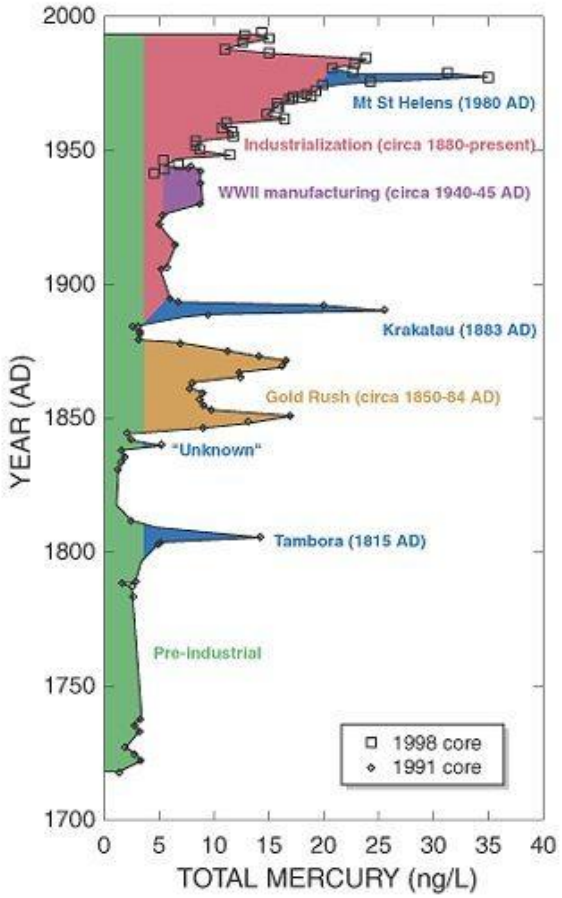

FIGURE 2-6. MERCURY DEPOSITION BY YEAR IN ICE CORES (KRABBENHOFT AND SCHUSTER, 2002)

Symptoms of long-term mercury poisoning due to moderate doses of mercury vapour include impaired cognitive skills, tremors, sleep disturbance, irritability, nervousness, memory loss, headaches, amongst others (Hinton et al., 2003, Langford and Ferner, 1999a, Liang et al., 1993, Ngim et al., 1992). Long-term, low-level mercury vapour exposure has been characterized by symptoms of fatigue, irritability, loss of memory, vivid dreams and depression (Veiga and Baker, 2004). In short to moderate duration exposure to

considerable levels of mercury vapour, kidneys are the most affected organs while the brain is the dominant receptor in long-term exposure to moderate levels (Suzuki, 1979). When exposed for short-term to high mercury levels, the individual can develop bronchitis and pneumonitis as well as serious neurological problems (Jones, 1971). These problems are relatively common in AGM sites, in particular amongst those working directly with amalgam burning and gold smelting; usually they do not use personal protective equipment or appropriate filters to condense mercury vapour. As a result, severe health impacts are observed in such workers (Spiegel and Veiga, 2010).

Observations by Counter et al. (2002), Counter (2003) and Counter et al. (1998) revealed that those health effects are already identified in children of Ecuadorean artisanal gold mining regions.

González-Carrasco et al. (2011) conducted an assessment of mercury contamination in the city of Portovelo, Ecuador. Air samples were collected in 11 different places around the city covering an area of approximately 25 km^2. The measurements of mercury concentration were obtained by using a Lumex Zeeman atomic absorption and background correction RA-915, provided by UBC- CERM3. The results showed values as high as 13,000 ng/m^3 near processing centers. A much higher concentration of mercury was measured in the air during the dry season, as usually observed in this kind of monitoring program. This difference is probably caused by lower evaporation from the soil and a larger condensation of mercury by the constant rainfall as observed by Wan et al. (2009) in an air contamination study in Changbai, China. It is important to note that this value is 130 times higher than

the recommended maximum exposure level of 100 ng/m³ (Beate et al., 2010). Although such large values were observed in measurements near processing plants, at the urban area, levels ranging from 214 to 574 ng/m³ were also obtained near gold shops that melt the gold *doré* with residual mercury. Normal background atmospheric levels of mercury in the air of rural areas are in the range 2-4 ng/m³ and between 10 and 20 ng/m³ in urban areas (Veiga et al., 2004a), so the observed levels are 11 to 57 times above background.

Another site heavily impacted by mercury pollution is Antioquia, in Colombia. This country is ranked as the 3rd highest polluter in the world by ASGM after China and Indonesia, being the 1st polluter with respect to mercury polution. In the five main gold producer municipalities in Antioquia, Segovia, Remedios, Zaragoza, El Bagre, and Nechí, values of mercury vapor in the air ranged from 100-10,000 ng/m³ in streets to up to 943,000 ng/m³ inside processing centers and gold shops (Cordy et al., 2011).

Although metallic mercury is dangerous when inhaled, it has low toxicity when ingested because the gastro-intestinal absorption of metallic mercury is lower than 0.01% of the dose (Langford and Ferner, 1999). Percutaneous vapor absorption is also low, being approximately 2% of the amount absorbed by respiratory exposure (Hursh et al., 1989).

Metallic mercury vapour is the main intoxicant in the mining sites. According to Marins et al., (1990) the metallic vapors emitted in amalgam burning and gold refining are deposited no farther than 1 km from the source. Some mercury precipitates in water courses, but the majority is deposited into neighbouring soil and vegetation. This deposited metal is eventually re-

emitted by forest fires to be re-deposited elsewhere. Meech et al., (1998) investigated the mobility of metallic mercury in the ASGM environment when reacting with organic acids. The conditions for methylation of the mercury bound with organic acids are still poorly understood. The bioavailability of these mercury complexes to earthworms was investigated by Hinton and Veiga, (2002) and by Hinton and Veiga, (2009) in a Brazilian ASGM site. It was demonstrated by these authors that the formation of mercury-organic complexes (i.e. mercury-humates and mercury-tannates) contributes to mercury bioaccumulation. They presented the hypothesis that sulphate-reducing bacteria in the intestines of an organism is capable of methylating mercury-organic complexes.

Roulet et al., (2000) examined the mobility of lithogenic and other sources of mercury deposited on sediments in the Brazilian Amazon River watershed. They claim that erosion and other human earthworks unrelated to mining are the most important factors contributing to the influx of mercury into water streams. However, they admit that "bioavailability of sediment-bound mercury would be expected to be low given its strong association with sediment particles". In fact, this low bioavailability of mercury bound to hydrous ferric oxides, abundant in the Amazon, was confirmed by Hinton and Veiga, (2009) using earthworm bioassay. It was indicated that these clay-fraction minerals have the ability to reduce mercury bioavailability of the organic-rich soluble mercury compounds. Regardless of the source of mercury in the rivers, high levels of mercury have been observed in fish in the Tapajós and other Amazon rivers and this is a health concern for the riverine populations (Mergler et al., 2007, Roulet et al., 1998).

Adler-Miserendino (2012) confirmed that erosion plays an important role in loading rivers and creeks in the Brazilian Amazon region with particulate sediments that contains lithogenic mercury. Using isotopic analysis, she concluded that particles containing mercury are carried in water streams for hundreds of kilometres. But she had a different perspective analyzing mercury isotopes in the Puyango-Tumbes watershed of Ecuador and Peru. She identified that most mercury in the sediments collected in the Tumbes River, 150 km from the artisanal gold mines, came from the mining operations in Portovelo, Ecuador. In many assessments of mercury transport with sediments (Mergler et al., 2007, Miserendino, 2012, Telmer and Veiga, 2009, UNEP, 2013), there are no studies or evidence of the methylation or bioavailability of mercury bound with the transported sediments. Complexed mercury ions or precipitates (from cyanide) are quite likely to be part of a pathway to methylation and subsequent bioavailability.

2.3.2. METHYLMERCURY

Metallic mercury released to water streams cannot be uptaken by aquatic biota since its solubility in water and subsequent bioavailability is low (Boudou and Ribeyre, 1997, Langford and Ferner, 1999). A serious contamination problem is generated when a bacteria mediated process can form methylmercury, which is readily absorbed by the aquatic biota and biomagnified in the trophic chain (Morel et al., 1998).

Then methylation reaction takes place by the transfer of an alkyl group ($-CH_3$) from an organic compound to the mercuric complex ion such as $Hg(OH)_{2(aq)}$ or an organic mercuric complex such as mercury fulvate or humate (Meech et al., 1998). Methylation occurring by the action of sulfate-reducing

bacteria (SRB) is the most accepted theory for increased methylmercury availability in natural- or human-contaminated water systems (Gilmour and Henry, 1991). Laboratory experiments with high concentration of mercury (0.5 mM) showed that methylation happens intensively in the presence of the SRB in a range of sulfate concentrations (Choi et al., 1994).

The cycle of mercury in a water system is shown in Figure 2-7, where it can be seen that mercuric species in anoxic sites in the presence of SRB are methylated. Generation of methylmercury occurs and then becomes bioavailable to be immediately incorporated into phytoplankton. Phytoplankton is the basis for aquatic trophic species feeding (Kainz et al., 2006) and it readily accumulates methylmercury and Hg (II) because these are highly reactive compounds, unlike the metallic form of mercury (Kainz et al., 2002). Methylmercury biomagnifies in the aquatic food chain because other mercury compounds are accumulated in cellular membranes and such materials are not digested by predators. The absorption of methylmercury occurs by ingestion of contaminated food and not by contact with contaminated water or sediment (Boudou and Ribeyre, 1997). Cabana and Rasmussen, (1994) and Morel et al., (1998) show that fish in higher levels of the food chain, show higher levels of methylmercury in muscle tissue.

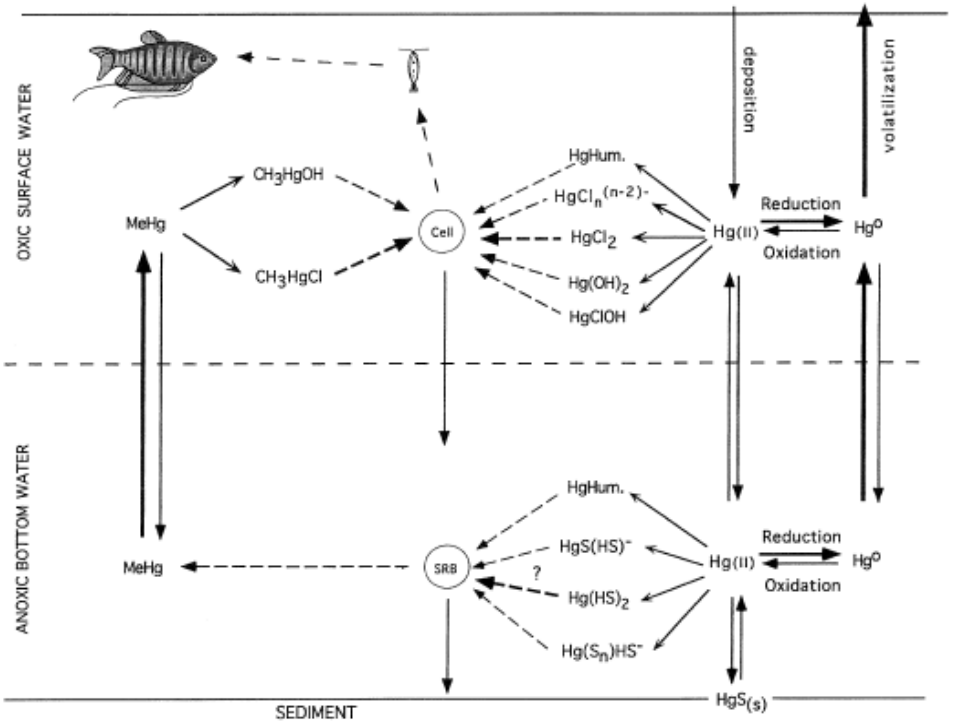

FIGURE 2-7. MERCURY CYCLE (MOREL ET AL., 1998)

The first large epidemic contamination of humans by methylmercury happened in Minamata, Japan in 1956 where methylmercury and mercuric compounds were released to the Minamata Bay, contaminating fish and shellfish. Such an incident officially shows 2,265 victims leading to 1,784 deaths (Harada, 1995). This is not an episode related to mining, but most of the symptoms of human intoxication by methylmercury such as cardiovascular, brain and nervous system effect were first discovered from studies on the Minamata epidemic (Mergler et al., 2007). The core impact of this human contamination was the congenital effects, not recognized until 1972. Exposed to methylmercury through placental absorption, infants began to exhibit severe cerebral palsy even though their mothers showed little or no manifestation of poisoning. The effects on foetuses included: mental retarda-

tion, cerebellar ataxia, primitive reflexes, dysarthria and hyperkinesia (Harada, 1978).

The main acute intoxication effects to high doses of methylmercury are neurologic, notably brain and nervous system damage resulting in distal sensory disturbances (visual perception problems), constriction of visual fields, ataxia (lack of voluntary muscle coordination), dysarthria (motor speech disorder), auditory disturbances and tremors (Clarkson et al., 2003, Kosatsky and Foran, 1996, Veiga et al., 2004a)

Cardiovascular effects include elevated blood pressure, alteration in heart rate variability, acute myocardial infarction and other coronary heart diseases caused by chronic methylmercury contamination in humans (Chan and Egeland, 2004, Stern, 2005); recent research however only found a correlation between methylmercury contamination and a relative risk increase of myocardial infarction (Guallar et al., 2002, Pesch et al., 2002).

Grandjean et al., (1999) investigated chronic intoxication with low to moderate levels of methylmercury in fish-eating children in the Tapajos river system downstream of the oldest and largest Amazonian artisanal gold mining activities. The amounts of methylmercury ingested and absorbed were determined by analyzing the mercury content in the hair of 351 children ranging in age from 7 to 12 years. The cohort was composed of people from 4 different communities at increasing distance from the mining activities. People in these villages eat fish at least once a day and they are not in touch with any kind of industrial activity using mercury. The primary source of mercury is fish. In the communities closer to the ASGM sites, children have hair mercury levels averaging 11 μg/g, above the safe neurotoxicity limit of 10

μg/g (World Health Organization, 1990). Around 90% of the children showed results above the safe limit in the regions served by the Tapajos River, after the confluence with the Amazon River. In the same region, similar results were found by Akagi and Naganuma, (2000).

1.2.1. MERCURY CYANIDE

The metallic mercury release observed in regions where AGM, and particularly processing centers operate as, described in section 2.3.1 is usually associated with a practice where payment to the processing centers is made with the gold-rich tailings generated after gravity concentration followed by amalgamation process. Mercury contaminated tailings are then reprocessed using cyanide leaching to extract the residual gold not trapped by mercury (e.g. non-liberated or very fine gold)

Cyanidation of mercury-contaminated tailings is a relevant source of pollution as mercury cyanide is formed and released into the environment. It is well known that soluble mercuric ion species are easily methylated in the environment (Jensen and Jernelöv, 1969). However the degree to which mercury cyanide species in sediment are methylated is not well-understood. If significant, this can lead to direct bio-accumulation in aquatic organisms. At many sites where mercury-contaminated tailings are leached with cyanide, nearby fish contain high levels of mercury (McDaniels et al., 2010). Rodrigues-Filho et al. (2004) analyzed 31 samples of carnivorous fish from a small lagoon in the Brazilian Amazon that was receiving effluent from a cyanidation operation. The average total mercury concentration of the samples was 4.16 ± 5.42 mg/kg and a small fish (15 cm) sample showed levels of 21.9 mg Hg/kg (Sousa and Veiga, 2009), probably a new world record.

Mercury, like gold and other metals, forms soluble complexes with cyanide, such as $[Hg(CN)_4]^{2-}$ which is stable at pH levels above 8.5 and $Hg(CN)_{2\ (aq)}$, stable at pH levels below 7.8 (Flynn and Haslem, 1995). Gold cyanide complexes formed during leaching can be adsorbed on activated carbon and removed from the leaching system. $Hg(CN)_2$ is more easily adsorbed on the activated carbon than $[Hg(CN)_4]^{2-}$ (Adams, 1991). Since gold cyanidation occurs at pH levels between 10 and 11, little mercury will report to the activated carbon. So, the tailing left behind when most of the gold is extracted from solution using carbon, is rich in mercury-cyanide soluble complexes in solution or non-dissolved metallic mercury droplets. This was observed in Portovelo where miners use cyanide to leach mercury-contaminated tailings. It was concluded by Velasquez et al. (2011) that less than 4% of the initial mercury that entered the cyanidation tank with the tailings was adsorbed and removed by activated carbon while about 51% of the mercury remained in solution. The rest of the mercury, not dissolved by cyanide, remains as droplets with the tailings that is discharged into the local water streams. The same authors report that zinc in the Merrill-Crowe process is much more efficient at removing mercury from the cyanide solution than is activated carbon. Unfortunately, it is a common practice to burn the zinc shavings releasing all volatile metals into the urban environment. Levels up to 100,000 ng/m^3 were analyzed in the atmosphere during this zinc-burning process (Velasquez et al., 2011).

Worldwide, all tailings left at processing centers are leached with cyanide. In some cases, the cyanidation is conducted in percolation vats. The percolation of mercury-contaminated tailings has been witnessed in Brazil, China, Co-

lombia, Ecuador, Mozambique, Nicaragua, Peru, Venezuela, Tanzania and Zimbabwe (Veiga et al., 2013).

As the concept of processing centers spread around the world generating local contamination in the form of air pollution from amalgam burning and creating long-reach impacts by spreading metallic mercury, mercury cyanide and causing methyl-mercury to be produced in the water-shed; attention should be given to organizing this industry so that it does not generate such a large impact on the exposed populations. Moreover, contamination can be observed hundreds of kilometers away from the mercury source, as occurs in the Tumbes River system where mercury released in Ecuador travels to Peru (Miserendino, 2012) generating international tension.

3. DEFINITIONS AND DIFFERENCES BETWEEN ARTISANAL MINING AND SMALL-SCALE MINING

Many research project combine artisanal gold mining and small-scale gold mining into one large group usually identified by ASGM (Artisanal and Small-scale Gold Mining). While this may be a reasonable simplification, some characteristics are particular of each of the two clusters: artisanal and small-scale. It is also difficult to study social impacts and the relationship models between individuals within the larger group. Because of such differences and lack of efficient definition in different jurisdictions in developing countries, the present work examines each cluster individually. Although Artisanal Gold Mining (AGM) is usually differentiated from Small-scale Gold Mining (SGM) by its size, this is not always the case since it is not uncommon to find artisanal operations processing thousands of tonnes per day of material (Veiga, 1997) nor it is unusual to observe small, but sophisticated operations processing 10 to 20 tonnes a day, as observed in Portovelo, Ecuador (Veiga et al., 2013). The term artisanal gold miner is used to encompass all small, medium, large, informal, legal and illegal miners who use rudimentary processes to extract gold from any kind of secondary or primary mineral deposit (Veiga, 1997). The term "artisanal" seems to define the way in which the miners operate and not necessarily the size of the operation. It is known for example that there are artisanal operations in the Brazilian Amazon processing more than 1000 tonnes of colluvial ore per day (Sousa and Veiga, 2009).

Few countries have clear regulations or definitions of artisanal mining but almost all that do relate the term to the size of the operation. In Peru, Law

27651 of 1992 (Peru, 1992) provides definitions based on production capacity. For example a large mine is one that exploits and processes more than 5000 tonnes of ore per day (tpd) and a medium mine processes from 150 to 5000 tpd. The Peruvian legislation discriminates between hard rock and alluvial deposits but just for small and artisanal operations. A mine processing from 25 to 150 tpd of hard rock or 200 to 3000 m^3 of alluvial ore is considered small and those processing less than 25 tpd of hard rock or less than 200 m^3 of alluvial material are classified as artisanal.

The Ecuadorian Mining Law of Jan 29, 2009, defines three types of mining: Artisanal, Small, and Large scale (Vergara, 2009). According to what is stated in Chapter 2 Art. 138 of this law, a small-scale mine is one with a capacity to exploit and process up to 300 tonnes of ore per day. Everything above this capacity is considered large-scale and subject to high taxes.

The Ecuadorian Law establishes a fuzzy difference between small-scale mining and artisanal mining. In Special Decree 120 of Nov 4th, 2009, (Delgado, 2009) small mines are not clearly defined but are identified as those that exploit an area no larger than 300 ha and do not mine for subsistence. Artisanal miners are individuals, families, or cooperatives that mine for subsistence. They are not subjected to royalties (Art. 134). It is interesting to note that according to this Law, this type of mining activity is characterized by the use of simple and portable tools. The result of mining is only to cover basic family needs. In this case, the level of production of an artisanal miner cannot exceed 0.5 tonnes/d (hard rock) or 40 m^3/d for alluvial ores.

In Chile a small mine is considered to be the one that mines and processes 300 tpd or less (Rodriguez, 2005).

It seems relevant to comment that legislation in the majority of these countries do not clearly define artisanal mining. The distinction from small-scale mining is usually unclear and very often is based on the size of the operation. This is the source of legal problems. The term artisanal mining, as it is defined in the Colombian, Ecuadorian, and Peruvian legislations should be replaced by "micro-mining", since the term "artisanal" does not necessarily imply small mining but certainly implies operations using rudimentary techniques.

Figure 3-1 shows, in a schematic way, the complexity of the problem in defining artisanal and small-scale mining. It is understandable why the judicial systems in developing countries cannot find common ground to establish clear policies for the sector.

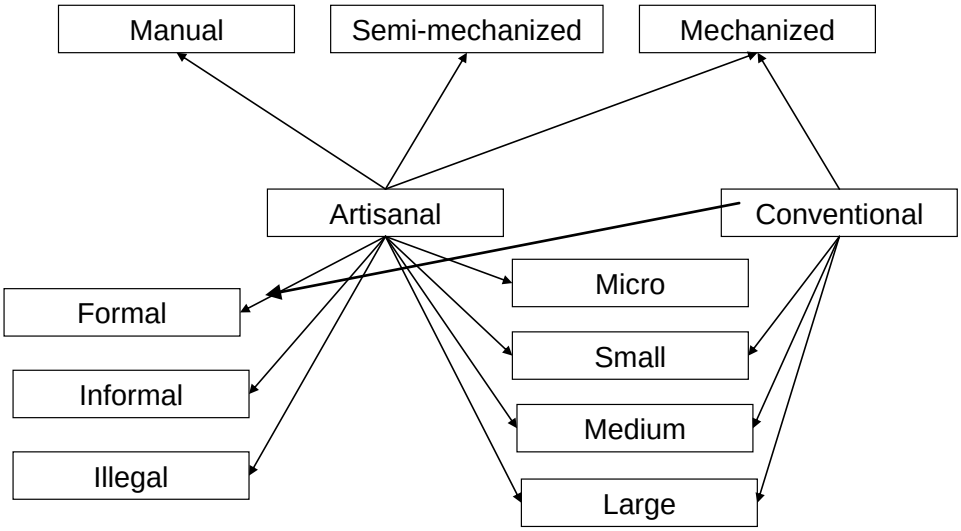

FIGURE 3-1. SCHEMATIC DEFINITION OF DIFFERENT TYPES OF MINING OPERATIONS

The way in which a mine operates is viewed in this work as the main difference between artisanal and conventional mining. Even with high mechanization, artisanal miners do not establish reserves, do not use short- or long-term mining plans, do not establish safety procedures, do not control metal-

lurgical recoveries, etc. Unfortunately governments do not recognize that the solution to organizing artisanal miners should proceed through education of the miners. This includes technical, economic, environmental, legal, and social education.

For simplification purposes, the term "micro-mining" (MM) is used from now on to identify mining activities processing less than 5 tonnes of ore/day. The term "artisanal" is restricted to characterizing those operations using primitive techniques, independently of their size. A definition of the mining types based exclusively on the size of the operation is proposed in Table 3-1. This suggested definition is based on the existing legislations in different countries, field observations in Ecuador and Brazil, and literature review (Veiga, 2004a, Veiga 2004b)

Definition	Tonnes/day
Micro	<5
Small	5 - 300
Medium	300 - 1000
Large	>1000

TABLE 3-1. PROCESSING CAPACITY DEFINITION

FIGURE 3-2 SHOWS THE DISTRIBUTION OF ASGM IN THE WORLD ACCORDING TO MERCURYWATCH (2013). IT CAN BE SEEN THAT VIRTUALLY ALL DEVELOPING COUNTRIES HAVE ARTISANAL GOLD MINING ACTIVITIES. (*) IS FOR WEEKEND PROSPECTORS

Table 3-2 and Figure 3-2 were assembled using data from Veiga et al., (2004). They list all countries in which scientific publications on ASGM have been issued.

Although artisanal mining is often characterized by extensive negative environmental impacts, this sector provides livelihoods for a growing number of people in Africa (Spiegel, 2009) and all over the underdeveloped world.

Australia(*)	Ecuador	Lesotho	Philippines
Benin	Ethiopia	Liberia	Russia
Bolivia	French Guyana	Madagascar	Rwanda
Brazil	Gabon	Malawi	Senegal
Burkina Faso	Gambia	Malaysia	South Africa
Burundi	Ghana	Mali	Sudan
Cambodia	Guinea	Mauritania	Suriname
Canada(*)	Guinea-Bissau	Mexico	Tanzania
Central Afr. Republic	Guyana	Myanmar	Togo
Chad	Honduras	Mongolia	Uganda
Chile	India	Mozambique	USA(*)
China	Indonesia	Nicaragua	Venezuela
Colombia	Ivory Coast	Niger	Vietnam
Costa Rica	Kenya	Nigeria	Zambia
DRC	Kyrgyzstan	Papua New Guinea	Zimbabwe
Dominican Republic	Lao PDR	Peru	

(*) IS FOR WEEKEND PROSPECTORS

TABLE 3-2. COUNTRIES WITH DOCUMENTED ASGM ACTIVITIES (ADAPTED FROM: VEIGA ET AL., 2004)

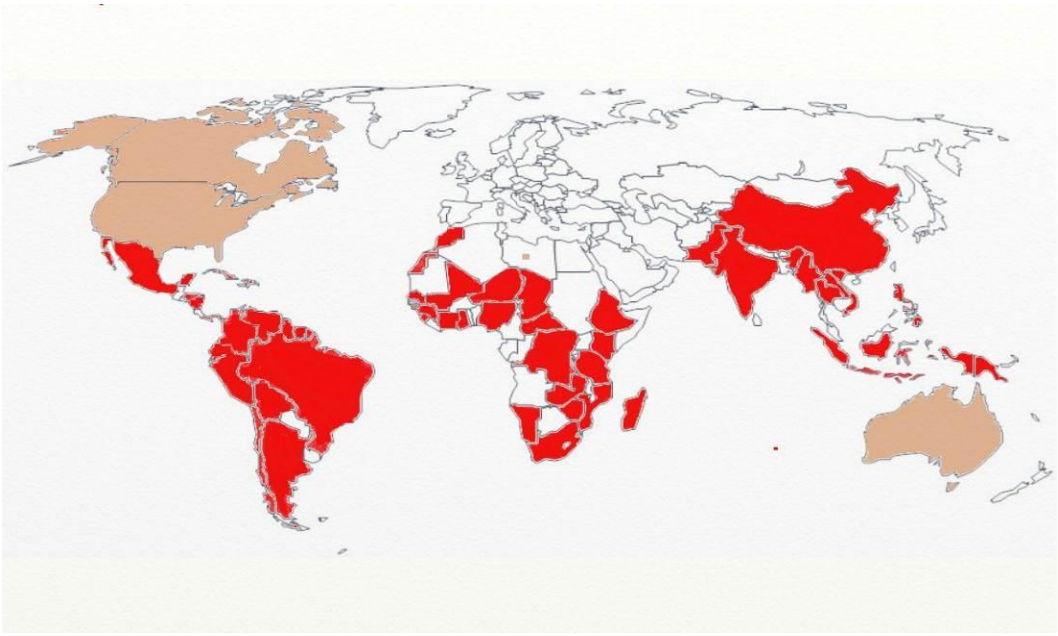

FIGURE 3-2. COUNTRIES WITH DOCUMENTED ASGM ACTIVITIES

3.2. MICRO GOLD MINING

Micro-mining (MM) is characterized by the informality of the practitioners, either due to ignorance or difficulties in complying with environmental and mining regulations. Generally, MM is exercised for simple subsistence. Some countries have legislation for this kind of worker in terms of taxation and licensing, but not much environmental attention is given by their governments.

Micro-miners use picks, chisels and dynamite to exploit gold from hard rocks and shovels in alluvial/colluvial deposits. Small and medium artisanal miners produce much more ore from alluvial and colluvial deposits using hydraulic monitors and excavators. Independently of the size of the operation, some miners concentrate gold by panning or sluicing followed by gold amalgamation or cyanidation or simply amalgamation of the whole ore without any previous concentration.

Most micro-mining activities are driven by the migration of people coming from other occupations, which is easily understood when there is no evidence of more profitable activities than gold mining in economically weakened countries. Heemskerk, (2001) conducted a study in Suriname showing that high prices and unemployment rates directly encourage MM as does high inflation, making hard currencies such as US dollars and gold to be attractive. Indirectly, a debilitated economy decreases the real value of wages. Between 1980 and mid-1990s for example, the real value of wages in Suriname dropped by 50% encouraging thousands of people to leave their homes in the cities or farms to become micro gold miners (Heemskerk, 2001).

MM may also serve as an important enhancement to rural economic activities, mitigating the decay of urban areas, particularly in urban slums (Sinding, 2005). The relation between MM and environmental destruction is definitely important, since this category of miners represent the largest contingent of individuals involved in artisanal gold mining. However, large artisanal operations do place more importance on environmental emissions, release of mercury and cyanide and pollution of rivers with sediments. This is an important point since many pollution reduction programs from artisanal miners focus only on micro-miners. The cost-benefit ratio of these interventions must be evaluated since MM are dispersed, usually do not work in cooperatives and have subsistence as their main driving force. MM are also those miners who mine as a temporary or seasonal activity (Hilson, 2009). It is important to note that lack of knowledge of micro-miners, in both the mining and processing steps of gold production, ends up generating a self-perpetuating cycle of poverty as illustrated by Figure 3-3.

This cycle of poverty is sped-up by exploitation of the miners by middlemen who pay an unfair value for the produced gold, by land owners who charge too much to allow micro-miners to operate, by processing centers that extract less than 30% of the gold of the ore mined by the MM, by the rudimentary techniques employed making the ratio produced-gold/worked hours too low, or by a combination of these factors.

A micro-miner is held in a condition of poverty if the activity only manages to feed their family or if their income is defined as the amount of money one makes selling the produced gold. Accordingly to Figure 3-3, one can consume all this income in the operating costs of this inefficient activity. The

problem is that the majority of the generated money is dedicated to subsistence only and nothing is left to be invested in education or other means of getting out of poverty (Sinding, 2005).

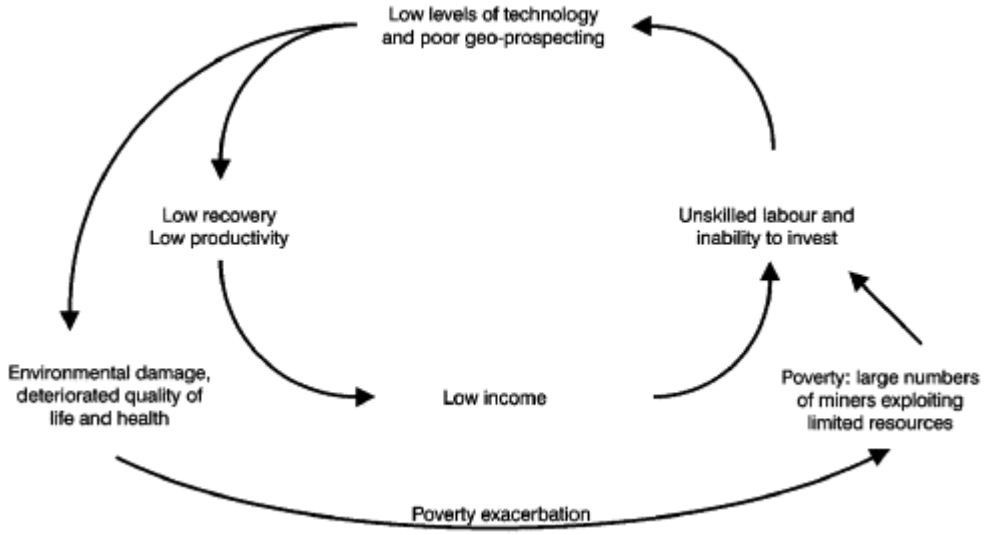

FIGURE 3-3. SIMPLE RELATIONSHIPS IN ARTISANAL MINING (HILSON AND PARDIE, 2006)

Successful action on reducing mercury pollution in MM is hard to be done, since the miners are spread around the world and are numerous and individually produce low amounts of gold.

3.3. SMALL-SCALE GOLD MINING

Large ore deposits are becoming hard to find. In Canada today for every 1000 explored deposits only 1 becomes a mine (Veiga et al, 2011). What to do with the other 999? Very infrequently exploration companies invest in a small operation to process less than 300 tonnes of gold ore per day. Usually junior companies abandon the area when they do not see the potential for a large deposit above 1 million ounces of gold. In developing countries, many

informal or illegal artisanal miners occupy the sites abandoned by exploration companies (Veiga et al., 2011).

In spite of the proliferation of the processing centers that exploit the miners and show little to no concern for the environment, it has been observed that a few small gold plants in Portovelo are doing efficient and responsible work. These plants have a processing capacity of 100 to 200 tonnes or ore/day. Owners of the processing centers are acquiring their own mines or making agreements with the miners to integrate them into their businesses. Engineers (many from Peru), operate either cyanidation of the whole ore or gravity and flotation concentration followed by cyanidation. At least four plants are positive examples for the other miners in Portovelo, Ecuador. The gold production ranges from 300 to 1000 g Au/d and mercury is not used. This provides jobs for a large contingent of local people and sustains other social activities in the community. Cyanide tailings are properly managed, being treated with hydrogen peroxide (that destroys free cyanide still available) and then impounded in impermeable dams,. Other minerals such as copper and silver are also extracted.

The modest evolution of ASGM witnessed in Ecuador is occurring thanks to strong investment from the private sector, some international training cooperation, increased presence of government regulators at the mining sites and the presence of good technical people interested in the small mining business. Unfortunately, not all governments in developing countries have the capacity to understand this important evolution and many avoid participation (McDaniels et al., 2010). They simply try to solve the pollution problem of artisanal miners by forcing formalization of the sector. Without edu-

cation and organization, this is only formalizing the pollution, as formal "companies" are releasing as much mercury and cyanide to the environment as the informal miners.

Small and responsible mining is definitely more socially acceptable by locals and more durable than large mines. Well-structured small mining companies in developing regions means more jobs and better use of the natural resource since small deposits are not of interest for large mining companies.

The Association of Processing Centers of Portovelo (APROPLASMIN) has been discussing with miners the possibility to implement a fair business for miners and processors. The goal is to build a more sustainable future for mining in the region. While it is still modest, this type of business in the industry of the processing centers is an interesting idea. These companies have been established either by means of associations of miners or as private companies with one or more investors. Competent administrators take care of hiring miners, operators, engineers, security guards, and other workers exactly like a large company. These small-scale gold mining companies employ from 10 to 200 people directly.

The case herein highlighted is related to the Cazaderos processing plant, located in Portovelo processing ore from their own mine, in Ponce Enriquez, a municipality 4 hours away. This company employs 20 people at the plant and 28 at the mine. This family company is owned by the patriarch of the family and three of his descendants who work at the plant and the mine. All of the family members work at managerial and administrative levels, while technical assistance at the plant is given by several engineers who work in 12-hour shifts, 6 days per week. All other employees are locally hired and

usually come from a mining environment, but have no specific technical education.

Almost all of the equipment in this plant was locally fabricated with designs copied by Peruvian engineers from books and other equipment manufacturing catalogues. The only exceptions are the jaw crushers (30 cm x 15 cm and 20 cm x 10 cm) imported from China and the 55 tpd ball mill (Ø120 cm x 120 cm) imported from Peru

The company has proper installations in order to use modern processing techniques such as a Falcon (Sepro) centrifuge, flotation and Carbon-in-Pulp (CIP) cyanidation leaching. The engineers designed a clever flowsheet to separate chalcopyrite from pyrite by flotation, in order to sell the chalcopyrite concentrate (containing gold) and only leach the pyrite concentrate with cyanide. A centrifuge is used at the discharge of the ball mill instead of the more usual and indicated location at the underflow of the hydrocyclone (Laplante, 2000). This unit recovers 60 – 70% of the gold in the feed, according to the mass balance conducted daily by the technical staff.

Centrifuging, as a gravity separation method, often used in large industrial gold processing but the technology was implemented in Cazaderos only 3 years ago. A general flowsheet of the process is shown in Figure 3-4. The ore is fed using a small front-end loader and firstly classified by a 100 mm grizzly screen. The oversize is separated to be broken by a worker with a sledgehammer. The -100 mm fraction feeds a small Chinese jaw crusher with capacity of 8-10 tonnes/hour working in open circuit crushing to a top size product of 38 mm. A conveyor belt transports the material to a second jaw crusher of smaller capacity – 2 to 3 tonnes/hour – working also in open cir-

cuit. A 12 mm vibrating screen separates the coarse material to be crushed in this second crusher while the -12 mm material goes directly to the main silo. The crushing circuit works only 2-3 hours a day, since it has a higher capacity than the ball mill.

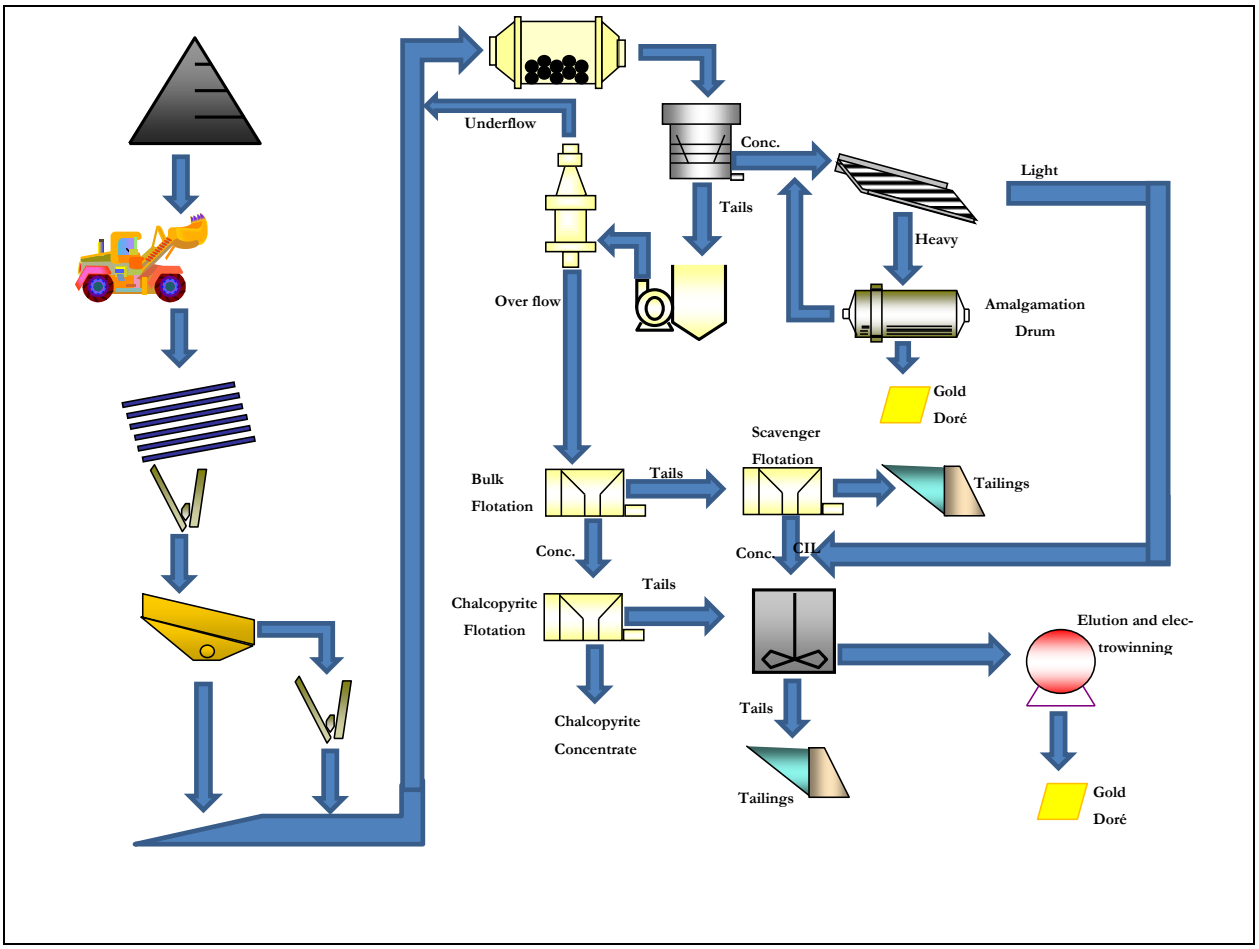

FIGURE 3-4. CAZADEROS PLANT FLOWSHEET

A 48-hour capacity silo feeds the ball mill via a conveyor belt. Ground material then feeds the SB400 Falcon (Sepro) centrifuge and the tailing goes to a hydrocyclone. The cyclone cuts the flow at 200 mesh (75 microns) with the coarse fraction returning to the ball mill and the fine fraction sent to the flotation circuit.

High gold content material is trapped in the centrifuge which is cleaned every hour. Global gold recovery in the centrifuge is around 70%. The heavy material is stored to be sluiced. Sluicing takes 8 hours per day and is conducted under supervision of a trustful person (owner's brother) since the sluice box concentrate assays between 2000 and 3000 ppm of gold. The tailings from the sluice are contained in a pool to be leached with cyanide together with the pyritic concentrate generated in the flotation circuit.

Fines coming from the grinding circuit feed a circular flotation cell (a copy of the Outkumpu cell) with a processing capacity of 5 tph; potassium amyl xanthate and frother (MIBC) are added with the pH remaining natural at around 7. Different frothers and xanthates are often used depending on local availability. This bulk flotation process floats both pyrite and chalcopyrite (around 30 – 40% gold recovery). The flotation concentrate of the bulk flotation flows to a conditioner where the pH is raised to 10 with lime. At this pH chalcopyrite will still float, while pyrite is depressed. The tailing from this flotation step, low in chalcopyrite but rich in pyrite, proceeds to be leached with cyanide. The tailing from the bulk flotation proceeds to a scavenger step to recover additional pyrite. The scavenger concentrate is sent for leaching while the tailings are the final tailings in the circuit, sent for disposal.

As copper minerals consume cyanide and there is gold associated with chalcopyrite, this two-step floatation works well in order to separate pyrite from the cyanide consumer chalcopyrite, generating a saleable copper/gold/silver concentrate. Gold contained in chalcopyrite concentrates is paid by the smelter with a discount of 10%, while copper is paid with a discount of 12%.

Different contaminants add penalties to the value paid accordingly to their quality and quantity.

The pyrite concentrate and sluice-box tailings proceed to a cyanidation tank. Cyanidation at pH 10.5 usually takes around 30 hours and consumption of cyanide is around 20 kg/tonne of concentrate. After leaching, activated carbon is added to adsorb gold for 12 hours. When adsorption is complete, the final tailings are pumped to geomembrane-lined tailings ponds located inside the property, together with tailings generated in the flotation circuit.

In spite of being a small family company with a good processing plant, relatively good technology and capital to invest, some poor practices are still observed. The rich material concentrated in the sluice-box does not go to cyanidation. Intensive cyanidation or even direct smelting could be a solution to extract gold from this concentrate but instead, they apply amalgamation in a *chancha*. Each drum is loaded with 200-250 kg of material, a few pebbles and one ounce (28 g) of mercury. The drums work for 12 hours and then the amalgam is recovered by panning and squeezed in a piece of cloth. The amalgam is then mixed with brown sugar as the operators believe this will clean the amalgam from other "impurities". The final amalgam is wrapped in aluminum foil to avoid contact with the iron of the retort and burned. In this process, 98% of the mercury is recovered. While good attention and care are paid in the amalgamation step, the amalgamation tailings that still contain gold and pulverized mercury are sent to cyanidation. As described before this generates mercury-cyanide complexes that can be transformed into methylmercury or directly bio-accumulated by aquatic biota. Although tailings are stored in a geomembrane-lined pond, process water possibly car-

rying harmful complexes is discharged into the Amarillo River since there is no cyanide destruction or any further treatment to the leaching circuit effluents that are discharged to the tailings dam.

3.3.1. SGM CONTAMINATION

Even though SGM has great potential to be the next trend in gold mining, when miners have access to technology and capital, this can also create problems. SGM operations even using non-artisanal techniques, may take advantage of the loose regulations and enforcement to pollute the environment.

For example in the city of Portovelo where AGM and SGM and processing centers coexist, mercury pollution plays a relevant role. It is hard to establish emissions and releases from different groups of processors, but it is known that there are about 108 processing plants in the town along the river. From the total AGM, SGM and processing centers, 21% are dedicated to grind, concentrate and amalgamate the gold, 58% perform cyanidation of tailings or gravity concentrates, and 15% combine flotation with cyanidation. About 94% of the processing centers concentrate the ore prior to using mercury which significantly reduces mercury emissions to the environment (Velásquez-López et al., 2010). Incorporation of flotation was an important step in providing miners with a process that increased gold recoveries substantially while also producing a sellable copper concentrate. Despite these innovations, mineral recovery is still inefficient in most of these areas. Waste management is a major challenge in creating true sustainability in this sector (technical efficiency together with social and environmental responsibilities).

Tailings management is controlled by the Ecuadorean environmental agency but such control needs to be improved. Most of the SGM are located not more than 100 m from a water stream and several tailings dams are located right beside the river, being poorly built with river boulders lined with pebbles, sand and geomembrane. Dams are usually 10 to 20 m high, hundreds of meters long and 50 m wide. The proximity of the dams to the rivers causes a "natural" erosion of the tailings in the aquatic system. The major impact is visual but many plants do not destroy cyanide and mercury/cyanide contaminated material is directly released into the river.

The rivers are very steep and run fast. They have never hosted many species of fish, but currently there are absolutely no fish. Appleton et al. (2001) showed that the level of river contamination is an issue in the region. Although surface water that is contaminated by As, Cu, Hg and Zn may pose problems to aquatic biota and humans, the major concern is the potential effect of these metals on the commercial banana plantations and shrimp farms downstream.

Concentrations of potentially harmful elements in bottom sediment at the mining site and downstream, as indicated by Miserendino (2012), suggest the likelihood of remobilization of these metals and bioaccumulation. As the majority of the plants dump their untreated tailings into the local rivers during the night, color changes of the water from a dark brown early in the morning to a lighter brown at the end of the day is observed.

Dumping tailings in the river is a common practice of MM, SGM and processing centers in the Zaruma-Portovelo region and appears to be accepted by the local community. In fact it has been witnessed by locals that the city

hall also dumps the municipal garbage into the same rivers. Little reaction by population against these poor environmental practices has been observed. It was estimated by Velasquez et al. (2010 and 2011) through mass balances that hundreds of tonnes of cyanide and about 1.5 tonnes of mercury is released every year from the plants in Portovelo-Zaruma into the environment. From this estimate, it was suggested that 70% of the mercury is evaporated and 30% is released with tailings. Since most plants are currently selling copper concentrates with gold to Peru, it is expected that a reduction in the mercury and cyanide releases from the levels reported three years ago has occured, but this needs be quantified.

In conclusion it is observed that small-gold mining can also have artisanal behaviour even using advanced processing techniques. The environmental pollution from SGM is definitely more relevant than a single micro-miner (or even a group of them). The example above makes more difficult for regulators to establish clear policies for micro, small, medium and even large artisanal gold miners since the efficiency and better techniques do not define better practices. The pollution is not only a simple matter of using better techniques but also an ethical commitment and a management policy. The concepts of corporate social responsibility are still far away from the artisanal operations independently of their sizes. Like in any other industry, there are individuals that incorporate environmental and social-sound procedures into their practices but others take advantage of the poor legislation and poor governance of the local communities to reap benefits at the costs of the environment and health of citizens. What is surprising is the tolerance and acceptance of these poor practices by the locals, although this is proba-

bly a result of the involvement of family members and friends in this polluting scheme.

4. THE LEGAL APPROACH

As shown in Chapter 1, the environmental impact of ASGM is well documented (González-Carrasco et al., 2011, Guimaraes et al., 2011, Harari et al., 2011, Li et al., 2009, Spiegel and Veiga, 2010). Irresponsible operations and management practices have a global impact on air, land and water. The environmental impacts also have implications on the socio-economic aspects of the AGM communities. Approximately five million women and children are directly employed in AGM (Veiga et al., 2004) and are victims of the pollution. Hinton et al., (2003b) highlighted cases in which women are left with the dirty work of amalgamation as the male miners do not make them aware of the dangers of mercury intoxication.

Indeed socioeconomic status remains a persistent and robust determinant in variation of rates of illness and death (Williams and Collins, 1995). Adverse health consequences may also negatively influence family dynamics, lifestyle and behaviour, and manifest a depressed social structure (Swain et al., 2007). Poverty is a major cause of global environmental problems as a result of long-lasting social and economic injustices (Kohlhuber et al., 2006). In most instances, poorly-resourced governments are the sole bodies able to direct regional activities in order to regularize and facilitate environmental improvements in what for decades has been treated as an illegal industry (Hilson, 2002).

The formalization approach has been the target of the large majority of the federal governments of developing countries facing problems of informal or

illegal[3] artisanal mining. Chen, (2005) stresses that informal employment comprises 50 to 75% of non-agricultural employment in developing countries. When the agricultural sector is taken into consideration as an informal activity, the proportion of informal employment can reach 93% of the total employment as in India, for example. Then, why are the governments so interested in formalization of the informal miners instead of formalizing other sectors of the economy as well? The answer is simple: taxes. Paraphrasing Hinton et al. (2003), an artisanal miner, micro, small, medium or large will not pay a dollar to be formalized or to adopt a new cleaner procedure if they do not see two dollars as a result of this. Unfortunately governments are not interested or do not have the technical capacity to deal with artisanal miners. The lack of government authorities in artisanal mining areas to replicate the training and educational initiatives, as well as to provide orientation to miners on how to obtain legal mineral title is a main impediment to establish formalized mining sector (McDaniels et al., 2010)

The idea of approaching the artisanal mining problem by means of formalization is spread throughout developing countries but good examples of successful actions are hardly found. While this approach has value in dealing with small, medium and large artisanal mining, it fails in dealing with micro miners since this activity is usually driven by poverty and subsistence. Organised companies can be required to implement cleaner production methods and follow the laws. However, it is hard to see governments enforcing the

[3] Illegal mining is identified when the activity is conducted without a proper title, authorization or concession issued by the competent authorities (Ávila, 2002), while informal mining is a set of deficiencies in environmental management, technical assistance and development, access to information and acceptable working conditions(Hentschel et al., 2002).

laws and providing technical advice to micro-miners on how to apply cleaner techniques. Corruption and a lack of qualified personnel is another impediment to establishing better practices (MMSD, 2002, Song and Mu, 2012).

In theory, the formalization of artisanal miners is the first step to cleaner production since this creates a sense of ownership to the mineral title and a formal title can give miners transferable capital (Siegel and Veiga, 2009). In a practical way, formalization is an intricate process for artisanal miners, in particular for the micro and small miners. The bureaucracy for formalization is heavy in all countries and not all miners can fulfill the legal requirements. It is important to note that, in most countries, the formalization at the Ministry of Mines does not imply that a legal condition will be satisfied since the environmental permits must also be obtained from the Ministry of Environment (Sousa et al., 2011). Unfortunately, formalization has been seen by artisanal miners as a way of the governments to charge taxes to informal miners (Veiga, 2011).

The number of miners formalized is usually fails to make a significant difference in the disorganization and pollution observed in artisanal mining sites. For example, after three year of promotion and planning, the formalization of 450,000 Peruvian artisanal miners[4] (Expreso Peru, 2012) was a fiasco due to the bureaucracy. Peruvian authorities recognize that out of the 50,000 miners that applied to obtain a formal mineral title, only 27,000 will be formalized in short-medium term (Mineria Al Dia, 2013). It seems logical that before formalizing, governments should invest in educating and organizing the miners. Otherwise, without legal and technical assistance, as well as eco-

[4] There is no official estimative of the number of artisanal mining workers in Peru.

nomic incentives, the governments are just formalizing pollution. A similar failure to formalize artisanal miners is mentioned by Güiza Suárez and Aristizabal, (2013) in Colombia, where 87% of the mining operations is illegal. Between 1993 and 2008, after government efforts to formalize the mines, from 3,631 applications, only 23 mines were legalized, or approximately 1% of the applications.

The MM have a somehow different story in the formalization process as their activities are driven by subsistence. This means that the MM usually have neither the education nor the capital to work properly and they are not in a condition to stop their activities to be trained or formalized (Spiegel and Veiga, 2006, Veiga et al., 2004a). This sometimes makes it impossible to teach MM how to employ better methods to produce more gold. The legal approach cannot be divorced from the realities faced by miners. The typical approach of international and local environmental and development agencies is political and palliative, therefore not sustainable in terms of creating sound poverty alleviation programs. Hilson (2007) argued that there is poor understanding of artisanal mining dynamics and international organizations try to implement inappropriate support schemes and interventions. An example is that many interventions try to bring sophisticated industrial processes to artisanal miners without properly evaluating the education level of the miners, adaptability to the processes, or in some cases, consulting them at all (Hinton et al., 2003).

In mining sites where MM are abundant a low life expectancy due to many factors such as starvation, war, and the HIV/AIDS is observed. In these

cases, the hazards of mercury pollution are secondary concern, even when miners are aware of mercury intoxication (McDaniels et al., 2010).

International development agencies rarely dedicate time to find out about the needs of the miners (Siegel and Veiga, 2010), in particular the micro-miners. This was also criticized by Hilson (2005), who suggested that any intervention should first assess "the number of people operating in AGM regions, their origins and ethnic backgrounds, ages, and educational levels". The main problem is that when the assistance is unwanted, it is not sustainable. Ernesto Sirolli shouts with passion when he says: "if people do not wish to be helped, leave them alone. This should be the first principle of aid." (Sirolli, 1999).

The inefficiency of the formalization approach in resolving the environmental and socio-economic problems of the artisanal mining activities is also exemplified in Ghana. Historically in Ghana there was no investment in long-term economic plans and little emphasis was put on income distribution and social improvement. Almost all foreign exchange earnings were depleted by 1970 (Hilson and Potter, 2005). In Ghana, artisanal gold mining has expanded dramatically in recent years. The country is currently Africa's second largest gold producer after South Africa, with gold exports accounting for more than 40% of the country's total export earnings (Carson, 2005, Mate, 1999). Between 300,000 and 500,000 people are currently involved in artisanal gold production at all size levels. This represents two thirds of Ghana's total gold miners (Carson, 2005). Micro-mining provides a vital source of income to tens of thousands of people who have few means for survival in the formal economy (Carson, 2005). In 1989, as part of the restructuring of the minerals

sector, artisanal sector was formalised through enactment of Law 218 - *The Small Scale Gold Mining Law*. Over 600 co-operative and individual small-scale miners were registered (Akabzaa and Darimani, 2001). According to this law a miner should have all procedures made and all fees paid before starting to operate in an area. This leads firstly to disbelief from the miners in the fact that there is actually gold at the site designated by the government. Secondly, most of the miners cannot afford to pay in advance for the mineral title. The government announced on October the 24th, 2008, that it had set up 44 areas or 500,000 acres for the small miners, but the feasibility of having a mine has yet to be determined (Tschakert, 2007). Some examples show that when renewing the exploration license after 3 to 5 years, miners face more difficulty as preference is given to large companies; the artisanal miners are not able to stay in their operations any longer if a large company requests the same area. Because of this unfair law, a large number of miners still remains illegal in their previous operations or operate illegally in other areas with no belief in formalization. Numerous conflicts have been documented between companies and artisanal miners (Hilson, 2007).

Another example is from Brazil. Most gold deposits in the country are located in the Amazon, a very isolated and remote area (Figure 4-1). There are about 200,000 artisanal miners producing approximate 6 tonnes of Au/a (Sousa et al., 2011). Most artisanal operations are of small or medium size. Due to the remoteness of the gold deposits it is not easy for micro-miners to operate but there are some. The Brazilian government has been criticized by not paying attention to regulating MM and SGM and placing efforts only on legislation with poor or no technical support to the miners (Barreto, 2003, Sirotheau and Barreto, 1999, Procópio, 2011). As described by Sousa et al.,

(2011), legislation in the country is largely influenced by media and public opinion. Laws are usually written and approved in major cities with no previous consultation or field work with the miners. Without proper preparation, education, and enforcement, the laws about artisanal mining end up being not only useless, but in fact, an obstacle to development in remote mining localities. Sousa and Veiga, (2009) investigated the effectiveness of Law 97507/89 that forbids use of mercury and cyanide without previous licensing. The authors found that 99.3% of the assessed sites in Tapajos basin do not have any license to use mercury and cyanide but all miners work with these reagents routinely and with no major problems other than the high cost to buy them in the black market. To obtain an artisanal mining license, the applicant is required to obtain an environmental license issued by the State where the operation will happen. Sousa et al., (2011) note that Brazil is one of the few countries that have characterized environmental damage as a crime punishable with fines or jail. However a major criticism is that poor enforcement makes transgressors unafraid of punishment.

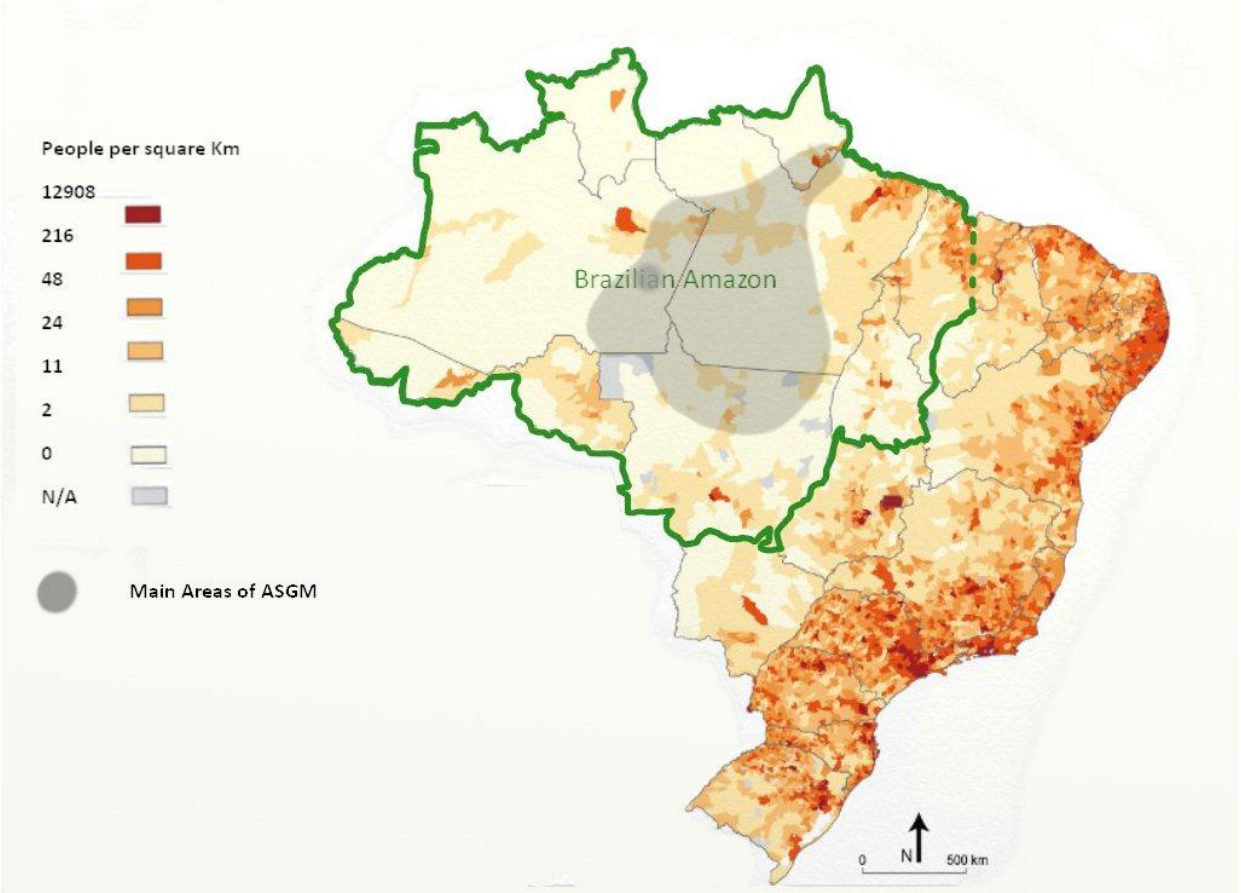

FIGURE 4-1. BRAZILIAN AMAZON, POPULATION DENSITY AND ASGM MAIN AREA

There is in Brazil and other developing countries a disparity between law and reality. Most of the inconsistency comes from concepts such as "sustainability" and "clean processes" created and used in the developed world which are not adapted or even understood in developing countries. Good laws are important but the intricate structure of the laws and the lack of enforcement make them useless. More attention should be thrown at training and organization so the enforcement of laws can be done effectively.

If enforcement is ineffective, the law will not produce the expected results. In addition, the fact that a region is vast and uninhabited, such as the Amazon, a large budget must be invested in order to create a useful and working enforcement system. This is a double-edged sword problem, since the region

is so large, probably the income generated by tax revenue would not be enough to pay for the necessary enforcement (Sousa et al., 2011).

As existing regulations and legal system have failed to resolve problems associated with artisanal miners it is clear that new and simpler processes are needed. Problems such as illiteracy, lack of confidence in the government, fear of bureaucracy, etc. must be addressed before requests, in terms of legalization, are created. Furthermore, technical assistance is needed whether the miner is legal or not, because impacts can be caused by all types of miner. As mentioned before the legal approach cannot come before the education and organization of the miners.

5. THE TECHNOLOGICAL APPROACH

A practical approach to the environmental problems caused by artisanal mining is one that fights the causes and consequences of the problems – the poor techniques used by the miners. Despite restrictions in law, artisanal mining will occur, particularly in areas where law enforcement is nonexistent. In addition, enforcement is not simple since the majority of the artisanal operations are located in the remote areas of developing countries (Hilson, 2002b).

Introducing or modifying technologies is the basis of the technological approach. It tries to solve – or alleviate – the environmental and health contamination by changing miners' behaviour or methods. New technologies must be cleaner than the ones being used by the miners, but it is of extreme importance that they are at least as profitable and simple as the ones being substituted. Artisanal miners usually do not consider environmental and health impacts to be as important as the profit generated by their operations. Technology for them is a simple tool and they rarely pay attention to simple concepts such as technical assessment of the gold recovery. Production is determined by the volume of material processed and not by the efficiency of gold extraction.

An example of the technical approach to resolve the pollution from AGM was the Global Mercury Project (GMP) conducted in Brazil, Indonesia, Lao PDR, Sudan, Tanzania and Zimbabwe, with financial support from the Global Environment Facility (GEF) and executed from 2002 to 2008 by the United Nations Industrial Development Organisation (UNIDO). In addition to the policy side of the project, the focus was to bring technical solutions to

reduce mercury emissions and releases to artisanal miners' attention. About 300 trainers and 30,000 miners and community members were trained in the six pilot countries. The reach of the pilot sites worked by the GMP team was, however, minuscule compared with the magnitude of the problem on a global scale. (McDaniels et al., 2010).

A recent initiative was proposed by the Norman B. Keevil Institute of Mining Engineering at the University of British Columbia, the Universidad Tecnica de Machala, Ecuador and the Dept. of Mining Engineering of the University of São Paulo, Brazil. This initiative is also supported by the US Department of State. The ITCAM (International Training Centre for Artisanal Miners) in Ecuador is a project-based, educational-research initiative to be created in the town of Portovelo, Ecuador. This unique start-up concept will be unlike any other technical assistance program in the world. The main proponents of the ITCAM-Ecuador project intend primarily to provide miners with education in ore processing, environmental impacts, economics, and the social aspects of mining, while at the same time serving as an international hub to train miners from neighbouring countries. This project is a response to the difficulty of training and working with miners without continuous presence of trainers in the field.

Technological education is not always enough to drive significant changes in the artisanal miners' techniques. Capital is needed as well. When a relatively costly "new" technology is considered, the idea of offering micro-credit to miners is raised (Hinton, 2006). The bureaucratic process to obtain formal credits usually makes the micro-credit hard to obtain or ineffective at creating substantial change. However, successful cases have been observed in

Namibia and Mozambique, where low interest rates, slow payments and low levels of bureaucracy encouraged formal micro miners to improve their operations (Hinton, 2006, Siegel and Veiga, 2009). Organizing cooperatives improves their purchasing power or capacity to obtain credit to solve technological problems. The introduction of technological improvements must coincide with the constant presence of trainers (McDaniels et al., 2010) and the training program must earn recognition from the artisanal miners. In fact, maintaining presence in the community, promoting dialogue and participation as a means of building trust are critical steps for any community development work (Veiga et al., 2001, Homan, 2010). Xavier et al., (2013) suggest that any approach to working with the community needs presence in order to develop sustainable and trustworthy relationships with the locals. Absentee trainers soon are forgotten.

The Table 5-1 gives suggestions of the steps needed to implement a technical intervention program in a mining community. It is important to identify the community, how they are organized, what they lack and what can be used as a link to solve the identified sources of problems. Being known and trusted before, during, and after the intervention is fundamental in order to have the proposed solutions adopted and implemented by the miners. Considering that miners usually have no education, it is important to understand that knowledge transfer is a slow process. As discussed before, the most important reason for a miner to change their way of working is to produce more gold; one should explore this aspect when introducing cleaner procedures. As presence is important for capacity building, trainers should always be available to assist miners; ideally, they should also have a similar cultural background to that of the miners.

Concept	Action
Know the miners	Identification of the miners' strength and weakness
Identify Problems	Identification of the main technical needs to improve production
Be known	Create links of friendship and trust with miners and their community members
Show off	Demonstrate the efficiency of the "new" technology to extract gold from the tailings of the miners; this creates credibility
Educate patiently	Start slowly the education process, explaining why the "new" technology works better
Do not complicate	Use technical concepts and language that are familiar to local miners
Leave to them the decision	Demonstrate more than one "new" technology and leave with the miners to decide what is good for them
Be flexible	Check the receptivity of the miners to "new" technologies and be prepared to demonstrate more than one option
Pick the right people	Training the trainers (select local leaders)
Gold is more important for them	Show first the economic advantages of the "new" technology and then show the reduction in environmental and health impacts

TABLE 5-1. SUGGESTED STEPS FOR INTERVENTION

It is of extreme importance to know how to approach and engage a mining community, but it is even more important to understand how different technologies and concepts work before introducing them as a technological intervention in any community. A myriad of proposed cleaner gold processing technologies are proposed in the literature. The next chapter investigates a few of these, selected by their being the most discussed in technical papers. Their application has been done (or proposed) in the majority of the artisanal mining sites.

6. MYTHS AND REALITIES IN GOLD EXTRACTION

Various methods aiming to eliminate or reduce mercury utilisation in artisanal and small-scale gold processing have been proposed. Most methods involve a concentration process since it is costly or impractical to perform the procedure on the whole ore. The list below, based on the works of Jønsson et al., (2009), Hilson and van der Vorst, (2002), Sousa et al., (2010), Vieira, (2006) and Appel and Na-Oy, (2012) compiles the methods studied in the subsequent sections of this Chapter.

- Conventional cyanidation – Gold-containing mineral is mixed with a cyanide solution, which dissolves the gold that is either adsorbed on activated carbon in pulp or precipitated with zinc after liquid/solid separation. While cyanide itself is toxic and dangerous if not handled properly, it is not a bio accumulative toxin and its destruction is relatively easy to conduct. If performed with caution, the method is efficient, but requires considerable investment and skills.
- Intensive Cyanidation – Some companies have developed different methods to extract gold from concentrates using high concentrations of cyanide and an oxidizing agent. Ball mill intensive cyanidation is a simplified process used to leach concentrates in a ball mill in the presence of an oxidizing agent. A literature review and some laboratory tests are presented.
- Borax extraction – High grade gold concentrate is melted with borax. Borax forms a slag with oxides and silicates promoting the separation of gold by gravity. The process requires practice and a very rich gold concentrate. It is used in certain parts of the Philippines.

- Chlorine leaching – Gold concentrate is treated with a solution of concentrated hydrochloric acid and sodium hypochlorite. Subsequently, the solution is treated with reagents to precipitate gold. The method requires some skill, and access to required chemicals. The iGoli process, devised by the South African Institute Mintek is discussed.
- Electro-leaching – Basically the same process of chorine leaching, but chlorine is produced from sodium chloride in solution by means of electrolysis. Laboratory tests were conducted using this technology in order to evaluate its potential.
- Other lixiviants – Haber (Haber, 2013), Yestech (Yestech, 2013), and other manufacturers offer different alternatives to mercury and cyanide as a "green" gold-leaching agent. Commercial products are available as a panacea for the mercury pollution problem. Their composition though is not disclosed, therefore it is hard to investigate what are their real efficiencies and impacts. In addition, it seems that the strategy of these companies is to keep the miners as their perpetual clients using their proprietary reagents.

The main myth of all, however, is that the replacement of mercury use can be achieved by simply introducing an alternative chemical extraction process. In all these methods to replace/reduce mercury use, the main challenge is actually to apply an efficient way to **concentrate** gold by either gravity separation and/or flotation. As discussed before, even this simple change of applying amalgamation to concentrates rather than to the whole ore substantially lowers mercury used and lost.

6.1. CONCENTRATION IS THE SOLUTION

Myth: Concentration loses gold….amalgamation of the whole ore is better.
Reality: It is not easy to concentrate gold…when you don't know how to do it.

As mentioned before, amalgamation of the whole ore in ball mills, on copper plates, or spreading mercury on the ground or in sluices boxes is the main cause of mercury loss in AGM (Veiga el al., 2013). Concentration methods are important tools to improve gold recovery and lower emissions and discharges by avoiding a large amount of material to flow for gold extraction, but their introduction must be done gradually to complement education and capacity building.

Mercury is dispersed during the grinding process, loses its coalescence and so, small droplets are lost to the tailings. Concentration of the mineral(s) of interest is definitely the main step to reduce mercury (or cyanide) pollution. Concentration is achieved by exploring the inherent characteristics of both the gangue (undesirable) minerals and the minerals (or metals) of interest. In gold ore concentration for example, the typical methods used in gold concentration are:

- Gravity methods such as shaking jigs, tables, centrifuges and sluice boxes that use the difference of specific gravity between gold and different minerals. Gold, silver and other heavy minerals report preferentially to the heavy concentrate while the rest of the minerals to the light fraction. However, there is a limitation of size in gravity separation Except for the centrifuging, gold particles below 65 # (300 microns) usually are poorly concentrated by most of the gravity methods used by artisanal miners.

- Flotation is a common method of concentrating gold and gold bearing sulphides, and can extend size limitation down to 10 microns. It works by exploiting differences in the particular surface properties of different minerals (or metals). These properties differences are achieved with a reagent that adsorbs preferentially onto certain particle surfaces to render them hydrophobic. Bubbles injected into the system attach to these particles and so they float. Gold and sulphides float while gangue minerals remain in the pulp. Different reagents and conditions can be set to selectively float gold and different sulphides.

When considering a concentration method, it is important to note that no process is perfect, thus recovery will be lower as the concentration ratio – the initial mass divided by the concentrate mass – increases. If gold ore is concentrated, inevitably some gold is going to be lost to the tailings of such a process. Figure 6-1 shows a hypothetical graph that exemplifies the antagonistic behaviour of gold recovery and grade of the concentrate. A large mass of concentrate implies that the grade is low but the recovery is high and vice-versa. The choice of an operation point in this curve, in which a concentrate from a sluice box or centrifuge must be discharged, is an economic decision.

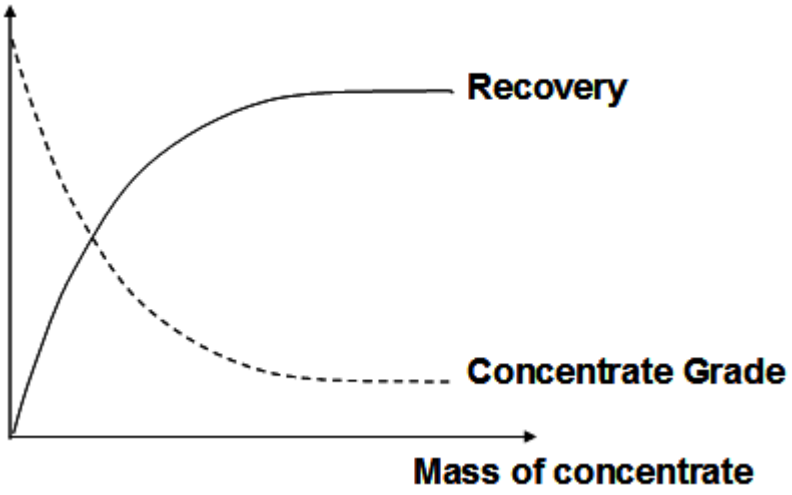

FIGURE 6-1. HYPOTHETICAL RELATIONSHIP BETWEEN MASS OF CONCENTRATE AND GOLD GRADE AND RECOVERY

Introducing concentration principles to artisanal miners is not an easy task as many miners believe that concentration is a waste of time and gold losses are too high. In fact, this is true when miners use inadequate grinding, classification and concentration processes. This lack of knowledge together with bad past experiences strengthen these perceptions. This is the rationale behind the amalgamation of the whole ore: it works; it is cheap; and the miner immediately sees gold in his hands. It is important, therefore, to show that, in a well-developed concentration process, reducing the mass of the material to less than 10% of its original mass (for example) will lower by a larger extent the cost of leaching and/or amalgamating the concentrate. It is also important to show that the concentration is not as inefficient as they think. At the end of the process – a well conducted one – there will be more money being generated when concentrating the ore and, as a positive side-effect, less mercury or cyanide will be used. Although this is routine for mining engineers working in large companies, it is a challenge to introduce these concepts to AGM workers, because of the lack of education in the sector.

Gold extraction is usually easier and more effective when dealing with a small mass of concentrate due to the presence of less impurities and the higher gold grade in the material. More technical knowledge is needed to understand when a concentration process is efficient and how to use a more complex piece of equipment.

Liberation of the mineral of interest from the gangue is not part of the vocabulary of artisanal miners. This idea is probably inherited from alluvial operations, where in most cases (but not always) gold is naturally-liberated in the water stream. Working with secondary gold deposits, most miners believe that gold is also liberated. However, for secondary (alluvial, colluvial and eluvial) deposits, gold particles are either partially liberated or completely trapped with gangue minerals (Spiegel and Veiga, 2010). It is rare to see any AGM grinding these types of ores. There are many different types of grinding systems used by AGM. Ball mills are not very popular as they are expensive and require skill to set up and operate properly. When grinding ores in a ball mill, artisanal miners rarely use classification (screening or hydrocycloning) which lowers grinding efficiency. It is common to find this misbelief among miners that a circulating load is a bad thing for milling. They believe that returning between 100 to 400% of the material to the mill feed is a waste of equipment. However, a high circulating load can maximize output rates at a narrower range of product size (Mular et al., 2002) that is at a greater degree of liberation. The classification size must be investigated, since it is ore dependant. Technical knowledge is therefore necessary in order to obtain an efficient milling system.

Many miners, in particular in South America and West Africa use hammer mills to grind hard rock below 0.5 to 2 mm, but these mills were actually devised to grind soft rocks such as limestone. As a result, as observed in Venezuela, hammers must be changed after grinding 1.5 - 2 tonnes (Veiga et al., 2005) because of excessive wear.

In Africa, notably Zimbabwe, the use of stamp mills is also common. Their capacity is very low, ranging from 0.2 to 0.5 tph, depending on the hardness of the ore. The product size is discharged through a 0.6 – 0.8 mm mesh, usually too coarse a size to obtain enough gold liberation (Veiga, 2004a, Veiga, 2004b, Veiga et al., 2009). Many ores are not well liberated until 300 microns and some only are liberated at very fine sizes (30 microns or less)

Unlike other minerals, gold does not need to be completely free to be concentrated. Gold associated with quartz can increase the specific gravity of the particle that reports to the concentrate. Unfortunately, when using amalgamation, as the gold is not liberated it is not trapped by mercury and lost with the amalgamation tailings, but when concentrated, these particles can be reground (and gold liberated) at a relatively low cost.

6.1.1. GRAVITY CONCENTRATION

Different pieces of equipment use different methods to promote gravity separation. Sluice boxes use a flowing-film, centrifuges use a fluidised bed in a spinning bowl and panning relies on settling time difference between particles allied to a flowing-film separation (Burt, 2000, Burt, 1984, Wills and Napier-Munn, 2005).

Panning is one of the oldest and most primitive methods of gravity concentration; panning was probably used to concentrate gold in the Egyptian Empire around 1500 B.C (Klemm et al., 2001) and classic books on mining – Natural History by C. Pliny, 79 D.C. and De Re Metallica by G. Agricola, 1556 – illustrate its use. Due to its simplicity, panning is used worldwide by MM together with sluice-boxes, a type of equipment probably even older than the gold pan (Klemm et al., 2001). Panning is however a slow process. As observed in Portovelo, Ecuador, one person is able to pan around 200-300 kg of material per day. Even with great ability, the panner loses a substantial amount of gold (mainly fine gold) to the tailings. Another downside of this technique is that amalgamation of the gold contained in the panned concentrate usually is conducted inside the pan right after the concentration. This practice allows mercury to be lost to the environment, when panners throw away the amalgamation tailings.

While panning is usually conducted by MM, sluicing is often used by larger artisanal operations, at a capacity as large as 500 tpd (Burt, 2000). In a sluice box, a flowing-film process washes away the unsettled particles, while the denser ones settle at the bottom of the sluice in the riffles, carpet or any other lined surface. There is some turbulence constantly freeing less dense particles from the bottom to be carried away. Sluices, as any other kind of gravity separation equipment, work better when feed size is in a narrow range and the gold particles are sufficiently liberated. With respect to efficiency, Burt, (1984) has identified that the majority of gold, especially coarse particles (above 0.25 mm), is trapped in the first meter of the sluice while finer particles, retention of gold in the following meters is low. The same author identifies an ideal slope to be in the range of 10 – 15° depending on the material.

Artisanal miners, however, do not understand these principles and/or do not have access to a proper comminution system. Moreover, it is also a common misbelief among AGM that the longer the sluice the better it will perform to trap fine gold, when in fact, what is recovered in the lower end of the sluice simply contaminates the concentrate, lowering its grade.

As sluices can process large amounts of material, bad practices can lead to large amounts of contamination. In the Brazilian Amazon, where sluices are fed colluvial or alluvial ore at rates of 500 tpd it is a common practice to spread metallic mercury in the water basin where the hydraulic monitor operates. There is a belief that mercury will "find" gold *in situ* and then carry it to the sluice where it will be trapped. More likely, however, the mercury will be suctioned by the pump and then trapped in the riffles, where amalgamation may then occur. Inevitably this process by using too much mercury loses much of it to the environment (Araujo, 2006).

Because operation and construction are simple, sluices are good options to introduce at mine sites where no concentration takes place. Building a sluice box is inexpensive. The main drawback is that gold recovery depends on the characteristics of the ore, the shape of the gold particles, and the degree of gold liberation. However, this is a definite improvement over amalgamation of the whole ore. The efficiency of such a piece of equipment varies with the mineralogy and is higher for coarser rounded gold particles such as those found in alluvial deposits. Efficiency drops with very fine or flattened gold particles, which tend to follow the water flow instead of sinking.

At the processing centers in Portovelo, Chilean mills discharge ground ore to a very long (around 10 m) and low slope cement sluice (~5%) covered with

cotton carpets. While this kind of mill is believed to be an appropriate technology[5], that is a cheaper alternative to ball-mills with relatively good performance, this type of sluice construction does not recovery noticeably more gold than using a shorter sluice. As a result, much more material is retained in the carpets. Miners discharge and clean the carpets every hour, generating a large amount of concentrate (between 200 and 300 kg per day). Miners need then to further reduce the mass of the concentrate by panning in order to obtain an amount where manual amalgamation is possible (usually 15 to 20 kg). Amalgamation is then conducted with addition of brown sugar, believed by the miners to increase gold extraction by amalgamation. However, the tailings may contain as much as 74% of the initial gold, according to Veiga et al., (2009). The amalgamation tailings are mixed with the gravity separation tailings to be leached with cyanide in the processing centers. Some miners conduct amalgamation of the whole sluice box concentrate in *chanchas* (small ball mills). While this probably helps recover more gold, definitely more mercury is lost to the tailings as mercury loses coalescence and is dispersed. As observed in Colombia, using similar amalgamation of the whole material in small ball mills, on average 46% of the mercury is lost with tailings and values as high as 82% were observed (Cordy et al., 2011).

Other miners in Portovelo leach the sluice box concentrates in rented cement tanks using a simplified Merrill-Crowe process (section 6.3.1).

[5] According to Schumacher, (1973), an appropriate technology is one that is affordable to be acquired and maintained by most of the workers, being therefore a means of generating wealth among the workers instead of retaining it in the hands of the richest owners.

At Cazaderos plant in Ecuador, a sluice box is used to further concentrate the centrifuge product from 1200 ppm/Au to 3200 ppm/Au. This process achieves a recovery of about 50% but this is not a loss in the plant, since the gold-rich tailings flow on to a CIP circuit. Figure 6-2 and 6-3 shows the centrifuge and the sluice at Cazaderos.

FIGURE 6-2. KNELSON-TYPE CENTRIFUGE

FIGURE 6-3. SLUICE AT CAZADEROS

Still not very popular, but also seen in Portovelo, are centrifugal concentrators. Some of the plants processing their own ore have introduced this technology in order to improve gold recovery by avoiding overgrinding of the metal. As a positive impact, this equipment also provides the companies with a faster return over the ore being processed, since usually most of the gold is recovered at this first step of concentration. The Canadian company Sepro manufactures a small sized (2 tph capacity) low cost centrifuge, Icon, which is sold in Ecuador for USD 8,000.

Sousa et al. (2010) studied a plant processing 28 tonnes of quartz-rich tailings from an AGM operation in the Brazilian Amazon, discharging the Icon concentrate every 30 min, a production of 41 kg/day with a gold recovery of 50%. In three passes of the tailings through the centrifuge the recovery was close to 65%, an increase of 30%.

A less efficient centrifuge is also popular in Portovelo. It is a rough copy of the Knudsen centrifuge which is a centrifuge that operates with no fluidised bed, processing 3 tph, at 30% solids running at very low centrifugal force (10 G compared to 150 G of an Icon centrifuge).

FIGURE 6-4. ICON CENTRIFUGAL CONCENTRATOR (SOURCE: WWW.ICONCENTRATOR.COM)

6.1.2. FLOTATION

At Cazaderos and various other plants processing their own ores in Portovelo it is common to observe flotation being applied as well. As gold in the region is usually associated with sulphides, notably pyrite, chalcopyrite and bornite, applying this technology makes sense. This process also allows separation of chalcopyrite and bornite (with some associated gold) that is sold as a copper concentrate directly to smelters in Peru or China. It is estimated that in the last 2 years, around 40 processing plants in Portovelo have introduced flotation cells with capacities ranging from 10 and 200 tpd of ore. This has reduced the use of mercury and cyanide in the region by perhaps 25%.

Most of the flotation equipment in the processing plants in Portovelo is provided by Chinese and/or local manufacturers at a cost as little as 1/5 the cost of comparable North American equipment. Because of their simplicity, circular flotation cells are also popular in the region and used for dual-purpose: as a conditioning tank and a first flotation step – either rougher or bulk – before the mechanical cells.

The main drawback of flotation as used today in Portovelo is the need for an operator to constantly inspect flows and levels in each cell. As flow rates are hardly stable in the small plants, an operator must continuously adjust the cell ports. In order to solve this problem a modified Jameson column flotation cell is being developed by a Brazilian company (DPSMS) to be tested in Portovelo. The main benefit of this equipment is the electronic control of flow to manage froth depth and tailings removal rate, relaxing the need for constant presence of an operator.

6.2. AMALGAMATION OF CONCENTRATES

Myth: Amalgamation can be easily replaced by another process.
Reality: The first step before elimination is reduction of mercury use.

Although most authorities and environmental researchers propose total elimination of mercury utilisation in AGM, assessment needs to be conducted beforehand to evaluate if this is indeed possible. Depending on the place, level of education, and capital available, it may be impossible to totally eliminate mercury and the half-done transition potentially generates more contamination than mercury alone, as in the case of mercury-cyanide releases, for example.

A suggested approach in this case, is the introduction of methods that make amalgamation more effective and protect both the operator and the environment against mercury emissions and releases.

In MM, as the amount of processed material is small and concentration by panning/sluicing is usually preferred, the problem consists in containing the mercury in the extraction process, with no discharge to the environment. As a positive side-effect this mercury can be recycled, saving money and generating an attractive financial incentive for miners to adopt these techniques.

Amalgamation is not a complicated process but gold must be exposed from the gangue to be effective. Mercury combines with gold to form a wide range of compounds from $AuHg_2$ to Au_8Hg (Taggart, 1945). The main problem causing mercury loss in amalgamating concentrates is "sickening" and loss of coalescence (or "flouring").

According to Beard, (1987), "sickening" is caused by mercury oxidation or impurities such as oil, grease, clay minerals, sulfates, and sulfides on mercury

surface while "flouring" is the division of mercury into small drops that can be caused by "sickening" or by mechanical forces such as grinding. "Flouring" increases the susceptibility of the droplets to "sicken" as it increases the specific area of the metal. Curiously, Beard, in 1987, actually promoted the use of mercury in ball mills and copper plates to amalgamate the whole ore and suggested to use amalgamation in conjunction with cyanidation.

The mechanisms of mercury "sickening" are not well understood, as there may be many causes for the lack of coalescence. In North America, prospectors recommend many methods to "clean" mercury including filtration of liquid mercury on a piece of cotton using a syringe to force the metal to pass through. This retains the oxidation product and other solid films on the surface of the sick mercury (Prospector's Paradise, 2012). It is believed by the miners in most AGM regions of the world that addition of "amalgamation aiding reagents" avoids sickening and flouring. In Portovelo, brown sugar or molasses are added during the amalgamation process. In other parts of the world, miners add lemon juice, caustic soda, baking powder, guava leaves, urea, cyanide, urine, or detergent (Veiga et al., 2006). There are no studies proving that such reagents work in increasing the capacity of mercury to hold together and avoid formation of droplets and neither are there studies about how these reagents work. Alterations of the mercury surface tension or adsorption of such reagents acting as surfactants are believed to be possible responsible mechanisms that prevent mercury coalescence.

Pantoja and Alvarez, (2000) suggested an electrolytic process to form sodium amalgam, making mercury more efficient in the gold amalgamation. This is similar to the industrial process to manufacture caustic soda. Actually, the

use of "charged mercury", as this process is popularly known, is recommended by the Nevada Prospectors Association (Ralph, 2013). Pantoja and Alvarez (2000) claim that flouring is actually promoted by sickening of the mercury due to formation of HgS. When mercury contacts sulphides, HgS acts as an inhibitor of amalgamation according to Gaudin (1939). The availability of S^{2-} is not explained by the authors but can be inferred by the presence of sulphides frequently associate with gold in ores.

The main objective of Pantoja and Alvarez, (2000) was to study the influence of mercury activation ("charged mercury") over gold recovery and mercury loss in the amalgamation process. Mercury is activated in an electrolytic process with a 10% NaCl solution. Metallic mercury is connected with a copper wire to the negative pole of a 12-volt battery and a rod of graphite that can be obtained from an old radio battery connected to the positive pole for 15 minutes (Figure 6-5). The activated mercury is much more coalescent than the sick mercury. As mercury and mercury salts are soluble at high pH values (Pence and Rarton, 2003; Meech et al, 1998) and sodium-amalgam forms sodium hydroxide in water, this is likely the mechanism by which the surface of metallic mercury is cleaned from oils and oxidized mercury. In fact, the effectiveness of the "charged mercury" process to amalgamate metals is noticed right away. When the negative charged copper wire is introduced in the metallic mercury it is dark, oxidized, and amalgamation does not occur. After some minutes of electrolysis, the wire becomes covered with a shiny silvery colour indicating that the amalgamation of mercury and copper is happening. This actually impresses the miners who witness the amalgamation power of the "activated" mercury.

FIGURE 6-5. MERCURY ACTIVATION PROCESS (PHOTO: R. SOUSA)

Pantoja and Alvarez, (2000) also investigated gold recovery and mercury losses when ore (or concentrate) was washed with detergent and lime prior to amalgamation. The results obtained were impressive, showing an increase of recovery from around 60% (using regular mercury) to 92% (using activated mercury). At the same time, mercury losses[6] were reduced from Hg_{lost}:$Au_{produced}$=0.5 to 0.086. When mercury is highly coalescent, there is less droplet generation to be carried with the tailings. Gaudin (1939) discussed the role of the amalgamation inhibitors. He mentioned that soluble substances can be chemically bonded to the mercury surface and this tends to flour the mercury. Actually, he suggested either the use of sodium cyanide or the formation of sodium amalgam (like in the Pantoja's method) to avoid the lack of coalescence. Priester and Hentschel (1992) mentioned that oils and greases form a film around mercury reducing its coalescence. Washing the ore with detergent or lime removes these oil or oxidation films.

[6] Mercury loss is the amount of mercury actually lost in the procedure, i.e., the mercury that or emitted to the environment after amalgamation is conducted. There is no use in identifying only how much mercury enters the system, since recycled mercury does not generate contamination (Veiga et at., 2004).

It is important to recycle mercury in artisanal mining operations. As the price of the metal is increasing, it seems that more miners are paying attention to this. Recycling also reduces the Hg_{lost}:$Au_{produced}$ ratio (Veiga et al., 2004).

The use of retorts has been extensively promoted in intervention programs in AGM regions. Jønsson et al., (2009) investigated in Tanzania, where the Hg_{lost}:$Au_{produced}$ ratio ranges between 1 and 2, why it is so hard to introduce retorts to artisanal miners. Tesha (2003) discussed the same issue and concluded that the main incentive to use a retort is financial, which is dictated by the difference between the cost of mercury and gold. Currently the international price of one kilogram of mercury (USD 95) is around 2 grams of gold (USD 104 in March 21, 2013). Even when sold in AGM sites at prices 3 to 4 times higher than the international price, mercury is still a cheap reagent when compared with the price of gold. This low price does not encourage miners to use retorts.

In the work of Jønsson et al., (2009) they pointed that the main reason why retorts were not fully adopted in an AGM after being introduced was the lack of presence of trainers at the site. This is justified by the fact that, after encountering technical problems such as gold becoming stuck on the retort or gold becoming red after contact with the retort, miners will usually stop using the device because they do not know how to solve these problems. The retort used in their project was a cheap-to-build model invented by Professor Raphael Hypolito (Veiga et al., 1995). As shown in Figure 6-6, the device consists of a bent piece of steel water pipe welded to a curve where a cap is threaded. Heat is applied to the cap where the amalgam is located and the end of the pipe is submerged in water such that the mercury vapours to

condense in the water recipient. Although very simple, this device is able to recover 95% of the evaporated mercury during the amalgam burning.

FIGURE 6-6. PIPE RETORTS – SOURCE: (JØNSSON ET AL., 2009)

The assessment study by Jønsson et al., (2009) investigated use of this steel water pipe retort with miners for 5 months and solved the problems they faced when using the equipment. There were two main problems encountered. Firstly, it takes a long time to evaporate all the mercury since the burning process occurs in a charcoal bonfire. This was solved by using a blower to raise the coal burning temperature. The second problem was the brown-reddish color of the retorted gold. While this does not influence the purity of gold, it makes the selling price lower. A partial solution to this was introduction of a small piece of stainless steel to support the amalgam inside of the retort. However this was only a partial solution since even when using the small holder, the gold does not achieve the characteristic golden color when produced. A contribution from one of the miners was crucial to solve this problem. He suggested lining the retort with ash before burning the amalgam. This generated a saleable gold with its common golden color. After 5

months all of the artisanal miners were using the retorts and all of them pointed out that the main benefit of the device was the ability to recycle mercury. The majority also noted the health benefits and a few understood environmental benefits. They recognized the higher consumption of charcoal when compared to open air burning but highlighted that the money saved in new mercury acquisition was more than enough to replenish the charcoal.

Veiga et al., (2006) listed a several reasons why miners end up not adopting retorts (Table 6-1).

Arguments	**Reasons**
retorts are expensive	miners do not know inexpensive options
it takes time (sometimes miners become vulnerable to bandits)	the retorting temperature is too low
experience is needed to operate retorts	lack of assistance in operation
gold is lost during retorting	with iron retorts, the amalgam is not visible so miners believe it has been lost
gold sticks to the retort crucible	this occurs when the temperature is too high or when the crucible is not lined with a thin layer of clay, talc or soot
recovered mercury loses coalescence	sometimes condensed Hg disintegrates into fine droplets
gold comes out brown from steel retorts	the cause is not known; it is probably due to a superficial reaction with iron

TABLE 6-1. ARGUMENTS FOR NOT USING A RETORT AT GMP SITES (ADAPTED FROM VEIGA ET AL, 2006A)

For most problems cited as an argument to avoid the use of retorts, there is a technical solution. The solutions for the problems cited in Table 6-1 are described in more detail below. It is important to note that a technical solution is not always the answer, in fact, as discussed before, the approach on how to implement new technologies to AGM is crucial.

Retorts are expensive – Although stainless steel retorts can cost as much as a few hundred dollars, it is possible to bring to the miners' attention retorts that can be built with water pipes (as mentioned above) and with kitchen bowls (Figure 6-7). These retorts do not cost more than USD 20, and this is easily paid back by the recycling of mercury.

FIGURE 6-7. KITCHEN-BOWLS RETORT IN COLOMBIA (PHOTO: M. VEIGA)

It takes too much time – Retort crucibles can be made with thinner steel walls that take less time to heat up. As many miners burn amalgams in bonfires, a manual air-blower bellows can also be built using a piece of pipe and a plastic bag to blow air to the fire to improve heat generation. The "*muvuto*" used by AGM in Zimbabwe is a plastic bag attached to a steel pipe that blows into a bonfire. The increased temperature reduced the amalgam burning time from 30 to 15 minutes (Veiga et al, 2006) (Figure 6-8).

FIGURE 6-8. AGM IN ZIMBABWE USING IMPROVISED BELLOWS TO INCREASE THE TEMPERATURE OF THE FIRE (PHOTO: S. METCALF)

Experience is needed to operate a retort – As everything else in mining, one needs to learn how to do an operation in order to do it properly. Training and presence is important so the difficulties do not drive the miners away from the technology. Miners are afraid to use pipe retorts as they can explode if at the end of the process the tip of the retort is not removed from the water tank. A vacuum is created at the end of the retorting process and water comes inside of the hot crucible creating high pressure (Prospector's Paradise, 2012).

Gold is lost during retorting – This is impossible to happen since the temperatures achieved in a retort (usually 500-600 °C) are not enough to vaporize gold, but the miners' perception is important. The kitchen bowls retort is ideally used when miners believe in this myth. The cover part of the retort does not need to be of glass, as this takes more time to cool down. However, it is important for the miners, at least once, to see inside of the retort during the burning process. With a glass bowl, the miners can see the amalgam col-

or changing from silvery to golden indicating that the mercury is evaporating and condensing on the glass cover.

Gold sticks to the crucible – Training would show the best materials to use between the amalgam and the surface being heated. Clay, ash or aluminum foil work well for this. In the case of aluminum foil, as in Portovelo, the *doré* comes out of the retort with pieces of aluminum attached. This goes to the borax slag when the *doré* is melted.

Recovered mercury loses coalescence – This problem is related to possible oxidation of the condensed mercury. The "sick" mercury forms many droplets. In fact most miners try to clean the condensed mercury with lemon juice or vinegar but when they cannot fully recover the coalescence of mercury, they end up throwing the mercury away.

Gold comes out of the retort brown in color – In fact, using cast iron retorts, gold *doré* comes out with a brown-reddish color. The retort crucible can be lined with ash or clay or aluminum foil to avoid this effect (Jønsson et al., 2009). Stainless steel or enamelled steel crucibles can also solve this problem and the *doré* comes out yellow giving better perception to the miners that the retort is working properly. Another solution proposed by MINTEK (2005) is to hammer the *doré* to form a yellow disk. In this process the surficial brown film is removed.

FIGURE 6-9. KITCHEN BOWL RETORT (PHOTO: GMP-UNIDO)

Most miners use a piece of cloth to squeeze off the excess mercury in the amalgam. The pathway of metallic mercury to humans through the hand skin is usually not relevant when compared with vapour, but it should be prevented as this can cause allergic dermatitis (WHO, 1991). Pryor, (1965) stated that "the wetting of gold by mercury is not alloying, but a phenomenon of moderately deep sorption, involving some interpenetration of the two elements". In this case it is possible to extract more mercury from the amalgam if more strength is applied in the squeezing process.

Then, instead of manual squeezing, it is possible to use a centrifuge to force the amalgam through a piece of cloth, where a "dry" amalgam is retained and the excess liquid mercury passes through. This kind of equipment can be easily built locally at low cost, using wood, pipes, a bicycle wheel or steel scrap. Figure 6-10 shows a manual centrifuge made of wood and PVC pipes. A hand squeezed amalgam contains 50 to 60% mercury, while an amalgam "squeezed" in a centrifuge can have as little as 20% of mercury (Veiga et al, 2006).

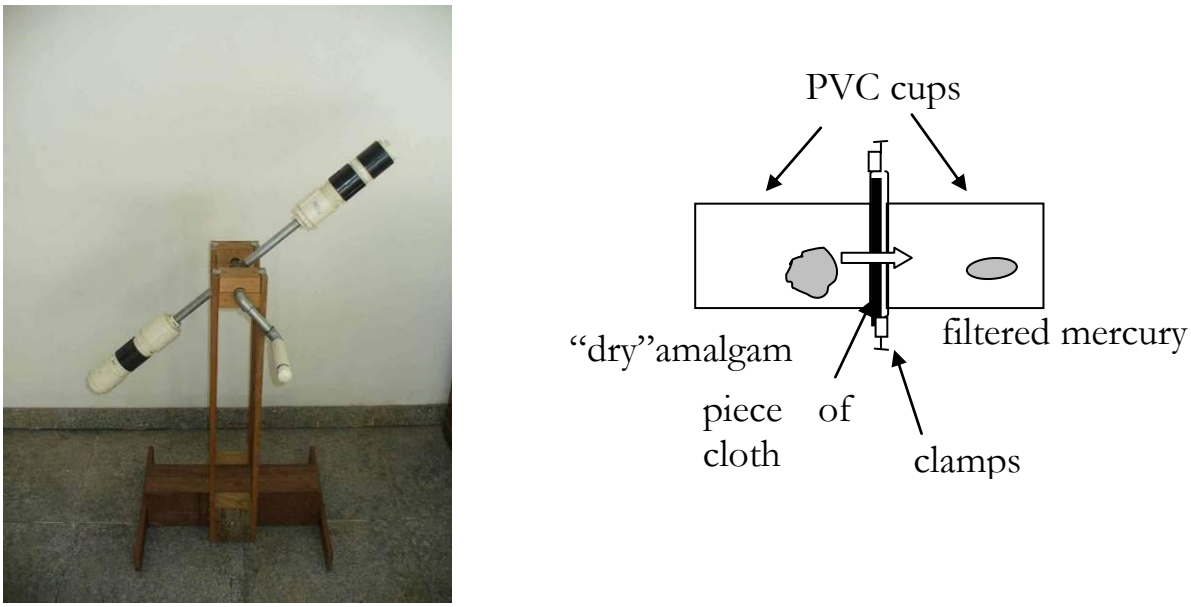

FIGURE 6-10. A MANUAL CENTRIFUGE TO FILTER AMALGAM (PHOTO: EDMUNDO VEIGA)

Amalgamation drums are also an interesting solution to avoid manual mixture of mercury with gravity concentrates. Their construction is easy and inexpensive, usually made from a plastic or glass cylindrical container with a tight lid and some heavier material inside, such as a steel chain, to revolve the pulp and amalgam (Figure 6-11). A crank handle attached to the container or to a supporting roll over a table makes it easy to spin the container without contacting mercury.

FIGURE 6-11. AMALGAMATION DRUM WITH A MANUAL LEVER (PHOTO: B. SUSILORINI)

Cleaner practices in the use of mercury should always be considered, in particular when there is no easy substitute available, usually due to lack of technical knowledge by the miners. In this case, reducing mercury emissions and discharges are definitely an important contribution, but measure should be implemented to prepare AGM for future replacement of mercury.

6.3. REPLACING MERCURY

Myth: Artisanal miners can always learn how to use other extraction methods
Reality: Without concentration, without education or without capital, all alternative methods are impractical, unavailable or expensive for artisanal miners

The process widely applied by owners of the processing centers to replace mercury use (usually aimed to improve gold recovery) is the cyanidation of gravity separation tailings. Unfortunately, miners do not have access to this technique as it requires access to capital and knowledge. The conventional cyanidation of tailings is discussed as follows.

6.3.1. CONVENTIONAL CYANIDATION

The extraction of gold by cyanide is governed by the following reaction, proposed by L. Elsner in 1846:

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

Several more recent studies have improved the knowledge of the intermediary steps leading to this reaction. However, this simplified view provides good understanding of the basics in the cyanidation process.

Cyanide leaching is conducted under controlled pH. The generation of HCN at pH lower than 10 is dangerous for the operators since such a gas is highly

toxic. In an industrial environment, lime is added to the leaching solution in order to control pH between 10 and 10.5 (Marsden, 1992).

This process also leaches other metals and the presence of sulphides (notably copper and iron sulphides) increases consumption of cyanide. Natural organic carbon also presents a challenge, by adsorbing gold cyanide complex ions from solution (similar to activated carbon).

A few common methods to leach gold by cyanide are used both as an industrial application and in AGM in various parts of the world. For example: CIP (Carbon-in-Pulp, where carbon is added to the pulp after leaching is conducted), CIL (Carbon-in-Leach, where carbon is added to the pulp during leaching) and vat leaching (where leached is conducted by flooding a vat with a circulating flow of cyanide solution).

Vat leaching is becoming a popular process in AGM sites. Although the process is relatively slow, some people use these percolation vats due to the low cost and simplicity. The material (usually tailings from gravity concentration and amalgamation) is kept soaked by a constant flow of solution. The solution is filtered at the bottom of the tank by a layer of sand and rags. It then passes through PVC columns with either activated charcoal or zinc shavings and returns to the vats.

A simplified Merrill-Crowe process was also observed in Portovelo at various sites. There are 160 Merrill-Crowe agitated tanks leaching gold from gravity concentrates with 1 to 5 g/L of NaCN with cyanide consumption of 1.5 to 5 kg of NaCN per tonne of tailings. The cement tanks usually have a leaching capacity of 14 m^3 of dry material per batch. The total leaching process takes from 3 to 5 days (depending on the operation) and the pH is not

well controlled (Velasquez, 2010). The process does not use filtration; the clarification of the solution is obtained by settling the solids in the cyanide tank by interrupting the agitation. Pulp then settles down and the clarified solution is siphoned through PVC columns with zinc shavings to precipitate the gold. This process should be conducted under vacuum, but the operators have no knowledge or access to equipment to do this properly. This simplified method results in losses of 5 – 10% of the leached gold, not extracted from the discharged solution (Veiga et al., 2009). Recycling this solution would minimize this losses.

In different countries of the world AGM uses different procedures for cyanidation, accordingly to what is available in the locality and the miners' capital. In remotes areas of the Brazilian Amazon, the operators use mostly vat leaching (Sousa et al., 2010), whereas in less isolated areas in Ecuador CIP or CIL in agitated tanks is more popular due to more knowledge and capital.

In Portovelo, there are 225 leaching tanks using CIP in many processing centers to leach tailings from the gravity concentration left by the miners. These tanks range from 30 to 40m³ in size, taking between 15 and 20 tonnes of material per batch. Plants have anything between a single tank to as many as 8 tanks in series with carbon in counter-flow. Sodium cyanide consumption is between 1 and 4 g/kg of ore, in extreme cases, leaching bornite-rich material, may leadto consumption rates as high as 15 g/kg of ore. Gold-loaded carbon desorption is conducted at 90 °C, in 10-20 g/L of NaCN and 6.4% ethanol solution (Velasquez, 2010).

The reality in AGM is that transitioning from amalgamation is rarely conducted in one step. It is common to observe mercury amalgamation being

conducted in part of the material, followed by further extraction by cyanide leaching being conducted on the mercury contaminated tailings. There is a myth among miners that amalgamation is still needed, either because it will recover more gold than cyanide leaching alone, or because it is important to have gold in their hands as soon as possible. Indeed, mercury amalgamation is the fastest method to produce gold, but recovery is low. Owners of the processing centers take advantage of this and quickly give gold to the miners using amalgamation while making more money with the gold left in the tailings.

Cyanide leaching is the process typically applied by large-scale mining companies and it has great financial advantage (in recovery, costs and effluent treatment) over mercury amalgamation. In industrial operations, Eissler, (1888) reported that the usual extraction achieved using mercury is 60% while, currently observed extraction achieved by cyanidation is usually higher than 90%. In fact, AGM extract much less than 60 percet of gold with amalgamation even from gravity concentrates. Veiga et al., (2009) have reported recovery of 26% of gold when miners from Portovelo amalgamated concentrates from sluice boxes.

Transition from mercury amalgamation to cyanide leaching in AGM is usually incomplete. This has the major implication in terms of toxicity of the mercury cyanide complexes produced. While cyanide leaching is clearly the most suitable process to extract gold, the transition from one technology to another must be introduced carefully.

More and more companies processing their own ores in Portovelo, are concentrating gold and copper minerals (by gravity separation and/or flotation)

prior to the CIP step. This is partially explained by the high content of chalcopyrite and bornite in the ore that is sold to smelters abroad. As an intelligent decision, gold and pyrite (usually a gold bearing sulphide in the region) are also floated separately, lowering the amount of material to be leached. One company (Soderec) which currently conducts cyanidation of 150 tpd of the whole ore, is now building a flotation plant to process rich-copper ores from its deposit. Old tailings are also planned to be reprocessed.

An intervention should first address the beliefs and the reasons behind the technology being applied and work together with the people involved in understanding the best way to use the new methods. In the examples cited above, technical training is needed in order to show that there is no gain in recovery by using amalgamation followed by leaching. Financial assistance and education can be used to avoid the necessity of "quick money" from amalgamation. The mercury-cyanide contamination generated by the processing centers is a far more complicated problem, it has social roots, a deeper investigation and interaction is needed to organize the business to operate in a less environmentally harmful way. Miners have no access to proper equipment and at the same time they are deprived of the opportunity to make enough money to invest in their own future processing facilities. The contamination problem, the generation of mercury cyanide and other cyanide complexes, was discussed before, and these are direct consequence of this kind of labour division.

6.3.2. INTENSIVE CYANIDATION

One of the limitations of cyanide leaching is the rate of gold dissolution controlled by the low dissolved oxygen content in the solution. This level de-

termines the maximum gold dissolution rate in the presence of enough cyanide in solution. Kudryk and Kellogg (1954) described how the dissolution occurs:

$$4Au + 8CN^- \rightarrow 4Au(CN)_2 + 4e^-$$

$$O_2 + 2H_2O + 4e^- \rightarrow 4OH^-$$

$$4Au + 8CN^- + 2H_2O + O_2 \rightarrow 4Au(CN)_2^- + 4OH^-$$

Under atmospheric leaching conditions, the dissolved oxygen content of a leaching solution must therefore be maintained high to maximize the dissolution rate of the gold. Kudryk and Kellogg (1954) also showed that the reaction is not any faster by operating at cyanide concentrations above 0.01% NaCN. However, practical levels for safe operation are usually 0.025% NaCN, in order to be sure that the process is not run at levels lower than 0.01%, since cyanide is consumed by other species in the ore.

Some companies have developed pieces of equipment for gold extraction using intensive cyanidation. While these companies offer a "solution package", the process is highly dependent on proprietary reagents and spare parts. Most processing plants dislike this dependency and are afraid of a shortage of reagents or technical assistance.

Gekko Systems is an Australian based company that manufactures a system called Continuous InLine Reactor. It conducts multiple steps in one single piece of equipment: a dewatering cone receives the gold concentrate slurry and lowers its water content to around 30%. After thickening, the pulp flows to a reactor drum where extraction takes place. Then a dewatering cone reduces pulp water content and washes it by counter current decantation. The

pregnant solution is sent to an electrowinning cell, where gold is deposited onto the cathodes and the solution is re-fed to the reactor drum. The use of a reactor drum makes aeration of the solution very efficient, increasing the leaching rate. Gold extraction as high as 99% is reported by the company (Gekko, 2003). Another solution implemented by the company is the elimination of activated carbon or zinc shavings use to recover gold from the cyanide solution, since it is directly deposited in the electrowinning cell.

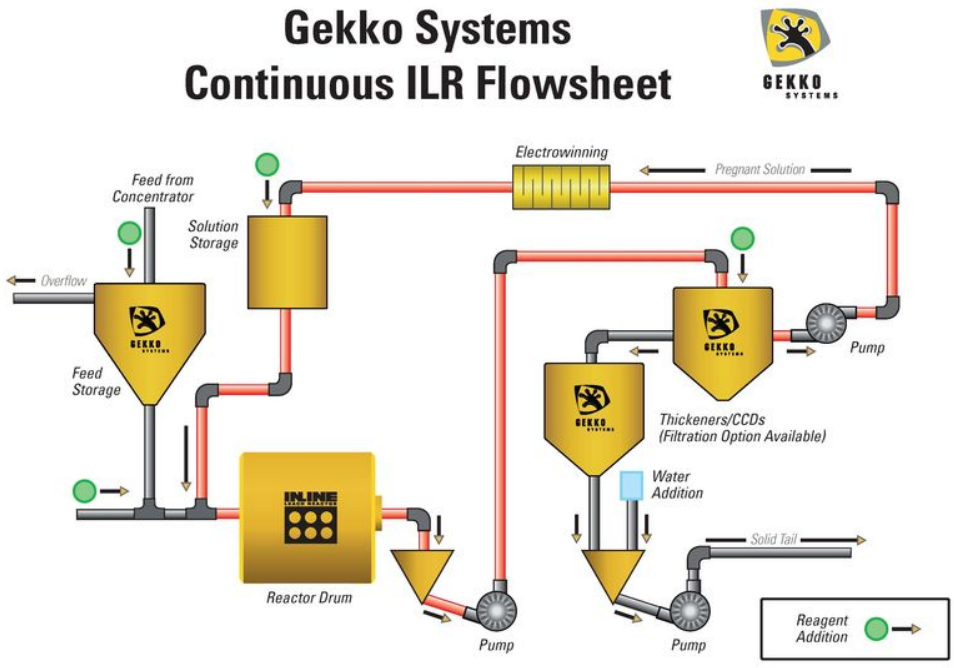

FIGURE 6-12. GEKKO INLINE REACTOR FLOWSHEET (WWW.GEKKOS.COM)

Acacia is a similar product, manufactured by Knelson concentrators in Langley, BC, Canada. It follows the same principle of thickening of gravity concentrates followed by an intensive leaching and subsequent electrowinning of the pregnant solution.

FIGURE 6-13. KNELSON'S ACACIA SYSTEM (WWW.KNELSONGRAVITYSOLUTIONS.COM)

Veiga et al. (2009) proposed an intensive cyanidation method for small-miners using a simplified methodology that does not require expensive equipment. The methodology suggests that the concentrate generated by a small-scale plant could be intensively leached by cyanide, by adding an oxidizing agent (such as H_2O_2) in a small ball mill. Intensive leaching uses more cyanide per kilogram of material, as the operation is conducted with a strong oxidizing agent on concentrates that consumes cyanide. On the other hand, as the gold grade in the concentrate is high, the cyanide spent per mass unit of gold extracted is not much higher than in conventional cyanide leaching.

Grinding, leaching, and gold adsorption onto activated charcoal take place in different steps when regular cyanidation processes are applied. In the proposed method, the material is not finely ground avoiding high costs in comminution, which is usually the largest expenditure in any mineral processing plant. The non-liberated gold can be concentrated and the particles reground later to expose the gold to cyanide. The gravity or flotation concentrate is added to a batch ball mill, where cyanide, H_2O_2 and lime (to regulate the pH), are added to a pulp of 35% of solids. A sturdy perforated PVC cap-

sule filled with activated charcoal inside a nylon screen is also added to the mill. The material is ground with steel balls for 24h in the presence of the capsule. Another possibility to avoid damaging the capsule is to previously grind the material with cyanide solution, remove the steel balls and add the capsule with activated charcoal. The concentration of H_2O_2 and NaCN are usually 0.3 g/L and 20 g/L, respectively. The amount of activated carbon is equal to 60 g per gram of Au contained in the concentrate feed.

Sousa et al., (2010) achieved a gold extraction equal to 95% using the method described above in a SGM operation in Brazil to recover gold from old tailings. The gravity separation recovery with an Icon centrifuge was 50% due to lack of a grinding circuit. The mill-leaching process achieved more than 95% gold extraction in 24 hours. With further improvement in gold concentration by adding a grinding process as well as another centrifuge in series and flotation of the gravity tailings, the recovery is expected to increase dramatically, based on the recoveries observed in Ecuador for a similar flowsheet.

FIGURE 6-14. PVC CARTDRIGE IN A BATCH BALL MILL (SOUSA ET AL., 2010)

In spite of recovering three times more gold than by amalgamation, in Ecuador the process is difficult to introduce because, as explained before, the tailings are left as payment to the owners of the gold processing plant.

Using the same process applied by Sousa et al. (2010), the author of the present work has evaluated gold recovery for the centrifuge concentrate obtained in Cazaderos plant in Portovelo. As mentioned before the concentrate from centrifuge is further concentrated in a low angle sluice box and then amalgamated in "*chanchas*". The study on amenability of the ore to intensive leaching aimed to propose a new, mercury-free method to be applied at this plant, using basically the same pieces of equipment they currently have.

The mains minerals present in this ore are quartz, pyrite, chalcopyrite and calcite (Appendix II), while the gold grade is 1118 ppm (Appendix III).

Tests were conducted following the same procedures as Sousa et al. (2010). For the first test, the source of oxygen was hydrogen peroxide. For the second test, Oxiclean® (having 30% sodium percarbonate) was used as an oxygen source (1 g of Oxiclean® is equivalent to 0.09 g of hydrogen peroxide). Sodium percarbonate-based bleaches are easier to obtain than hydrogen peroxides, mainly in small communities; therefore its behaviour was investigated.

Both tests were conducted using a laboratory scale rod mill, with 400 g of ore and 400 ml of cyanide solution at 20 g/L and 0.3 g/L of hydrogen peroxide (or equivalent amount of Oxiclean®, which has around 20% hydrogen peroxide by weight). After approximately 24 hours pulp samples were collected. Solids were separated from the liquor by centrifuging and analysed for gold.

Results were similar to those obtained by Sousa et al. (2010) with a little disadvantage for the test using Oxiclean®. The lower extraction observed when using Oxiclean® is attributed to the leaching taking place at a pH higher than optimum. As the carbonate elevates the pH (to above 11), further investigation in how to keep pH low (at 10.5) is suggested.

Maximum extraction when using hydrogen peroxide was 98.5%, while using the sodium percarbonate-based product an extraction of 86% was obtained for the same 24-hour test-time. The kinetics can be observed in the Figure 6-15.

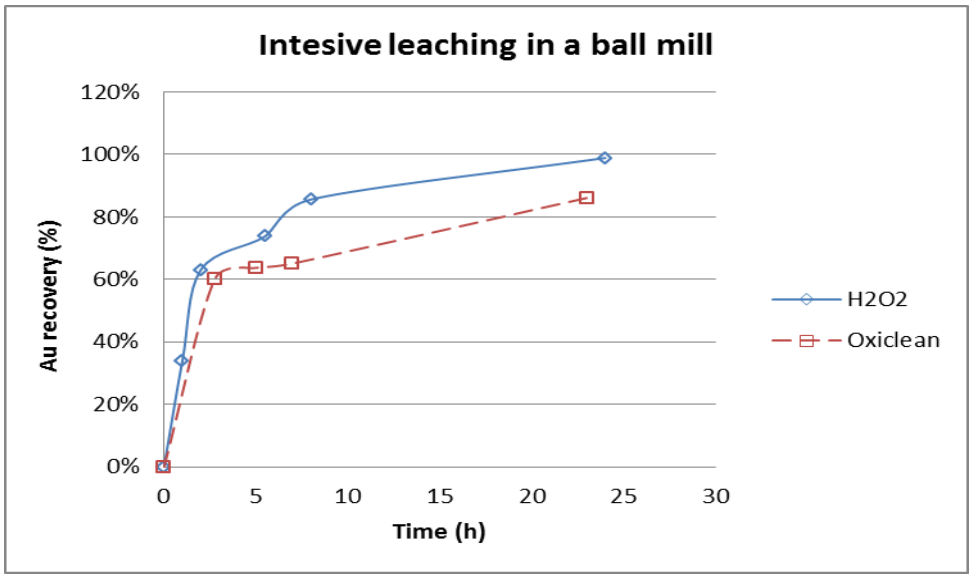

FIGURE 6-15.GOLD EXTRACTION BY INTENSIVE LEACHING

6.3.3. BORAX EXTRACTION

The use of borax and a flux to melt gold ores in fire assaying is very old technique, but the use of borax alone has been publicized recently as a way to replace amalgamation (Amankwah et al., 2010). Gold extraction is applicable to specific ores and a high gold grade concentrate is needed, otherwise gold is lost with the slag (Veiga, 2011). Borax extraction is only efficient with

very low sulphur content concentrates and in materials that are not ground too fine (Appel and Na-Oy, 2012). The technique uses the ability of borax of lowering the melting point of the minerals to around 1063 ˚C, which is a temperature reachable using charcoal and a blower or a butane or propane torch (Appel and Na-Oy, 2012). The same work indicates that in order for the process to be effective, the concentration process must achieve high grades of gold in the concentrate. Appel and Na-Oy, (2012) procedure is shown below:

1. Concentrate the ore by gravity (sluice box, followed by panning). 30000 ppm Au (or 3% Au) is an ideal grade to be achieved by this step
2. In a plastic bag, add a mixture of 1:1 of borax and concentrate, a few drop of water and put the bag in a ceramic crucible
3. Using a torch or a bonfire increase the temperature to above 1063 °C
4. Separate the gold bead from the slag by hammering it after cooling

As explained in Section 6.1, there is a balance between grade and recovery. As no concentration method is totally efficient, the higher the grade achieved in the concentrate, the lower the recovery obtained in the process. Concentration ratio of 10 is usual in the industry and usually does not represent a large loss of gold recovery. For example, in a rich ore with 30 ppm of gold, a concentration factor of 10 would bring the grade to, ideally, 300 ppm, which is still low to feed a direct smelting process. The necessary factor of 1000 (to generate a 3%/Au concentrate) is way too high and would compromise the gold recovery to unacceptable levels. This is important to take into account because a proposed method will only be adopted by any miner if it returns more profit than the previous method. It seems that the borax

method is useful to be applied to alluvial ores if gold is liberated, making it easier to obtain a concentration ratio higher than 1000 without sacrificing recovery too much. However, the method is not practical for those miners producing more than 10 kg of concentrate per day, since the mass of material to be melt becomes impractical.

Site specific investigation was conducted with a complex ore from Portovelo that was being processed at Cazaderos Plant, Ecuador.

As mentioned before, this plant still uses amalgamation of rich concentrates produced by two gravity methods: centrifuge and sluice box. The grade of such a concentrate was around 3300 ppm of gold, which is not totally adequate for direct smelting, but further concentration would not be accepted by the local miners as they are aware they would lose more gold. Currently they are amalgamating this concentrate. The main components are sulphides, as shown in the XRD in Appendix II. Another problem in this specific location, is the energy consumption. Usually this plant produces 175 kg of gravity concentrate per day and the energy spent to melt this amount of concentrate at 1063 ˚C with borax would be considerably higher than the energy used in amalgamation.

A sample of the sluice box concentrate was taken and quartered into 50 g sub-samples. One sub-sample proceeded to fire assay resulting in a gold grade of 3330 ppm while silver grade was 1170 ppm. This result indicates that each 50g sub-sample contains 0.165 g of gold and 0.06 g of silver.

Several tests of direct smelting with borax were conducted with no or little success. Only 3 tests are worth to be analyzed and discussed:

For the first test, a ceramic crucible was heated with an acetylene-oxygen torch and a bed of borax was sprinkled over the bottom of the crucible to prevent molten material to adhere to its surface. After the bed was formed, a mixture of 50 g of borax and 50 g of concentrate was added to the crucible and melted with a propane torch. In about 7 minutes, all the material was melted and transferred to a ceramic cup to solidify. After the material solidified the cup was broken in order to obtain the gold bead. This test was unsuccessful because there was no separation between the gold and the slag. This is due to the high amount of sulphides in the sample.

For the second test the same procedure was adopted, but KNO_3 was also added to the material to be melted in 1:1:1 (Concentrate:Borax:KNO_3) proportion. KNO_3 is a strong oxidizing agent that was added with the aim of transforming the sulphides into oxides for a better separation between the gold and the slag. The solidified product of this test looked better than the previous one, with a well-defined interface between the slag and the metal. However, the slag was not vitreous and showed 1160 ppm of gold. The metallic bead analyzed 6165 ppm of gold. Base metals were present in the bead, differently than a regular fire assay. This happened because no lead oxide (or litharge) was added and the concentrate had too little amount of gold to create enough weight to bring all gold to the bottom. In a fire-assay analysis, the lead oxide is reduced to metallic lead that carries the gold to the bottom of the crucible. Another function of the litharge is to prevent base metals from reducing and entering the metallic (gold) phase (McGuire, na). No litharge was added because using litharge would make the process exactly the same as a fire assay procedure and introduce a potential contaminant, lead. The

idea was to compare the initial proposed simple borax method of gold extraction to amalgamation.

A third test was conducted by adding SiO_2 and $NaHCO_3$ in order to further increase the separation of gold from the slag. In this test a proportion of 1:1:1:0.25:0.25 (Concentrate:Borax:KNO_3:SiO_2:$NaHCO_3$) was used. Silica is added to generate a matrix where the metals can be dissolved, while sodium bicarbonate is used to control pH (avoiding generation of gold chloride) and decrease viscosity (a brief description of the functions of these flux components is shown in Appendix IV). The formation of a well-defined interface between slag and the gold bead was again noticed while the slag showed a more vitreous aspect, indicating that a higher amount of metal was separated at this time. In fact, the assay of slag showed 840 ppm of gold while the bead contained 6935ppm Au. Figure 6-16 shows the bead of gold combined with large amounts of iron, copper and other metals. The bead was then dissolved in hot nitric acid to obtain metallic gold.

FIGURE 6-16. SEPARATED BEAD AND SLAG IN THE CUP

The following table shows the recovery achieved by the borax method. This was calculated based only on the smelting process.

Test	Grade Au (ppm)		Mass (g)		Au Distribution (%)	
	Bead	Slag	Bead	Slag	Bead	Slag
2	6165	1160	40	61	77.7	22.3
3	6935	840	45	67	84.7	15.3

TABLE 6-2. RECOVERIES FOR DIRECT SMELTING TESTS

It is important to note that the gold recovery must take into account the recovery of the concentration method as well. In the Cazaderos case the gold recovery was around 70% for the centrifuge and 50% for the sluice box, as reported by the company. In the best case scenario, the global recovery would be 29.6%. Comparing direct smelting with the current amalgamation method, the gold recovery would typically be slightly better with the direct smelting, but impractical and costly (energywise) to be conducted for 175 kg of concentrate per day. In addition the slag must be recycled to the cyanidation process.

The main problems observed in the borax process were:

1. It is hard to be applied when large amounts of concentrates (an economical analysis should be conducted to evaluate the maximum amount). The size of the crucible increases and more expensive furnaces would be needed;
2. It is hard to reduce the mass of concentrates by 1000 times and keep the gold recovery at a reasonable level (it would be quite low);
3. The presence of sulphides in the concentrate requires additional reagents and knowledge of the operator to adjust the flux;

4. The gold retained in the slag is substantial and must be recycled through a cyanidation process. Some miners in Indonesia amalgamate the produced slag (Veiga, 2011);
5. The amount of gold in concentrates with less than 3% Au (as recommended by Appel and Na-Oy, 2012) is too little to provide weight to the gold bead, in order to create a distinguishable interface between gold and slag.
6. The amount of energy required to melt large amounts of concentrate is higher than the energy used in the amalgamation and amalgam burning.

The process seems useful for a narrow range of artisanal miners and types of ores, e.g. those without sulphides. In fact, it is useful for micro-miners producing very small amounts of gold per day and working with alluvial ores in the water streams. In this case the main questions related to this method are:

1. Are those micro-miners the main mercury polluters?
2. Is it practical for these micro-miners producing vey small amount of gold per day to acquire borax, bicarbonate, potassium nitrate , air blowers, etc. to melt high grade gold concentrates?
3. Is it easy to achieve 1063 °C without the use of a propane torch or an air blower?
4. Is it worth spending millions of dollars to train micro-miners how to use this process instead of using the funds to deal with larger polluters?

The answers for these questions are "no" and in the context of mercury pollution, all the investment in training and publicity of this method seems to be a grain of sand in a desert of problems. Moreover, it is likely that the concentration ratio needed to achieve a rich-enough concentrate (ameanable to direct smelting with borax) will compromise too much the recovery. Therefore, by yielding less profit to the miner, this method probably will be quickly abandoned after introduction.

AGC, (2012) states similar conclusions about the borax method. The article criticizes the high grade necessary for the method to work as an extraction method. At the same time, it differenciates gold extraction replacing amalgamation from purification of gold, adding borax which is a method widely used by the gold industry.

6.3.4. CHLORINE LEACHING

Chlorine leaching was applied extensively in the 1800s as a form of treatment of fine gold and sulphide bearing ores where amalgamation and gravity concentration were inefficient. This process used to be conducted after roasting the ore to oxidize the sulphides, namely galena and copper sulphides. It was not effective if the gold grains contained more than 50% of silver due to the formation of an impenetrable coating of silver chloride. The solubility of this salt is 1.6 mg/L at 20 °C (Lide, 2012).

In the early 1900s, cyanidation became popular because of its advantages over chlorination, as it promotes faster extraction rates, leaches silver and gold in the same step, and uses simpler equipment with no need to be corrosion resistant (Eissler, 1888)

The early procedure was conducted in a wooden tank pumping chlorine gas through the dump ore (about 6% moisture) inside a covered vat for a time varying between 12 and 18 hours. Then the vat was opened and a shower of water carried the extracted gold to the bottom of the tank. The solution was collected and sent to the precipitation vat where iron sulfate was introduced to precipitate the gold. The silver chloride generated was washed from the ore in the vat after the gold pregnant solution was all removed. In order to do so, a solution of sodium hyposulfite washed the ore, washing the silver away. To this solution was then added calcium polysulphide, precipitating then silver sulphide that was roasted and then melted with scrap iron and borax. Recoveries achieved by this process were no higher than 80% (Eissler, 1888).

Chlorine based methods are currently a popular means of gold refining, as for example the Wohlwill and the Miller processes. The former process takes place in an electrolytic cell where the impure *doré* serves as an anode, and 99.999% thin sheets of gold are used as cathode. Chloroauric acid ($HAuCl_4$) serves as an electrolyte. When current is applied to the cell, gold and other metals dissolve at the anode and pure gold coming from chloroauric acid by ion transfer is plated onto the cathode. When the anode is totally dissolved, the process is complete and the gold deposited at the cathode is 99.999% pure (Wohlwill, 1910). In the latter process, pure chlorine gas is blown through a crucible full of molten impure gold where it reacts with most metals forming chloride compounds which float to the slag. This makes the gold 99.95% pure at the end of the process.

Considering how well these processes work for refining gold, Vinals et al., (1995) studied the kinetics of aqueous chlorination of suspended particles of gold as an initial approach to apply this technology in mineral processing. The main problem faced by the engineers in the 1800s was the difficulty of conducting leaching in agitated steel tanks due to corrosion. Combined with the high solubility of chlorine in water, this leads to a faster dissolution rate of gold than in the vat process.

For the oxidation/reducing of gold at anodic sites in chlorine leaching, the following pathway is shown by Marsden and House, (1992):

$$AuCl_2^- + e^- \leftrightarrow Au + 2Cl^-$$

$$AuCl_4^- + 3e^- \leftrightarrow Au + 4Cl^-$$

$$2Au + 2Cl^- \leftrightarrow 2AuCl + 2e^-$$

$$AuCl_2^- + 2Cl^- \leftrightarrow AuCl_4^- + 2e^-$$

One conclusion that Vinals et al. (1995) obtained based on studies of the pH, total chlorine, and total chloride indicates that the gold dissolution is indirectly controlled by the concentration of the trichloride ion (Cl_3^-). This concentration (for a fixed total chlorine concentration and without gold in solution) increases as pH decreases, achieving the maximum value at pH 1 or lower, and being absent at pH higher than 5. Gold dissolution follows the same pattern.

MINTEK, a South African research center, has developed a method to extract gold from gravity concentrates based on chlorine leaching. The iGoli gold extraction system is intended for artisanal use in substitution for amalgamation. It consists of a small (30 liters) agitated leaching tank, where the

miner adds a 1:1 mixture of pool acid (HCl @ 33%) and bleach (sodium hypochlorite @ 16%).

The formation of chlorine gas in this system is given by:

$$NaClO + 2HCl \leftrightarrow NaCl + Cl_2 + H_2O$$

Leaching occurs under agitation while the excess chlorine gas generated is bubbled into a NaOH solution, where it is converted into NaCl and H_2O. This process leaches other metals present in the concentrate as iron, arsenic and copper. However, after filtration, only gold is precipitated from the obtained liquor, by adding sodium metabissulfite:

$$3\ NaHSO_3 + 3\ H_2O + 2\ HAuCl_4 = 3\ NaHSO_4 + 8\ HCl + 2\ Au$$

Calcium chloride, sodium hydroxide and apatite should be added to the effluent solution after gold is recovered to precipitate the other metals in solution to an inert form. This method can be a good substitute for amalgamation of concentrates as it recovers more gold. Recoveries as high as 98% were achieve by Mahlatsi and Guest, (2003). On the other hand, the process is chemically complicated and technical knowledge is crucial to solve extraction problems that might appear using different ore concentrates. The effluent treatment is another preoccupation, there is an extra cost associated and again, technical knowledge is needed. There is also no recovery of silver in this system, since chlorination of this metal generates an insoluble compound (AgCl). High silver content ores will also respond poorly to chlorination because of the formation of a passivation AgCl layer around natural gold/silver alloys.

6.3.5. ELECTRO-LEACHING

The electro-chlorination or electro-leaching has been proposed by Sobral et al., (1996) as a potential candidate to substitute cyanide in large operations and in AGM since the process is relatively simple, safe and environmentally friendly.

The idea of a small-scale electrolysis cell able to: a) produce chlorine and/or bromine from sodium chloride and/or sodium bromide solutions; b) leach the gold in an agitated pulp; and c) plate it on the cathode, seems to be a good approach to extract gold from concentrates. In fact Sobral et al., (1996), studied this method to leach mercury from contaminated tailings and noticed that gold was also dissolved and recovered on the cathode.

In order to investigate this further, a laboratory scale electrolysis cell was built at UBC. The cell was designed to leach 25 mg per batch of pure gold (99.99%) at size -200 # +100 # (-150 +75 microns)(Sigma-Aldrich brand) using sodium chloride and sodium bromide in different concentrations. The leaching rates were determined each situation. The tests were conducted for durations of 1, 2 and 3 hours.

The apparatus consisted of a 250 ml beaker, a magnetic stirrer, a pair of graphite electrodes, a controlled DC source and a couple of multi-meters to have an accurate reading of the voltage and current applied in each test (Figure 6-17).

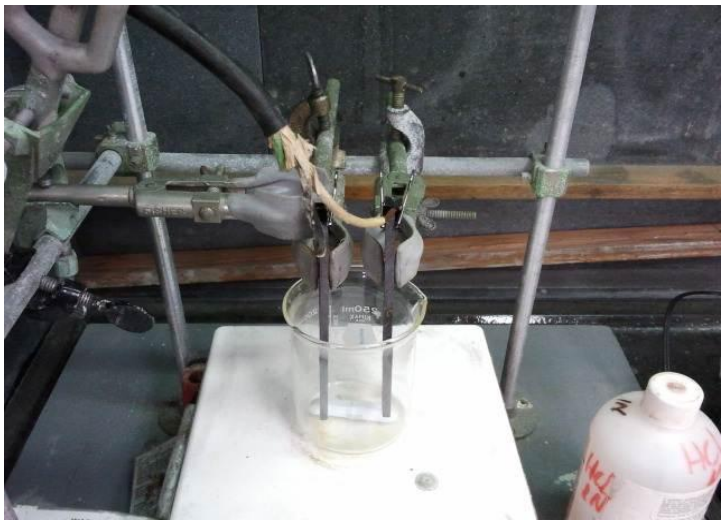

FIGURE 6-17. LABORATORY-SCALE ELECTROLYSIS CELL

Putnam, (1942) registered a patent exploring the enhanced properties of bromine as a gold leaching agent. He patented a process of leaching gold with 1.0 g/L of sodium bromine combined with sodium chloride at higher concentrations – as high as 200 g/L – stating that the process is much faster than leaching gold only with sodium chloride – both in a hydrochloric acid solution.

Different concentrations and proportions of electrolyte (NaCl and NaBr) were tested and the gold extraction versus time of leaching is shown in Figure 6-18. For these tests a very short time of leaching was needed to dissolve all gold grains. Every 15 minutes the stirrer was stopped, the pH adjusted and the gold grains were visually inspected. Gold dissolutions of 100% and 92% were achieved in two hours at pH 1.5, using 30 g/L of NaCl + 1 g/L of NaBr and 30 g/L of NaCl, respectively. A potential difference of 4 V was applied between the electrodes. Current varied from 0.18 to 0.80 A, depending on the voltage and electrolyte concentration.

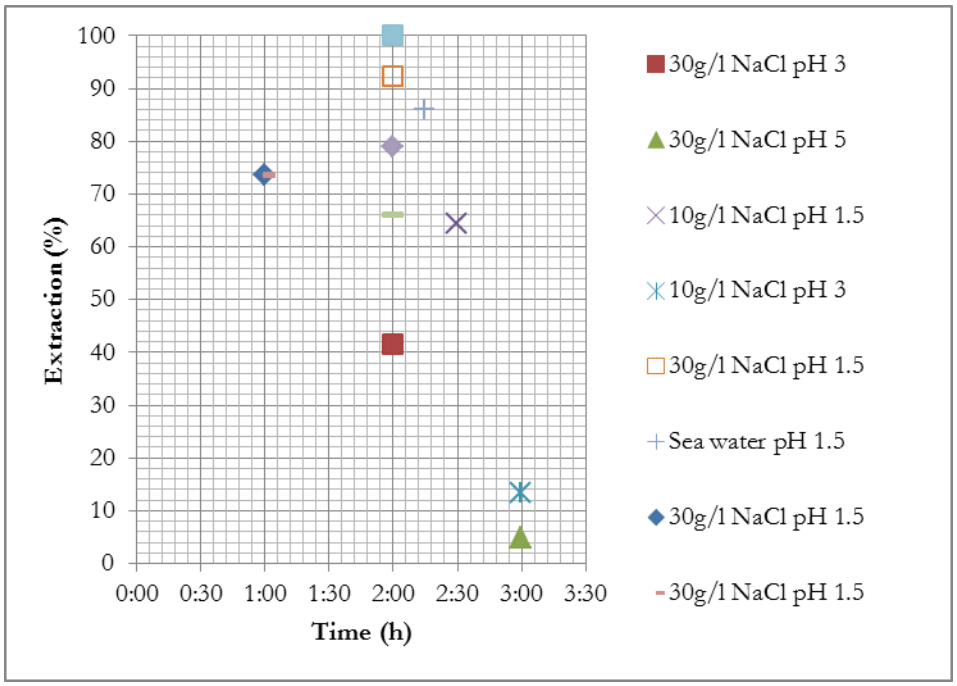

FIGURE 6-18. EXTRACTION OF GOLD FROM PURE METAL VS. TIME

By difference between the amount of gold introduced into the cell and what remained in solution after the test, it was possible to calculate the amount of gold plated into the cathode.

The initial idea was to have a one-step process that could leach the gold and have it deposited onto the cathode where it could be mechanically scraped. For the test where the best rate of dissolution was achieved, the conditions and results are shown in Table 6-3. All the original gold was leached in this test with 74% being deposited on the cathode in two hours showing that the process could be suitable for leaching gold concentrates.

30g/l NaCl + 1g/l NaBr pH 1.5		Time (min)	
		0	120
Metallic Au	(mg)	24.8	0
Au in solution	(mg)	0	6.46
Au deposited	(mg)	0	18.34
	%	0	73.95

TABLE 6-3. TESTS OF ELECTRO-LEACHING OF PURE GOLD

Electro-leaching was used to leach the centrifuge concentrate obtained at Cazaderos plant, containing 1118 ppm of gold. Around 30 tests were conducted both at UBC and at the Cazaderos Plant in Ecuador. The fire assay conducted at the Cazaderos plant, indicated 1200 ppm of gold, while the test conducted at ACME labs, shown in Appendix III; indicated 1118 ppm. As the differences are small between those results, it is assumed that other analyses conducted in Ecuador are also reasonably accurate.

The presence of galena and copper sulphides does not interfere with the chlorination process if a well conducted roasting step precedes the chlorination (Eissler, 1888). However, the chlorination is a potential oxidative treatment for such minerals as showed by Arslan and Duby (1997):

$$2FeS_2 + 15ClO^- + H_2O + 2H^+ \rightarrow 2Fe^{3+} + 4HSO_4^- + 15Cl^-$$

$$2FeS_2 + 15HClO + H2O \rightarrow 2Fe^{3+} + 4HSO_4^- + 15Cl^- + 13H^+$$

According to Arslan and Duby (1997), oxidation of the host sulphide mineral (to access gold locked in the crystal lattice of a sulphide) through chlorination is possible. The authors indicated that for a material containing 5% of pyrite, total oxidation occurs in 24 hours of electro-chlorination in a cell similar to the one employed in the present study.

For the lab tests using ore from the Cazaderos plant, the same apparatus used for leaching pure gold was assembled to leach 20 g of concentrate. This amount of concentrate was selected because it was the largest amount the cell could maintain in suspension.

The Table 6-4 shows the conditions of the tests using Cazaderos concentrate. Control of pH was done by using HCl at 10%. Voltage applied was 4 V. Table 6-5 lists the results of these tests.

Test #5 was a control test, conducted with pure gold, in order to evaluate if the behaviour in the Ecuadorean lab was the same as at UBC.

Test #6 was conducted with 6.34 mg of a *doré* with 80% gold and 20% silver. The deposited weight was very low (0.8% recovery), as well as the gold in solution after the test ceased, indicating that the main problem was in the gold leaching, and not in the electrolytic deposition. At the end of the test, 99.2% of the *doré* was still in the metallic form. The remaining material was completely covered by a white compound, probably silver chloride, an insoluble salt that blinds access of the leachant to gold.

Tests #8 to #12 were all conducted with 20 g of centrifuge concentrate containing 1118 ppm gold. Each one of these 5 tests was conducted at a different current or with a different electrode area in order to evaluate the influence of these conditions on gold extraction and deposition. For currents ranging from 0.4 A to 2 A, no significant change in extraction rates was observed. Some material was deposited on the cathode but no significant amount of gold was leached. Using a hand magnet it was possible to observe that iron was a constituent of the cathode deposit.

	Time (h)	Aui (mg)	ml	NaCl (g/L)	NaBr (g/L)	pH
test #5	2	21.1	200	5	5	1.5
test #6	3	6.34	200	30	0	1.5
test #8	2	24	200	30	0	1.5
test #9	2	23.3	200	30	0	1.5
test #10	2	21.8	200	30	0	1.5
test #11	2	21.6	200	5	5	1.5
test #12	4.2	22.4	200	30	0	1.5

Note: Aui = gold contained in the feed mass, ml = amount of water, Time = time of leaching

TABLE 6-4. ELECTRO-LEACHING TEST CONDITIONS

	Auf (mg)	Au Sol (ppm)	Au Sol (g)	Au Sol (% of fed gold)	Au Dep (mg)	Au Dep (%)	Au Left (mg)	Au Left (%)
test #5	2.56	12.1	0.0024	0.01%	18.51	87.80%	2.57	12.20%
test #6	6.29	1.80	0.00036	0.01%	0.05	0.80%	6.29	99.20%
test #8	23.9	1.08	0.00022	0.00%	0.02	0.10%	23.9	99.90%
test #9	23.2	1.25	0.00025	0.00%	0.02	0.10%	23.2	99.90%
test #10	21.7	1.25	0.00025	0.00%	0.02	0.10%	21.7	99.90%
test #11	21.6	0.80	0.00016	0.00%	0.02	0.10%	21.6	99.90%
test #12	22.4	1.80	0.00036	0.00%	0.02	0.10%	22.4	99.90%

Auf= Initial mg of gold entering the test, Au sol= Concentration of gold in solution after test, Au Dep= Deposited gold after test, Au Left = Gold not leached

TABLE 6-5. ELECTRO-LEACHING TESTS RESULTS

A test was also conducted for 24 hours with low results of gold deposition. The test conditions were 30 g/L of NaCl and pH 1.5. The amount of gold leached was below 0.15%.

In tests 6 to 12 it can be said that, because of accuracy of the fire assay and sampling variance, the actual gold extraction was basically nil.

In the literature, studies of electro-chlorination of gold ores are less frequent than experiments using pure gold, but a few papers that investigate interactions of gold with other metals and minerals in the pulp are mentioned as

follows. It can be concluded that chlorine extraction performs better on sulphide-free ores.

Van Meersbergen et al. (1993) studied galvanic interactions and passivation film formation on metallic gold while being leached in a bromine solution. The authors identified and measured the interactions between gold and different metals and minerals in an electrolytic cell. The main finding is that copper, iron, galena and pyrrhotite lower the gold leaching by 86, 90, 19 and 43% respectively. This result agrees with the ones obtained in Cazaderos, as copper and iron are available in the chlorine leaching that material. Metallic iron was actually deposited onto the cathode.

A hierarchy of cathodic deposition has been suggested by Van Meersbergen et al. (1993), in which the less noble metals such as iron and copper are leached preferentially to gold. In this case, gold only is leached at a higher rate when other less noble metals are already leached out from the system.

Ximing et al. (1992) investigated chlorine leaching of gold-bearing sulphide concentrates by bubbling chlorine gas into a HCl solution. The authors reported recoveries as high as 98.9% of the gold from the concentrate when the system operated at 90 °C with a very high consumption of chlorine, as high as 4.5 kg of chlorine per kg of ore. This result was only achieved in a roasted ore, where the sulphides are oxidized.

Greaves et al. (1990) also investigated chlorine as a substitute for using cyanide on several different types of gold ores. The recoveries in high-sulphide ores were 10.2% for a 35% sulphide content ore and 6.4% for a 20% sulphide content ore. When pre-oxidized, these values increased to 20.4 and 70%.

Chlorination is claimed by Diaz et al., (1993), Vinals et al., (1995), Ximing et al., (1995), Bard et al., (2006) and Vieira (2006) as being a good candidate to substitute amalgamation, but this process seems to be applicable only in the case of simple gold ores (i.e., very low sulphides content).

6.3.6. ORE CHARACTERIZATION IN AGM

Myth: one process fits all
Reality: each ore needs technological characterization (process mineralogy)

An AGM operation, as with any other conventional ore processing, is subject to design problems, variability of ores (including their location) and engineering problems. As a conventional processing plant relies on studies and characterization of their ore, an AGM operation would also benefit from this service. The main problem in establishing sound and efficient processing plants is the availability of good engineers and trained technicians. This problem is aggravated by the myth that one solution fits all types of ores, when the mineralogy, the location and available technology play the main role.

However, this scenario seems to be changing as characterization services are starting to become known to AGM miners. A report presented in Appendix V0 was product of a study conducted on a mineral sample from an AGM operation in Costa Rica. Today this operation involves amalgamating the concentrate generated by a Chilean mill feeding a sluice box. Gold recovery obtained is around 45%.

A simple characterization procedure is proposed. This procedure can be realized locally using relatively inexpensive equipment. The technique makes

possible for the miners to understand how proper concentration equipment can help increase recovery and consequently increase profits:

- A representative sample is taken from the mine or from the current mill feed;
- The crushed or ground sample is riffled into 1kg samples;
- GRG (Gravity Recoverable Gold) tests are conducted on the material using at least 3 kg of sample;
- Tailings from the GRG test are tested for flotation under different conditions;
- Based on the GRG and flotation test, a flowsheet simulation is then conducted.

The diagram in Figure 6-19 exemplifies the steps of the proposed characterization procedure.

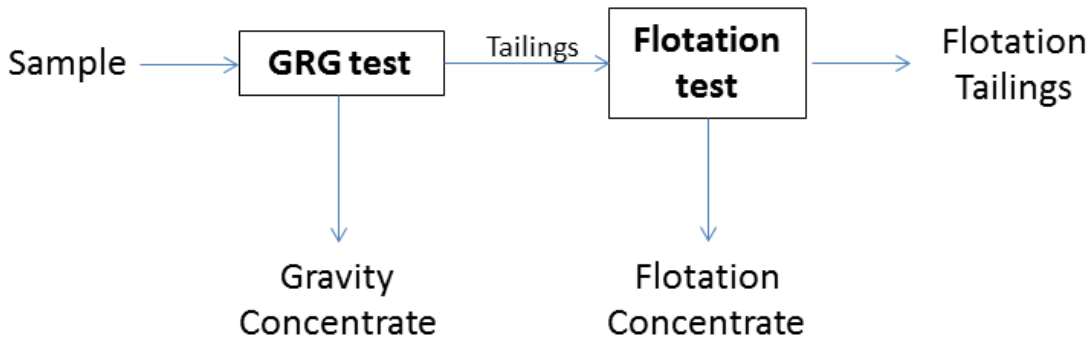

FIGURE 6-19. SIMPLIFIED CHARACTERIZATION FLOWSHEET

GRG test is conducted by passing a sample through the centrifuge, collecting the generated concentrate and regrinding the tailings for a short period of time. The reground material feeds the centrifuge again and tailings are re-

ground. Part of the tailings is screened in a 100 mesh screen. After screening and weighed, this sample is re-composed, combining the -100 and +100 mesh fractions, and used for the centrifuge tests. This repetition of grinding and reprocessing occurs for 4 to 8 times. A chart of recovery versus size is then plotted to identify the minimum particle size to which the gravity concentration responds (Figure 6-20). For the material from Costa Rica, Table 6-6 shows the evolution of the gold recovery with the number of tests, where the particle size in each test is getting finer as observed by the % of material retained in the 100 mesh screen.

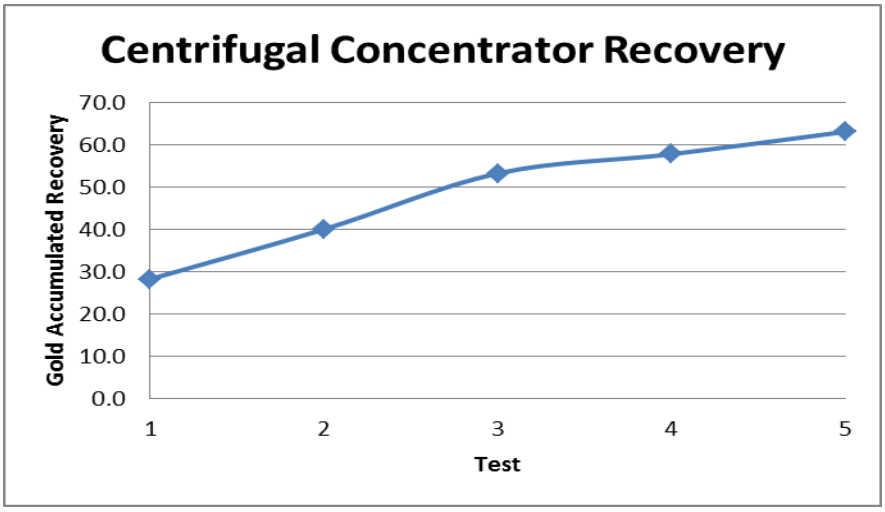

FIGURE 6-20. CENTRIFUGAL CONCENTRATOR RECOVERY

Test	% Retained in 100# Tyler	Conc. Mass (g)	Grade (ppm/Au)	Mass (%)	Recovery (%)	Accumulated Au Recovery (%)
1	54.3	87.4	119.0	3.0	28.3	28.3
2	50.3	74.5	58.1	2.6	11.8	40.1
3	43.7	60.2	80.8	2.1	13.2	53.3
4	39.1	68.7	24.3	2.4	4.5	57.9
5	33.3	68.9	28.2	2.4	5.3	63.1
Tailings	33.3	2526.9	5.4	87.5	36.9	

TABLE 6-6. FEED AND RECOVERY FOR GRG TEST

It was determined that for 70% of material passing 100 # (150 microns), the gold recovery was maximized at 63%.

The tailings from the GRG test 5 were then floated in three different conditions, as shown in Table 6-7. The best recovery was achieved at 50 g/tonne of PAX and 10 g/tonne of Aerofloat 208 at pH 6 in the rougher step. For the scavenger step 25 g/tonne of PAX and 5 g/tonne of Aerofloat 208 at pH 9 yielded slightly better recovery than the other tests. A further scavenger step was conducted at the same reagent concentrations, at pH 10, as shown in Table 6-8.

Test	**1**	**2**	**3**
pH Rg	6	6	6
pH Sc1	9	9	9
pH Sc2	10	10	10
PAX Rg (g/t)	100	50	25
PAX Sc1 (g/t)	50	25	10
PAX Sc2 (g/t)	50	25	10
Aeroflot 208 Rg (g/t)	25	10	5
Aeroflot 208 Sc1 (g/t)	10	5	3
Aeroflot 208 Sc2 (g/t)	10	5	3

TABLE 6-7. CONDITIONS OF FLOTATION TESTS

	Au (ppm)			**Gold Recovery**		
Test	**1**	**2**	**3**	**1**	**2**	**3**
CR Flot R	57.3	50.2	94.9	30.1%	44.3%	11.8%
CR Flot Sc1	29.2	15.3	42.2	10.5%	8.8%	5.5%
CR Flot Sc2	30	16.2	14.4	15.8%	10.4%	3.2%
CR Flot Tails	3.5	1.6	7.8	43.6%	36.4%	79.5%

TABLE 6-8. RESULTS OF FLOTATIONS TESTS

PAX is a good collector for fresh sulphides (with fresh non-oxidized surfaces) while Aerofloat (sodium dibutyl dithiophospate) helps in recovering tar-

nished surfaced sulphides (Marsden, 1992). Both collectors are indicated for native gold (Marsden, 1992).

An overallcalculated recovery of 86.5% was achieved in this characterization, although the concentration ratio in the centrifuge was relatively low (from 3 kg of feed, 375 g of concentrate was produced).

A second phase simulating a flowsheet was then conducted. At this time, 6 kg of ore was previously ground until 75% passes through 100 # (150 microns) and two triple passes were conducted in the centrifuge. This concentrate recovered 31.5% of the gold, the tailings proceeded to flotation at the best condition determined before, yielding a 58.2% unit recovery.

An overall recovery of 70.4% was achieved in the flowsheet simulation. This result, yielded a recovery 45% higher than the current applied process yields. Further fine tuning in the tests conditions is probably likely to increase the recovery.

Intensive cyanidation of the gravity and flotation concentrates, following the hydrogen peroxide-procedure described in Section 6.3.2 was investigated on the produced concentrates with a recovery of 90% in 24 h. This extraction is considerably higher than the one usually achieved with mercury in AGM.

This simple procedure conducted in the lab with 40 kg of a representative sample can provide a small miner with the following information:

- How important and how fine the grinding needs to be done.
- How a new processing technology can increase their profit.
- How much gold they can extract in a gravity separation.

- What is the flotation procedure in order to recover fine gold.
- What is the recovery expected after investing in new equipment/process.

This type of information and service is essential in transforming AGM into responsible SGM. This particular case shows that it is possible to recover more gold, in a more profitable operation, while not using mercury. This technology can be introduced in Small-, Medium- and Large-scale artisanal mines by providing this sector with education and capacitation.

7. CONCLUSION

The study shows that amongst the interventions observed in AGM, there is a lack of understanding of the mercury elimination problem, the culture and the organization of an artisanal mining community before beginning to work. This is considered to be the most important part of a successful mercury pollution reduction program.

Characterization of the polluters is important to determine, not only by size, but by technology, what mining sectors investment would return more noticeable results. Artisanal mining, in both small and large scale, is the main releaser of contaminants. However, most efforts are put into bringing solutions to micro-miners (MM), a sector that involves the largest contingent of people who are working for subsistence and not necessarily releasing the largest amount of mercury. This kind of investment does not yield the best results, since the cycle of poverty in MM is the main factor that generates pollution and prevents the acquisition of new technology.

The findings of this work are:

- Borax extraction is clean and simple, but its application is restricted, even for MM. This method is not suitable to large amounts of concentrate or ores with sulphides.
- Borax extraction requires a rich concentrate, not always viable to be produced at acceptable levels of recovery. The grade of gold needed for the process to work is likely to incur in low recovery in the concentration step. This would drive miners away from the method, in search for a more profitable one, such as mercury amalgamation.

- Chlorination extraction is technologically complicated and also not applicable to ores containing sulphides.
- Intensive cyanide leaching is a simple and efficient method, applicable to complex ores and relatively large amounts of concentrate.
- Concentration of gold is the key step to reduce mercury use and release in AGM.
- Introduction of low-cost retorts is probably the best solution for MM.

Elucidation of the myths and realities serves as a guideline for creating a project in mercury pollution reduction. Based on the experience acquired in Ecuador and Peru, allied to case studies in several parts of the world, most commonly observed errors in this approach are listed in Table 7-1.

Myth	Reality
Amalgamation of the whole ore is better to extract fine gold	Fine gold is usually lost in rudimentary gravity concentration. Fine and unliberated gold particles are also unlikely to amalgamate
Concentration of gold is inefficient	Usually miners do not know how to do it
To liberate gold, miners need to grind very fine before concentration and this is difficult	Gold can be partially liberated and yet be concentrated. More grinding can be done after pre concentration
Long sluices recover more gold	There is no noticeable improvement in gold recovery after the first two meters of sluice
Circulating load in a ball mill is a waste of equipment	A more precise output size range is obtained using a high circulating load
Amalgamation can easily be replaced by another process	Mercury use needs to be first reduced, then eliminated
Artisanal miners can learn how to use other extraction methods	Without concentration, without education or without capital, all alternative methods are impractical, unavailable or too expensive for artisanal miners
Cyanide leaching of amalgamation tailings is more efficient	Mercury-cyanide complexes can be formed, exacerbating pollution
One process fits all	Process design depends on ore, location, and availability of technology

TABLE 7-1. MYTHS AND REALITIES COMPILATION

Another conclusion is the importance of presence in any intervention on an artisanal mining site. Trainers should be always present to solve technical problems that may appear to the miners. Ideally trainers should share the same cultural background as the miners, in order to develop trust.

Based on observations in Ecuador, one can see that an effective way to reduce contamination is by producing a metal-rich (Au, Cu, Ag) concentrate instead of metallic gold. Copper smelters can purchase concentrate from small-to-large mining operations, paying around 85 to 95% for the precious metals and other useful metals contained therein. The spread of this type of

operation would benefit miners and reduce mercury pollution. The SGM are only responsible for concentration, selling concentrates to smelters . For this to occur in larger scale, as observed in Portovelo, Ecuador, technology, education, and organization should be provided to the mining communities able to produce such concentrates. Access to capital is also important since this model entails payment for the product after a couple months, instead being paid on the spot for metallic gold.

Technical and economic studies on this production chain would be welcome by providing means of connecting small companies to large smelters.

The newly discovered gold deposits in the last 10 years have not matched the rate of gold production in the same period (SNL, 2012). The inventory is decreasing as it becomes increasingly difficult to find large deposits enticing enough to attract large companies. Although Schodde, (2011) debates this tendency, by observing that existing deposits are measured only to a certain depth, and deeper exploration is likely to increase proven deposits inventories in the next 15 years. However, he also notes that small to medium sized deposits are increasing in number.

It is expected therefore, that more and more small- to medium-sized mines and processing plants will start to operate over the next decade. The uptrend of gold prices also supports this theory. As the value of the metal rises, there will be an increasing interest in exploiting it in different areas of the world.

As an option to large scale companies to adapt to this new situation, there is the possibility of association with small scale cooperatives. This would enhance the possibility for them to exploit areas where no large operation is

feasible. On the other hand, liability and other legal problems in associating with small companies remain problematic.

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APPENDIX I – CAZADEROS PLANT PICTURES

FIGURE 0-1. BALL MILL AT CAZADEROS PLANT

FIGURE 0-2. SECONDARY CRUSHER AND SILO AT CAZADEROS

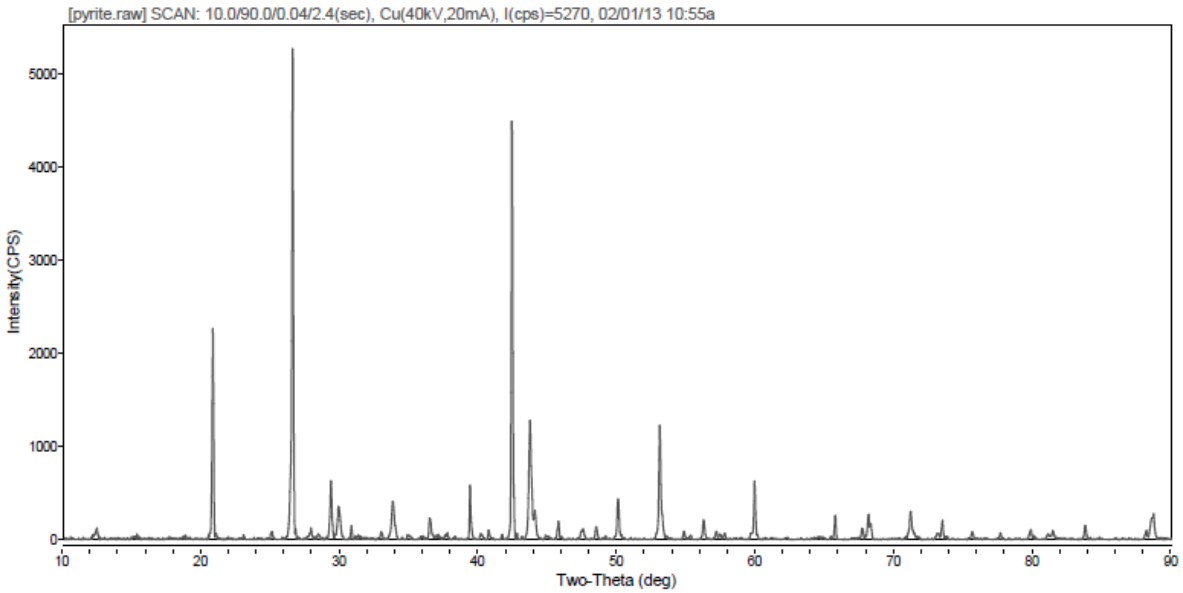
FIGURE 0-3. PRIMARY CRUSHER AT CAZADEROS PLANT

FIGURE 0-4. CENTRIFUGES AT THE BALL MILL DISCHARGE AT CAZADEROS PLANT

FIGURE 0-5. DENVER-TYPE FLOTATION CELL AT CAZADEROS PLANT

FIGURE 0-6. CYANIDATION TANKS AT CAZADEROS PLANT

APPENDIX II – CAZADEROS ORE XRD RESULT



Identified Minerals:

Main:
Pyrite
Quartz

Minor:
Chalcopyrite
Calcite

APPENDIX III – CAZADEROS CENTRIFUGE SAMPLE ASSAY

Acme Analytical Laboratories (Vancouver) Ltd.

www.acmelab.com

PHONE (604) 253-3158

Client: **UBC Mining Engineering Dept.**
Room 430A - 6350 Stores Road
Centre for Environment Research
Vancouver BC V6T 1Z4 Canada

Project: None Given
Report Date: January 14, 2013

Page: 2 of 2 Part: 1 of 1

CERTIFICATE OF ANALYSIS VAN12005274.2

	Method	1T	1T	1T	1T	1T	1T	1T	1T	1T	1T	1T	1T	1T	1T	1T	1T	1T	1T	1T	1T
	Analyte	Mo	Cu	Pb	Zn	Ag	Ni	Co	Mn	Fe	As	U	Au	Th	Sr	Cd	Sb	Bi	V	Ca	P
	Unit	ppm	ppm	ppm	ppm	ppb	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	%	%
	MDL	0.05	0.02	0.02	0.2	20	0.1	0.2	2	0.02	0.2	0.1	0.1	0.1	1	0.02	0.02	0.04	1	0.02	0.001
G1	Prep Blank	1.27	7.68	18.01	44.2	<20	69.8	6.7	686	2.55	11.3	2.5	<0.1	9.5	702	0.04	0.37	0.18	49	2.10	0.071
HEAD	Tailing	2.58	4851	1187	186.5	262240	302.8	457.6	355	35.56	12081	<0.1	1118	0.1	16	3.49	76.45	331.0	71	2.01	0.009

APPENDIX IV – FIRE ASSAY FLUXES

Obtained from (Fulton, 1907)

- KNO_3 – Oxidizing agent
    - ◊ Used to oxidize sulphides and generated metals (except PGM and Au) to their nitrate salts.
- $NaHCO_3$ – pH Control, Thinner
    - ◊ Keeps gold from evaporating as $AuCl_3$ under heat
    - ◊ Reduces viscosity of slag
- SiO_2 – Slag matrix
    - ◊ Dissolves and incorporates most metals oxides
- Borax – Thinner and SiO_2 melting point reducer
    - ◊ Silica: 1723 C → 800 C
- Litharge (PbO) – Collector
    - ◊ Reduces to Pb and washes down the metals in suspension (Au and PGM)
- Carbon –Flour, Coal, Sugar
    - ◊ Reducer for the litharge. These are added to the charge in sufficient quantity to produce the proper size of lead button in the crucible assay. It often happens that an ore will contain reducing agents, chiefly sulphides, so that it becomes unnecessary to add an extraneous agent. In fact, it may contain an excess of reducing agent, requiring an oxidizing agent to destroy the excess.

APPENDIX V – CHARACTERIZATION OF AN AGM ORE

According to the proposal issued by UBC, this study was conducted on a sample sent to UBC laboratories. Size analysis and gold assaying were conducted, indicating that most of the gold is contained in the coarse fraction (+35#) and in the fines (-325#), but there is still 38% of the gold is contained in between these size classes.

GRG (Gravity Recoverable Gold) test indicate that with 70% of the material pasing through 100# the optimum recovery is obtainedle, where about 63% of the gold in the head sample is recovered. Given the principles of operation of a centrifuge it can be inferred that most of the gold contained in the coarser fraction is GRG while the fines should be passing by the concentrator and will have their response to flotation studied next.

Riffling and Sampling

Two different samples were collected and sent from Costa Rica by the client and work on the head sample were initiated by drying the material and homogenizing it in prismatic heaps (stacked, reclaimed and re-stacked in order to achieve the best possible homogenization). The dry material weighted 39,143 grams and was sampled as the following table shows:

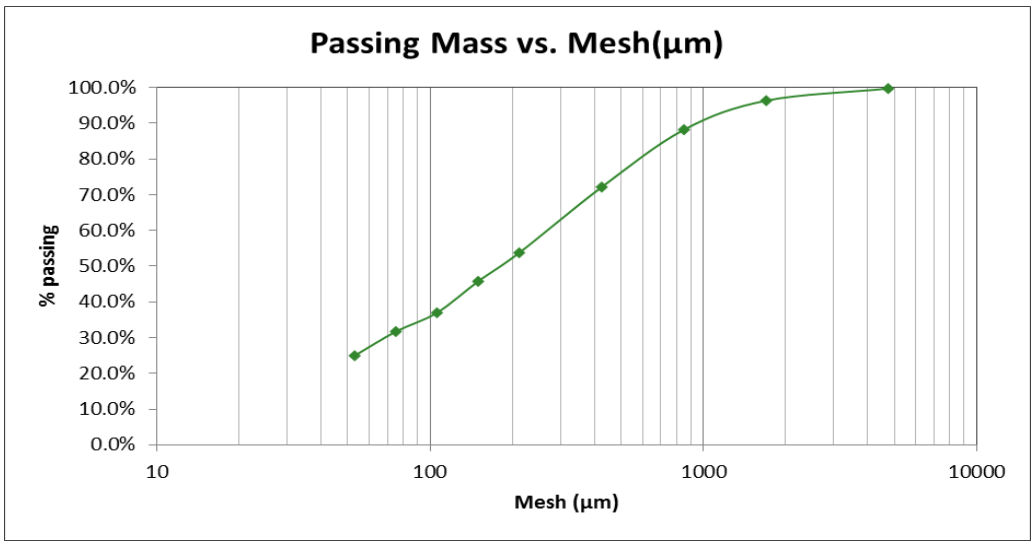
The material was riffled according to the table below:

Mass (g)	
20200.0	1005.0
1098.0	950.0
974.0	968.0
957.0	1090.0
1052.0	965.0
1006.0	1000.0
987.0	966.0
938.0	964.0
1000.0	1019.0
961.0	1043.0
Total Mass	39143.0

Grain Size Analysis

Using one sample, analysis in wet media was conducted. A chemical analysis by size fraction in terms of contained gold will be conducted in order to evaluate if there is a preferential distribution of gold according to the particles sizes as sent by the client. The table and chart below show the size distribution of the material:

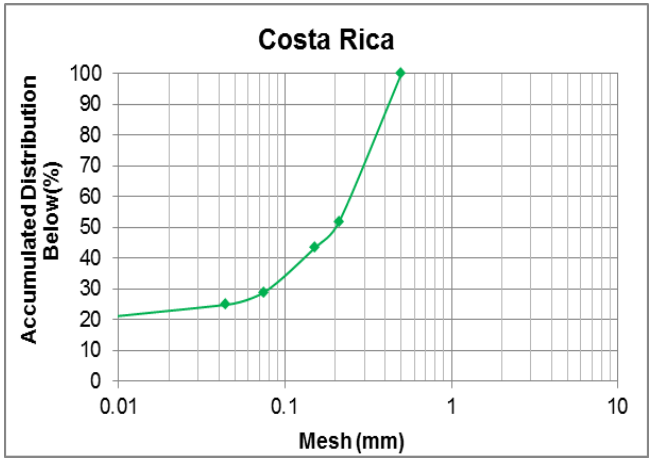
Mesh Tyler	Mesh (µm)	Retained Mass (g)	% Passing
4#	4750	3.0	99.7%
10#	1700	34.6	96.3%
20#	850	82.9	88.2%
35#	425	164.0	72.1%
65#	212	187.6	53.7%
100#	150	81.7	45.7%
150#	106	89.7	37.0%
200#	75	53.6	31.7%
270#	53	69.0	24.9%
325#	45	11.1	23.9%
Pan		243.5	0.0%
Final Mass	1020.8		
Losses	3.0%		

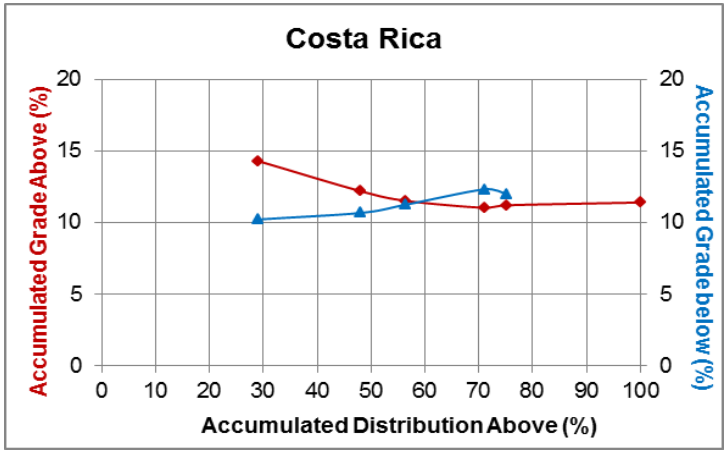


Size Analyses with Au Assay

Size separation from previous section generated six composite samples that were sent to SGS Canada to have gold content analysed by fire assay. The table and charts on the next table compiles the information obtained. It is observed that gold is present through the whole sample as received by UBC; distribution is relatively concentrated in the +35 and -325 fraction, where approximately 62% of the contained gold is.

Size Class (Mesh Tyler)	Mass (g)	Retained (%)	Accumulated Retained (%)	Accumulated Passing (%)	Grade (ppm/Au)	Accumulated Grade Above (ppm/Au)	Accumulated Grade Below (ppm/Au)	Distribution (%.Au)	Accumulated Distribution Above (%.Au)	Accumulated Distribution Below (%.Au)
+35	284.5	29.0	29.0	100	14.3	14.3	10.2	36.4	36.4	63.6
-35 +65	187.6	19.1	48.1	51.9	9.0	12.2	10.7	15.1	51.4	48.6
-65 +100	81.7	8.3	56.5	43.5	7.7	11.5	11.3	5.6	57.0	43.0
-100 +200	143.3	14.6	71.1	28.9	9.1	11.0	12.3	11.7	68.7	31.3
-200 +325	40.1	4.1	75.2	24.8	14.3	11.2	12.0	5.1	73.9	26.1
-325	243.5	24.8	100.0	0.0	12.0	11.4	-	26.1	100.0	0.0
Total	980.7				11.4					

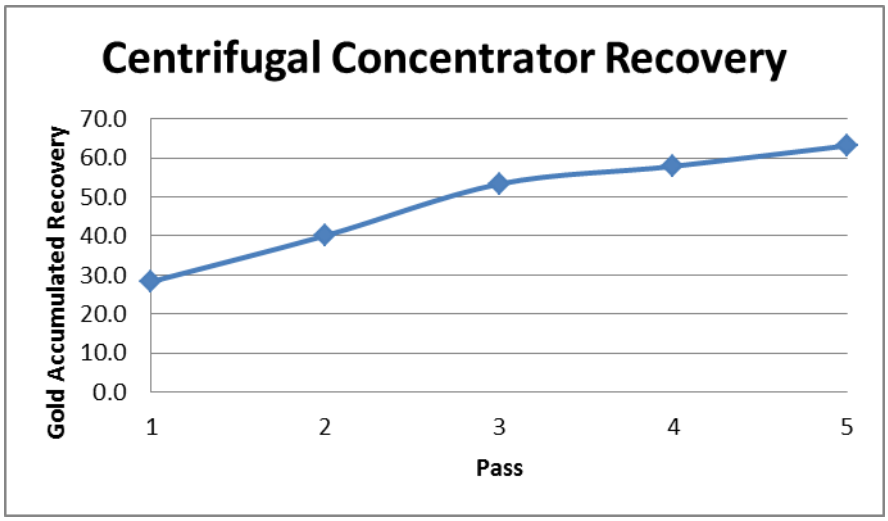




Centrifuge Concentration Tests

A Knelson laboratory centrifuge with capacity to recover ~40cm³ was used to evaluate GRG (Gravity Recoverable Gold) in the sample. In order to do so, 3 samples were reunited weighting 3024g and a first pass through the centrifuge was conducted, a concentrate was recovered and the tailings were ground in a laboratory rod mill for 7 minutes. The first concentrate was stored while the tailings were passed through the centrifuge one more time, the second concentrate was stored, while the tailings of the second pass were further ground and further concentrated. This procedure was repeated until 5 concentrates and one final tailing were obtained. The masses, partial size and gold content analysis for each pass are shown in the table below.

Pass	% Retained in 100# Tyler	Conc. Mass (g)	Grade (ppm/Au)	Mass (%)	Recovery (%)	Accumulated Recovery (%)
1	54.3	87.4	119.0	3.0	28.3	28.3
2	50.3	74.5	58.1	2.6	11.8	40.1
3	43.7	60.2	80.8	2.1	13.2	53.3
4	39.1	68.7	24.3	2.4	4.5	57.9
5	33.3	68.9	28.2	2.4	5.3	63.1
Tailings	33.3	2526.9	5.4	87.5	36.9	



It is observed that the accumulated recovery starts to stabilize between the 4th and 5th pass, extrapolating this results, it can be inferred that a grind size where 70% of the material passes a 100# is ideal grinding size for such sample.

Flotation Tests

The remaining material collected as tailings from the centrifugal concentration step followed on to a preliminary investigation for flotation behaviour.

The material was homogenized and divided into three samples that followed on to flotation in Denver-type bench cells at different reagents conditions, forcibly higher than usual since the idea is to identify if there is any response to flotation while fine tuning of dosage will be conducted at a later time. Tests were conducted in a three step flotation (1 rougher and 2 scavengers at different pH values) in a 3 liter cell at a density of solids equal to 23%.

The conditions for the flotation tests are as below:

Test	1	2	3
pH Rg	6	6	6
pH Sc1	9	9	9
pH Sc2	10	10	10
PAX Rg (g/t)	100	50	25
PAX Sc1 (g/t)	50	25	10
PAX Sc2 (g/t)	50	25	10
Aeroflot 208 Rg (g/t)	25	10	5
Aeroflot 208 Sc1 (g/t)	10	5	3
Aeroflot 208 Sc2 (g/t)	10	5	3

The mass recoveries obtained are as follows:

	Mass (g)			%		
Test	1	2	3	1	2	3
CR Flot R	29.36	27.9	8.75	3.79%	3.55%	1.16%
CR Flot Sc1	20	18.1	9.2	2.58%	2.30%	1.22%
CR Flot Sc2	29.36	20.35	15.8	3.79%	2.59%	2.10%
CR Flot Tails	696.1	719.4	718	89.84%	91.56%	95.51%

The gold recoveries obtained are:

	Au (ppm)			Gold Recovery		
Test	1	2	3	1	2	3
CR Flot R	57.3	50.2	94.9	30.1%	44.3%	11.8%
CR Flot Sc1	29.2	15.3	42.2	10.5%	8.8%	5.5%
CR Flot Sc2	30	16.2	14.4	15.8%	10.4%	3.2%
CR Flot Tails	3.5	1.6	7.8	43.6%	36.4%	79.5%

Based on these results, condition 2 was chosen as the more appropriate, since it yielded a better gold recovery, at 63.6%.

Phase II – Simulation of a proposed flowsheet.

Based on the works from phase one, grinding was done in a single step to obtain 70% of the material passing 100#. This simulation used approximately 6 kg of sample.

As 6 kg of sample is a too large amount to the mill and centrifuge size, it was divided in 2 samples of 3 kg that were processed in parallel.

The centrifuging process was conducted by passing each sample 3 times by the equipment, then emptying the bowl to generate concentrate 1. This process was repeated generated concentrate 2. The 3 passes were intended to increase recovery for each concentrate, simulating an industrial centrifuge being fed slowly and at an optimum speed and water pressure. The generation of 2 concentrates simulate 2 centrifuges in series in an industrial operation.

Gravity concentration results are shown in the table below:

Pass	% Retained in 100# Tyler	Mass (g)	Grade (ppm/Au)	Mass Recovery	Gold Recovery	Accumulated Au Recovery
1	30.12	145.9	234.8	2.5%	27.5%	27.5%
2	30.12	133.1	37.6	2.2%	4.0%	31.5%
Tailings	30.1	5664.0	15.1	95.3%	68.5%	

Accumulated recovery was much lower than that obtained in phase 1. This is caused by the proportionally much larger amount of concentrate obtained in phase 1. Fine tuning on the discharge time in industrial equipment needs to be conducted and recovery is expected to be in between 31.5% obtained in phase 2 and 63.1% obtained in phase 1.

Tailings of the gravity concentration were sent to flotation using condition 2 previously chosen. Again, as the mass of material was too big for the laboratory equipment, it was divided into 3 samples. Conditioning was conducted for 5 minutes at 50% solids (in mass), while flotation was conducted at 30% solids (in mass).

Flotation results are shown below:

Product	Mass (g)	Mass Recovery	Grade (ppm/Au)	Au Recovery	Accumulated Au Recovery
CR Flot 1 R	423.96	8.01%	71.35	46.3%	46.3%
CR Flot 1 Sc1	172.38	3.26%	26.57	7.0%	53.3%
CR Flot 1 Sc2	196.37	3.71%	16.13	4.9%	58.2%
CR Flot 1 Sunk	4502.3	85.03%	6.07	41.8%	

Flotation recovery was in line with results obtained in phase 1 where 63.4% gold recovery was obtained. Those values are expected to be similar, because flotation is not limited to a fixed amount of concentrate as a centrifuge does.

The global mass balance is shown below. Total recovery of gold in the concentration process was 70.4%.

	Product	Mass (g)	Mass Recovery	Grade (ppm/Au)	Au Recov-ery	Accumulated Au Recovery
Flotation	CR Flot 1 R	423.96	8.01%	71.35	46.3%	46.3%
	CR Flot 1 Sc1	172.38	3.26%	26.57	7.0%	53.3%
	CR Flot 1 Sc2	196.37	3.71%	16.13	4.9%	58.2%
	CR Flot 1 Sunk	4502.3	85.03%	6.07	41.8%	
Centrifuge	1	155.9	2.6%	234.8	28.8%	27.5%
	2	143.1	2.4%	37.6	4.2%	31.5%
	Tailings	5644.0	95.0%	15.1	67.0%	
Global	Centrifuge	299.0	5.0%	140.5	31.5%	31.5%
	Flotation	792.7	13.3%	47.9	39.0%	70.4%
	Tailings	4502.3	75.8%	6.1	29.6%	