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System-Wide Evaluation of Gold and Silver Recovery and Mercury Losses in an Artisanal Amalgamation–Cyanidation Circuit: A Diagnostic Approach to Mercury-Free Processing

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Abstract

Despite decades of intervention efforts, mercury amalgamation remains widespread in artisanal gold mining (AGM), yet no prior study has presented an integrated metallurgical balance quantifying gold (Au), silver (Ag), and mercury (Hg) simultaneously across both the amalgamation and cyanidation stages of a single processing circuit. This study addresses this gap by evaluating an artisanal amalgamation–cyanidation operation in Pernambuco, Brazil, through systematic sampling, fire assay and ICP-OES analyses, grain size distribution, and SEM-EDS mineralogical characterization. The whole-ore amalgamation (WOA) stage recovered 44.7% ± 11.7% of Au and 11.5% ± 15.4% of Ag from a feed grading 2.96 ± 1.53 ppm Au and 8.09 ± 1.65 ppm Ag. Subsequent vat cyanidation of the Hg-contaminated tailings recovered 75.8% ± 4.32% of the remaining Au and 32.9% ± 9.24% of Ag, raising overall recoveries to 86.6% ± 2.89% for Au and 40.6% ± 4.58% for Ag. Although both stages contribute meaningfully to gold recovery, under the operating conditions evaluated, cyanidation contributed more substantially to overall Au recovery than whole-ore amalgamation. SEM-EDS identified free amalgamated electrum particles in the tailings, attributed to excessive pulp density (≈43% solids) impairing particle settling on the amalgamation plates. These inefficiencies led to estimated mercury losses of 262 g/day, yielding a $Hg_{(lost)}$:$Au+Ag_{(produced)}$ ratio of 5.5. Based on this diagnostic evidence, replacing whole-ore amalgamation with optimized comminution followed by gravity concentration is identified as a technically promising mercury-free pathway requiring pilot-scale validation.

Keywords: artisanal gold mining (AGM); whole-ore amalgamation (WOA); cyanidation; gold recovery; metallurgical balance; mercury losses

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Academic Editor: Rajesh Kumar Jyothi

Received: 17 June 2026
Revised: 17 July 2026
Accepted: 24 July 2026
Published: 27 July 2026

Correction Statement: This article has been republished with a minor change. The change does not affect the scientific content of the article and further details are available within the backmatter of the website version of this article.

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1. Introduction

Artisanal gold mining (AGM) is characterized by the low level of technology employed in ore extraction and processing. In the broader literature, this term is often used interchangeably with artisanal and small-scale gold mining (ASGM); however, in the context of this text, the first will be preferentially adopted because it emphasizes the low-technology nature of operations rather than their scale, which is more descriptive of the operational profile studied here. The activity ranges from rudimentary, labor-intensive manual techniques to the use of medium to heavy machinery, and often involves both simultaneously. It is estimated that worldwide approximately 15 to 20 million people, including 4 to 5 million

women and children, are directly involved in AGM across more than 80 countries [1], which results in estimated 248 to 838 tonnes of mercury released to the environment each year, making the activity the main anthropogenic source of mercury to the environment [2]. Metallic mercury is the main reagent used in AGM due to its ability to amalgamate free gold particles present in the ore [3]. Amalgamation is an easy to use and long-term tradition in gold mining [4]. A simple procedure like an open-air bonfire burning of amalgam allows miners to literally have gold in their hands by the end of each working day.

Efforts by governments, academia, and NGOs to curb mercury use in ASGM have spanned more than four decades [5]. The adoption of the Minamata Convention on Mercury in 2013, which entered into force in 2017, established a binding international framework for signatory nations. Under Article 7, gold-producing countries, such as Brazil, committed to developing and implementing National Action Plans to reduce and, where feasible, eliminate mercury use in ASGM. A search in the Global Environmental Facility (GEF) Projects Database demonstrates that in the past 25 years around $185 million dollars have been used in projects related to this topic [6]. As reported by Veiga and Fadina [7] and Martinez et al. [8], most of these projects were related to mercury monitoring in the environment, public health issues, and government policy strategies, with minor focus on technologies that could effectively reduce Hg use. Some projects succeed in their initial stages but fail to produce lasting results, primarily due to insufficient continuity and government commitment in developing countries. Additionally, AGM personnel are predominantly composed of poor people who live a nomadic life in pursuit of gold deposits for subsistence. Environmental and public health arguments rarely influence miners' day-to-day decisions. Moreover, introducing unfamiliar equipment without sustained technical education and government oversight frequently reverses the gains of intervention projects [7]. When governments in developing countries reinforce their control efforts with more energetic actions such as imprisonment and destruction of artisanal miners' equipment, conflicts can erupt, further hindering efforts to establish trust and implement sustainable practices [9–11].

In Latin America, AGM operations predominantly exploit secondary deposits, including alluvial, colluvial, and eluvial ores, primarily because the presence of liberated gold simplifies extraction and processing. These secondary ores generally exhibit low grades, typically below 0.5 ppm Au, necessitating gravity concentration (e.g., sluices) followed by amalgamation of the gravity concentrate [12]. Conversely, primary deposits are targeted by a smaller number of miners who possess the capital and technical resources to manage the increased complexity of hard-rock processing, which involves comminution stages such as crushing and milling to liberate gold from the mineral matrix.

For primary ores, with grades above 3 ppm Au, "whole-ore amalgamation" (WOA) is a common process observed in Latin America, Africa and Asia [13]. In parts of Africa, miners processing smaller batches of primary ore (500 to 2000 kg/operation) often prefer hammer mills for size reduction followed by sluice box concentration [14].

While interventions by government and researchers have increased awareness regarding formalization and improved practices regarding processing techniques, operational complexities remain. WOA is a common technique used to capture free gold particles either using hammer-mills and mercury-coated surfaces (e.g., Hg coated copper plates) or ball mills with mercury [15]. Despite being obsolete, inefficient and a polluting technique, amalgamation of gravity concentrates reduces the use and losses of mercury, when compared to the WOA practices [16].

Influenced by intervention projects and practices from larger mining operations, the incorporation of cyanide leaching to treat tailings and recover residual gold is increasingly observed in many artisanal operations worldwide. When a cyanide solution percolates

through heaps or vats, depending on the specific setup, the solution is filtered and gold is either adsorbed by activated carbon or precipitated with zinc shavings [17]. In Africa and most Latin American countries, zinc shavings are used to precipitate the dissolved gold from cyanide solutions; however, this is inefficient as the operation is conducted in the presence of air. In Brazil, the resulting gold-bearing solution is typically passed through activated carbon columns to adsorb the dissolved gold, with final recovery steps often conducted off-site. Operators are often aware of cyanide's acute toxicity under acidic conditions, yet a critical knowledge gap persists regarding the environmental risks of mercury–cyanide interactions [18]. Discharging Hg-contaminated cyanidation tailings into waterways can generate highly mobile, bioavailable mercury–cyanide compounds, particularly under anaerobic conditions [19].

Given that gold is the primary driver of AGM activities, economic arguments centered on gold recovery efficiency can be highly persuasive when advocating for changes in processing techniques. Quantifiable evidence of gold losses provides a compelling economic case against rudimentary practices such as whole-ore amalgamation [5,14,20,21]. Quantifying gold recovery rates in AGM operations highlights the potential benefits of eliminating mercury use altogether. When gold is coarse and liberated and the concentrate exhibits favorable mineralogy (e.g., low sulfide content), gravity concentration can yield a high-grade product suitable for intensive leaching or direct smelting [22,23]. Furthermore, the generation of mercury-free gravity tailings facilitates safer subsequent processing through secondary leaching methods, including cyanidation, for the recovery of residual gold. Consequently, the implementation of a rigorous metallurgical balance within artisanal operations serves as a critical, cost-effective diagnostic tool for operator education and the identification of optimization opportunities to enhance recovery while minimizing mercury consumption and its associated environmental hazards.

To our knowledge, no prior study has presented an integrated metallurgical balance quantifying Au, Ag, and Hg simultaneously across both the amalgamation and cyanidation stages of a single artisanal processing circuit. This study addresses this gap by presenting data from a diagnostic intervention in a Brazilian AGM processing plant, quantifying gold and silver extraction efficiencies and mercury losses across the full circuit, and providing a replicable diagnostic framework for evidence-based mercury elimination strategies.

2. Methodology

The methodology comprised field sampling and laboratory analysis of three process streams (sluice feed, amalgamation tailings, and cyanidation tailings), complemented by direct observation of operations and interviews with plant personnel. The approach covered: (1) operational documentation of the comminution, amalgamation, and cyanidation stages; (2) quantitative determination of Au and Ag recoveries and loss distribution; (3) mineralogical characterization of tailings; and (4) estimation of mercury losses in tailings and during amalgam burning.

2.1. Overview of the Study Site

The study was conducted at an artisanal gold processing site in Serrita, Pernambuco State, Brazil (Figure 1). According to the Brazilian Institute of Geography and Statistics (IBGE), Serrita occupies an area of 1535 km^2, and has a population of approximately 18,207 (in 2022) inhabitants and a GDP per capita equivalent to US$2158 or roughly US$180 per month [24]. The town lies within the "Sertão Central" mesoregion, characterized by a semiarid climate with recurrent drought and "caatinga" (scrub forest) vegetation.

This site was selected because its hard-rock WOA circuit, employing hammer mills, copper-amalgamation plates, and vat cyanidation, is representative of operations common

throughout Latin America. It is also one of the few accessible sites where both processing stages are co-located, enabling an integrated Au-Ag-Hg balance across the full circuit. The semiarid climate and water scarcity constraints are typical of many AGM regions in North East Brazil and analogous contexts worldwide, making the findings relevant beyond this single operation.

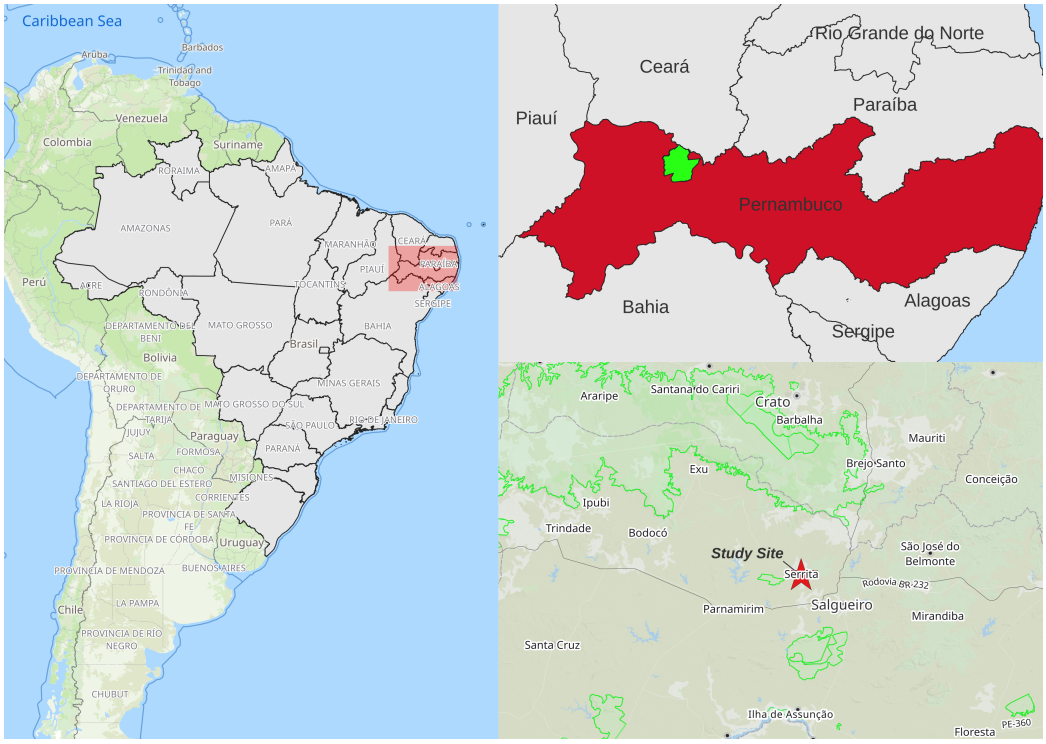


Figure 1. Localization of the studied site in Serrita, Pernambuco State, Brazil (adapted from OpenStreetMap© data).

The gold mineralization within the Serrita and Salgueiro municipalities in Pernambuco, Brazil, has been known since the World War II and it is an example of orogenic (highly deformed by tectonic forces) gold systems within the Borborema geological province. The ore is hosted by metasedimentary rocks of the Salgueiro Complex (mainly phyllites, schists, and metagraywackes) and Neoproterozoic granitic rocks of the Serrita Suite (quartz-monzodiorites, granodiorites, and tonalites) [25,26]. The gold ore is strictly controlled by shear zones and the mineralogy is characterized by presence of pyrite, arsenopyrite, and chalcopyrite and in some areas, galena. Gold from the Salgueiro has from 6.80 to 51.5 wt% of Ag [27]. The artisanal miners extract the semi-oxidized ore rich in iron oxides.

The studied processing facility belongs to a legal company that operates its own mine employing 24 full-time local workers engaged in various stages of mining and ore processing. Water for the operations is supplied by a natural lagoon situated near the processing plant. Tailings are disposed of in a dam, apparently with no contact with local drainage.

The operational workflow at the processing site consists of several sequential steps, beginning with ore extraction and culminating in gold recovery through amalgamation and cyanidation. As illustrated in Figure 2, the primary ore is transported from an open-pit mine situated approximately 30 km from the processing facility. Upon arrival, the ore is stockpiled and ore comminution starts with manual breaking of larger blocks of rock prior to being processed through: (1) jaw crusher; (2) screening in a 5 mm vibrating screen; (3) further reduction in a second jaw crusher; (4) grinding in a hammer mill (H3); and

(5) final grinding in five hammer mills (H1) with internal opening of 1 mm, being manually fed by operators.

The ground ore is subjected to whole-ore amalgamation (WOA) using copper plates coated with mercury placed on five concrete sluices positioned at the discharge of the final hammer mills. Each sluice is 340 cm long and 52 cm wide, covered with copper–mercury plates, with a gradient sloping of approximately 10°. The tailings, contaminated with mercury, are collected in a designated pit, typically filled within six to seven months, and subjected to a cyanidation process. This involves leaching the amalgamation tailings with a sodium cyanide solution (concentration not disclosed by the operators) with the leachate being filtered at the bottom of the vats with assistance of cotton rags. The solution passes through activated carbon columns, and it is recirculated for a minimum of 30 days. The loaded carbon is sent to a non-disclosed facility in the state of Bahia for elution and final gold recovery by electrowinning. Gold production by cyanidation is sporadic, typically occurring every six months or more, once sufficient tailings have accumulated to fill the vats.

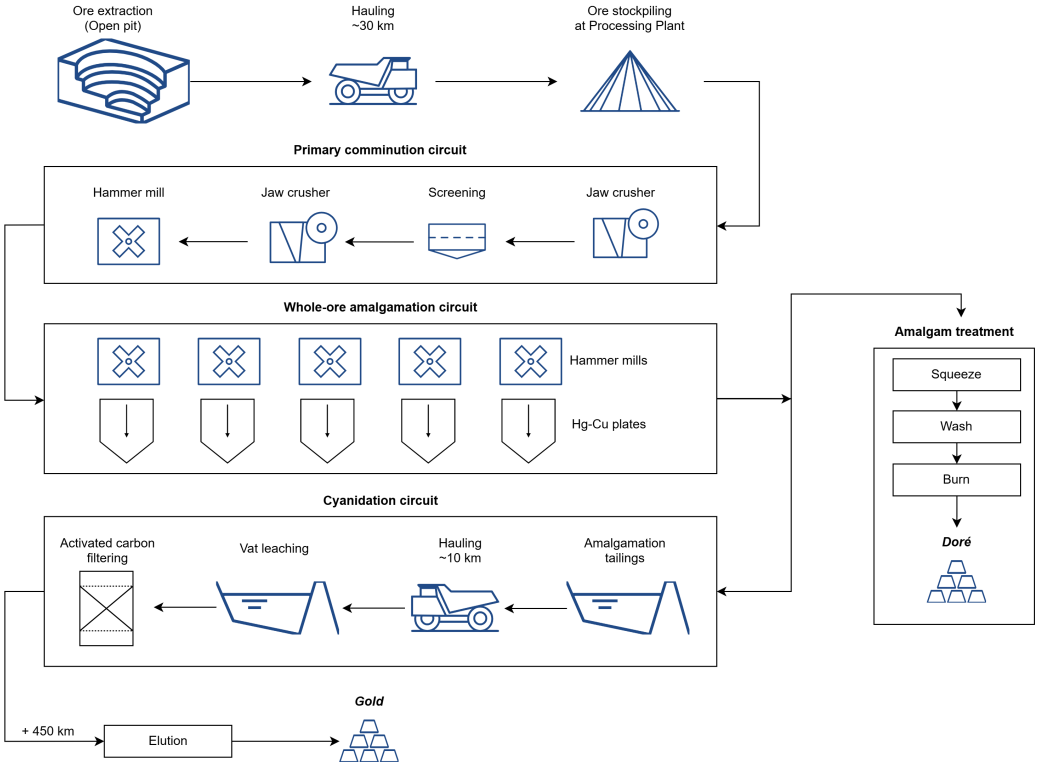


Figure 2. Operational workflow of the processing plant (adapted from Silva et al. [28]).

Throughout the sluicing cycle, the plant manager sporadically applies additional mercury ("azougue" in the local jargon) to the copper plates. At the end of each working day (typically an 8-hour cycle), the amalgam is collected by scraping the copper plates, squeezing the material through a porous cloth, washing, weighing, and storing it. The accumulated amalgam is typically burned at the end of each month, or as per the directives of the mine owners. This burning process is conducted using a torch in a steel retort within a residential area located in the center of the Serrita municipality. No information was disclosed by the processing operators regarding the total mercury used or bought by them.

The managing operator of the comminution and amalgamation plant was unable to provide precise annual ore processing figures but estimated a daily throughput of around 15 to 20 tpd, with 18 tpd being a plausible average, as the operational capacity varies with the characteristics of the ore, equipment and personnel. According to him, approximately

250 to 300 trucks, each carrying around 15 tonnes of ore, are transported to the cyanidation plant annually (3750 to 4500 tonnes/a).

Cyanidation of the amalgamation tailings is conducted by vat leaching without prior knowledge of the tailing's gold and silver grades. Additionally, no systematic control is implemented to regulate the concentration of sodium cyanide applied during the leaching process. Instead, the operator relies on a syringe filled with an unknown reagent to "assess" cyanide levels, possibly being a solution of silver nitrate ($AgNO_3$) with potassium iodide (KI) as indicator, which turns to a yellowish turbidity when Ag^+ reacts with iodine anions. Control of pH is made with soda ash (sodium carbonate) and pH measuring strips, keeping it around 10.5. The cyanide solution in the vats and recirculation tanks remains directly exposed to sunlight, subject to vaporization and oxidation of the reagents. There was no available estimate regarding the total amount of sodium cyanide used at the site.

Adjacent to the primary processing site, an artisanal gold mining cooperative operates, comprising at least 40 small processing units, with more than 300 miners, that also employ whole-ore amalgamation. These units vary in scale, utilizing between three and eight sluices with copper-amalgamation plates each. Due to the restricted availability of mercury, which, in Brazil, depends on multiple government permits, its procurement is conducted through illegal means, though the specific source and annual acquisition volumes remained undisclosed.

2.2. Field Sampling and Laboratory Analysis of Au, Ag and Hg

Based on the established AGM literature, a three-day sampling campaign was conducted at the processing site to provide a diagnostic snapshot of routine circuit performance rather than a long-term production audit [14,20,21]. During the WOA stage using sluices with Hg-Cu plates, feed and tailings pulp samples (250 mL each) were collected every 15 min over a 4-h period and deposited into separate 10-L buckets to form daily composite samples. Once sampling was complete for each stream, solids were allowed to settle, excess water was decanted, and the material was sun-dried. Additionally, representative composite samples were collected from multiple points in the exhausted pile of tailings to represent the final cyanidation tailing sample. For each process stream sampled, dry composite samples weighing approximately 1.5 to 2.0 kg were obtained. Subsequent preparation included homogenization, splitting, and submission of triplicate 50 g subsamples of each process stream (n = 9 analyses in total, being 3 per stream) to an SGS laboratory for gold (Au) and silver (Ag) content analysis. Mercury (Hg) concentrations were also evaluated in sluice and cyanidation tailings. Although longer campaigns are required to capture seasonal or monthly ore fluctuations, this nine-replicate dataset provides a statistically robust basis for the recovery estimates, that capture both analytical and short-term operational variability over the consecutive days.

Gold analyses were conducted using the SGS FAA505 protocol, which involves fire assay with a 50 g sample charge followed by quantification via atomic absorption spectrometry (AAS). In this method, the pulverized sample is melted with a mixture of fluxing agents including lead oxide (PbO), borax, and flour as a reducing agent, allowing gold and other precious metals to be collected into a lead button. After cupellation at high temperature (≈950 °C), a precious metal bead is recovered and dissolved in aqua regia. The gold concentration in the resulting solution is then determined using AAS.

For total digestion of the solids, a finely ground aliquot of approximately 0.25 to 0.5 g was digested with a mixture of hydrochloric (HCl), nitric (HNO_3), perchloric ($HClO_4$), and hydrofluoric (HF) acids, enabling near-total dissolution of silicate and sulfide minerals. The resulting solution was analyzed by Inductively Coupled Plasma Optical Emission Spectrometry (ICP-OES, Agilent 5110 ICP-OES, Agilent Technologies, Santa Clara, CA,

USA) to quantify the elements. Silver and mercury concentrations were determined by this method using the SGS protocols GE_ICP40Q and ICM14B, respectively. Mercury was also analyzed in some feed samples by flameless AAS at the SATC Technology Center, Criciuma, Santa Catarina State, Brazil.

2.3. Grain Size Analysis

Grain size distribution analyses of three sluice tailing samples were carried out using 1 kg sample, previously sun-dried. The samples were collected from the tailing buckets related to the three days of sampling, and wet screen analysis was performed by passing the slurry through a stacked series of sieves with decreasing aperture sizes: 600, 300, 150, and 75 µm. Each sieve was gently agitated to allow finer particles to pass through to the appropriate level. The material retained on each sieve was collected, rinsed, and then transferred to drying plates. Following oven-drying, the sized fractions from each sieve were weighed and analyzed for gold (Au) and silver (Ag) by fire assay-AAS. This distribution data identifies the specific grain size fractions where precious metals were not captured by the amalgamation plates.

2.4. Mineralogical Analysis

One kilogram of a composite tailing sample (from the 3-day buckets) was screened to obtain the grain size fraction, −106 +53 µm. This fraction was concentrated in bromoform (s.g. = 2.89) and the magnetic material was separated with a hand magnet. The mass of this non-magnetic heavy material was 0.34% of the original sized fraction. The non-magnetic fraction was analyzed in a Scanning Electron Microscope (FEI Quanta 400 FEG, FEI Company, Hillsboro, OR, USA) equipped with Energy Dispersive X-ray Spectroscopy (EDS Bruker XFlash 660, Bruker Nano GmbH, Berlin, Germany) and X-ray Diffraction (Bruker D8 ADVANCE, Bruker AXS GmbH, Karlsruhe, Germany) at the Center of Mineral Technology (CETEM) in Rio de Janeiro, Brazil. The objective was to characterize the primary heavy minerals and determine the specific points of precious metal loss.

2.5. Pulp Density Measurement in Sluices

To evaluate the pulp density during sluicing, measurements were conducted following the approach proposed by Veiga and Gunson [15], using a portable digital scale and a 250 mL laboratory beaker. The beaker was filled with slurry collected from the feed of the sluice, where the flow is representative of the operational pulp conditions. The mass of the slurry-filled beaker was measured. With the known total volume of the beaker and the measured mass of the slurry, the mass of solids contained in the pulp was estimated through Equation (1):

$$M_s = \frac{M_p - V_p}{1 - \frac{1}{\rho_s}} \quad (1)$$

where M_s is the mass of solids, M_p is the mass of the beaker filled with pulp, V_p is the total volume of the beaker, and ρ_s is the specific gravity of the solids, which is assumed to be 2.7 in most ores processed by sluices [15]. The % of solids in the pulp was then calculated by Equation (2):

$$\%\ \text{Solids} = \frac{M_s}{M_p} \cdot 100 \quad (2)$$

2.6. Statistical Treatment of Data and Metallurgical Balance of Au and Ag

For each process stream, arithmetic means ($\bar{G}$) and standard deviations (s) of Au and Ag grades were computed from the replicate analyses and used as inputs to the metallurgical balance calculations.

Metallurgical balances for gold and silver were performed based on procedures adapted from Balzino et al. [20] and Anene et al. [21]. As the whole-ore amalgamation sluice retains only the precious metals, it is reasonable to assume that the mass of feed is very similar to the mass of tailing. Therefore, the % of gold (Au) and silver (Ag) recoveries were calculated based on the grades of the feed and tailings streams using Equation (3):

$$\% \text{ Recovery} = \left(1 - \frac{\bar{G}_{tail}}{\bar{G}_{feed}}\right) \cdot 100 \quad (3)$$

where $\bar{G}_{tail}$ stands for the average gold or silver grade of the tailings and $\bar{G}_{feed}$ is the average Au or Ag grade of the feed.

The standard deviation of the recovery (SD_{rec}) was determined through the propagation of uncertainty associated with the analytical measurements of feed and tailings grades. The analytical data used in the recovery calculations were obtained from independently prepared and homogenized composite samples, each subjected to triplicate subsampling and laboratory analysis. These conditions, coupled with the fact that each sampling campaign was conducted on a different operational day using distinct ore types, support the assumption that the feed and tailings grades can be treated as statistically independent random variables [29]. This assumption allows the use of the propagation of error formula for a ratio of independent variables, which estimates the SD of recovery (%) as presented in Equation (4):

$$SD_{rec} = \sqrt{\frac{s_{tail}^2}{n \cdot \bar{G}_{feed}^2} + \frac{\bar{G}_{tail}^2 \cdot s_{feed}^2}{n \cdot \bar{G}_{feed}^4}} \cdot 100 \quad (4)$$

where $\bar{G}_{feed}$ and $\bar{G}_{tail}$ represent the average Au or Ag grade of the feed and tailings samples as well as s_{feed} and s_{tail} stand for the standard deviation of the feed and tailings, respectively.

Recovery uncertainties are expressed as 95% confidence intervals (≈2× SD) [30]. The overall recovery $R_{total} = R_{amal} + (1 - R_{amal}) \times R_{cyan}$ combines the amalgamation and cyanidation stage recoveries. While the stage recoveries are calculated from physically independent sample sets (different feed-tailings pairs, collected at different times, analyzed in separate batches), supporting the treatment of their measurement errors as independent, the mathematical combination introduces a dependency that is not captured by the simple independent-variable propagation formula. As a consistency check, overall recovery was also calculated directly from the feed and final cyanidation tailings grades: $R_{total} = 1 - G_{final}/G_{feed}$. This alternative calculation yields 86.7% ± 3.12% (2 × SD), which is in close agreement with the value obtained by stage combination (86.6% ± 2.89%), confirming that the propagation approach does not materially misestimate the combined uncertainty.

3. Results and Discussion

3.1. Metallurgical Balance of Au and Ag

The metallurgical balance of gold and silver, computed from the three-day sampling campaign, is summarized in Table 1. Feed grades represent the arithmetic mean of nine analyses per stream (three daily composites × three replicate subsamples each), with uncertainties expressed as 2 × SD.

The metallurgical balance reveals distinct recovery efficiencies for Au and Ag at each processing stage. The initial WOA stage, processing feed at 2.96 ± 1.53 ppm Au and 8.09 ± 1.65 ppm Ag, yielded recoveries of 44.7% ± 11.7% (2 × SD) for Au and 11.5% ± 15.4% for Ag. The low Ag recovery suggests that silver is not solely present as

electrum or native silver amenable to amalgamation; it likely also occurs as silver minerals or fine inclusions within gangue phases that are unresponsive to mercury [31].

Table 1. Gold and silver grades and percentage recoveries in amalgamation, cyanidation, and overall recovery.

	Gold				Silver			
Process	$\bar{G}_{feed}$	$\bar{G}_{tailing}$	% **Recovery**	**2 × SD**	$\bar{G}_{feed}$	$\bar{G}_{tailing}$	% **Recovery**	**2 × SD**
Amalgamation	2.96	1.64	44.7	11.7	8.09	7.16	11.5	15.4
Cyanidation	1.64	0.40	75.8	4.32	7.16	4.80	32.9	9.24
Overall recovery	2.96	0.40	86.6	2.89	8.09	4.80	40.6	4.58

Different studies report % Au recovery by amalgamation to be around 20% to 30% [7,32–34], however, given the adverse pulp density documented in Table 2 (42.6% ± 17.6% solids), the 44.7% recovery is notably higher than these reference values. The plant manager reported that this thicker slurry is an intentional effort to conserve water, a scarce resource in the region. However, such high and inconsistent pulp densities significantly increase slurry viscosity, impede particle stratification, and hinder the effective settling of dense minerals like gold to the plate surfaces, leading to gold losses in the sluice tailings. Therefore, achieving an average amalgamation recovery approaching 45% under these suboptimal upstream conditions could be interpreted as relatively effective, and this suggests that the gold particles are predominantly free and coarse under the comminution conditions at this site.

Table 2. Percentage of solids in the sluice feed pulp.

Sample	**Mass of Pulp (g)**	**Volume of Pulp (mL)**	**Mass of Solids (g)**	% **Solids in Pulp**
1	347	250	154	44.4%
2	313	250	100	32.0%
3	380	250	206	54.3%
4	362	250	178	49.1%
5	328	250	124	37.8%
6	347	250	154	44.4%
7	370	250	191	51.5%
8	303	250	84	27.8%
9	341	250	145	42.4%
Average	343	250	148	42.6%
2 × SD	51.1	0	81.1	17.6%

Subsequent cyanidation of the amalgamation tailings achieved gold and silver recoveries of 75.8% ± 4.32% and 32.9% ± 9.24%, respectively. The reported confidence intervals reflect the precision of triplicate fire-assay measurements propagated through the recovery equation; they capture analytical and short-term sampling variability but do not encompass the full operational uncertainty arising from intermittent cyanidation cycles (every six or more months), unmonitored NaCN concentrations, uncontrolled solution exposure, or feed-grade fluctuations over the accumulation period. Gold recovery from cyanidation occurs intermittently, every six months or more, once sufficient tailings have accumulated to fill the vats. The operation processes 3750 to 4500 tonnes of amalgamation tailings annually. The high cyanidation recovery demonstrates that a significant portion of the gold not captured during amalgamation, whether due to incomplete liberation or inefficient plate retention, remained accessible to cyanide leaching. The low overall silver recovery, despite the fine grind, is consistent with either refractory silver mineralogy or leaching conditions not optimized for silver dissolution [35].

The calculated overall recoveries combining the contributions of both stages, amalgamation and cyanidation reached 86.6% ± 2.89%, indicating that the combined process effectively captures a large majority of the gold from the feed. This suggests the ore contains gold amenable to both amalgamation and cyanide leaching. However, the overall silver recovery stands at 40.6% ± 4.58%, confirming that the process, as configured, is less efficient for silver extraction. This relative inefficiency may partly reflect operational priorities: plant management reportedly treats silver as a secondary product, and the cyanidation circuit is not optimized for Ag extraction.

A temporal mismatch must be acknowledged: the cyanidation feed consisted of amalgamation tailings accumulated over at least six months prior to the sluice sampling campaign, and the calculated overall recoveries assume that ore head grades and mineralogical factors remained consistent during that period. This assumption is geologically reasonable, as all processed ore originates from the same shear-zone-hosted deposit (Salgueiro Complex) and the plant does not blend ore from multiple sources. Nevertheless, this operational gap should be considered when interpreting the results. Future studies should prioritize multi-campaign sampling in which all three streams are collected contemporaneously.

3.2. Grain Size Analysis

The gold distribution across grain size fractions of the tailings is shown in Figure 3. The feed material, ground by hammer mills with a 1 mm screen opening, is relatively fine: 45% passes 150 µm and 59.1% of the mass is distributed between −600 and +75 µm. The gold distribution (average of three days of sampling) indicates that 33% (range = 26.1 to 41.7%) of the gold was lost in the −75 µm fraction. This fine fraction likely comprises two populations: (a) fine liberated gold particles that passed through the amalgamation plates without contacting the Hg surface, and (b) fine gold locked within gangue that was not exposed to amalgamation. The higher grade of gold in the tailing fraction +600 µm (3 ppm), comparable to the average feed grade of 2.96 ppm, raises two hypotheses: (i) coarse liberated gold particles that were not captured by the amalgamation plates, or (ii) fine gold inclusions locked within coarse gangue particles.

Amalgamation is primarily effective for recovering liberated gold particles larger than 75 µm [36]. The fact that approximately 67% of the lost gold reports to the +75 µm fractions could reflect either incomplete liberation or inefficient capture of liberated particles. Given the high pulp density documented in Table 2, the latter mechanism appears to be at least partially responsible.

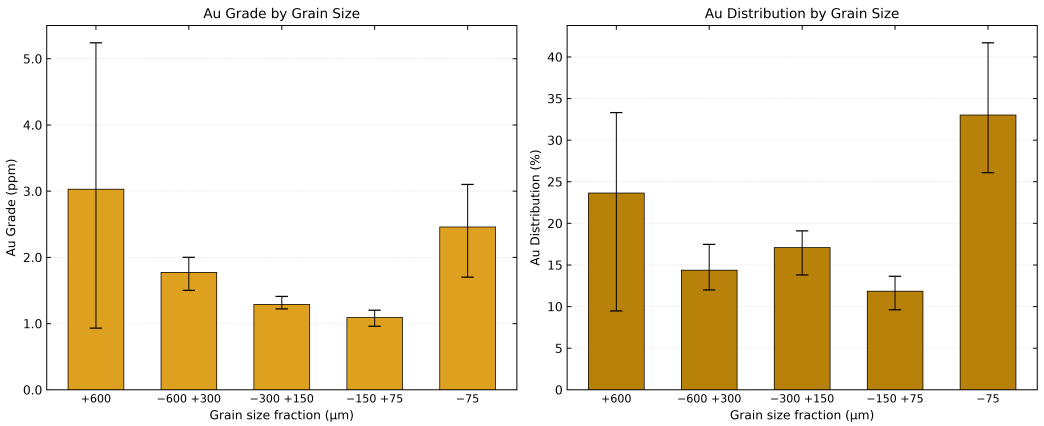


Figure 3. Gold grade and distribution in grain size analysis of the average results of three amalgamation tailings (range bars represent minimum and maximum values of the three daily campaigns).

Silver distribution across the three tailing samples reveals a significant enrichment in the fine fraction; specifically, the −75 µm fraction yields a grade of approximately 13 ppm Ag and accounts for 44% of the total silver lost (Figure 4). The silver distribution differs from that of gold, suggesting that gold particles are not the sole carriers of silver in this deposit.

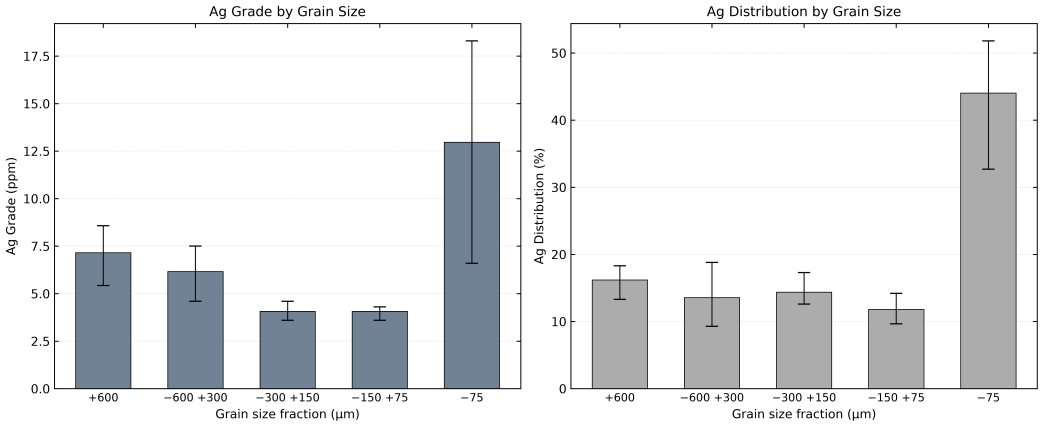


Figure 4. Silver grade and distribution in grain size analysis of the average results of three amalgamation tailings (range bars represent minimum and maximum values of the three daily campaigns).

3.3. Mineralogical Study

The XRD Rietveld refinement of the non-magnetic heavy fraction (−106 + 53 µm) is presented in Figure 5. Among the heavy and sulfide phases identified, goethite (6.4%) and arsenopyrite (4.8%) were most abundant, followed by pyrite (4.5%), ilmenite (3.8%), hematite (3.0%), almandine (2.9%), galena (1.7%), chalcopyrite (0.9%), and monazite (0.7%). SEM-EDS additionally identified pyromorphite, scheelite, apatite, and spessartine in trace amounts.

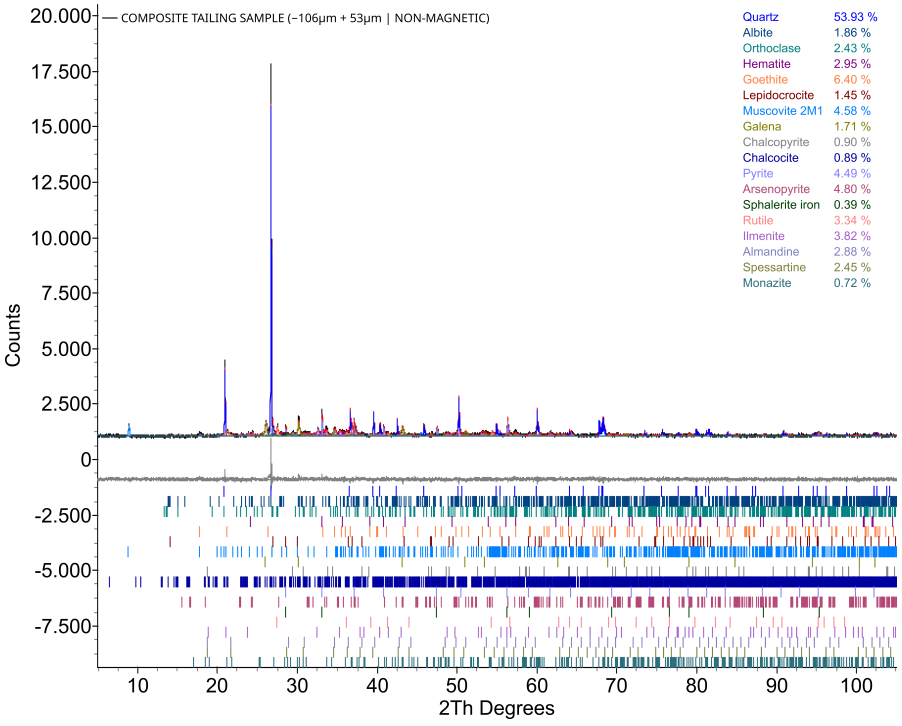


Figure 5. XRD Rietveld refinement of the non-magnetic heavy fraction (−106 +53 µm). Phase abundances are from the Rietveld refinement.

Several free gold particles of electrum (Au-Ag) were observed, with some featuring mercury at grain boundaries. Flameless AAS analyses of three feed samples yielded Hg

grades of 0.05, 0.95, and 1.0 ppm, making the presence of a primary mercury mineral in the ore unlikely. The mercury detected at electrum grain boundaries is therefore interpreted as anthropogenic.

The observation of Hg at electrum grain boundaries (Figure 6) provides further insight into this physical detachment mechanism. If Hg were present as a physically adsorbed metallic film, EDS would show a continuous or patchy coating across the particle surface. Instead, the Hg signal is concentrated at grain boundaries, a pattern characteristic of amalgam that formed on the particle surface and was then mechanically sheared off under the high shear forces in the dense pulp, with residual Hg remaining in micro-crevices. This confirms that amalgamation occurred on the plate but physical retention failed due to excessive pulp viscosity. A definitive discrimination between liberated and locked gold populations would require automated mineral liberation analysis (e.g., QEMSCAN or MLA), which is recommended for future work.

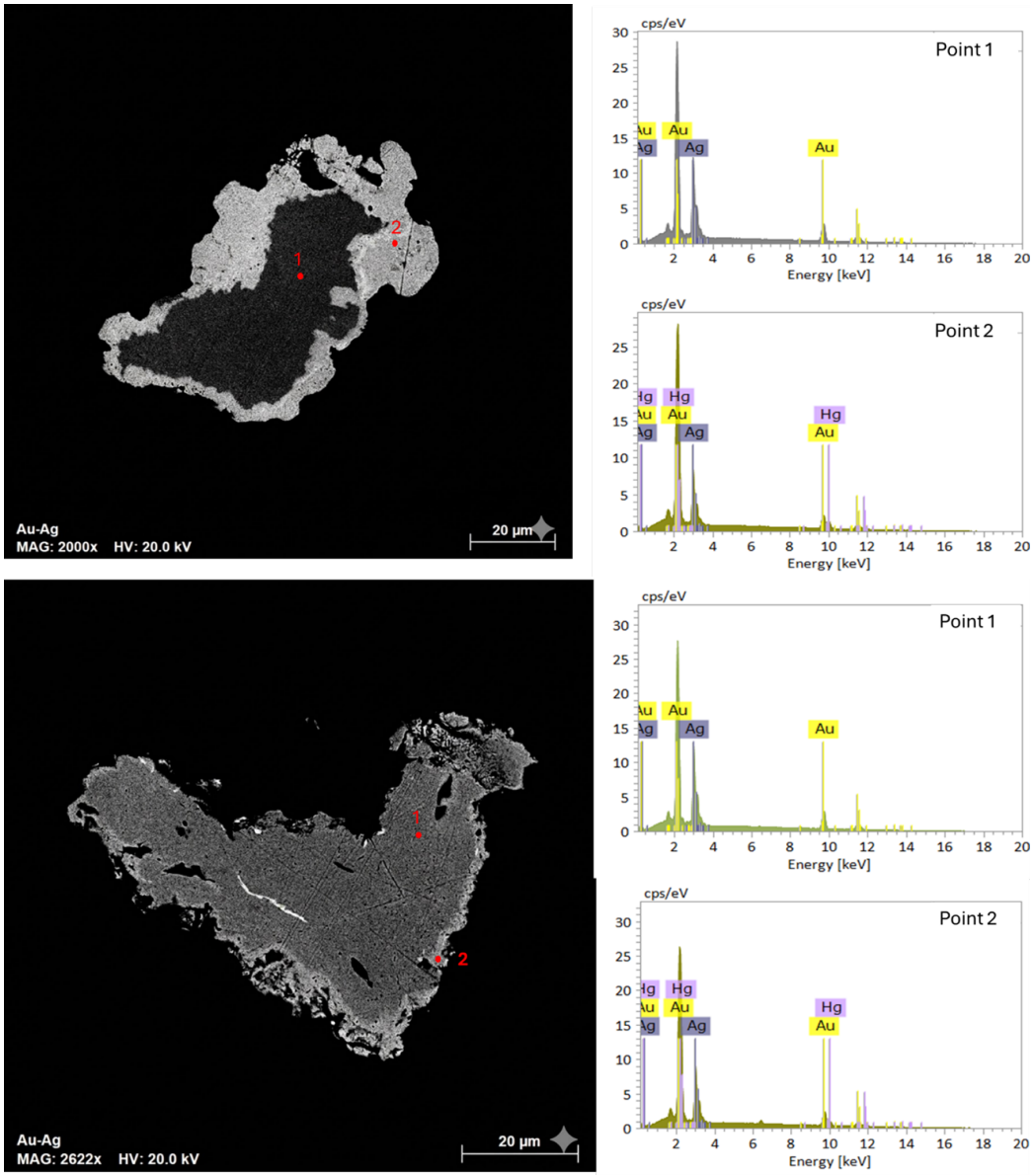


Figure 6. Back-scattered images and EDS spectra of Au, Ag and Hg. Free grains of electrum (images from CETEM, Rio de Janeiro, Brazil).

EDS analysis of 16 points across four electrum grains revealed an average composition of 26.8 ± 15.8% silver and 55.3 ± 27.3% gold (2 × SD). Beyond electrum, silver was

also detected in copper sulfides, pyrite, sphalerite, and galena, as well as discrete silver sulfide phases.

ICP-OES analyses of nine feed samples could not detect sulfur above 0.02%. Based on elemental grades and mineral stoichiometry, the estimated total sulfide content of the feed is 0.049%, comprising chalcopyrite (0.023%), arsenopyrite (0.011%), and galena (0.010%).

To complement these direct observations, elemental correlation analysis of the feed ICP-OES data offers preliminary insight into the mineralogical hosting of silver. Geochemical correlation coefficients are suggestive but do not constitute mineralogical proof. Silver does not exhibit the strongest correlation with gold ($r = 0.80$), instead showing higher affinity for Pb ($r = 0.87$) and S ($r = 0.89$), consistent with the SEM-EDS identification of galena and silver sulfide as silver-bearing phases. The correlation with As ($r = 0.83$) suggests a possible association with arsenopyrite, though not confirmed by SEM-EDS. Correlations with Cr ($r = 0.92$) and Fe ($r = 0.90$) raise the hypothesis of Ag in chromite, reported by Beurlen [25] in the Borborema Province but not identified by XRD or SEM. The correlation with Ni ($r = 0.84$) suggests pentlandite as a potential silver-bearing phase [37]. A more detailed mineralogical study is recommended to test these hypotheses. The small sample size ($n = 9$) warrants caution in interpretation.

3.4. Assessment of Plant Operational Capacity and Mercury Losses from Doré Outputs

Estimating daily processing throughput is a persistent challenge in AGM operations, where operational records are scarce and ore grades are variable. Common approaches, such as volumetric tracking and time-based capacity estimation, are both prone to inaccuracies. As an alternative or complementary approach, metallurgical reconciliation allows for throughput estimation by back-calculation using measured outputs and established process parameters [38–40].

Amalgam was collected from the copper plates over four operational days (total mass: 460 g; average: 115 g/day). On the fourth day, the accumulated amalgam was retorted using the operators' conventional method, with the exhaust tip submerged in a water bottle to condense mercury vapors. The resulting doré weighed 190 g (average: 47.5 g/day) and 210 g of mercury was recovered by condensation. By difference, the amalgam contained 270 g of mercury, of which 60 g was lost to the atmosphere during retorting (average: 15 g/day); the retort thus captured approximately 80% of the amalgam mercury. Mercury may not be completely lost from the amalgam during the retorting process, so that the resulting doré retains residual mercury, which is then released when the gold is melted at the shop prior to sale.

To reconcile the analytical data with the field-observed data, a linear model integrating the expected doré mass (calculated) was based on the average WOA recoveries of Au and Ag (these obtained from feed and tailings sampling), through Equation (5):

$$\text{Mass of doré}_{(\text{calculated})} = (\text{TPD} \cdot \text{Au}_{\text{feed}} \cdot \%R_{\text{Au}}) + (\text{TPD} \cdot \text{Ag}_{\text{feed}} \cdot \%R_{\text{Ag}}) \quad (5)$$

where TPD stands for the plant's ore throughput in tonnes per day; Au_{feed} and Ag_{feed} are the average Au and Ag feed grades; and $\%R$ is the % recoveries of Au and Ag in the WOA circuit, respectively.

Figure 7 depicts the modeled relationship between varying TPD values (10 to 25 tpd) and mass of doré (calculated) in Equation (5). The calculated values were plotted against the average doré production from the four-day sampling campaign at the studied site (47.5 g doré/day). A sensitivity analysis incorporating plausible doré impurities demonstrates that minor variations in product purity can shift the inferred throughput, therefore, the importance of accurately characterizing doré composition when reconciling metallurgical balances. The results indicate that producing 190 g of doré over four days (or

47.5 g per day) corresponds to a daily throughput of approximately 19 to 21 tpd, considering as high as 5 to 10% impurities in the doré. This is not far from the operational capacity stated by the plant manager (10 to 20 tpd), however, despite its practical value, this throughput estimate faces several uncertainties. A primary concern is ore grade variability, that can lead to daily fluctuations in metal content that may not be fully captured by limited sampling campaigns. Additionally, the model assumes that average recoveries derived from laboratory tests remain constant regardless of throughput. In practice, factors such as changes in pulp density, residence time, and mercury amalgamation efficiency can reduce recoveries at higher processing rates. Furthermore, the analysis of doré impurities only accounted for residual mercury. The presence of other components like copper or gangue inclusions would affect both the mass and assay results, introducing further errors into the back-calculated throughput.

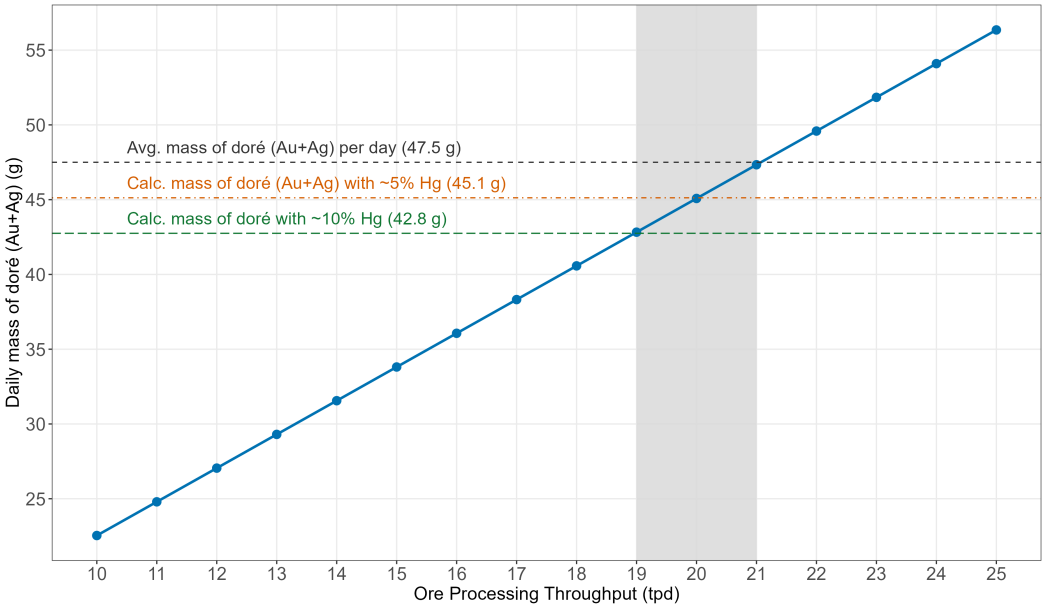


Figure 7. Estimated Au and Ag production via whole-ore amalgamation as a function of processing throughput. The hatched area indicates the most probable ore processing throughput of the studied site.

Mercury losses were estimated by multiplying the Hg grade in tailings by the plant's daily throughput. Six subsamples from three WOA tailing composites were analyzed by ICP-OES, yielding an average Hg concentration of 12.3 ppm ± 24.4 ppm (2 × SD). The large standard deviation relative to the mean reflects substantial spatial heterogeneity in the tailings, arising from: (i) the sporadic addition of liquid Hg to the copper plates during the sluicing cycle, which creates a highly heterogeneous Hg distribution; (ii) possible "nugget effect" whereby individual Hg droplets carry disproportionate mass; and (iii) the small number of analyses available (n = 6). This variability is itself an informative finding, demonstrating that Hg distribution in WOA tailings is far from uniform, with implications for environmental sampling design at analogous sites. All mercury loss figures presented in this section are estimates derived from metallurgical mass balances and do not represent direct measurements of gaseous or aqueous Hg emissions. Based on the 19 to 21 tpd throughput estimate, this corresponds to approximately 247 g of Hg lost daily with WOA tailings, plus an estimated 15 g/day during amalgam retorting.

The ratio of mercury lost to gold produced ($Hg_{(lost)}$:$Au_{(produced)}$) is an important parameter for quantifying mercury releases in AGM operations [41]. This ratio is highly variable, influenced by the amalgamation method, operational scale, miner skill, ore grade,

and use of recovery technologies such as retorts [2,20]. Whole-ore amalgamation typically results in ratios > 1, whereas concentrate amalgamation with retorts can reduce this to as low as 0.01 [3,15,42]. Table 3 places the present study's results in the context of published AGM metallurgical data from other regions.

Table 3. Comparison of Au recovery and Hg:Au ratios from selected AGM studies.

Study	Country	Ore Type	Process	Au Recovery (%)	Hg:Au Ratio
This study	Brazil (NE)	Primary, orogenic	WOA + CN	86.6 (overall)	5.5 [a]
Anene et al. [21]	Nigeria	Primary	Conc. Amal.	20–30	3.35
Balzino et al. [20]	Brazil (Madeira)	Alluvial	Conc. Amal.	N/R	0.1–0.2
Cordy et al. [43]	Colombia	Primary	WOA	N/R	14.6
García et al. [44]	Colombia	Primary	WOA (post-interv.)	N/R	6.5
González-Vásquez et al. [34]	Ecuador	Primary	WOA	20–30	N/R
Government of Suriname [45]	Suriname	Mixed	WOA	N/R	3.3
Yoshimura et al. [46]	Global avg.	Mixed	Mixed	N/R	1.2–4.6 [b]

[a] Expressed as $Hg_{(lost)}$:Au + $Ag_{(produced)}$. [b] Regional averages: Africa 1.96, Lat. Am. 4.63, Asia 1.23. N/R = Not reported; WOA = Whole-ore amalgamation; CN = Cyanidation; Conc. Amal. = Concentrate amalgamation.

In the present study, due to the presence of Ag in the samples, we used the $Hg_{(\text{lost})} : Au + Ag_{(\text{produced})}$ ratio, which yielded a value of 5.5 (Table 4). This elevated ratio may be attributable to the limited sample size ($n = 6$) for Hg analyses with high variability (12.3 ± 24.4 ppm, $2 \times$ SD), or potential cross-contamination from supplementary Hg additions during sluicing operations. Nevertheless, the value falls within the range reported for WOA operations in Latin America [46] and is consistent with the high mercury losses characteristic of amalgamation plate systems [3]. Caution is warranted in interpreting this ratio, as low precious metal production inherently inflates it regardless of the absolute mercury loss rate.

Table 4. Mercury mass balance for the four-day sampling campaign in the studied site. All Hg loss values are estimates derived from metallurgical mass balances.

	Throughput (tpd)	No. Operation Days	Avg. Hg in Tailings (g/t)	Hg lost in Tailings (g)	Mass of Amalgam (g)	Mass of Doré (g)	Hg Lost in Evaporation (g)	Total Hg Lost (g)	$Hg_{(lost)}$: Au + $Ag_{(produced)}$
4-day sampling	20	4	12.3	987	460	190	60	1047	5.5
Daily averages	20	1	12.3	247	115	47.5	15	262	5.5

3.5. Limitations and Future Work

Key limitations include: (i) a temporal mismatch between the sluice and cyanidation sampling campaigns, requiring an assumption of feed-grade continuity; (ii) a limited dataset ($n = 9$), especially for the Hg balance where only six subsamples were available; and (iii) throughput estimated by back-calculation from doré mass rather than direct measurement. Future work should incorporate larger multi-campaign sampling, direct throughput measurement, Hg speciation analysis in cyanidation effluents to assess mercury–cyanide complex risks, and pilot-scale gravity concentration testing to validate the proposed mercury-free pathway.

4. Conclusions and Recommendations

This study presents the first integrated metallurgical balance quantifying Au, Ag, and Hg across both amalgamation and cyanidation stages of an artisanal processing circuit. Under the operating conditions evaluated, amalgamation recovered 44.7% of the gold, cyanidation extracted 75.8% of the gold in the amalgamation tailings, and the combined process overall extraction was 86.6% ± 2.8%. The overall recovery links temporally separated

sampling campaigns under the assumption of feed-grade continuity. Silver overall recovery was limited to 40.6% ± 4.58% due to refractory mineralogy. Mercury losses, estimated via metallurgical mass balance, were 262 g/day, yielding a $Hg_{(lost)}$:Au + $Ag_{(produced)}$ ratio of 5.5. SEM-EDS confirmed that excessive pulp density (≈43% solids) caused physical detachment and subsequent loss of free amalgamated electrum to the tailings.

While these findings derive from a diagnostic assessment of a single circuit, the metallurgical and mineralogical evidence supports a strategic shift away from mercury use. The following recommendations are presented as evidence-based hypotheses requiring pilot-scale validation across broader processing scales: (i) Optimize comminution for improved gold liberation and replace whole-ore amalgamation with gravity concentration. (ii) Implement gravity concentration via centrifugal concentrators or shaking tables to produce high-grade concentrates for direct smelting or targeted leaching. This equipment can be coupled with closed-loop water recirculation using settling ponds or thickeners, reducing net water consumption compared to the current once-through sluicing system in water-scarce semiarid regions. (iii) Optimize the existing cyanidation circuit to treat these mercury-free gravity tailings directly. (iv) Prioritize gold recovery over silver, given the refractory silver mineralogy and operational economics.

The metallurgical balance framework demonstrated here remains highly transferable to other operations, serving as a cost-effective diagnostic tool to identify processing inefficiencies and build an evidence-based case for mercury elimination.

Author Contributions: Conceptualization, E.M.S. and M.M.d.V.; methodology, E.M.S. and M.M.d.V.; formal analysis, E.M.S. and M.M.d.V.; investigation, E.M.S. and M.M.d.V.; writing—original draft preparation, E.M.S.; writing—review and editing, M.M.d.V. and G.D.T.; supervision, M.M.d.V. and G.D.T.; funding acquisition, M.M.d.V. All authors have read and agreed to the published version of the manuscript.

Funding: This research was funded by NSERC-Natural Sciences and Engineering Research Council of Canada grant RGPIN-2020-06125.

Data Availability Statement: All data is contained within the article.

Acknowledgments: The authors would like to express their sincere gratitude to John Simas and Jackson for their invaluable support, specifically for allowing the execution of this research within their processing plant and for providing infrastructure for the sample collection and insights into their processing workflow. Special thanks are also extended to Maria do Carmo S. Barreto from Maná Agroindustry for her vital support regarding infrastructure and the logistics of sample shipments. Furthermore, we are deeply grateful to Felipe Emerson André Alves and Flávia Gomes de Souza, from the Multi-user Laboratory for Technological Characterization (LMCT) at CETEM, Rio de Janeiro, for their expertise in conducting the mineralogical analyses and for the insightful discussions that significantly enriched this work.

Conflicts of Interest: The authors declare that the study was conducted in the absence of any commercial or financial relationships that could be construed as a potential conflict of interest. The views expressed are solely those of the authors and do not necessarily reflect those of USP, UBC, or NSERC.

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