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Efficiency of Mineral Processing in Rwanda's Artisanal and Small-Scale Mining Sector

Quantitative Comparison of Traditional Techniques and Basic Mechanized Procedures

Imprint

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Efficiency of Mineral Processing in Rwanda's Artisanal and Small-Scale Mining Sector

Quantitative Comparison of Traditional Techniques and Basic Mechanized Procedures

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Kigali, February 2017

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About this Report

Artisanal and small-scale mining in Rwanda often involves manual mineral processing techniques at the mine site. It is commonly known that these traditional processing techniques tend to be inefficient and, depending on the style of mineralization, may lead to loss of ore particles of economic interest to the tailings or waste fractions. Usually, mineral concentrates are only upgraded to export grade once the product reaches Kigali. At this stage, valuable ore components may have already been lost due to inefficient processing at the mine level. The present study was commissioned by BGR in partnership with the Rwanda Natural Resources Authority in order to evaluate the extent of this issue at a pilot scale. The study sampled representative selection of Rwandan ASM operations (cassiterite, coltan and wolframite) and quantifies the efficiency of both traditional mineral processing and mechanized procedures. Further, the study evaluates and compares the economic impact of these different processing approaches at the mine level.

About the BGR Module of the German Support Program to the ICGLR

The regional German support program to the International Conference on the Great Lakes Region (ICGLR) is implemented both at the regional and at the national level of selected ICGLR member states. As one of the key countries in the ICGLR process, BGR has supported Rwanda since 2011 in implementing the Regional Certification Mechanism and in formalizing the artisanal and small-scale mining sector. BGR’s long-time project partner in Rwanda used to be the Rwanda Natural Resources Authority which recently underwent organizational restructuring into the Rwanda Mines, Petroleum and Gas Board. The BGR project runs until 2019; more information can be found at www.bgr.bund.de/mineral-certification.

Abbreviations

3T	Tungsten (W), Tin (Sn), Tantalum (Ta) ores
Al	Aluminum
As	Arsenic
ASM	Artisanal and Small-Scale Mining
BGR	German Federal Institute for Geosciences and Natural Resources
Fe	Iron
Fe_2O_3	Hematite
Fe_3O_4	Magnetite
$FeWO_4$	Ferberite
G&A	General and Administrative Expenses
GMD	Geology and Mines Department
ICGLR	International Conference on the Great Lakes Region
K	Potassium
MLA	Mineral Liberation Analysis
$MnWO_4$	Hübnerite
NPV	Net Present Value
ppm	Parts per million (same as grams per ton)
QEMSCAN	Quantitative Evaluation of Minerals by Scanning Electron Microscopy
RINR	Regional Initiative Against the Illegal Exploitation of Natural Resources
RNRA	Rwanda Natural Resources Authority
ROM	Run of Mine
RWF	Rwandan Franc (in this report: 1 USD = 781 RWF)
Si	Silicon
SnO_2	Cassiterite
Ta	Tantalum
Ti	Titanium
USD	US Dollar

Glossary

Alteration	Transformation of a mineral's chemical composition and physical properties induced by weathering
Barren Rock	Material of no or low economic value contained in the ROM which has to be separated from target minerals, also referred to as waste/waste material
Cleaner stage	Reprocessing stage of a multi-stage process. Usually reprocessing of the concentrate fraction of a rougher stage
Comminution	Comminution is the process of reducing the grain size of a particle for example by crushing or grinding
Concentrate	Enriched product of processing, mostly heavy fraction of a final processing stage, containing a high grade of target minerals which allows for further beneficiation
d50	Refers to particle size distribution, specifically the mesh (diameter) where 50 % of the sample mass is passing the screen
Density/specific gravity	particle mass divided by particle volume
Feed	Ore which is treated in the processing
Grade/G	Amount of target minerals in a sample in percent with regard to total sample mass
Intergrown	ref. Interlocked
Interlocked	Opposite of liberated particles. Interlocked grains are valuable mineral grains (target minerals) that are still connected to a matrix of uneconomic minerals (barren rock)
Liberation	Liberated grains are those which only consist of a single mineral and are not connected to others
Magnetic susceptibility	Measure of intensity of magnetic properties of the material
Ore	Naturally occurring rock composed of gangue and minerals containing metals, which can be extracted economically
Recovery rate	Recovery rate is the amount of target minerals that is sorted to the concentrate fraction in percent of the total of target minerals contained in the feed. The fraction of target minerals not recovered in the concentrate is lost to the tailings.
Rougher stage	First processing stage of multi-stage process. Usually to pre-concentrate a feed
Run of mine (ROM)	Raw material (ore) that is produced in a mine, before any processing (upgrading) takes place.
Scavenger stage	Reprocessing stage of a multi-stage process. Usually reprocessing of the waste fraction of a rougher stage
Slimes	ultrafine particles that get suspended in water
Sorting	Process of separation material by its physical properties (ref. to density sorting)
Tailings	Waste material from the light fractions of the processing stages, containing not recovered target minerals
Target Minerals	Minerals of economic value which are to be extracted from ore. For the purpose of this report, target minerals are cassiterite, wolframite and tantalite (coltan), the "3T" ores.
Timbering	Supporting tunnels and other underground facilities with wooden pillars or other constructions from wood
Washing pan	Conical vessel for density sorting

Executive Summary

Mineral exports provide a major share to the export revenues of Rwanda. In 2015, 55 % of forex income generated by principal exports were related to mineral resources. While the sector is currently pushing towards new prospects of different nature like gemstones and iron ore, the 3T (Tin, Tungsten and Tantalum) minerals have been the most important products of Rwandan mining operations since the 1930s. Ore concentrates of 3T minerals provided 79 % of total mineral exports from Rwanda in 2015. The Government therefore considers mining as one of the key sectors to drive economic development and poverty reduction.

While there are few medium-scale mining operations, the major share of Rwanda’s total mineral production is generated by artisanal and small scale mining (ASM) operations. These small operations tend to use manual mining methods, with few or no mechanized mining equipment. This labor-intensive approach provides significant employment for low-skilled workers and gives benefits in terms of flexibility and applicability, considering challenges related to fluctuating mineral markets and infrastructural conditions. However, this mining practice is also thought to be inefficient and unstructured. Apart from deficits in geological knowledge about deposit size and the occurrence of target minerals of economic interest, small-scale operations often do not monitor operational data sufficiently, except for the quantities of produced and sold mineral concentrate. Acquiring sufficient data in order to evaluate and improve single stages of the mine workflow can therefore be challenging.

Apart from the ore deposit’s size and accessibility of the ore body, mineral processing is known to be a major impact factor for the economic feasibility of mining operations. Even though a deposit might contain good ore grades, it will not be considered feasible if the contained valuable minerals cannot be separated from barren rock and gangue to produce a marketable mineral concentrate. Hence, every professional mining operator includes a detailed test series and review of the planned processing scheme in feasibility planning of a mining operation. For such industrial operations the processing efficiency and recovery rate is therefore evaluated before the actual production starts.

Due to low financial investment and lack of management competence in certain key areas (such as exploration and mine planning), ASM operations tend to lack substantial knowledge on the deposit characteristics and the expected production capacities. A proven concept to optimize mineral processing at the deposit scale is often not available, implying risks for losing target minerals of economic interest in the process. Importantly, any theoretical mineral processing concept also needs to consider practical and economic feasibility based on the individual mining operation. Making uninformed investment decisions with regards to mineral processing equipment might otherwise imply the risk for the investor to lose money.

This study is based on a concept developed by the Geology and Mines Department (GMD) of the Rwanda Natural Resources Authority (RNRA) in partnership with the Federal Institute for Geosciences and Natural Resources of Germany (BGR). The study’s objective is the quantitative comparison of the efficiency and economic impact of both artisanal and basic mechanical mineral processing procedures. The processing efficiency in terms of recovery rates, throughput and operational expenditures of artisanal and mechanical processing procedures are assessed. Further evaluation of cost effectiveness of the competing processing approaches is done based on a detailed appraisal of operating data of three mine sites selected as case studies for this analysis. As part of the study assignment, two RNRA counterparts received on-the-job training and were directly involved in all sampling activities and processing tests overseen by the contracted consultant and BGR.

The team participating in this research identified three representative ASM operations, which meet different operation sizes and concepts, as well as the main ore types found in Rwanda. Two of the selected mines are mining deposits of pegmatite-hosted mineralization, one producing a cassiterite concentrate (Mine 1) and the other producing a mixed cassiterite and tantalite (coltan) concentrate (Mine 2). The third mine produces a wolframite concentrate from hydrothermal mineralization in quartz veins. Between 800 kg and 1,400 kg of bulk samples were taken from run of mine material (ROM) of the three mines. Miners homogenized and split the bulks representatively into two sub-samples each. For each mine one sub-sample was treated with artisanal processing measures while the other sub-sample was processed at a mechanical plant. During the processing tests all recoverable products of individual processing steps were sampled. The chemical and physical analysis of these samples allowed for a thorough evaluation of the processing tests.

For all mines the properties of the sampled ROM were identified. It was found that the two pegmatite ores, while showing slight differences in mineralogical composition, varied broadly in terms of their grain-size distributions. Further, within the different grain size fractions, the target elements of economic interest showed fundamental disparities in individual distribution. For mine 1 (cassiterite) the target minerals were distributed mainly in medium-sized and coarse fractions of the ROM. In contrast, the material of mine 2 (mixed Sn-Ta) showed an intense concentration of target minerals in fine grain-sized fractions. For Mine 3 (wolframite), target elements were found to be mainly contained in very coarse grain-size fractions.

Artisanal processing tests were performed according to the typical procedure of regular mining operations. Density sorting was the major processing technique, as it is the most common concentration approach for the targeted minerals. At the artisanal operations density sorting was applied in washing pans (Mine 1 - cassiterite) and ground sluices. Mine 2 (mixed Sn-Ta) used a linear ground sluice, formed as an artificial stream with multiple stages. Mine 3 (wolframite) implemented density sorting in a circular ground-sluice and washing pans. These “traditional” processing techniques were compared to a set of mechanical processing steps kept consistent for all three mines. Density sorting in the mechanical processing schemes was applied by shaking tables and in one case (mine 3 – wolframite) by a spiral concentrator as a pre-sorting stage. Additionally to the density sorting measures, a magnetic separator was used to clean and separate the final products of the mechanical tests. It should be emphasized that the employed mechanical processing approach suffered from the density sorting equipment not being sufficiently adjustable. Therefore, the mechanized processing tests are considered to give a basic indication, rather than optimized results.

The artisanal processing tests demonstrated no or little preparation of the ore (ROM) prior to density sorting. The operational scheme of Mine 1 (cassiterite) included hand-picking of coarse tailings followed by comminution of the grains which have been considered to be rich in target minerals. In contrast, for the mechanical processing intensive preparation steps were applied prior to density sorting. The ore material was screened with mesh sizes of 2 mm or 3 mm, depending on its properties. Over-sized grains were comminuted by different measures. However, while such preparation steps are necessary to properly operate mechanical processing equipment, they might be negligible for artisanal processing techniques; but probably result in a decreased recovery rate. The results of the processing tests showed different outcomes reflecting different mineralization styles and ore characteristics. While artisanal and mechanical processing schemes both proved to be efficient for the ROM of mine 1, recovery rates and concentrate grades achieved for mine 2 and mine 3 can be considered as insufficient. However, in two out of three tested cases mechanical processing outperformed artisanal processing. This is remarkable given that the artisanal processing schemes have been used for the respective materials for a long time, while the mechanical processing approach was a first and not optimized concept.

The gathered operational data in combination with the quantitative outcomes of the processing tests were processed for economic case studies. Annual profit and net present value of individual mines for ten years of operation were calculated in different market scenarios regarding pricing of the produced minerals. For the mechanical processing, investments in different types of equipment was simulated in order to evaluate the cost effectiveness of such investments.

The combined evaluation of raw material, processing schemes, operational data and financial analysis allowed for an integrated interpretation of the findings. Apart from inefficiencies of the actual processing tests, further implications on unfavorable surrounding conditions could be identified. The basic results for the processing tests and economic calculations are shown in the table below. As can be seen, in most cases, mechanical processing schemes led to higher concentrate grades and profits than manual artisanal processing schemes.

Feed #	Processing	Ore Type	Feed Grade	Concentrate Grade	Recovery rate of target mineral	Profit/ loss [USD/a]
Mine 1	Artisanal	Cassiterite	2.17% SnO_2	68.51% SnO_2	60.10% SnO_2	113,000
	Mechanical			71.19% SnO_2	72.84% SnO_2	127,000
Mine 2	Artisanal	Mixed Sn-Ta	0.46% SnO_2 0.20% Ta_2O_5	41.26% SnO_2 9.90% Ta_2O_5	24.50% SnO_2 13.40% Ta_2O_5	65,000
	Mechanical			36.32% SnO_2 10.20% Ta_2O_5	16.95% SnO_2 10.97% Ta_2O_5	46,000
Mine 3	Artisanal	Wolframite	0.50% WO_3	21.41% WO_3	22.32% WO_3	-12,000
	Mechanical			34.07% WO_3	26.02% WO_3	-400

Annual profits and losses refer to calculations at the current (2015) mineral pricing scenario.

Results in the table show two notable features. First, for mine 2 artisanal processing was found to be more efficient, leading to a higher profit for application of artisanal compared to mechanized measures. When analyzing the properties of the ore of mine 2 it was found that the major share of target minerals are contained in the fine grain size fractions. Even though mechanical processing proved to be able to recover a larger share of the fine grained material, it was not able to outperform the artisanal approach in this case. However, it is most likely that a detailed adjustment of operating parameters (calibration) of the mechanical equipment would lead to a significant increase in processing performance. The outcome of the mechanical processing tests should therefore be considered as low-case scenarios. The second anomaly is the apparent lack of profitability of mine 3. The results show an investment in mechanical processing equipment would be beneficial, but still not profitable with regards to the feasibility of the whole operation. Mine 3, being the smallest of the sampled operations, had already reduced mining activity due to low profitability of the operation, prior to the field work for this study. The mine’s negative profitability inferred from the above model calculations can therefore be assumed to be accurate.

Findings of this study include hands-on solutions as well as more sophisticated recommendations. Small-scale miners should in any case apply comminution to coarse-grained ore particles instead of considering them as waste. Chemical analysis demonstrated that coarse grains rejected by artisanal miners due to their supposed low value and not containing significant target minerals still comprised substantial amounts of ore target minerals. Other operations did not even process or evaluate coarse grains, partly leaving major shares of the total contained target minerals unprocessed and unrecovered and, hence, lost as "waste".

As the table above indicates, two out of the three artisanal operations recovered far less than half of the target minerals contained in the processed ore. Therefore, most target minerals are left in the tailings of artisanal processing. It remains to be clarified whether target minerals are recoverable from tailings by additional processing. However, in any case, the tailings as well as coarse-grained waste material should be considered a potential asset and, therefore, be stored properly. A waste management concept would allow reprocessing of prospective materials at a later stage, while decreasing environmental impacts such as erosion and siltation of surface waters.

As the previous two points are giving advice on artisanal processing practices, mechanization of mineral processing in Rwanda was found to be potentially beneficial to reduce losses in the processing of ores. However, regarding the implementation of mechanical processing, various challenges were found. Firstly, the infrastructural conditions on most ASM sites are not necessarily suitable to put mechanical processing equipment in place. Apart from sufficient and steady electricity and water supply, mechanical processing equipment is in need of a minimum amount of raw material to be processed constantly. Especially smaller operations will be challenged to constantly deliver sufficient amounts of ore feed. A solution to this problem might be a centralized processing plant, which can be supplied with material from different mine sites to work continuously and cost efficient.

However, as is proven in this study, the evaluation and testing of a suitable processing scheme is critical, prior to investing in processing equipment. Unfortunately, the number and availability of analytical equipment as well as laboratory scale processing equipment is very limited in Rwanda. To accelerate implementation of advanced processing equipment the accessibility of such equipment needs to be improved. The establishment of a testing plant, including analytical equipment for evaluating and fine-tuning the performance of different processing schemes, should therefore be considered.

The availability of local testing and analytical equipment would further be beneficial for assessing potential secondary deposits related to storage sites of artisanal waste and tailings. When planning to reprocess waste material, detailed processing tests and chemical analysis should be applied in combination with geological evaluation. There are already a number of such prospects in place. Mining was introduced to Rwanda at the beginning of the 20th century. Since then, numerous small- and medium-scale operations have produced tailings and waste, some of which have been stored properly whereas others have been dumped unsystematically. There are already examples for industrial reprocessing of waste sites implemented in Rwanda. Further understanding and unlocking the mineral potential of historic mine waste and tailings could hence play an important role in accentuating the mining sector's economic development potential in Rwanda.

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1. Introduction

In 2015 mineral exports accounted for 55 % of the revenues for principal export from Rwanda. During the very same year, minerals of the so called 3T group provided 79 % of these exports. The artisanal and small-scale mining (ASM) sector in mostly rural areas generated a major share of the nation's mineral production. Therefore, a remarkable percentage of Rwanda's foreign exchange income is generated, and a major contribution to the national economy is done by ASM. Hence, with improved mining practices and stability of this sector the employment rate can be increased, poverty reduced and steady forex income generated for the Republic of Rwanda.

Small scale mining is characterized by low-tech and low-investment operations. Even though this sector exists since ancient times, practices and protocols have changed little. Apart from a lack of knowledge about geological features of the exploited deposits, mining practices are often unstructured and exploitation concepts are missing. Furthermore, one of the main disadvantages when it comes to developing ASM operations is the lack of operational data regarding production. The monitoring of ASM operations done by the mine managers is often limited to the costs for labor and the quantity of produced concentrates. For improving mining practices, additional monitoring and data is necessary. For example, as long as there is no knowledge about the amount of ore which has been processed, the efficiency of artisanal processing cannot be evaluated.

Focusing on improving mining practices in ASM, the Rwanda Natural Resources Authority / Geology and Mines Department (RNRA/GMD) in cooperation with the Federal Institute for Geosciences and Natural Resources of Germany (BGR) developed a concept to attempt a first evaluation of processing efficiency in the national ASM sector. This research aims to gain further understanding of ASM activities in the region to support the implementation of the Regional Initiative against the Illegal Exploitation of Natural Resources (RINR) of the International Conference on the Great Lakes Region (ICGLR). BGR is supporting the national Authorities of Rwanda in charge of Natural Resources in their efforts to improve mining practices, implement Due Diligence measures and formalize the ASM sector, since 2008. The cooperation started with the very first regulatory body of the mining sector in Rwanda, the Rwanda Geology and Mines Authority (OGMR), and continued with the development of the Geology and Mines Department (GMD). During this partnership several studies have been conducted to assess the status and characteristics of the national ASM sector.[1]

The study at hand reports on the implementation of the latest concept developed by GMD and BGR. It compares processing methods of ASM with basic mechanical processing techniques. The intention to conduct this study is based on the assumption that artisanal processing measures are leaving mentionable amounts of target minerals in artisanal tailings, mostly left to erosion. Therefore, it was tested if mechanical processing measures are capable to decrease losses of target minerals during the first mineral beneficiation steps. Additionally, potential financial gains from applying improved processing equipment have been evaluated.

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[1] http://www.bgr.bund.de/EN/Themen/Min_rohstoffe/CTC/Mineral-Certification-Rwanda/Downloads/downloads_rw_node_en.html

In order to compare the efficiency of artisanal and mechanical processing methods, processing tests have been carried out with both processing approaches. For the artisanal tests, the traditional processing measures have been carried out on the ASM sited. Afterwards, basic mechanical processing schemes have been applied to the run of mine material (ROM), sampled from the very same production as for the artisanal tests. The potential set-up of mechanical equipment was matched to be suitable for application in medium sized ASM operations. Therefore, rather than the maximization of possible recovery rates, a decrease in losses of target minerals was sought.

To allow an integrated view, the study gives a basic introduction to the geological features of Rwandan 3T deposits and the main minerals which are mined in ASM. As all test and sampling campaigns aim to cover a representative share of the whole issue, selection of representative mine sites was a major part of the preparations for this study. Apart from geological properties of potential mine sites, operational data and production practices have been taken into account.

In order to generate a common understanding of reviewed processing measures, observed and applied techniques will be introduced briefly to the reader in the methodology part of this study. To further evaluate the performance of the processing tests, detailed knowledge of mineralogical and physical properties of the processed material is necessary. Therefore, the processed material was analyzed with chemical and physical methods to allow for a discussion of its properties and characteristics. Following the findings of this discussion, the performance of the processing tests will be presented and discussed.

The research closes in an economic analysis, based on the findings of the processing tests and gathered operational data of the involved ASM operations. Calculations for this analysis considers different processing approaches (artisanal and different mechanical set-ups) in different mineral pricing scenarios to evaluate the economic feasibility of the operations and potential investments in mechanical processing equipment.

The conclusive review of all tests, findings and calculation led to a number of implications for points of improvement of the current practices. While some recommendations refer to the general situation and facilities in the Rwandan ASM sector, others might be easy and quick to implement under current surrounding conditions. Hence, this study provides a baseline to access points of further interest and basic recommendation for improving mining practices and will hopefully proof to be beneficial to the national ASM sector.

2. Features of Rwandan Geology and 3T Ore Deposits

Rwanda is situated north east within the Kibara Belt which is evolved between two pre-Mesoproterozoic domains: Firstly, the Archaean-Palaeoproterozoic Congo Craton to the West, and secondly the Archaean-Palaeoproterozoic Tanzania Craton to the East and South. Recently, Tack et al. subdivided the traditionally known Kibara Belt into two segments of the same age: the Karagwe-Ankole Belt (KAB) (Rwanda, Burundi, Maniema and Kivu in the DRC) and the Kibara Belt (type area Katanga region DRC). The two segments are separated by the palaeoproterozoic Rusizian terrane which is an extension of the NW-SE trending Palaeoproterozoic Ubende Belt in Tanzania (Tack, 2010). Figure 1 shows a simplified sketch map.

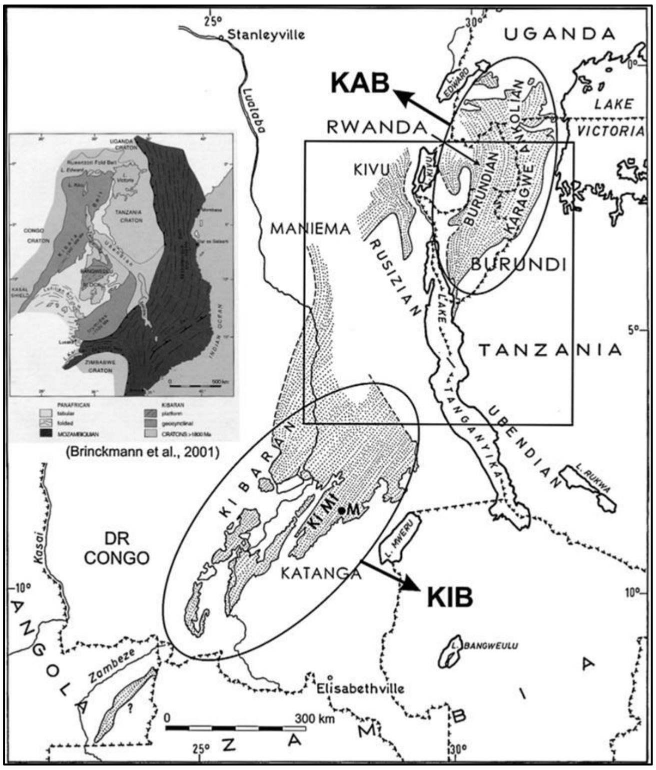

Figure 1: Sketch Map of the "Kibara Belt senso lato", showing the General NE–SW Extension of the Belt

The stated area is hosting a large metallogenic province of numerous ore deposits, e.g. such as Nb, Ta, Sn, W, Au. Also, there are good signs indicating the presence of base metals. The origin is discussed either as granite related or as metamorphogene.

The mentioned mineralization occurs in different styles as primary granitic pegmatite, quartz vein/ shear zone hosted in different metasediments or sediment related as secondary deposits (placers, either alluvial, eluvial, or colluvial), and in lateritic environments.

The basic ore geology of the Kibaran has been summarized by W. Pohl (Pohl, 1987) and re-evaluated by Biryabarema, Pohl and Lehmann (Pohl et al., 2013) in three types as follows.

Basic ore geology of the Kibaran Belt:

1. Pegmatite hosted tin and tantalum mineralization,
2. Hydrothermal tin and tungsten mineralization in quartz veins, and
3. Hydrothermal gold quartz vein and breccia related mineralization.

Kibaran pegmatites were found to carry mainly Nb, Ta, Sn, Li, Rb, REE, Cs, and Be. Varlamoff described in detail the pegmatites and their mineralogy in relation to the granite cupolas in Rwanda (Varlamoff, 1972). Hydrothermal alteration and subsequent kaolinization is assumed to be the major driver for these occurrences. In Rwanda, most economically important pegmatites can be classified as LCT- type pegmatites (Li-Cs-Ta bearing).

The hydrothermal Sn- and W- quartz mineralization occurs in veins and veinlets forming stockworks and vein fields. Positioning is clearly controlled by anticlines and domes which acted as anticlinal fluid traps. Target mineral concentrations here tend to increase towards the contact zone of veins and host rock. The major vein fields are found separated from the pegmatite districts. However, rare transition zones of both mineralization were described, as well (Varlamoff, 1972). As a matter of fact, the major tin districts (Tin Belt) are clearly separated from the tungsten districts (Tungsten Belt).

Mineralization of quartz veins do differ in size of mineralized crystals. Mostly, the thicker the vein, the higher the chance for coarse grained minerals. However, for pegmatites associated properties could be observed. Depending on the intensity of alteration and kaolinization, occurring target minerals differ in size. A more intensely altered pegmatite tends to bear smaller grained barren rocks and target minerals.

Cassiterite

Cassiterite ($\mathbf{Sn}O_2$) is the world's primary tin ore. Cassiterite has a yellow brownish to black color and a light brown to light grey streak. It crystalizes tetragonal and fractures subconchoidal to uneven. Cassiterite is brittle with a Mohs-scale hardness of 6-7. The main sorting criterion that will be used within the test series is its high specific gravity of around 7 g/cm³. A secondary sorting criterion is the nonmagnetic property of cassiterite, which allows separation from other ferro- or paramagnetic dense minerals, notably tantalite. Tin is used mainly for the production of solder, tin plating as well as diverse chemicals and pigments. The economic unit to determine the value of cassiterite concentrate is Sn.

Wolframite

Wolframite ($(Fe, Mn)\mathbf{W}O_4$) is an iron manganese tungstate mineral. It is a mixture of the minerals ferberite (Fe^{2+}-rich) and huebnerite (Mn^{2+}-rich). Wolframite is monoclinic and has a dark gray to brownish color with a reddish brown streak. Its fracture is uneven and rough. Hardness on the Mohs-scale is between 4 and 4.5. Its specific gravity is 7 to 7.5 g/cm³ and represents the main sorting criterion in the processing of wolframite ores. Tungsten is mostly used as an alloy for heavy duty steels in construction or machines and tools. The economic unit to determine the value of wolframite concentrate is WO_3.

Tantalite / Coltan

Tantalite ($(Fe, Mn)(Ta)_2O_6$) has a dark brown to black color and a brownish-red to black streak. Its Mohs-scale hardness is 6-6.5 and its specific gravity is >8 g/cm³. It fractures subconchoidal and is orthorhombic. The main sorting criterion for tantalite is its high specific gravity. The low paramagnetic properties of tantalite allows separating it from ferro- and nonmagnetic minerals such as cassiterite (Okrusch & Matthes, 2009). Coltan is an informal trade name for minerals of the columbite-tantalite series and additional Ta- and Nb-bearing minerals. Tantalite is usually the main Ta-bearing mineral. Columbite ($(Fe, Mn)Nb_2O_6$) is the main Nb bearing mineral. Other Ta-bearing minerals included in coltan in various proportions are microlite, wodginite and tapiolite. The economic unit to determine the value of tantalite concentrate and term used in this study is Ta_2O_5.

3. Methodology

To evaluate potentials of applied and alternative processing practices one month of field work was carried out in Rwanda. During this field work three ASM operations have been visited, their mineral processing evaluated and mechanical processing applied to the very same run of mine (ROM) which has been produced on the respective ASM sites. Furthermore, the collected sample material and products of the processing tests have been analyzed chemically, physically and optically. This chapter gives an overview of the methodology of these activities. To find a common understanding of basic non-chemical processing techniques which are practiced or applicable in ASM, this study also provides an introduction to such techniques. The descriptions consist of observations during the field work and literature review in case of the mechanized processing techniques and equipment. The field work was performed in cooperation with technical staff of the GMD on training purpose to increase understanding of scientific field work and data collection practices. Therefore, at least two GMD technicians were included actively in sampling, processing tests and collection of operating data.

3.1 Limits of Data Acquisition and Test performance

The study at hand presents a first approach into investigating the status quo of the processing capabilities of the ASM sector in Rwanda. It provides indications on how the current situation might be improved, based on the acquired data. It has to be noted that only three out of about 280 active ASM operations have been evaluated and only one sampling point per operation could be tested. Additional tests for different types of deposits and different ores should be considered to gain a more complete image of the current situation.

Mechanical processing tests have been conducted on shaking tables that could not be optimized for processing of low grade ores. Those shaking tables did not provide sufficient adjustment options in respect to the processed material. Up-to-date equipment provides such options. A few minor improvements, for instance better water regulation and reduced amplitude and frequency of the shaking tables, are very likeable to improve the measured performance in terms of concentrate grade and recovery rate sharply. Results would probably be further improved if prescreening for the shaking table would be applied wet and maybe with even more and therefore narrower grain size fractions.

The results of the mechanical processing tests can therefore be assumed as low case scenarios. As mentioned in the title of the study this gives a comparison of basic mechanical processing techniques and artisanal processing measures. An approach with modern processing equipment and detailed adjustment would therefore most probably give even higher recovery rates.

Operational data of the sampled mining companies has been acquired by interviews with the respective mine management during on-site visits. The given statements could seldom be verified independently.

Recovery rates and economic feasibility

To value the relation between recovery rate and economic feasibility, it is mandatory to understand that a recovery rate of 100 % is not achievable with any of today’s mining practices. Also, it is not desirable, as investments and efforts which are necessary to achieve the highest possible recovery rate tend to exceed the gains which are generated by the additional recovery in comparison to a scenario with a lower recovery rate. In general, the reasonable investment and efforts in processing increases with the value of the target mineral and therefore with the additional gains generated by an increased recovery rate. For Gold, which is one of the most valuable mineral resources, recovery rates of industrial operations might be as low as 80 % (SrkConsulting, 2015) and can still be considered good practice. However, such recovery rates will only be achieved with extensive effort and application of multiple processing techniques like physical sorting and chemical leaching in different processing stages. For industrial iron ore projects, recovery rates as low as 40 % (Micon, 2012) are common, depending on the deposit type and which recovery rate is considered to be the most economic feasible one. Considering sustainability it is of course eligible to recover the highest possible amount of a not-renewable resource when exploiting it. Therefore, regardless of the actual performance of processing, waste material and tailings containing target minerals should always be considered a potential asset.

3.2 Mine Site Selection and Evaluation

To evaluate suitable mine sites the updated national mine concession database was reviewed to identify operations with a constant and sufficient production capacity. From the mines suiting this criteria the type of mineralization, applied processing techniques and accessibility of the mines had to be considered further. Therefore, a team of GMD and BGR staff visited 12 mine sites, collected basic information and rated them. In hindsight, suitable mine sites have been identified by focusing on specified criteria and a most broad variety of characteristics to represent the most common deposit types and processing practices (Table 1).

Table 1: Characteristics and criteria for selecting suitable mine sites

Characteristics	
Type of deposit /mineralization	To give a representative overview, each of the most common deposit types was covered speaking of one pegmatite hosted deposit with cassiterite mineralization, one pegmatite hosted deposit with tantalite mineralization, and one hydrothermal quartz vein deposit with tungsten mineralization.
Applied processing technique	To give a representative overview, each of the most commonly practiced artisanal processing technique was covered, speaking of panning, ground sluicing and concrete sluicing.
Criteria	
Accessibility	The mine site had to be accessible by truck to transport bulk samples
Capacity	Manpower and production of the mine had to be sufficient to provide 1.5 t of bulk sample and process a share of it within two days
Infrastructure	Access to water supply and electricity grid

Figure 2 shows indicated locations of the selected mine sites. As for the integrity of the assessed operations, the report has anonymized the gathered data. This was done to ensure to not cause any disadvantages of any sort to the cooperating companies. However, the selected mine sides were chosen to represent an average of the Rwandan small scale sector. Apart from being able to produce a certain amount of minerals, the properties of the three mines are varying broadly. The amount of employees, access to infrastructure (electricity and water), corporate organization (privately owned or cooperative) and of course performed processing technique differ.

Within this, one the main criteria was to cover the most common deposit types, in respect to their occurrence in ASM operations. As most of the ASM operations are exploiting pegmatite hosted deposits, two of the selected mines were producing out of such nature. Within those, one mine is producing cassiterite and one mine is producing a mixed Sn-Ta concentrate of cassiterite and tantalite. Also, the regarding pegmatites of these mines showed alteration of different intensity. Further, as the second most common artisanal mined deposit type is quartz vein hosted, one more mine of this kind was selected, which is producing wolframite.

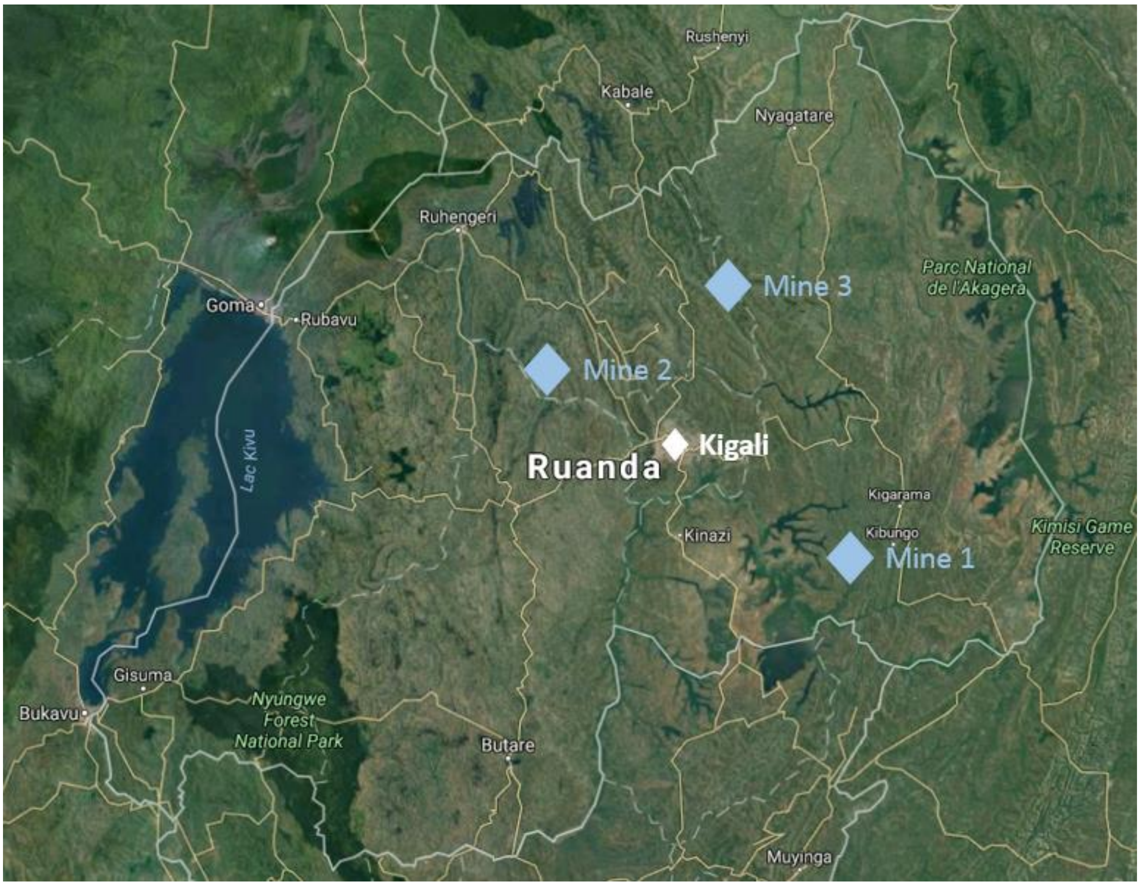

Figure 2: Map of Rwanda with indicated locations of the selected mine sites (google maps)

For artisanal processing in Rwanda, it is assumed that the majority of operations does still use ground sluices in combination with panning. The ground sluices do differ in properties and set up. Therefore operations with different ground sluicing concepts have been selected, while one of the mines processed exclusively with panning.

For the planned financial analysis, it was also worthwhile to gain insights in operations of different size in terms of employees and composition of expenditure. Therefore the selected mine sites are differing operational expenditures in terms of water and electricity supply, as well as personnel costs.

Table 2 shows the characteristics of the selected mine sites. As can be seen the characteristic vary broadly. However, it has to be kept in mind that three mine sites are only giving little insights to the whole ASM sector of Rwanda. The in-depth evaluation of the selected mines was performed to allow implementations by data gathered in different operational scenarios.

Table 2: Selected mines sites and some of their characteristics

Mine site characteristics	Mine 1	Mine 2	Mine 3
Mineralization	Pegmatite hosted cassiterite	Pegmatite hosted cassiterite and tantalite	Hydrothermal quartz vein hosted wolframite
Processing technique	Panning	Ground sluicing (circuit)	Ground sluicing (linear)
Capacity (pre-concentrate)	2-3 t/month	1.5-5 t/month	Up to 1 t/month
Permanent staff /casual workers	350 / 30	8 / 42	6 / 9
Water supply	With pump from well	Well upside the hill	Rain water and purchasing gallons
Energy supply	Diesel generator	Public power grid	none
Active Tunnels	58	19	5
Water costs	none	none	150 RWF/gallon
Diesel / water consumption per day	20 l / not measured	none / not measured	none / 50 l

After selection of suitable mine sites, the actual field work was performed. A team consisting of the consultant, two to three technical staff from GMD and the national representative of BGR visited the selected mines sites. During this, focus was set on representative sampling and applying measures of scientific field work.

3.3 Sampling and Sample Analysis

On each of the three ASM operations the team took a bulk sample of ROM out of the productive tunnels or stock piles.[2] The total sample mass was determined by weighing the bulk samples gradually and measuring its humidity in hindsight. To retain two homogenous samples, workers mixed (homogenized) the bulk samples multiple times with shovels on a flat surface. After this, workers formed a pile or an elongate shape and split the samples by riffling or quartering (Figure 3).

Figure 3: Workers are splitting the bulk samples representatively after homogenization. Left: riffle splitting, half of the sample goes to the right, the other half to the left; Right: Quartering, by quartering the pile and combining the quarters opposite to each other, two representative samples are formed

For later processing tests, splitting the samples to two uneven subsamples was necessary, as a higher sample mass was needed for mechanic processing tests than for the artisanal processing. Therefore, splitting and mixing of samples was performed two more times, after taking away half of the representative sample of the previous splitting. Applying this, the team produced two representative samples with a mass ratio of about 3:1 of each bulk sample.

For further analysis and sample description, taking additional samples during and after the processing tests was mandatory. During processing tests, samples were taken of all products of the individual processing stages in order to determine processing performances of the applied processing techniques.

Grain size and Mineralogical Analysis

When the sample composition and/or distribution of target minerals was of special interest, for example in the case of feed material, concentrate or waste, sieve analysis was applied on the samples. This gave the opportunity to understand the mass distribution and properties of the different grain-size fractions. Also, fractions of interest were analyzed optically with reflected and transmitted light microscopy to identify target minerals and the state of intergrowth of target minerals and barren rock[3].

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[2] *At some of the operations processing was not performed on a daily basis. Therefore, workers are producing in the tunnels for several days and store the produced material in stock piles.*

[3] *Material of no or low economic value contained in the ROM which has to be separated from target minerals, also referred to as waste/waste material*

Geochemical Analysis

In total, 270 samples have been send to a certified laboratory for analysis. The samples have been produced during the processing tests, as well as by sieve analysis of the ROM. The contracted laboratory[4] is accredited by ISO/IEC 17025:2005 for its analytical procedures and certified by ISO 9001:2008 for its quality management. Depending on the expected elements and the regarding concentrations, different analytical techniques have been applied, such as Inductively Coupled Plasma Atomic Emission Spectroscopy (ICP-AES, for the elements Al, As, Fe, K, Mn, Si, Sn, Ti, W) and Inductively Coupled Plasma Mass Spectrometry (ICP-MS for the elements Nb, Sn, Ta, W). For quality control of the measured concentrations, 22 blank samples and 20 standard samples have been analyzed with the 270 product samples. Additionally Xtraccon Limited applied X-ray fluorescence spectroscopy (XRF) to crosscheck the results.

3.4 Artisanal Processing

This chapter gives a brief description of observed artisanal processing techniques. It is not thought to give a conclusive overview about practiced techniques in the whole sector, but gives a common understanding of the processes which are evaluated in this study.

Panning

Panning is an artisanal processing method which sorts particles by their specific gravity. The washing pan (black vessel Figure 4) is usually a conical vessel in which the material is filled and mixed with water. The washing pan is moved in circles to gently agitate the material. When moved properly with adequate speed in rotatory motion, material with low specific gravity gets spilled over the edge of the washing pan. Material with higher specific gravity tends to stay near the center of the pan. Panning is usually performed in washing ponds that are specifically build to reduce water consumption and collect tailings for future reprocessing. Washing pond sizes usually vary between 1.50 m x 1.50 m and 2 m x 2 m, with depths of about 0.6 m. They may be covered with plastic sheets and filled to about 50 % with water. Panning has also been observed to be performed using barrels as washing ponds.

Figure 4: Panning process in prepared washing pond

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[4] *http://msa.a2bigs.com/*

Ground sluicing

Ground sluicing is a basic artisanal density sorting method, as can be seen in Figure 5, showing circuit ground sluicing. One person [Figure 5 #1] feeds the material to a regulated water channel, which was formed from soil and might be reinforced with sand bags. The second person [Figure 5 #2] shovels the material up against the water stream. By this, the single grains of the feed are separated based on differences in their mass densities. In this example, the water stream is supplied by a third person [Figure 5 #3], who pours water from a basin into the channel. Once the ore is processed, the sluice is cleaned and the concentrate removed by hand.

Figure 5: Circuit ground sluicing with material feeding (1), moving of the material in the opposite direction of the water flow (2) and supply of the water stream (3)

Another type of ground sluice is shown in Figure 6 (linear ground sluice). The difference to the ground sluice in Figure 5 is the absence of a recirculation concept of the water. Hence, a higher quantity of water is consumed while processing. Usually, a natural spring is fed into the sluice via gravity or a continuous flow of processing water is generated by pumps. This technique also allows to use more than the usual two sorting stages. In Figure 6, first three stages of the ground sluice are shown. At first the material gets fed on top of the stream [Figure 6 #1]. Then it passes through the first step and is concentrated at the first stage [Figure 6 #2]. The light material that passed the first stage goes down another step and gets concentrated again at the second stage [Figure 6 #3] and afterwards at the lase stage [Figure 6 #4]. Like this, up to 10 stages or even more can be used. The concentrate mass, grade and grain size usually decreases with increasing number of the stage. The material gets scrubbed naturally by passing through the stream and little waterfalls (steps). Between each stage, water velocity is adjusted by building dams with shovels, rice bags, stones, wheat or similar accessories.

Figure 6: Linear ground sluicing. (1) Feeding of the ROM and water; (2) first concentration stage and narrowing of water stream/reducing water mass and velocity with a shovel; (3) second concentration stage; (4) possible third concentration stage.

Hand Picking

Hand picking is a visual evaluation of single grains. It is commonly used on coarse pieces of the processed ROM, which cannot be processed efficiently by panning or sluicing. Every rock is evaluated visually and its density estimated manually. The efficiency of this process depends on the experience of the performing miner. He decides whether the rock might contain valuable minerals or is to be treated as waste. A picture of the visual evaluation of the rocks is shown in Figure 7.

Figure 7: Visual evaluation and handpicking of coarse rocks after panning

Air Classification/Tap and Blow/Winnowing

Figure 8: Artisanal air classification (Tap & Blow). The blue arrows are showing the air flow, created by the worker; the red arrows are showing the movement of the pan.

After the drying of the concentrate, artisanal air classification (Figure 8) might be used as the final concentration step. Applicability of this technique depends on the grain size of the concentrate. It is only suitable for relatively low grain sized pre-concentrates. For artisanal air classification (or winnowing) the material is repeatedly thrown in the air with the pan and caught again (Tap). Light particles accumulate at the edge of the material cone and can be carefully blown away (blow) by the worker as visualized by the blue arrows in Figure 8. Throughput for the manual air classification is estimated to be as low as 30 kg/h.

Artisanal Magnetic separation

If the concentrate contains magnetic impurities, a manual magnet like the one shown in Figure 9 is used for the final processing step. By moving the magnet along the concentrate magnetic particles are attached by magnetic force. By using a separate vessel or a textile as cover for the magnet, the magnetic particles are then manually removed. With this, magnetic impurities, such as hematite or alike minerals, are separated from non-magnetic cassiterite ore and paramagnetic coltan ore.

Figure 9: Artisanal magnetic separation of a mixed concentrate with manual magnet (red circle)

3.5 Mechanical Processing Techniques

This chapter gives a brief description of the potentially applicable mechanic processing techniques in the ASM sector. However, equipment used for the processing tests were of basic standard, to ensure representative testing. The used equipment cannot be considered as up to date or high end machines, while the following gives an introduction in common standards of mechanical processing equipment.

Screening

Screening separates grain-collectives into separated fractions of specific grain size compositions. By screening grains are separated depending on the size of the particles. A porous screen bottom allows grains that are smaller than its meshes to pass through, while grains with a bigger size are remaining on top of the screen. There are multiples reasons to screen a feed before processing.

Table 3: Benefits of applying screening measures prior to processing

To Prevent coarse grains from getting into processing machines which do not have the capacity to handle coarse grains and might take damage	!
To separate coarse grain sizes, which are not fully liberated in order to comminute and therefore liberate them	!
To create narrow grain size fractions to optimize efficiency of separation processes. Most processing techniques proved to be more efficient when narrower grain size fractions are used	!

Comminution

Comminution is a collective term for processes which reduces the grain sizes of a material. In mineral processing comminution can also be referred to as crushing (for coarse material) and grinding (for fine material). For materials which are not completely liberated at their original grain size, comminution is highly recommended. The concept of interlocked or intergrown value particles within a barren rock matrix can be explained on base of Figure 10. On the left side, a coarse particle that consists of barren rock (yellow parts) and value material (grey parts) is shown. After comminution there are usually three different kind of particles, as can be seen in Figure 10. Those are pure barren rock, remaining intergrown value material (which has to be liberated to recover contained target minerals) and pure liberated value material.

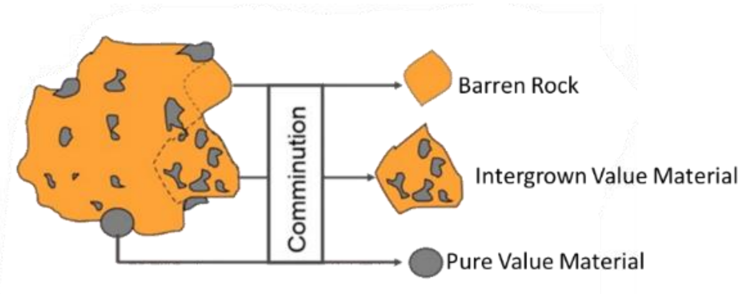

Figure 10: Schematic drawing of concept of intergrown and liberated particles

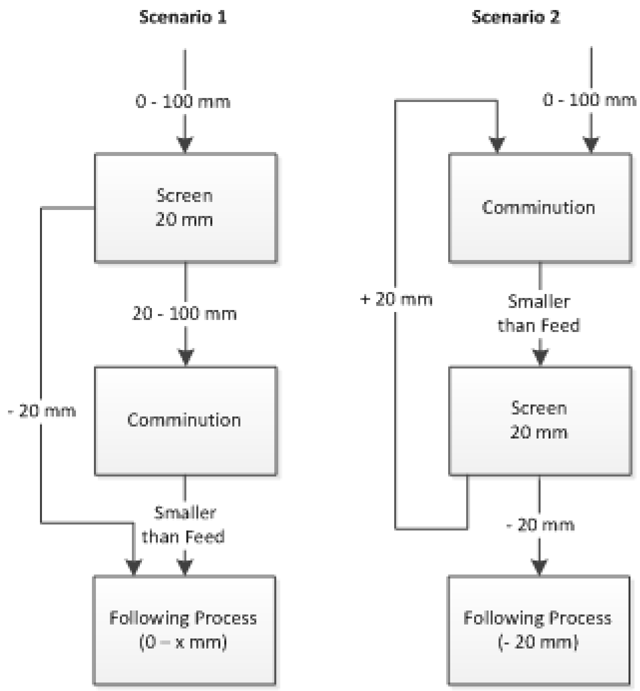

Figure 11: Different scenarios for applying comminution

Comminution may further be necessary when the original grain size exceeds the maximum feed size for a specific process. Also, it is used to produce the optimum grain size range for a specific separation process. In this case, comminution is combined with classification. For example, a screen can be applied before comminution to relieve the grinder or crusher and separate the coarse fraction. Only the coarse fraction will then be fed to the crusher or grinder. All particles that are already fine enough are bypassed (Figure 11, scenario 1). This is also possible as a cycle, in which the comminution product is screened again to make sure all particle sizes were decreased to a specific measure. In this case, the coarse screening product is then recirculated to the crusher/mill until all grains are smaller than the meshes of the screen (Figure 11, scenario 2).

Comminution

Before applying comminution to large quantities of material it is critical to evaluate the process. There is a risk of overgrinding the material. In this case, the grain size is decreased below the size which is suitable to perform the planned processing technique. In some cases, the material can even become too fine to be processed by density sorting, so the valuable elements cannot be recovered without chemical treatment.

Spiral Concentrator

Spiral concentrators are commonly used for density separation of particles smaller than 1 mm. The feed material is mixed with water to produce so called liquid pulp. The pulp gets fed onto the spiral and runs downwards in three to five circles, depending on the type of spiral. Due to their inertia heavy particles accumulate on the inside of the spiral while lighter particles are pushed to the edge of the spiral (Figure 12).

Figure 12: Spiral concentrator fed with liquid pulp.

In the spiral two streams can be identified which occur more or less rectangular to the main stream which runs down in the middle of the spiral. A second stream is located at the bottom of the spiral directing to the inside and a third stream flows on top of the pulp directing to the outside of the spiral (Figure 13). Because of centrifugal forces the thickness of the pulp stream is rather low on the inside (2-3 mm), while it is rather turbulent and high on the outside (7-16 mm) (Schubert, 1989).

In the inner side of the spiral, there are so called splitters collecting particles with a higher density. The splitters are manually or automatically adjusted to split right between the heavy and the light fraction. The separated fraction which was collected by the splitters runs towards the middle of the spiral and gets recovered. The border of the two fractions is usually not a clear cut, as coarser particles of higher density overlap with smaller particles of lower density. Therefore, most spirals have the option to create a third, middling fraction between heavy and light material, which might be recirculated to gain additional concentrate. Also, the feed should consist of grains which are more or less the same size, to increase the efficiency of this process. Comminution and screening should be used before feeding a spiral, to produce a narrow variation of grain sizes.

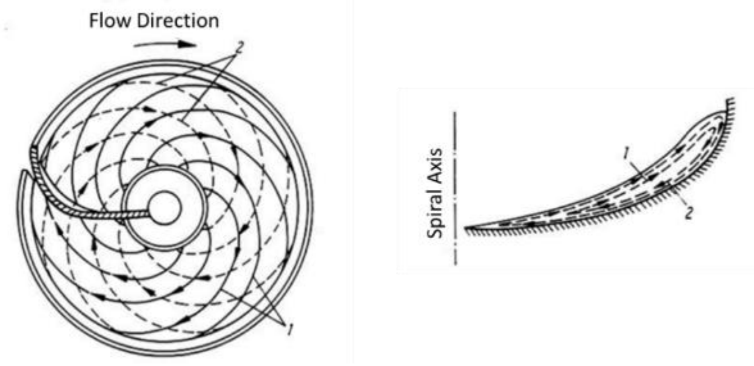

Figure 13: Schematic drawing of flow directions inside a spiral concentrator. Lower stream (2) directing to the inside of the spiral and top stream (1) directing to the outside of the spiral. (Schubert, 1989)

Shaking Table

Shaking tables are widely used in mineral processing by density separation. For a good result, it is recommended to use particles smaller than 2 mm for this process (Schubert, 1989). The principle of a shaking table can be seen on the schematic drawing in Figure 14.

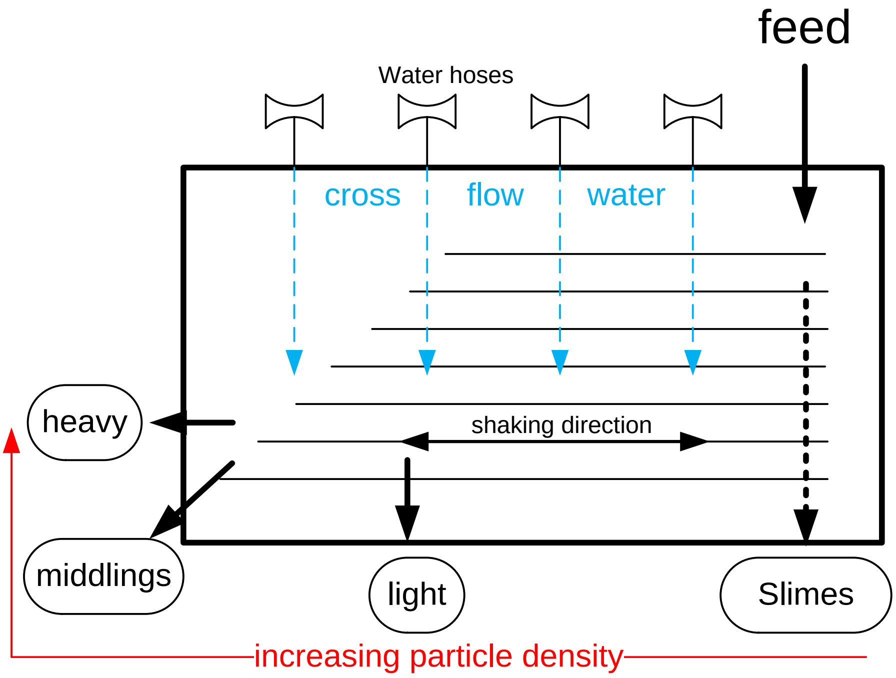

Figure 14: Schematic drawing of the principle of operation of a shaking table.

Shaking tables are consisting of a plate with riffles that is inclined with the lower edge in opposite of the feed side. An engine moves the plate back and forth. Frequency and amplitude of this movement can be adjusted. Cross-flow Water is fed onto the table from water hoses, from which the intensity can be adjusted, as well. This water stream creates flow energy across the table, which leads slimes (very small and low density particles) to be suspended from the table, following the water flow (#3 Figure 15). Light particles flow rather to the light and middling section of the table (#1 Figure 15), whereas heavy particles are less affected by the water flow and therefore follow the shaking direction of the table (#2 Figure 15).

Generally, apart from slimes, small particles move further lengthwise than coarser particles of the same density. This can lead to overlapping of coarse dense and small light particles. To avoid this effect, it is crucial to create a narrow grain size fraction[5], before feeding the shaking table. For using shaking tables efficiently a proper calibration is mandatory. Also, the whole plate should be moistened by water and stay moistened during the whole process. In general, there are a few options to adjust the shaking table in respect to the feed (Table 4).

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[5] For example, instead of feeding one grain size fraction 0-2 mm, two grain size fractions, 0-<1 mm and >1-2 mm, could be fed, by separating the 0-2 mm fraction by a 1 mm screen before feeding the table.

Table 4: Aspects to consider in order to optimize shaking table performance

Inclination of the plate	!
Intensity of the cross water flow	!
Frequency and amplitude of the shaking movement	!
Amount of feed that is fed onto the plate	!
Width of captured concentrate flow	!

If these parameters are adjusted correctly and the feed is of the right grain size the streams of slimes, light/middling fraction and concentrate are flowing separately from each other along the plate (Figure 15). The size of the gap between the slimes and the heavy fraction might differ. Therefore, the capturing of the concentrate is the last crucial adjustment. This is usually done with a slide plate or cone, which can be moved to different positions, to recover different amounts of the heavy fraction (Schubert, 1989). The adjustment of a shaking table needs some experience. With multiple attempts a proper adjustment might even be performed by trial and error.

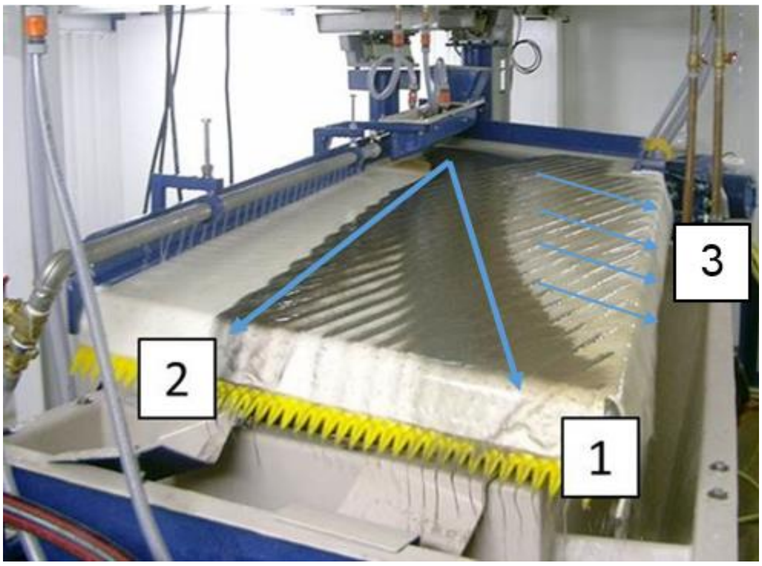

Figure 15: Modern shaking table with separated flows of middling fraction (1), heavy fraction (2) and light/slimes fraction (3)

Magnetic Separation

Sorting criteria of magnetic separators is the magnetic susceptibility of the sorted material. Magnetic particles can be separated from nonmagnetic particles, as well as particles with a certain differences in magnetic susceptibility from each other.[6] For example, separation of high magnetic magnetite from low magnetic coltan and nonmagnetic cassiterite can be performed. In Figure 16 a magnetic separator with variable magnetic forces and multiple sorting stages is shown. This concept allows to produce five different products. Depending on the feed, the first product has very high magnetic properties, like iron or magnetite (#2 in Figure 16). The second product has ferromagnetic properties and therefore might be a medium magnetic product, like magnetite or hematite (#3 in Figure 16). The third and fourth product are low magnetic minerals like tantalite or wolframite (paramagnetic, #4 in Figure 16). The last product is the nonmagnetic fraction, which includes all minerals that do not show any magnetic susceptibility (#5 in Figure 16). When using a magnetic separator basic knowledge of the magnetic properties of the processed material is needed. Also, magnetic separators should not be fed with too much feed at once, in respect to their specific throughput capacity.

Figure 16: Magnetic separator with multiple electromagnets of variable magnetic intensity. The feed section is at (1) where the material is put on a vibratory feeder. At (2) the high magnetic fraction is separated by a low energy magnetic field. At (3) another high magnetic fraction is separated. At (4) low magnetic materials are removed and at (5) nonmagnetic material remains.

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[6] *The intensity of the attraction to magnets depends on the physical properties and chemical composition of the grains or minerals.*

3.6 Economic analysis

The economic analysis performed is based on the gathered operating data of the mine sites. After taking samples at the sites, an interview with the mine management was carried out. By this, basic data regarding the operations performance and costs have been collected. It is worth mentioning, all given information are based on the knowledge and impressions of the mine management. Therefore, in many cases it could not be double checked if given information was correct. However, exceptional unlikely statements have been averaged or excluded. The survey included the aspects shown in Table 5.

Table 5: Type of operational data gathered on-side

General Information	Costs	Mining Practice
Infrastructure	Per worker	Ore grade
Concession type	Royalty	Concentrate grade
Ore type	Revenue tax	Mine lifetime
Deposit type	Land tax	Number of tunnels
Mine sites	Patent fee	Production rate
Number of workers/gender	Environmental fee	Mining method
Number of teams	For fuel	Transporting method
Number of workdays	For water	Processing technique

After field work and analysis of the processing tests, multiple calculations could be performed based on a combination of this data and the observed processing performance of different processing techniques. Also, based on concentrate prices of the last 10 years, three different pricing scenarios (high, medium and low) were assumed. This approach allowed to give preceding calculations of economic sustainability for different processing and investment scenarios as can be seen in Chapter 8.

4. Description or Raw Material Samples (ROM)

In order to derive implications for processing performance and specific challenges or the processed material, a detailed sample description and analysis is necessary. Therefore, the team took representative sub-samples of each collected ROM. Due to the high mass of the bulk samples (Table 6) producing representative sub samples was challenging and has been achieved by multiple mixing and splitting steps.

Table 6: Dry masses of bulk samples

Sample	Total sample mass	Mechanic Processing	Artisanal Processing
Mine 1	852 kg	732 kg	220 kg
Mine 2	1413 kg	1065 kg	348 kg
Mine 3	1111 kg	857 kg	254 kg

By analyzing the sub-samples of the ROM as described in chapter 3.3, details about the mineralogical composition, grain-size distribution and distribution of the target elements within the different grain-size fractions could be learned. Which will be presented in the following. A more detailed sample description can be found in chapters 1 to 3 in the annex.

4.1 Grain Size Distribution

Sieve analysis has been performed on all bulk samples. As to the different mineralogical background of the sampled deposits, bulk samples are showing different grain size properties. As presented in chapters 3.4 and 3.5 the different processing methods partly rely on the right grain-sized material to work efficiently. Therefore, grain size distribution of the processed material has a major impact on processing procedures. For example, when a feed shows a narrow distribution of grain sizes, it is more likely to achieve a good recovery rate. On the other hand, when the feed contains a wide range of different grain sizes, it might be necessary to apply multiple screening and comminution steps, before sorting it.

Table 7 shows the basic grain size distribution properties of the sampled ROMs. The ROMs are showing distinct varying grain-size distributions. While about 86 % of the ROM 1 is composed of grain sizes between 63 µm and 20 mm, only 69 % of the ROM 2 lies in this range. For Mine 3 the share of ROM in this grain-size fraction is even lower (about 60 %). With this, the samples represent a broad range of grain-size compositions. The ROM of Mine 1 shows an average distribution on grain sizes. While from the same deposit type, but in a different state of weathering, the ROM 2 can be described as rather fine grained. In opposite to this, the ROM of Mine 3 shows a very broad variation of grain sizes with high shares of it being coarser than 2 mm. As it is the only material from another deposit type, a different grain-size distribution was expected. However, comparing the d_{50} of the ROMs it becomes clear, the coarsest material was taken from mine 3 (d_{50} = 3 mm), while ROM 2 with a d_{50} of 250 µm is the finest material. Detailed graphs of the grain size distributions are included in the annex.

Table 7: Basic grain size distribution properties of the sampled ROMs

Mine 1 / Cassiterite	Mine 2 / Mixed Sn-Ta	Mine 3 / Wolframite
Pegmatite weathered	Pegmatite heavy weathered	Quartz vein hosted
d_{50} at 2 mm	d_{50} at 250 µm	d_{50} at 3 mm
12 % of mass below 63 µm	30 % of mass below 63 µm	25 % of mass below 63 µm
3.5 % of mass above 20 mm	>1 % of mass above 20 mm	15 % of mass above 20 mm
Fairly average distribution of grain sizes with a mentionable amount of fines	Grain distributions tends to finer grain sizes	Wide range of grain sizes with a big share of fines and a high amount of coarse grains

Note: the d_{50} particle size distribution is used as an indicator to estimate the average grain size. d_{50} is the size where 50 % of the sample mass is passing the screen. This means at this mesh size (d_{50}) half of the material is coarser, and half is finer-grained than the indicated diameter.

4.2 Distribution of Target Minerals

To evaluate the processing efficiency, it is necessary to understand the distribution of target minerals in the ROM, prior to any processing. Therefore, all grain-size fractions of the different feeds have been analyzed chemically and the findings have been put into graphs. In Figure 17 multiple graphs of the distributions and grades of the respective target minerals in the different grain-size fractions can be seen. Here, the x-axis shows the grain-size fractions. Blue columns indicate the percentage of target minerals contained in the respective fraction with regard to the total contained target minerals in the ROM sample. This distribution is quantified at the primary y-axis. The red dots show the target mineral grade in each fraction and are related to the secondary y-axis.

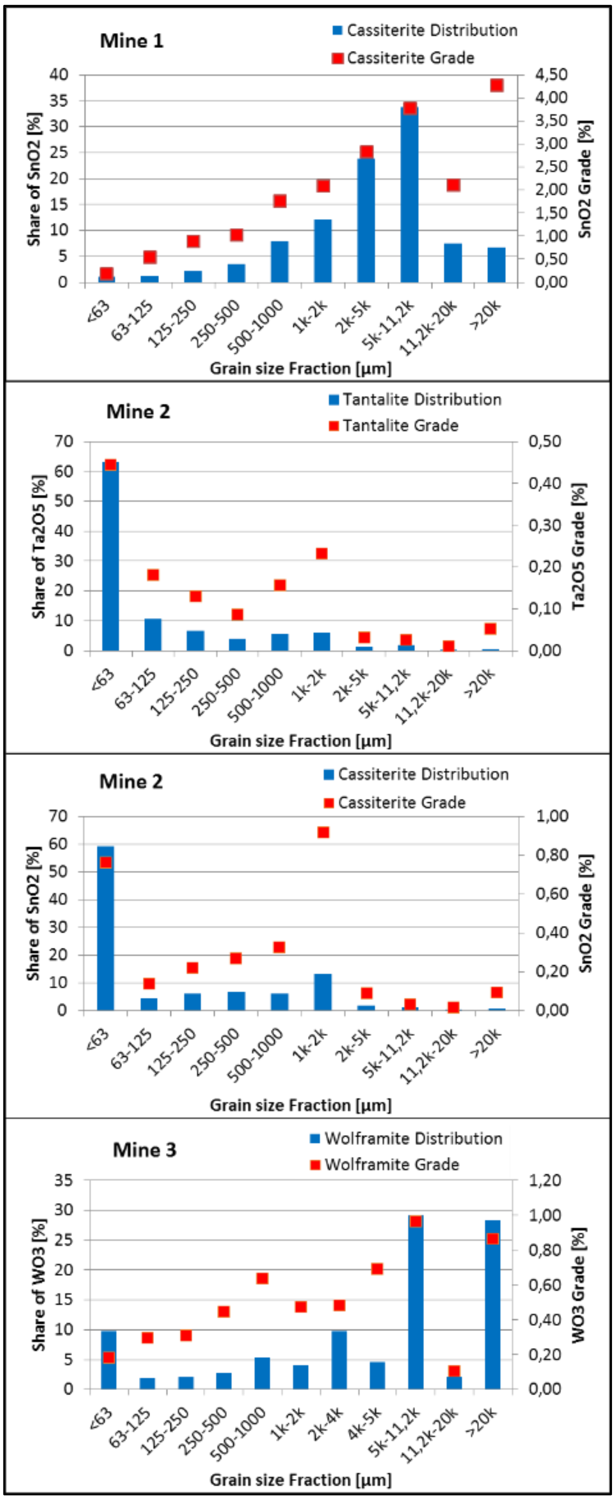

Figure 17: Distribution and grades of the respective target minerals in the grain-size fractions

The ROM 1 (mine 1 - Figure 17) shows a concentration of target minerals in the medium sized fractions between 0.5 mm and 11.2 mm. About 78 % of the total SnO_2 is contained in the grains of these fractions. 14 % of the target minerals are found in the coarse fractions larger than 11.2 mm, while only 8 % are contained in the fine fractions below 0.5 mm. Apart from a minor set-back in the fraction between 11.2 mm and 20 mm, the grade of SnO_2 in the material increases with increasing grain size. Therefore, the coarsest grains above 20 mm in size showed the highest grade of SnO_2 in this ROM (4.35 %).

For the ROM 2 (mine 2 - Figure 17), with about 60 % each, the largest share of target minerals (SnO_2 as well as Ta_2O_5) are contained in the fraction below 63 µm. For Ta_2O_5 additional 10 % are contained in the fraction 63 to 125 µm. Further 20 % of the Tantalum is contained in the fractions between 125 µm and 2 mm. Only about 5 % of the tantalite containing particles is coarser than 2 mm. For SnO_2 the distribution is alike, even though a slightly increased content can be observed in the grain-size fractions between 125 µm and 2 mm, were about 30 % of total SnO_2 is contained. For both target minerals the contents in the fractions above 2 mm are as low as 4 % of total target minerals in the sample. Also an anomaly has been found in the fraction between 1 mm and 2 mm, where the grade of Ta_2O_5 was slightly increased with 0.22 % and that of SnO_2 peaked with about 0.83 %.

For the ROM 3 (mine 3 - Figure 17) a way different distribution of the target mineral was observed. Here, most of the target mineral (about 85 %) was found in fractions above 0.5 mm. The analysis showed 10 % of the WO_3 are comprised in the fine grains below 63 µm. The fractions from 63 µm to 2000 µm combined contain about 15 % of the total WO_3. From 2-5 mm further 15 % are contained. With this, the coarsest fractions above 5 mm are containing 60 % of the target minerals. The highest WO_3 grade (9.6 % WO_3) was found in the fraction 5-11.2 mm. This fraction also contained the highest total amount of WO_3. In the fractions coarser than 20 mm the grade was about 8.5 %. It is noticeable that in the fraction between those two high grade fractions from 11.2-20 mm the lowest WO_3 grade (1 % WO_3) was found. However, the grade tends to increase with grain size, even though there are some variations in the medium grained fractions and the mentioned anomaly between 11.2 mm and 20 mm.

4.3 Composition of the Samples and Grain Properties

For the final assessment of the bulk samples properties an optical analysis has been performed via transmitted-light and reflected-light microscopy. By evaluating further data of chemical analysis and the visual nature of the grains, an approach to the mineralogical composition was made. Apart from observations in other grain-size fractions, the findings and figures here are representing the most relevant findings. First, the findings of the two pegmatite ores (ROM 1 and ROM 2) will be presented. Later in this chapter, a description of ROM 3 is given. A more detailed sample description can be found in the annex.

In Figure 18 a microscopic picture of the grain size fraction 63 - 125 µm of the feed is shown. The picture was taken with transmitted light to give an impression of the amount of mica in the feed. All clear parts of the picture are either mica or quartz. They can be distinguished from each other by their crystal structure. Quartz can be identified by its conchoidal fracture with cubic shapes (red circle), while mica shows a flaky shape with even fractures. The black parts are other minerals, which cannot be distinguished from each other in transmitted light. Next to the clear mica and quartz particles, cassiterite and some iron minerals like hematite and magnetite are present in the feed, which can be distinguished by the reddish-brown color. The cassiterite (SnO_2) grade of 2.17 % of ROM 1 was determined by chemical analysis.

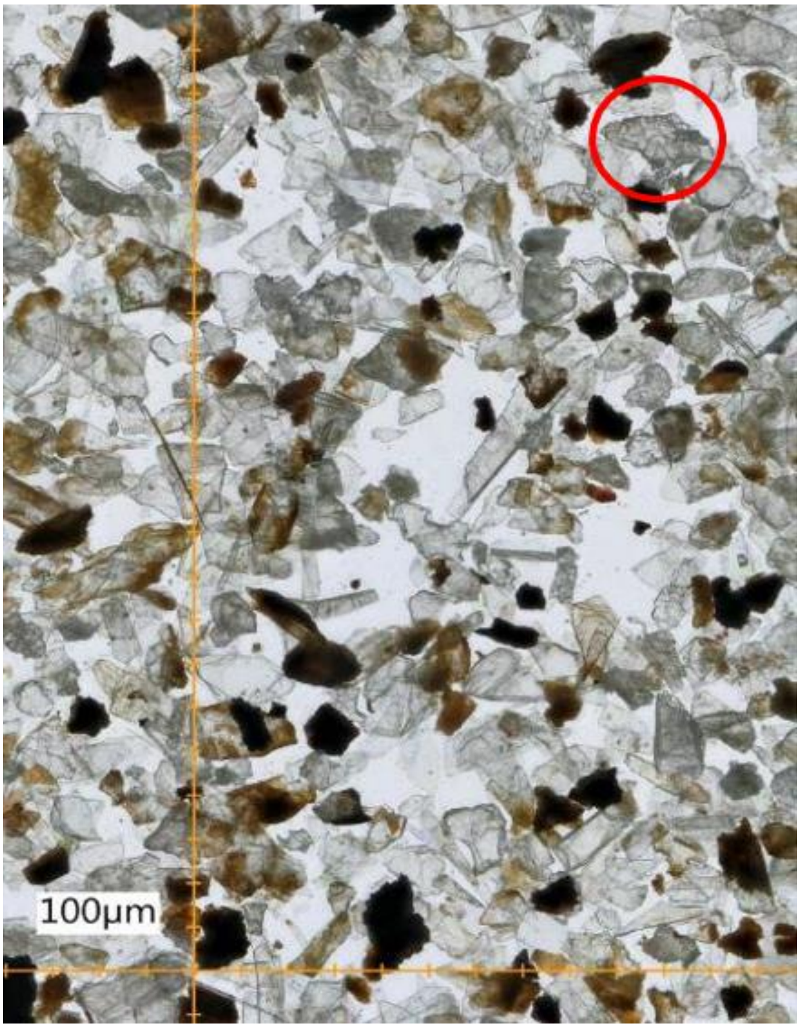

Figure 18: Transmitted light microscopy picture of the grain-size fraction 63-125 µm of ROM 1

Furthermore, concentrations of elements found were 4.5 % iron (Fe) which is mainly present in magnetite (Fe_3O_4) and hematite (Fe_2O_3) complexes. About 15 % aluminum (Al) is present in the feed, which in combination with about 6 % potassium (K) most likely as part of the mica. Mica was also found to be the dominant mineral in the feed. About 20 % silicon (Si) contained in mica and quartz as well as a minor amount of feldspar.

The feed does not contain as much mica as ROM 1. Nevertheless, the target mineral grade in ROM 2 visually seems to be lower. It has to be mentioned, that cassiterite and tantalite grains could not be distinguished from each other by microscopy analysis. There is a high quantity of quartz and a mentionable amount of feldspar present. A picture of the feed fraction between 125 and 250 µm is shown in Figure 19.

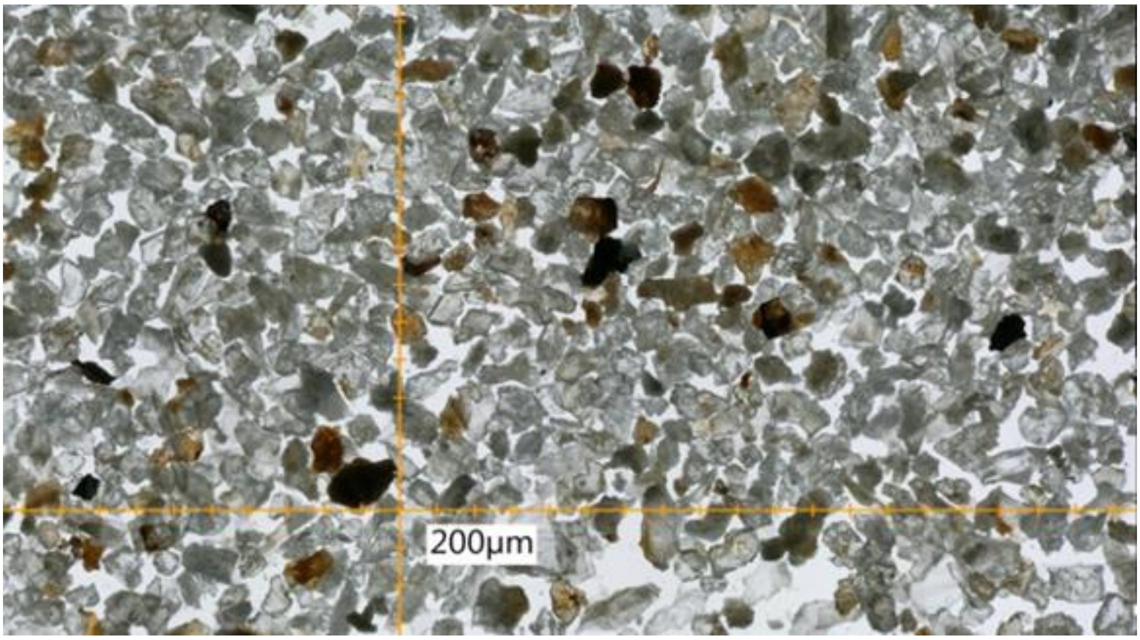

Figure 19: Transmitted light microscopy picture of grain-size fraction 125-250 µm of ROM 2

Chemical analysis of ROM 2 confirmed most of the visual findings. It shows a mineralogical composition similar to ROM 1. However, the grade of cassiterite was 0.46 %, lower than that of ROM 1. Amounts of magnetite and hematite were lower, as well. The Fe grade was measured to be 1.2 %. Also, the mica contained was estimated to be lower than that in ROM 1. While still 11 % of Al was measured, the analysis showed a grade of 1.5 % for K, which might be a result of advanced alteration. With 28 % of silicon, a higher amount of quartz can be assumed than that found in ROM 1. The grade of Ta_2O_5 was 0.199 %.

Figure 20: Picture of reflected-light microscopy of the grain-size fraction 2 mm to 5 mm of ROM 2

Figure 20 is showing reflected-light microscopy of the grain-size fraction 2 mm to 5 mm of ROM 2. The pictures shows barren rock grains of white to light-brown color, as well as a liberated grain of the target mineral with a dark blackish color. In the upper right a grain (red markings) with intergrown target minerals can be seen. The grain seem to be consisting of Ta_2O_5 or SnO_2 and quartz or feldspar. As stated in the previous chapter and can be seen, ROM 2 showed a low grade of target minerals in this particular grain-size fraction.

Figure 21 shows a picture of reflected-light microscopy on the grain size fraction between 500 µm and 1000 µm of ROM 3. Transparent and dark particles seem to be distributed evenly. The transparent particles are quartz. Different minerals appear as black grains. A major part of those was identified to be graphitic minerals by low hardness and a black streak. The overall WO_3 grade was determined to be about 5 %. The Fe grade of the feed was about 1.9 % and most probably represented by ferberite, goethite and limonite. Arsenic (As) is present with 0.03 %.

The presence of 0.63 % titanium (Ti) suggests that ilmenite is also contained in the feed. Silicon accounts for 26 % of the feed and aluminum and potassium are present with grades of 12 % and 4.8 % respectively. ferberite ($FeWO_4$) and hübnerite ($MnWO_4$) are commonly the main wolframite minerals in such deposit types. Since almost no manganese is present (0.001 %) in the feed and the streak color of the tungsten rich particles is brownish, ferberite is most likely the main tungsten mineral contained. Besides the mentioned minerals, some limonite covers several particles and goethite particles have been identified.

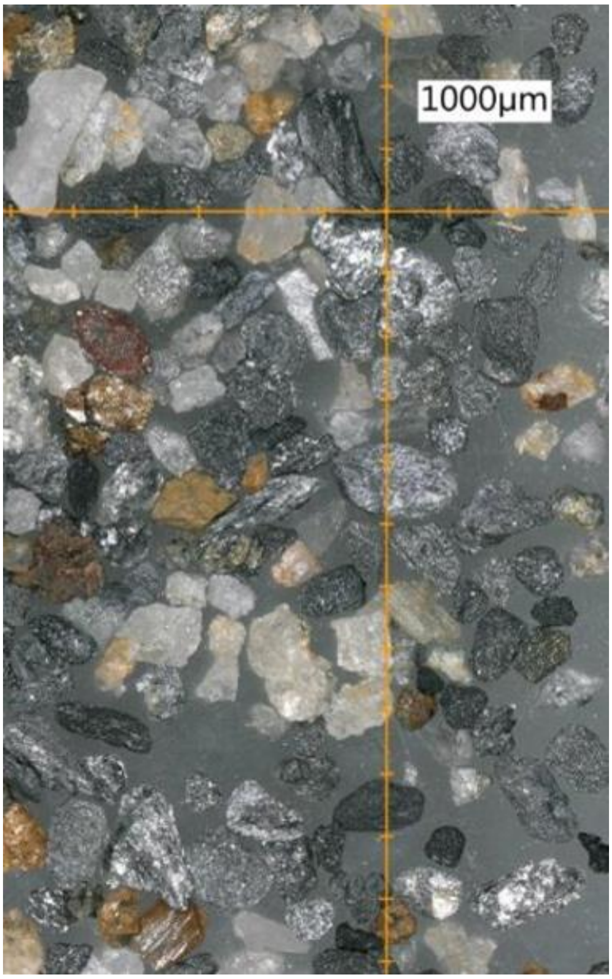

Figure 21: Picture of reflected-light microscopy of grain-size fraction 0,5-1 mm of ROM 3

5. Evaluation of Artisanal Processing Methods

The processing tests have been performed in the processing facilities of the sampled mine sites. Preparation steps as well as the methodology of processing have been recorded in detail. This chapter presents a brief introduction to the basic steps of the tests and their results. The miners of the respective operations performed the processing as in their daily practice. The team of consultant, GMD technicians and the local BGR representative took care of documentation and sampling during the field tests. The feed masses can be found in Table 6, a more detailed description of the performed processing tests is found in the annex (chapter 4 in the annex).

5.1 Artisanal Processing at Mine 1/Cassiterite

The processing scheme of mine 1 as shown in the flow sheet in Figure 22. A more detailed flow-sheet of the artisanal process is shown in the annex. In the first stage, the ROM is washed in a washing-pan (#1, see Figure 4) in a washing pond. While the light fraction of this first step is dumped as waste, the heavy fraction is sorted by hand picking (#2, see Figure 7). The coarse grains which were found to potentially contain target minerals are crushed and sorted. In the second stage the crushed material is washed with the rest of the heavy fraction from the first stage. After this, the heavy fraction is dried and then air classified (#3 see Figure 8). The tailings of the air classification have been recovered for reprocessing at a later point. During the tests the washing pond was lined with a plastic canvas. Therefore, the team recovered all tailings, except for those from the air classification for later reprocessing with mechanical measures.

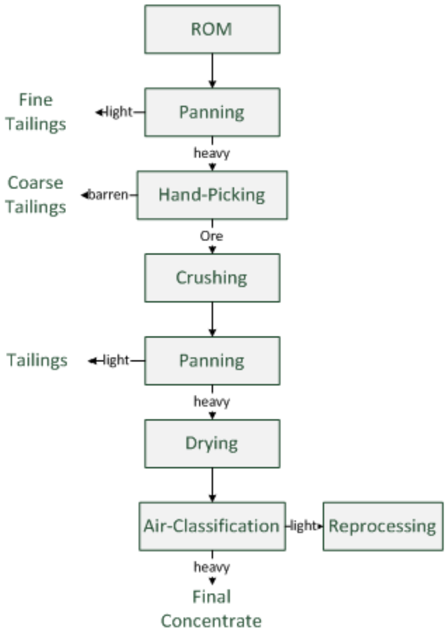

Figure 22: Flow sheet of the artisanal processing scheme of mine 1 (Cassiterite)

For the artisanal processing at mine 1 another observation has been made. The approach of panning in a washing pond gave the opportunity to recover all side products from the different processing stages. When reviewing the concentrations in the different products, it was found that the first panning stage of the artisanal processing removed about 82 % of the mass of the processed material into the waste material, while recovering about 84 % of the contained target minerals in the heavy fraction (see annex). This ratio can be considered as an efficient pre-sorting, leaving a heavy fraction for further processing with an SnO_2 grade of about 11 % for further processing.

The final results of the artisanal processing of ROM 1 can be seen in Table 8. A recovery rate of 60.10 % has been achieved. The concentrate had a SnO_2 grade of 86.98 % and can therefore be estimated to be a high quality concentrate. However, close to 40 % of the target minerals remained in the waste material. By further assessment it was found that most of these lost target minerals were contained in the tailings of the washing stages. The fine tailings of panning stage 1 contained 16.23 % of the total SnO_2 in the feed.

Additional 9.08 % were lost in the coarse tailings after the hand-picking and another 19.09 % has been found to have remained in the tailings of panning stage 2. The enrichment factor was around 40. The whole process was estimated to be capable of a throughput of 200 to 250 kg of feed per hour.

Table 8: Artisanal processing results of ROM 1 (Cassiterite)

Mine 1	SnO_2 grade [%]	SnO_2 recovery [%]	Mass recovery [%]
ROM/Feed	2.17	-	-
Concentrate	86.98	60.10	1.50
Waste	0.88	39.90	98.50

5.2 Artisanal Processing at Mine 2/Mixed Sn-Ta

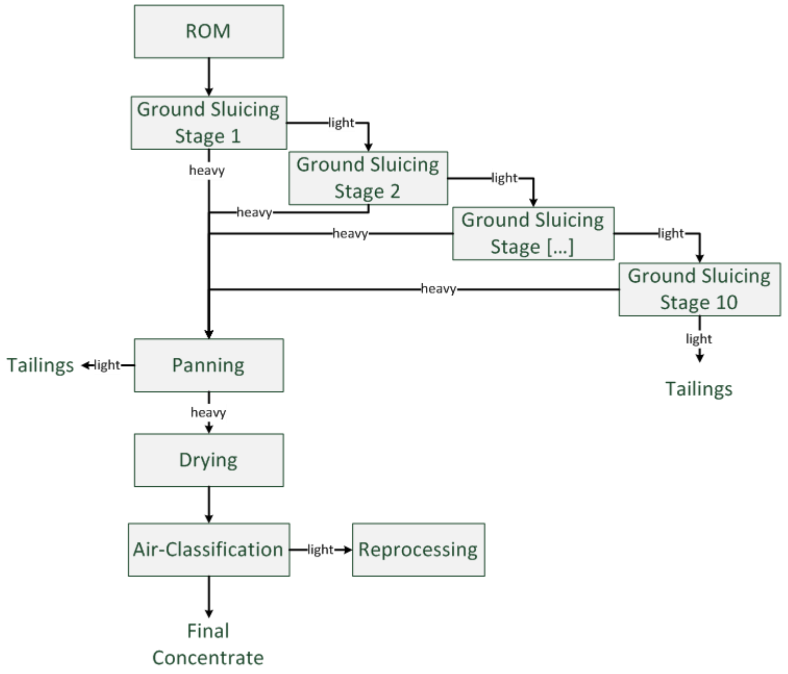

Figure 23: Flow sheet of the artisanal processing scheme of mine 2 (Mixed Sn-Ta)

As shown in Figure 23 the processing scheme of Mine 2 uses three basic sorting techniques. The first which is a linear ground sluice (#1) is therefore performed in up to 10 stages according the material processed. For testing purpose and due to the limited amount of ROM the ground sluice was worked on three stages. For later analysis, the heavy fraction of stage 2 and 3 have been combined, due to their low mass. One miner feeds the ROM directly into the first stage of the ground sluice, were another worker shovels the material against the water stream (see Figure 6). Some of the heavy material is already captured in stage 1, while the rest follows the light material in the water stream down to stage two, where the shoveling is repeated. This process is continued until all ROM was fed into the sluice and no new material arrives at the last operating stage. When the sluicing process is finished, the heavy fractions from the sluicing stages were then collected and processed by panning (#2). While the light fraction of the panning stage is considered as waste, the heavy fraction gets dried and air classified (#3). The light fraction of the air classification was recovered for reprocessing, while the heavy fraction was considered as final concentrate.

The waste material of the sluicing process was lost with the processing water flow and therefore no analysis of the artisanal waste could be carried out. During evaluation of the chemical analysis back-calculation led to the losses of target minerals and the probable grade of the waste material. However, as can be seen in Table 9 the artisanal concentrate showed a SnO2 grade of 41.26 %. This grade has to be classified as a below average cassiterite concentrate, which is not suitable for export. The recovery of SnO2 was 24.49 % which means that more than 75 % of the SnO2 was lost in the waste. With increasing the grade from 0.46 % in the feed to 24.49 % SnO2 in the concentrate, the enrichment factor was 50. For tantalite a recovery rate of only 13.35 % was calculated with a grade of 9.92 % Ta_2O_5 in the concentrate, which is not suitable for the international market. As the grade of Ta_2O_2 in the feed was only 0.19 %, the enrichment factor for Ta_2O_5 is 90 and can be estimated as very high.

Table 9: Artisanal processing results of ROM 2 (Mixed)

Mine 2	SnO_2 grade [%]	SnO_2 recovery [%]	Ta_2O_5 grade [%]	Ta_2O_5 recovery [%]	Mass recovery [%]
ROM/Feed	0.46	-	0.19	-	-
Concentrate	41.26	24.49	9.92	13.35	0.27
Waste	0.35	75.51	0.17	86.65	99.73

5.3 Artisanal Processing at Mine 3/Wolframite

As shown in Figure 24 the processing scheme of mine 3 uses two basic sorting techniques. It starts with circuit ground sluicing (#1 see Figure 5). The ground sluicing is only applied in one stage. After this the heavy fraction of the ground sluice was panned (#2, see Figure 4) in a rougher[7] and a scavenger stage[8]. In the first panning stage a final concentrate is created which gets dried and is then ready for sales. Tailings of the first panning stage get reprocessed in a second panning stage, where a second concentrate is created. The light tailings of the second panning stage are treated as final tailings. As can be seen in Table 10 the artisanal concentrate showed a grade of 21.41 % WO_3. As the feed had a grade of 0.54 % WO_3 and the concentrate summed up to 0.52 % of the mass of the feed, recovery rate in the concentrate was just 20.76 %.

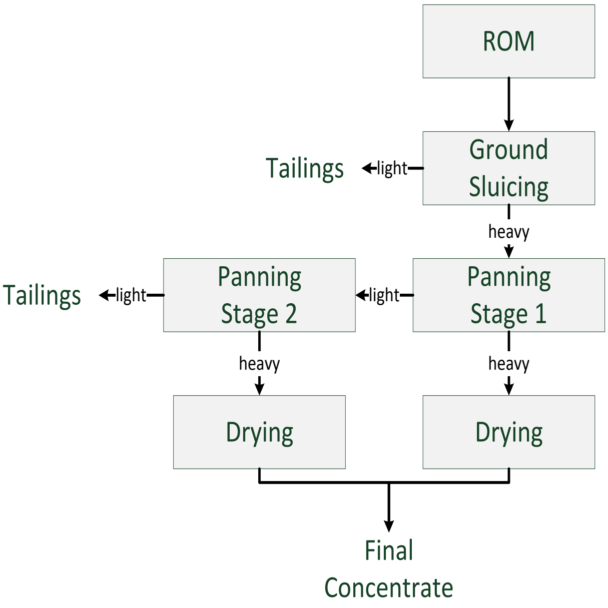

Figure 24: Flow sheet of the artisanal processing scheme of mine 3 (Wolframite)

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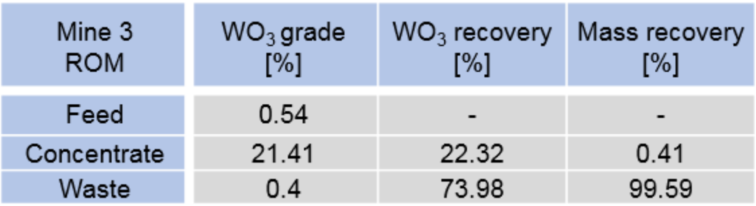
[7] *Rougher stage is the first treatment of the material with one specific sorting technique*

[8] *Scavenger stage is the retreatment of tailings of a rougher stage to improve recovery*

By this analysis it was assessed that 79.24 % of the target minerals remained in the waste of the artisanal processing practice. However, as a circuit ground sluice was used, the waste material could have easily been re-processed. In terms of this study and by information of the mine management it has to be assumed that reprocessing is usually not performed. With a concentrate grade of 21.41 % the enrichment factor for the target mineral was 40. Grade as well as enrichment factor can be assumed to be below average. The grade of the concentrate is too low to be marketable.

Table 10: Artisanal processing results of ROM 3 (Wolframite)



Mine 3 ROM	WO_3 grade [%]	WO_3 recovery [%]	Mass recovery [%]
Feed	0.54	-	-
Concentrate	21.41	22.32	0.41
Waste	0.4	73.98	99.59

6. Evaluation of Mechanical Processing Tests

The mechanical processing tests were performed in the processing facilities of Phoenix Metal Limited[9] in Kigali. As part of the preparation of the field work the BGR office in Rwanda contacted several companies running processing plants in the country. While few of the companies have not been willing to cooperate, three different processing plants could be visited. One of those was found to not be functional and oversized for the planned processing tests. Another plant only provided some equipment while lacking of suitable crushers, mills and a magnetic separator. The processing plant at Phoenix Metal, however, provided a good basis for a complete processing set-up, including preparation stages with different screen sizes and comminution machines, as well a multiple shaking tables and magnetic separators. The equipment was mostly old and worn, but well maintained. Some of the machines did not show any specification label and, therefore, have probably been constructed for the application at Phoenix Metal, exclusively. However, the set-up options and machines at Phoenix Metal Limited have been found to be the most suitable ones for the processing tests. Additionally the advantage of location at the outskirts of Kigali left this processing plant to be the best available option in terms of logistics.

Preparation steps as well as the methodology of processing have been recorded in detail. This chapter presents a brief introduction to the basic steps of the tests and their results. The consultant and the technical staff of GMD performed the tests with the given equipment. The feed masses can be found in Table 6, a more detailed flow-sheets of the mechanical processing including grades and recovery rates of most of the single processing steps can be found in annex (chapter 4 in the annex).

For all mechanical processing tests the preparation steps were kept consistent. After drying and determining the dry weight, the samples were sieved with a mesh size of 2 mm. After the sieving, two fractions, one coarser than 2 mm (> 2 mm) and one finer than 2 mm (< 2 mm), were available. Consequently, but were processed separately. Hence, for the fraction > 2 mm additional preparation steps, like further screening and comminuting, were necessary. However, the applied sorting techniques were shaking tables and magnetic separator, while for one testing series a spiral was used.

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[9] *http://phoenix-metal.com/*

6.1 Mechanical Processing of ROM 1/Cassiterite

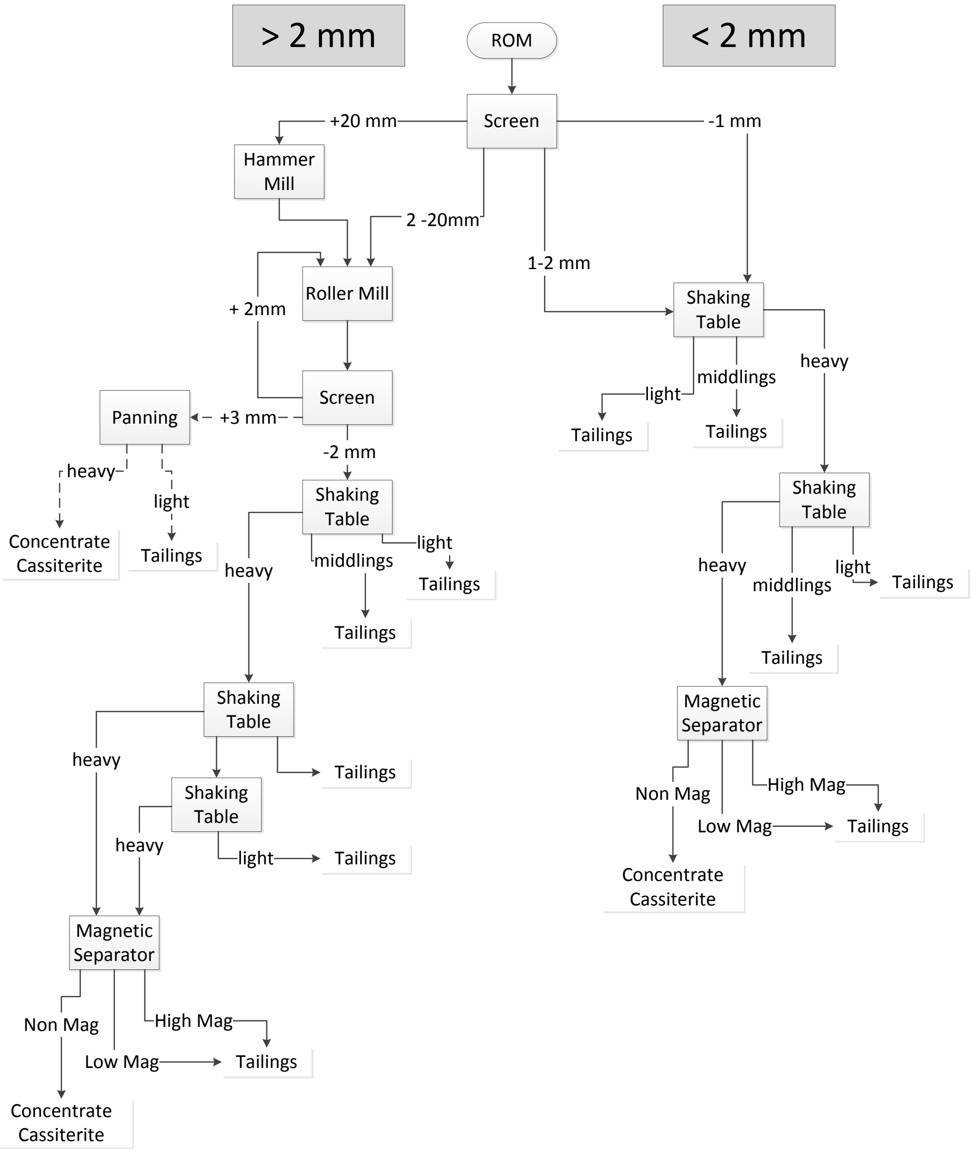


Figure 25: Flow sheet of the mechanical processing scheme of ROM 1 (Cassiterite)

For sorting the ROM of mine 1 gravity and magnetic separation was applied. The processing scheme can be seen in the flow sheet in Figure 25. First, the material was split in the fractions < 2 mm and > 2 mm and then processed separately. Multiple comminution and screening steps have been applied to fraction > 2 mm, before sorting. However, the basic sorting procedure was the same. The main sorting stages were done by shaking tables, where the heavy fractions of the first table stage have been used as feed for the second shaking table stage. For the fraction > 2 mm a third shaking table stage has been added, to reprocess the middlings of stage 2 (see Figure 14 and Figure 15). After the shaking stages, the material was dried and fed into a magnetic separator (see Figure 16). The concentrates from the shaking tables were combined to form a final concentrate, so the performance of artisanal and mechanical processing is comparable.

As can be seen in Table 11, the mechanical processing with all sorting measures achieved a concentrate grade of 90.30 % of SnO_2 with a recovery rate of 72.84 %. It has to be mentioned that the recovery rate was slightly higher, without applying magnetic separation (recovery 74.76 %).

Table 11: Mechanical processing results of ROM 1 (Cassiterite)

Mine 1 ROM	Gravity Separation			Gravity + Magnetic Separation		
	SnO_2 grade [%]	SnO_2 recovery [%]	Mass recovery [%]	SnO_2 grade [%]	SnO_2 recovery [%]	Mass recovery [%]
ROM/Feed	2.17	-	-	2.17	-	-
Concentrate	77.65	74.76	2.08	90.39	72.84	1.74
Waste	0.56	25.24	97.92	0.60	27.16	98.26

As a concentrate grade of 77.65 % is sufficient for export, in this scenario 25.24 % of the target mineral remained in the waste. For application of magnetic separation it is remarkably easy to recover the waste material easily and reprocess it with the next feed or processing step. With this, it is highly likely that further processing tests would have led to a concentrate grade of over 90 % and a recovery of about 75 %. The mechanical processing scheme showed to be capable of a throughput of at least 1 t of feed per hour.

For ROM 1 the mechanical processing has also been performed with recovered artisanal tailings from the washing pond (Table 12). The recovered tailings had a total mass of 180 kg and a SnO_2 grade of 0.6 %. With this, about 1 kg of pure SnO_2 was still contained in the artisanal tailings. The sorting measures basically followed the flow-sheet in Figure 25, with the difference being the material was not split with a sieve of 2 mm but of 3 mm mesh size. The two produced fractions of < 3 mm and > 3 mm were processed on three shaking table stages each and finally combined for magnetic separation. This reprocessing test can be considered as successful, as an additional 740 g of concentrate with a grade of 59.23 % SnO_2 was produced. Here the same observation was made as for the main mechanical processing test of Mine 1. By applying magnetic separation the concentrate grade was increased but the recovery rate decreased.

Table 12: Mechanical reprocessing results of artisanal tailings of mine 1 (Cassiterite)

Mine 1 Artisanal Tailings	Gravity Separation			Gravity + Magnetic Separation		
	SnO_2 grade [%]	SnO_2 recovery [%]	Mass recovery [%]	SnO_2 grade [%]	SnO_2 recovery [%]	Mass recovery [%]
Feed	0.60	-	-	0.60	-	
Concentrate	43.79	38.19	2.08	59.23	18.66	0.15
Waste	0.42	61.81	97.92	0.51	81.34	99.85

6.2 Mechanical Processing of ROM 2/Mixed Sn-Ta

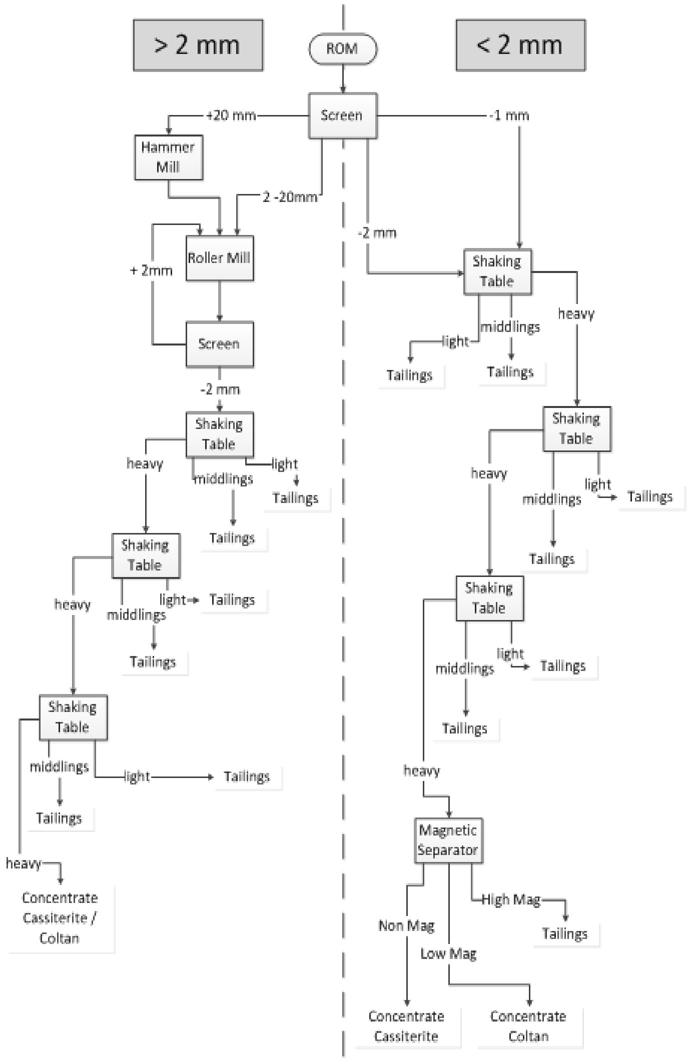


Figure 26: Flow sheet of the mechanical processing scheme of ROM 2 (Mixed Sn-Ta)

For sorting the ROM of mine 2 gravity and magnetic separation was applied. The processing scheme can be seen in the flow sheet in Figure 26. Again, the total sample was split in two fractions using a screen of 2 mm meshes. While the coarser fraction was comminuted and screened further before sorting, both fractions were sorted in three stages using shaking tables. From each shaking table, only the heavy fraction was fed to the next stage. After the third stage, the heavy fractions were combined and fed to a magnetic separator in order to clean out ferro-magnetic impurities and separate the contained tantalum and cassiterite from each other.

Results of mechanical processing of ROM 2 are shown in Table 13 and in the annex. The final concentrate had a Ta_2O_5 grade of about 10.2 % with a recovery of almost 11 %. The enrichment factor for SnO_2 was as high as 79. However, as to the low grade of the feed, the SnO_2 grade was 36.32 %, with a SnO_2 recovery of about 17 %. In the tailings fraction 0.18 % tantalite and 0.36 % SnO_2 was contained. With 83 % of the SnO_2 and 89 % of the Ta_2O_5 remaining in the waste, efficient recovery of the target minerals has not been achieved. As the waste material of the artisanal processing tests for mine 2 (mixed Sn-Ta) could not be recovered due to the applied processing technique (linear ground sluicing). Therefore mechanical reprocessing of the artisanal waste material was not performed.

Table 13: Mechanical processing results of ROM 2 (Mixed Sn-Ta)

Mine 2 ROM	SnO_2 grade [%]	SnO_2 recovery [%]	Ta_2O_5 grade [%]	Ta_2O_5 recovery [%]	Mass recovery [%]
Feed	0.46	-	0.19	-	-
Concentrate	36.32	16.95	10.19	10.97	21
Waste	0.38	83.05	0.1	89.03	97.92

6.3 Mechanical Processing of ROM 3/Wolframite

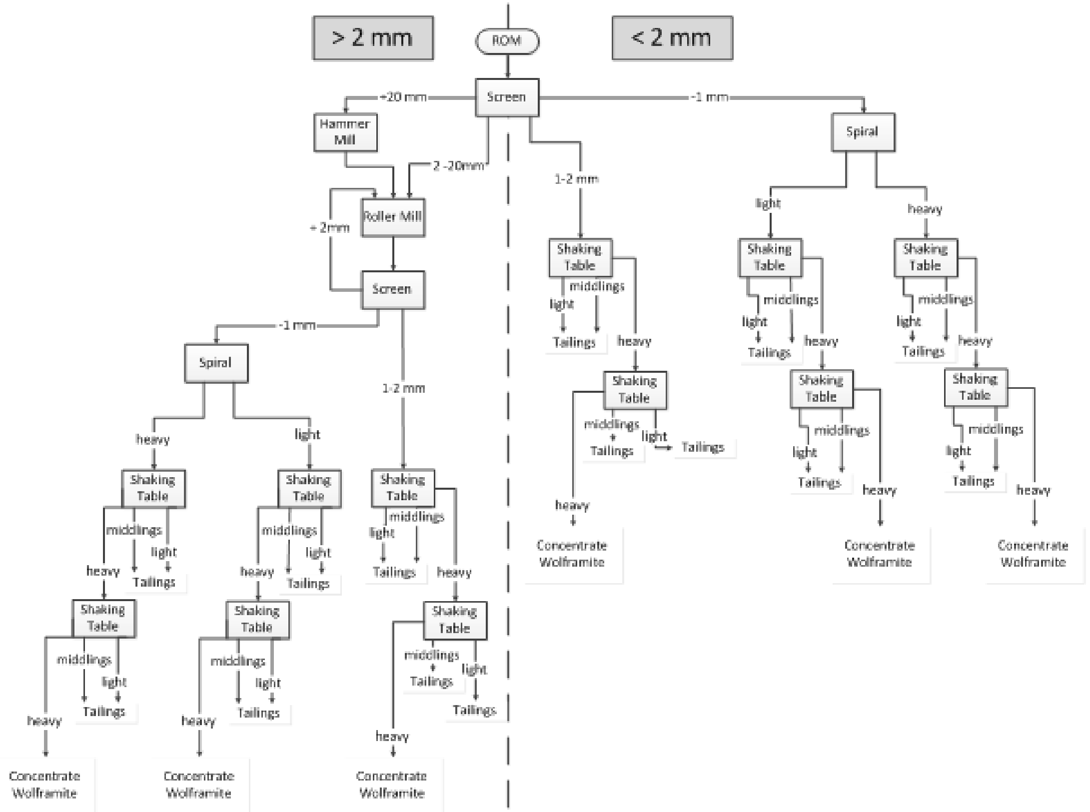


Figure 27: Flow sheet of the mechanical processing scheme of ROM 3 (Wolframite)

For sorting the ROM of mine 3 gravity separation was applied by shaking tables and a spiral. The processing scheme can be seen in the flow sheet in Figure 27. As a first step, screening of the material produced two fractions, again. In opposite to the previous tests, these fractions were split again by screening. The fine material of these fractions was fed to a spiral, before sorting on two shaking table stages. The spiral was applied to produce two further fractions with similar or more alike physical properties to operate the shaking tables with a higher efficiency. However, the final heavy products of the shaking tables was combined to form the final concentrate.

The results of the mechanical processing tests of ROM 3 are shown in Table 14. While the feed grade was about 0.54 % WO_3, the concentrate could be enriched by factor 63 to a grade of 35 % WO_3. The recovery rate was 26 %. In the tailings a grade of 0.4 % WO_3 was determined. Taking this into account the waste material remained with a grade of 0.4 % WO_3 and about 74 % of the total target minerals. The recovered waste material of the artisanal processing test has been recovered but was lost during the drying process. Therefore mechanical reprocessing tests with the artisanal waste material could not be performed.

Table 14: Mechanical processing results of ROM 3 (Wolframite)

Mine 3 ROM	WO_3 grade [%]	WO_3 recovery [%]	Mass recovery [%]
Feed	0.54	-	-
Concentrate	34.07	26.02	0.41
Waste	0.4	73.98	99.59

7. Review of Mineral Processing Performance

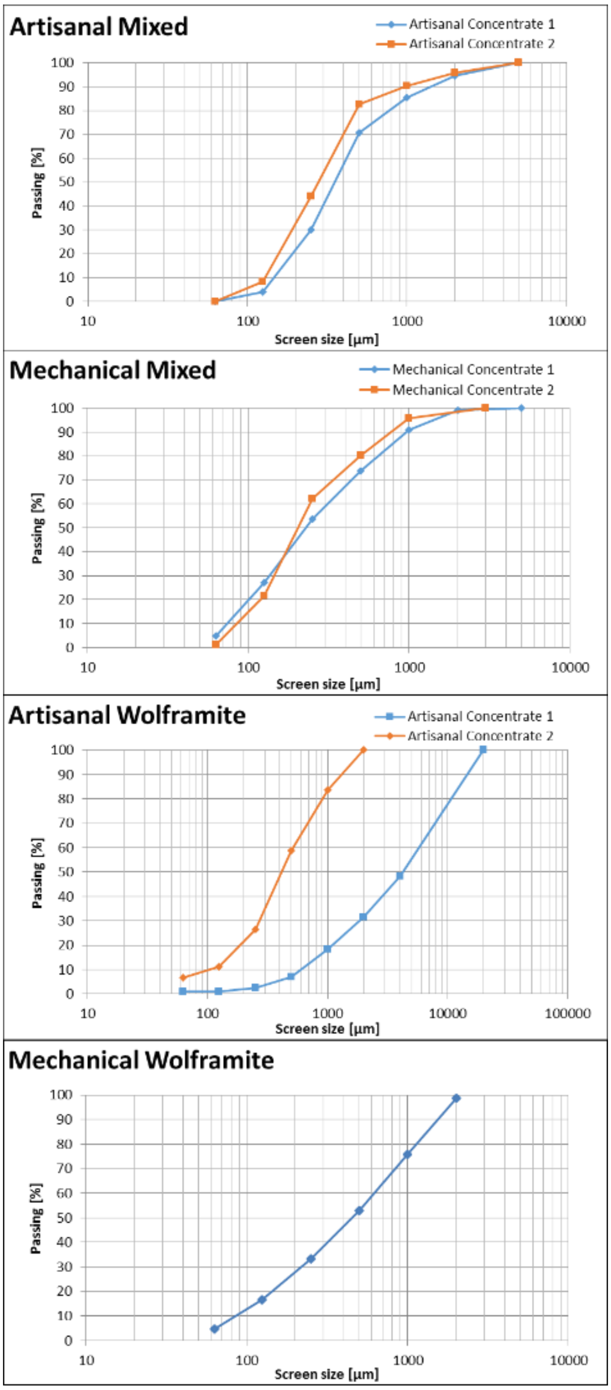


Figure 28: Grain size distribution of the concentrates produced in the processing tests of mine 2 (Mixed) and mine 3 (Wolframite)

For further assessment of the performance of the processing tests, all concentrates have been analyzed by analytic screening and geochemical analysis, as described in chapter 3.3. The findings of this analysis provided information to assess the differences in performance between artisanal and mechanical processing, as well as the main points of losses of target minerals. Figures in this chapter are only examples representing the most relevant finings.

In Figure 28 the grain-size distribution of selected concentrates is shown. On the x-axis the grain size is indicated in a logarithmic scale. On the y-axis the cumulative percentage of the total mass that is passing the screen at the specific screen sizes is shown. The concentrates are showing a more narrow distribution than the ROM, where grains occurred between 63 µm and 20mm.

More detailed, both concentrates of the Wolframite feed are showing coarser grain sizes than those of the mixed Sn-Ta feed. Also, while in case of the material from Mine 1 (Mixed Sn-Ta) concentrate 1 is slightly coarser grained than concentrate 2, this difference is even more obvious for the artisanal Wolframite concentrates. This indicated, for the processing scheme of mine 2 (Mixed Sn-Ta, ground sluice) and 3 (Wolframite, ground sluice), the average grain size of the products decrease with advance in processing stages. However, as mentioned in chapter 4.2, about 60 % of the target minerals of Mine 2 (Mixed Sn-Ta) and about 10 % of the target mineral of Mine 3 (Wolframite) were located in the grain sizes below 63 µm (see chapter 4.2). This fraction is more or less abundant in the ore concentrates. Only the first artisanal concentrate and the Mechanical concentrate of Mine 3 (Wolframite) are showing small quantities of this fraction.

This indicates that major losses of the target minerals have been suffered in the small grain-size fraction below 63 µm. This gets even more obvious in Table 15, where the basic grain-size distribution properties of the produced concentrates and sampled ROMs are show. As can be seen, all of the concentrates are showing no or only little content of grains below the size of 63 µm. Even though the mechanical concentrates of Mine 2 (Mixed Sn-Ta) and Mine 3 (Wolframite) managed to recover small amounts of this fine fraction, it is obvious that neither artisanal, nor mechanical processing was suitable to selectively process material with a grain-size of below 63 µm. When looking at Figure 28 it becomes obvious that the curves for the mechanical concentrates are shoving a steeper curve progression in the lower left than those for artisanal processing. This indicates that in the mechanical concentrates a higher share of small grain-sizes is present.

Table 15: Basic grain size distribution properties of the produced concentrates and sampled ROMs

	Mine 1 / Cassiterite	Mine 2 / Mixed Sn-Ta	Mine 3 / Wolframite
ROM	d_{50} at 2 mm	d_{50} at 250 µm	d_{50} at 3 mm
Artisanal Conc.	d_{50} at 2.3 mm	d_{50} at 250 and 350 µm	d_{50} at 400 µm and 4 mm
Mechanical Conc.	d_{50} at 500 µm	d_{50} at about 200 µm	d_{50} at 450 µm
ROM	12 % < 63 µm	30 % < 63 µm	25 % < 63 µm
Artisanal Conc.	0 % < 63 µm	0 % < 63 µm	0 and 6 % < 63 µm
Mechanical Conc.	0 % < 63 µm	0 and 6 % < 63 µm	4.5 % < 63 µm
ROM	3.5 % > 20 mm	>1 % > 20 mm	15 % > 20 mm
Artisanal Conc.	0 % < 20 mm	0 % < 20 mm	0 % < 20 mm
Mechanical Conc.	0 % < 20 mm	0 % < 20 mm	0 % < 20 mm

As can be seen in Table 15 and Figure 28, the coarsest grain size fraction which has been contained in the ROMs is missing in the concentrates, as well. In the case of mechanical processing and the artisanal processing at mine 1 (Cassiterite) this is due to the application of comminution. For Mine 2 (Mixed Sn-Ta) and Mine 3 (Wolframite) no comminution has been applied. This means for the last two artisanal processing tests, the coarse grain size fractions, including the contained target minerals have been lost. The amount of these losses will be presented more detailed below.

7.1 Performance of Processing Tests ROM 1/Cassiterite

Figure 29 shows the target mineral distribution and grades of target minerals in the different grain-size fractions of the sample material (ROM) and the concentrates of the processing tests (Artisanal/Mechanical) for mine 1 (cassiterite). It has to be noted that grain size fractions are defined differently for the concentrate as for the ROM. The presented mechanical concentrates are the heavy fraction of the final shaking table stages for the processed fractions < 2mm (Mechanical 1) and > 2mm (Mechanical 2). This was done to evaluate the actual performances of the density sorting, exclusively. However, as discussed below, both concentrates are missing most of the fine and coarse grain sizes.

As for the ROM the main share of target minerals is contained in the medium sized fractions, this observation is less critical than for the other two sampled mines. By having a look at the grades of the different grain size fractions in the artisanal concentrate (Artisanal in Figure 29), it stands out that all relevant fractions are showing a grade of more than 80 % SnO_2. On the other hand, the grades in the mechanical concentrates are varying from ca. 30 % and 80 %. This indicates the artisanal processing recovered only very high grade SnO_2 particles, while leaving grains with contained barren rock (intergrown) in the waste. This of course increases the total concentrate grade but decreases the recovery rate.

In contrast, the grades of the fractions of the first mechanical concentrate (mechanical 1) is showing stronger variations. This indicates that intergrown grains or minor amounts of barren rocks have been sorted into the heavy fractions. However, even though this less selective sorting resulted in a lower overall concentrate grade, a higher recovery rate has been achieved. Also, as can be seen in Figure 29, the recovery of target minerals in the grain-size fraction between 63 µm and 500 µm of mechanical processing is way higher than that for artisanal processing This might be due to a higher amount of target minerals in that fraction is generated by comminution and has therefore be assessed further.

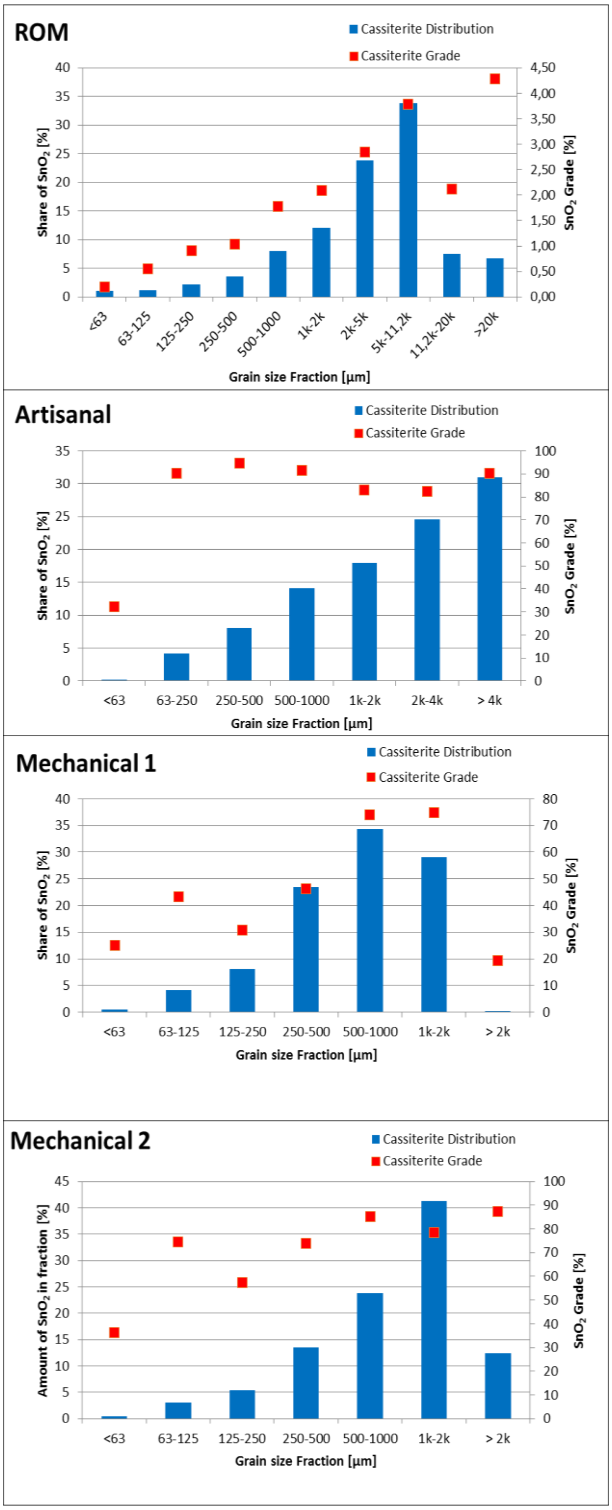
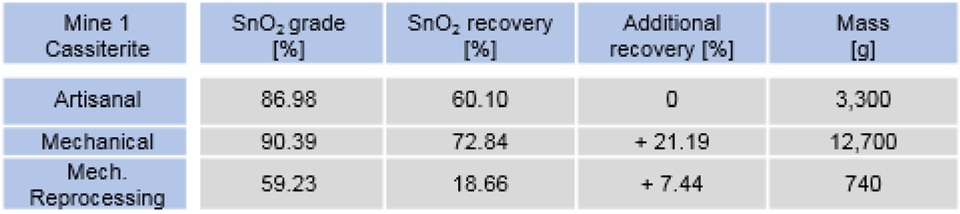


Figure 29: Distribution and grade of target minerals in the different grain-size fractions of the sample material (ROM) and the produced concentrates of mine 1 (Cassiterite)

For Mine 1 (Cassiterite) the artisanal waste material has been reprocessed with mechanical measures. As shown in chapter 6.1 the reprocessing led to mentionable additional gains of concentrate. More detailed, these experimental results clearly show that mechanical processing techniques allow for improving recovery of artisanal processing methods. The concentrate mass of artisanal processing was 3300 g with a SnO_2 grade of 86.98 %. Another 740 g were recovered with mechanical methods and a SnO_2 grade of 59.23 %. This represents increase in product mass of over 20 %. The combined concentrate would still have a grade of 78.5 % SnO_2 and therefore be a marketable product. When assuming a yearly production of about 40 t/a, the increase of concentrate mass by 20 % would account for further 8 t of 78.5 % SnO_2 per year.

Table 16: Performance of the processing tests by final concentrates of mine 1 (Cassiterite)



Mine 1 Cassiterite	SnO_2 grade [%]	SnO_2 recovery [%]	Additional recovery [%]	Mass [g]
Artisanal	86.98	60.10	0	3,300
Mechanical	90.39	72.84	+ 21.19	12,700
Mech. Reprocessing	59.23	18.66	+ 7.44	740

Table 16 shows the properties of the final concentrate of each test regarding mine 1 (cassiterite) are shown. The masses of the concentrates do of course not relate to the percentages of other properties, as in each processing test, other quantities of feed have been used (Table 6). Also, for the mechanical reprocessing the feed had another grade than the ROM. For calculating the values of "additional recovery" the recovery rate of the artisanal processing test have been taken as base value. However, the two main tests are showing good recovery rates and product grades. This is most probably due to the concentration of target minerals in the medium sized grain-size fractions, as those can be processed more efficiently.

The results for the two main tests are showing a better performance for mechanical processing compared to artisanal processing. While the artisanal processing has left about 40 % of the target mineral in the waste, the mechanical process lost only about 28 % and hence recovered 12 % more of the target minerals contained in the feed. Taken into account the performance of the artisanal processing, mechanical processing showed to exceed its performance of recovery by 21 %. Additionally, by reprocessing waste material from the artisanal processing test, about 19 % of the lost target minerals of the artisanal process could be recovered, leading to an increase in performance of the artisanal process of about 7 %. Taking into account both mechanical tests, the recovery rate of the artisanal processing test was exceeded by 28.63 %. The main observations of the test series of mine 1 (cassiterite) are shown in the box below.

Main Findings Mine 1/Cassiterite:

- High performance in terms of grades and recovery of both mechanical and artisanal processing due to the properties of ROM.
- Mechanical processing exceeds recovery of artisanal processing by 28.63 %.
- Fine grain size fractions could not be recovered with either processing measures.
- The artisanal sorting process was shown to be highly selective, only recovering high grade material resulting in a high concentrate grade but a reduced recovery rate.
- In terms of capacity the mechanical process showed to be capable of a throughput of about 1 t/h while the artisanal process has a capacity of a maximum of 250 kg per hour.
- Potentially higher efficiency of mechanical processing for small grain sizes (63 µm-500 µm)

7.2 Performance of Processing Tests ROM 2/Mixed Sn-Ta

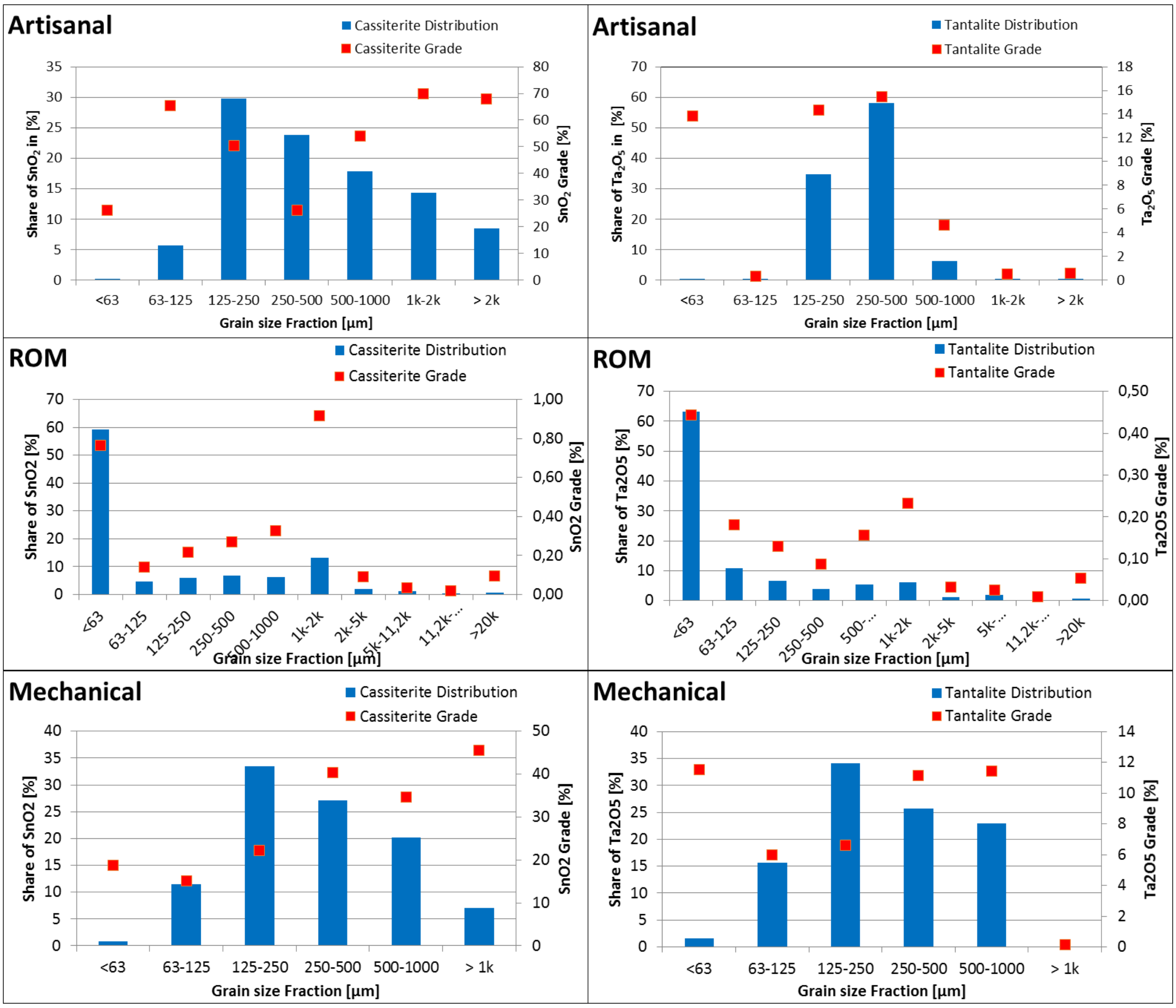


Figure 30: Distribution and grade of target minerals in the different grain-size fractions of the sample material (ROM) and the produced concentrates of mine 2 (Mixed Sn-Ta)

Figure 30 shows the target mineral distribution and grades of target minerals in the different grain-size fractions of the sample material (ROM) and the concentrate of the processing tests (Artisanal/Mechanical) for mine 2 (mixed Sn-Ta). Again grain size fractions are defined differently for the concentrate as well as for the ROM. Distributions and grades shown for the artisanal process are those of the combined concentrate of ground sluicing stages one and two. Distributions and grades shown for the mechanical process are those of the final concentrate of the sieved ROM finer than 2 mm. While both concentrates of the mechanical processing showed alike distributions of grain-sizes and grades, the concentrate of the ROM < 2 has been selected, as no comminutions was applied to it and therefore the grades in the grain-size fractions have only be affected by sorting.

From Figure 30 it is obvious that most of the target minerals in the ROM have been contained in the fine grain-size fractions below 63 µm. This fraction contains about 60 % of the total target minerals in the ROM (chapter 4.2). In contrast, the main share of target minerals in the concentrates (artisanal and mechanical) is found in the grain size-fractions 125 µm to 250 µm, or in one case in the fraction 250 µm to 500 µm. Therefore, the high amount of target minerals in the fraction below 63 µm was lost during the sorting process.

Even though the concentrates tempt to consist of smaller grain with increased number of sorting stages when linear ground sluicing is applied, it is highly unlikely that the fraction below 63 µm is recoverable with this process. Grains of such small sizes tempt flow with the process water, even with low stream velocity, instead of settling on the ground. Especially the usual practice of shoveling the material against the water stream will lead to the small particles following the turbulent water stream. Figure 30 shows again that the recovery of the mechanical product is proportionately higher in the grain-size fractions between 63 µm and 500 µm than that for the artisanal concentrates. As no comminution has been applied in the shown case for mechanical processing, a higher efficiency of the applied mechanical sorting measures than that of the respective artisanal processing for fine grained target minerals has been demonstrated.

Table 17: Performance of the processing tests by final concentrates of mine 2 (Mixed)

Mine 1 Cassiterite	SnO_2 grade [%]	SnO_2 recovery [%]	Additional recovery [%]	Ta_2O_5 grade [%]	Ta_2O_5 recovery [%]	Additional recovery [%]	Mass [g]
Artisanal	41.26	24.49	+ 44.48	9.92	13.35	+ 21.69	950
Mechanical	36.32	16.95	0	10.19	10.97	0	2,290

However, both sorting attempts (artisanal and mechanical) have not been able to process the ROM efficiently. As can be seen in the highest recovery rate achieved is that for SnO_2 with artisanal processing. Again, masses of the concentrates in Table 17 do not relate to the percentages of other properties, as in each processing test, another quantity of feed has been used (Table 6). The recovery of the artisanal processing test is 44 % higher than that of the mechanical processing. Even though this is a large number, both recovery rates cannot be described as good results. About 75.59 % of the SnO_2 was left in the waste while processing with a ground sluice. For Ta_2O_5 the highest recovery rate was 13.35 %, again achieved with artisanal processing. The mechanical processing only recovered 10.97 % of total Ta_2O_5 contained in the feed. Artisanal processing therefore recovered 21.69 % more target mineral than the mechanical processing.

Summed up, both tests showed that the applied sorting schemes worked not efficient. In both cases the majority of target minerals has been lost in the waste. This of course, is mainly due to the fact that 60 % of the target minerals are contained in the grain-size fraction below 63 µm and are therefore not recoverable by density sorting. However, about 40 % of the target minerals have been contained in coarser grain-sizes, and could have been recovered.

It is reasonable to assume that the target minerals in the medium grain-sizes are not liberated and therefore the difference in density between grains containing target mineral and grains of barren rock was not sufficient to separate them by sorting. For this instance it might be considered to use even more narrow grain size distributions when using the shaking table. Also the supply of process water should be reduced to increase the potential of target mineral containing grains to be sorted to the heavy fraction. If further liberation of the target mineral is sought, it has to be considered that comminution always produces grains which are smaller than the targeted grain size, due to brittle behavior of the minerals. Therefore, comminution might result in an additional share of target minerals in not recoverable grain sizes. This is also the main problem that was faced in the test series of mine 2 (mixed Sn-Ta). As particles of the grain size (<63 µm) are always very light, irrespectively their mineralogical composition, they tend to follow the water flow instead of settling on the ground. Density sorting is therefore not the appropriate sorting method for this kind of grain size. Flotation or leaching processes would be more suitable for such feeds.

These processes however are quite sophisticated, require advanced process management and also advanced waste water treatment. Therefore, those methods are not considered to be a good solution for ASM operations. The main findings of the tests series for mine 2 (mixed Sn-Ta) are shown in the box below.

Main findings Mine 2/Mixed Cassiterite-Coltan:

- Low performance of mechanical and artisanal processing due to the properties of ROM.
- Artisanal processing exceeds recovery of mechanical processing by 21.69 % (Ta) to 44.48 % (Sn).
- Concentrates lost most of fine grain-size target minerals as they could not be recovered with either processing measures.
- Higher recovery rates might be possible with *vernier* adjustment of shaking tables or additional stages of ground sluicing but most likely without reaching overall satisfying levels.
- Mechanical processing proved to be more efficient for fine grain sizes (63 µm-500 µm)
- In general, the processed ore material gives a challenge for density sorting measures. Further testing of multiple sorting techniques and processing schemes is necessary. Apart from chemical treatment advanced equipment like an upstream-classifier might be considered for improvement of recovery rate and concentrate grade.

7.3 Performance of Processing Tests ROM 3/Wolframite

Figure 31 shows the target mineral distribution and grades of target minerals in the different grain-size fractions of the sample material (ROM) and the concentrate of the processing tests (Artisanal/Mechanical) for mine 3 (wolframite). Again grain size fractions are defined differently for the concentrate as for the ROM. Distributions and grades shown for the artisanal process are those of the heavy fractions of the panning stages 1 and 2 (see chapter - 30 -0). Distributions and grades shown for the mechanical process are those of the combined final concentrate. However, it can be seen that the distribution and grades of the concentrates differ from that in the ROM.

For the artisanal process no comminution has been applied, therefore the main share of WO_3 was recovered in the coarse grain-size fractions, as the distribution of the target minerals was concentrated in the coarse fractions of the ROM, as well. For the mechanical processing all grains coarser than 2 mm have been comminuted and are therefore contained in the medium fractions of the concentrate.

One of the first observations which sticks out when looking at Figure 31 is probably the low grade of most of the fractions shown for the concentrate artisanal 2. Here, the light fractions of the first panning stage have been panned again, to recover left target minerals. This approach showed to be not very effective, as it produced only a small amount of a low quality concentrate. Therefore, the artisanal 1 concentrate might rather be evaluated closer. Comparing the distributions and grades of the mechanical concentrate and that of artisanal 1, it can be found that the mechanical process recovered a higher share of its concentrate in the small grain sizes. Similar observation have been done before, so this can be seen as an additional confirmation for the higher efficiency of mechanical processing measures.

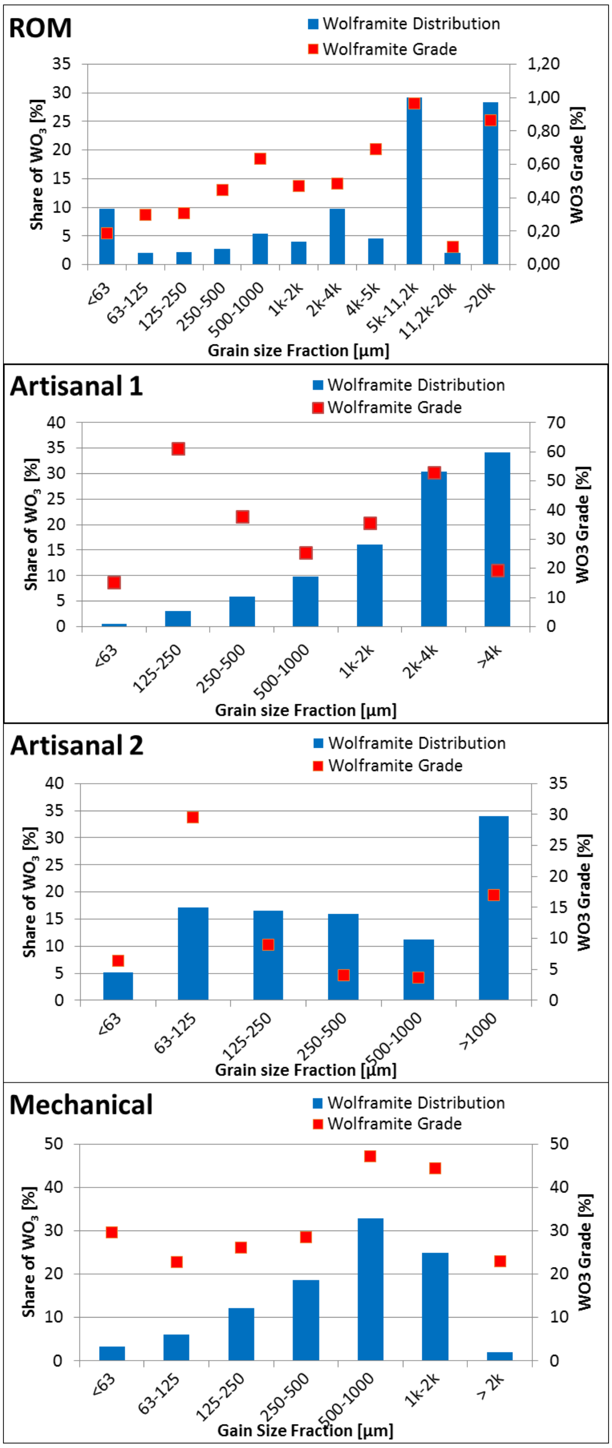


Figure 31: Distribution and grade of target minerals in the different grain-size fractions of the sample material (ROM) and the produced concentrates of mine 3 (Wolframite)

When evaluation the distribution in the concentrate artisanal 2 it might be assumed that a good efficiency has been achieved for smaller grain-sizes as well. Looking more closely, the low grade of the recovered fine grained fraction shows that there was no selective sorting performed. Further, the mechanical concentrate shows higher overall grades of WO_3 in the recovered fractions than the artisanal concentrates. It can therefore be assumed that the mechanical sorting process worked more selective than the artisanal processing. Most likely this is due to a higher amount intergrown target minerals in the artisanal feed, while the ROM was liberated by comminution prior to mechanical processing.

As can be seen in Table 18 both processing tests produced low quality concentrates. The highest grade and recovery rate was achieved by mechanical processing. Here, a concentrate with a WO_3 grade of 34.07 % and a recovery rate of 26.02 % has been produced. The recovery rate of the mechanical processing is about 16 % higher than that for the artisanal process. However, both concentrates are not marketable if not enriched further or blended with high quality concentrates.

Table 18: Performance of the processing tests by final concentrates of mine 3 (Wolframite)

Mine 1 Cassiterite	WO_3 grade [%]	WO_3 recovery [%]	Additional recovery [%]	Mass [g]
Artisanal	21.42	22.41	0	1,330
Mechanical	34.07	26.02	16.11	3,487

Regarding the artisanal processing, the low performance is probably explainable with the lack liberated target minerals and the high amount of target minerals in the large grain size fractions. When evaluating the target mineral distribution of the ROM (Figure 31), it can be seen that about 75 % of the WO_3 in the ROM is contained in the grain-size fractions coarser than 2 mm. As shown in the beginning of chapter 7 the artisanal concentrates of mine 3 (wolframite) are not containing any grains coarser than 2 mm and therefore did not recover any of the target minerals contained in the richest grain-size fractions. Further, as the target minerals tend not to crystallize exclusively in coarse grains their abundance in the coarse fractions of the ROM is most likely in an intergrown state. With these findings it is highly recommended to apply comminutions before artisanal processing, as an attempt to liberate the target minerals and recover those contained in the coarser grain-size fractions.

Even though the mechanical processing techniques do outperform the artisanal techniques, it is highly likeable that the results could be increased sharply by adjustment of the shaking table parameters. As the used equipment was lacking in options for adjustment, this assumption could not be tested. However, it is obvious that the performance of the mechanical processing is poor. Considering the distribution of target minerals in the grain-size fractions, it a larger share of those is most probably recoverable. Further testing of mechanical processing techniques is recommended.

Main findings Mine 3/Wolframite:

- Low performance of mechanical and artisanal processing.
- Mechanical processing exceeds recovery of artisanal processing by 16.11 %.
- Comminution of coarse grains should be applied before processing with artisanal measures.
- Possible higher recovery rates with vernier adjustment of shaking tables or additional stages.
- Mechanical processing proved to be more efficient in fine grain sizes (63 µm-500 µm)

8. Economic Analysis

The following chapter presents the results of an economic analysis regarding the profitability of each of the sampled mine operations. The calculations are representing the profitability of the operations in different scenarios, including the investment in mechanical processing equipment. It has to be noted that the representative nature of the results is limited as only three operations were taken into account and only one feed sample was taken at each operation. As ore deposits are usually heterogeneous, it is very likely that an additional feed sample from another tunnel of the same operation would lead to slightly different test results. Also, the calculations are depending on the information that was given during the interviews with mining company employees. Since not all necessary data could be gathered during these interviews several assumptions were made. Where necessary assumptions will be explained in footnotes.

The profitability analysis includes the comparison of the different processing techniques that were tested for each feed. Within each case a low-price, a current-price, and a peak price for the produced minerals was tested. In the low-price scenario the lowest average annual price within the last 10 years for the corresponding concentrate was assumed. In the current-price scenario the average export prices from 2015 were assumed for the concentrates. In the peak-price scenario the highest average annual price within the last 10 years for corresponding concentrates was used.

Additionally, to allow for a comparable results, the operational data details as shown in have been assumed for each evaluated mine:

Table 19: Fixed operational data which has been assumed for the profitability analysis

Fixed operational data	
Annual throughput	2,080 t (for 1t/h, 8h/d and 260d/y)
Evaluated timeframe	10 years
Maintenance costs	10 % of investment annually
Energy costs	0.15 $/kWh on grid, 0.29 $/kWh off grid[10]
Interest rate	15 %

Individual operational costs have been converted from RWF to USD[11]. General and Administrative Expenses[12] have been excluded, as such costs could not be evaluated in detail. The selling price of minerals for the mining companies has been calculated by the three price scenarios (by export prices) reduced by an exporter share of 16 %[13].

Prices which are paid to Mining companies are further influenced by the product grade/quality. For cassiterite export quality is assumed to be 64 % Sn in the concentrate. For wolframite 60 % WO_3 is the concentrate grade and for tantalite a grade of 26 % Ta_2O_5 is considered export quality. A bonus

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[10] Calculated from Diesel consumption generator: 0,26 l/kWh and diesel price of 1,10 $/l

[11] Exchange rate: RWF 781 = USD 1

[12] For example rent, insurance, utilities and managerial salaries

[13] The exporters share is assumed to be 16 % of the export price, even though this is not true for every mineral it gives a good average. This rate has been take out of the study by Estelle Levin Consulting (2014): Evaluation of Mining Revenue Streams and Due Diligence Implementation Costs along Mineral Supply Chains in Rwanda. Contract Study for BGR & RNRA; ISBN 978-3-943566-18-5; available online at: http://www.bgr.bund.de/EN/Themen/Min_rohstoffe/CTC/Downloads/rpt_mining_revenues_rwanda_en.pdf?__blob=publicationFile&v=4

was added to high quality concentrates, while low quality concentrate grades were charged with a penalty[14]. Concentrate grades and recoveries, as well as market prices and production were held constant over the period of 10 years. Grades and recoveries of each product were taken from the test series described in chapters 5 and 6.

For theoretical investment in mechanical processing several purchases have been assumed. The equipment list for mechanical processing is shown in Table 20. The list applies for all mechanical processing tests. The calculations refer to these tests and therefore in the scenarios for mine 1 and 2 no spiral concentrator and for mine 2 no generator is included in the calculations. The mentioned roller mill is a small model with a low throughput, which however is sufficient for the operation at hand. The shaking tables are models from China, which might need some additional improvement for optimization of the separation. Costs for those improvements are included in the maintenance budget. All prizes have been found by desk study and prices for equipment which had been available online.

Table 20: Equipment list for mechanical processing

Device	$ per unit	Units	Throughput [t/h]	Consumption	
				Energy [kWh/t]	Water [m³/t]
Roller Mill	25,000	1	1	5	0
Double Decker Screen	10,000	1	10	0.3	0
Pump	3,000	1	3	0	15
Shaking Table	3,500	3	1	1	3
Generator 50kW	12,000	1	n/a	n/a	n/a
Magnetic Separator	10,000	1	10	0.5	0
Spiral concentrator	2,000	1	1.5	0	3

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[14] Bonuses and penalties were proportional to the reduced or increased concentrate grade compared to the usual export grade. This practice was confirmed in interviews with local exporters. High quality concentrates are necessary to blend lower quality concentrates to exportable material and therefore in demand. For average trading grades the above mentioned study was referred to (see [12]).

8.1 Profitability Mine 1/Cassiterite

In Table 21 the profitability analysis for the artisanal process of mine 1 is shown. The received concentrated price for the mining company is about 9 % lower than the export price in each scenario. This is the result from a 16 % treatment fee which is charged by the exporter and a bonus since concentrate grade was 4.5 % higher than export grade. Applying the results of the processing tests with a processed feed of 2,080 t/y, a concentrate mass of 39.6 t was assumed.

Total annual revenues are varying between 149,248 USD in the low-price scenario to 501,820 USD in the peak-price scenario. Total annual costs are calculated to vary between 72,046 USD and 226,942 USD. The main cost factor has been identified to be personnel costs.[15] Water costs are not taken into account, as water supply was said to be granted cost free at the operation of mine 1. With the given parameters the operation would be profitable in all scenarios. Even for the low-price scenario for cassiterite concentrates (4.14 $/kg), the operation would still achieve an annual profit of about 50,000 USD.

In the second case for mine 1, whose data is shown in Table 1, it was assumed that some investment has been made in order to improve profitability, regarding to the equipment listed in Table 20, all machines have been purchased except for a magnetic separator and a spiral concentrator (mechanical setup 1 as shown in Table 22). In this scenario the total investment sums up to about 60,500 USD and is depreciated over 7 years with an annual depreciation rate of about 8,643 USD. The costs for mechanical processing (per kg) is slightly lower than it is for applying artisanal methods. The operational profit however is slightly higher as the recovery was about 10 % higher (with slightly lower concentrate grade) for the mechanical processing and therefore more concentrate is produced out of the same feed mass. Due to depreciation of the machines the net profit within the first 7 years is lower than when applying artisanal methods. This calculation does not take into account that the mechanical process could decrease personnel costs, as probably higher qualified and therefore more expensive personnel would be necessary to maintain and use the equipment. Also, transportation of ROM and the actual mining process would remain the same with the same demand for workers.

The profitability analysis of the third case for mine 1 is shown in Table 23 (mechanical setup 2). The only difference to the second case is the simulated purchase of a magnetic separator for the final processing step. As this step slightly reduces recovery but greatly improves the concentrate grade the results justify the investment of a magnetic separator of about 10,000 USD (ref. Table 20). By applying mechanical processing with magnetic separation the average current price per kg for the mining company could be improved by about 5 % to 8.49 $/kg. Furthermore about 6,500 kg additional concentrate could be produced per year, which results in a total annual profit in the current price scenario of about 128,000 USD. Compared to the 113,000 USD that are generated with artisanal methods, this gives an improvement of about 13 %.

As has been shown in chapter 6.1 the artisanal processing bears the potential to gain additional recovery by reprocessing the waste material from the artisanal processing.

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[15] For the case of Mine 1 personnel costs were assumed to be 40 % of the export price per kg, as it is an cooperative and workers are paid by share, as stated by the mine management.

Table 21: Profitability analysis of artisanal operation of mine 1

Mine 1/Cassiterite - Artisanal	**Unit**	low	current	high
Export Price	USD/kg	4.14	8.91	13.92
Price/kg for Mining Company	USD/kg	3.77	8.11	12.67
Feed per year	t	2,080	2,080	2,080
Concentrate per year	kg	39,595	39,595	39,595
Total Revenue	**USD**	**149,248**	**321,208**	**501,820**
Royalties	USD	5,970	12,848	20,073
Net Proceeds	**USD**	**143,278**	**308,360**	**481,747**
Total Personnel Costs	USD	65,570	141,118	220,467
Total Energy Costs	USD	5,490	5,490	5,490
Total Water Costs	USD	-	-	-
Total Fixed Costs	USD	985	985	985
Total Costs	**USD**	**72,046**	**147,594**	**226,942**
Operating Profit	**USD**	**71,233**	**160,766**	**254,805**
Depreciation	USD	-	-	-
Profit b Tax	**USD**	**71,233**	**160,766**	**254,805**
Tax	USD	21,370	48,230	76,441
Net Income	**USD**	49,863	112,536	178,363
Revenue per t feed	USD/ton	71.75	154.43	241.26
Revenue per kg conc.	USD/kg	3.77	8.11	12.67
Costs per t feed	USD/ton	34.64	70.96	109.11
Costs per kg conc.	USD/kg	1.82	3.73	5.73
Profit per t feed	USD/ton	23.97	54.10	85.75
Profit per kg conc.	USD/kg	1.26	2.84	4.50
Return	**USD**			
Interest rate	%	15.00	15.00	15.00
NPV over 10 years	**USD**	250,250	564,794	895,164
Total Invest	USD	-	-	-

Table 22: Profitability analysis of mechanical operation of mine 1, setup 1

Mine 1/Cassiterite – Mechanical 1	**Unit**	low	current	high
Export Price	USD/kg	4.14	8.91	13.92
Price/kg for Mining Company	USD/kg	3.29	7.09	11.08
Feed per year	t	2,080	2,080	2,080
Concentrate per year	kg	55,173	55,173	55,173
Total Revenue	**USD**	**181,733**	**391,121**	**611,044**
Royalties	USD	**7,269**	15,645	**24,442**
Net Proceeds	**USD**	**174,464**	**375,476**	**586,602**
Total Personnel Costs	USD	91,366	196,636	307,202
Total Energy Costs	USD	10,919	10,919	10,919
Total Water Costs	USD	-	-	-
Total Maintenance Costs		6,050	6,050	6,050
Total Fixed Costs		985	985	985
Total Costs	**USD**	**109,320**	**214,590**	**325,156**
Operating Profit	**USD**	**65,143**	**160,886**	**261,446**
Depreciation	USD	8,643	8,643	8,643
Profit b Tax	**USD**	**56,500**	**152,243**	**252,803**
Tax	USD	16,950	45,673	75,841
Net Income	**USD**	**39,550**	**106,570**	**176,962**
Revenue per t feed	USD/ton	87.37	188.04	293.77
Revenue per kg conc.	USD/kg	3.29	7.09	11.08
Costs per t feed	USD/ton	52.56	103.17	156.33
Costs per kg conc.	USD/kg	1.98	3.89	5.89
Profit per t feed	USD/ton	19.01	51.24	85.08
Profit per kg conc.	USD/kg	0.72	1.93	3.21
Return	**USD**			
Interest rate	%	15.00	15.00	15.00
NPV over 10 years	**USD**	**187,036**	**523,393**	**876,675**
Total Invest	USD	60,500	60,500	60,500

Table 23: Profitability analysis of mechanical operation of mine 1, setup 2

Mine 1/Cassiterite – Mechanical 2	**Unit**	low	current	high
Export Price	USD/kg	4.14	8.91	13.92
Price/kg for Mining Company	USD/kg	3.94	8.49	13.26
Feed per year	t	2,080	2,080	2,080
Concentrate per year	kg	46,176	46,176	46,176
Total Revenue	**USD**	**182,087**	**391,882**	**612,234**
Royalties	USD	**7,283**	15,675	**24,489**
Net Proceeds	**USD**	**174,803**	**376,207**	**587,744**
Total Personnel Costs	USD	76,467	164,570	257,106
Total Energy Costs	USD	10,919	10,919	10,919
Total Water Costs	USD	-	-	-
Total Maintenance Costs		7,050	7,050	7,050
Total Fixed Costs		985	985	985
Total Costs	**USD**	**95,421**	**183,524**	**276,060**
Operating Profit	**USD**	**79,382**	**192,683**	**311,684**
Depreciation	USD	10,071	10,071	10,071
Profit b Tax	**USD**	**69,311**	**182,611**	**301,613**
Tax	USD	20,793	54,783	90,484
Net Income	**USD**	**48,517**	**127,828**	**211,129**
Revenue per t feed	USD/ton	87.54	188.40	294.34
Revenue per kg conc.	USD/kg	3.94	8.49	13.26
Costs per t feed	USD/ton	45.88	88.23	132.72
Costs per kg conc.	USD/kg	2.07	3.97	5.98
Profit per t feed	USD/ton	23.33	61.46	101.50
Profit per kg conc.	USD/kg	1.05	2.77	4.57
Return	**USD**			
Interest rate	%	15.00	15.00	15.00
NPV over 10 years	**USD**	**230,146**	**628,187**	**1,046,255**
Total Invest	USD	70,500	70,500	70,500

8.2 Profitability Mine 2/Mixed Sn-Ta

In Table 24 and Table 25 the profitability analyses of artisanal and mechanical processing operation of mine 2 are shown. The received concentrated prices for the mining company are more than 50 % lower than the export price in each scenario. This is the result from a 16 % treatment fee which is charged by the exporter and an additional fee because of the low concentrate grades. Energy costs for mechanical processing were calculated with a price of 0.15 USD/kWh, as the operation is connected to the local power grid. Personnel costs for mine 2 were assumed to be 77 $/m per worker, as an average of the salaries for the different positions which have been stated by the mine management. Investments for equipment are the same as assumed for mine 1, including all machines stated in Table 20, except for a magnetic separator and a spiral concentrator.

Both assumed cases are profitable in all three scenarios. Artisanal processing however creates a higher net profit (17,000 USD in low-price to 96,000 USD in peak-price scenario) than mechanical processing (8,000 USD in low-price to 71,000 USD in peak-price scenario). This mainly results from a higher recovery rate as well as a higher concentrate grade for the artisanal processing tests. As discussed in chapter 7.2, the results of mechanical processing can most probably be improved by adjusting parameters of the sorting aggregates. Additional tests with the ROM of mine 1 are necessary to give a more suitable assumption for the applicability and cost efficiency of mechanical processing in this case.

Table 24: Profitability analysis of artisanal operation of mine 2

Mine 2/Mixed Sn-Ta - Artisanal	**Unit**	**low**	**current**	**high**
Export Price Cassiterite	USD/kg	4.14	8.91	13.92
Cassiterite Price/kg for Mining Company	USD/kg	1.44	3.10	4.84
Export Price Coltan	USD/kg	15.41	40.08	54.58
Coltan Price/kg for Mining Company	USD/kg	6.93	18.04	24.56
Feed per year	t	2,080	2,080	2,080
Cassiterite concentrate per year	kg	5,683	5,683	5,683
Coltan concentrate per year	kg	5,598	5,598	5,598
Total Revenue	**USD**	**47,006**	**118,585**	**165,017**
Royalties	USD	1,880	4,743	6,601
Net Proceeds	**USD**	**45,125**	**113,841**	**158,417**
Total Personnel Costs	USD	18,480	18,480	18,480
Total Energy Costs	USD	1,843	1,843	1,843
Total Water Costs	USD	-	-	-
		-	-	-
Total Fixed Costs	USD	492.09	492.09	492.09
Total Costs	**USD**	**20,815**	**20,815**	**20,815**
Operating Profit	**USD**	**24,310**	**93,026**	**137,602**
Depreciation	USD	-	-	-
Profit b Tax	**USD**	**24,310**	**93,026**	**137,602**
Tax	USD	7,293	27,908	41,281
Net Income	**USD**	**17,017**	**65,118**	**96,321**
Revenue per t feed	USD/ton	22.60	57.01	79.34
Revenue per kg conc.	USD/kg	8.27	20.87	29.04
Costs per t feed	USD/ton	10.01	10.01	10.01
Costs per kg conc.	USD/kg	3.66	3.66	3.66
Profit per t feed	USD/ton	8.18	31.31	46.31
Profit per kg conc.	USD/kg	2.99	11.46	16.95
Return	**USD**			
Interest rate	%	15.00	15.00	15.00
NPV over 10 years	**USD**	**85,405**	**326,814**	**483,414**

Table 25: Profitability analysis of mechanical operation of mine 2

Mine 2/Mixed Sn-Ta - Mechanical	**Unit**	**low**	**current**	**high**
Export Price Cassiterite	USD/kg	4.14	8.91	13.92
Price/kg Cassiterite for Mining Company	USD/kg	1.19	2.56	3.99
Export Price Coltan	USD/kg	15.42	40.08	54.58
Price/kg Coltan for Mining Company	USD/kg	7.21	18.74	25.52
Feed per year	t	2,080	2,080	2,080
Cassiterite concentrate per year	kg	4,517	4,517	4,517
Coltan concentrate per year	kg	4,474	4,474	4,474
Total Revenue	**USD**	**37,619**	**95,382**	**132,205**
Royalties	USD	1,505	3,815	5,288
Net Proceeds	**USD**	**36,115**	**91,567**	**126,916**
Total Personnel Costs	USD	6,468	6,468	6,468
Total Energy Costs	USD	2,808	2,808	2,808
Total Water Costs	USD	-	-	-
Total Maintenance Costs	USD	6,050	6,050	6,050
Total Fixed Costs	USD	492.09	492.09	492.09
Total Costs	**USD**	**15,818**	**15,818**	**15,818**
Operating Profit	**USD**	**20,296**	**75,748**	**111,098**
Depreciation	USD	8,643	8,643	8,643
Profit b Tax	**USD**	**11,654**	**67,106**	**102,455**
Tax	USD	3,496	20,132	30,737
Net Income	**USD**	**8,158**	**46,974**	**71,719**
Revenue per t feed	USD/ton	18.09	45.86	63.56
Revenue per kg conc.	USD/kg	8.33	21.12	29.27
Costs per t feed	USD/ton	7.60	7.60	7.60
Costs per kg conc.	USD/kg	3.50	3.50	3.50
Profit per t feed	USD/ton	3.92	22.58	34.48
Profit per kg conc.	USD/kg	1.81	10.40	15.88
Return	**USD**			
Interest rate	%	15.00	15.00	15.00
NPV over 10 years	**USD**	**29,483**	224,293	**348,482**
Total Invest	USD	60,500	60,500	60,500

8.3 Profitability Mine 3/Wolframite

In Table 26 and Table 27 the profitability analyses of artisanal and mechanical processing operations for mine 3 are shown. Assumptions for operation costs were mostly alike those in chapter 8.2.

The calculations showed that in case of artisanal processing the operation would only be profitable in the peak-price scenario. It is also noticeable that costs for water are one of the major cost drivers of the total production costs.[16] The case calculations for artisanal processing is somewhat questionable, as a throughput of 1 t/h was assumed. In reality however material is only processed once a week, which means that annual production is less than was assumed for the case study. If mining production could be increased to the desired amount of 1 t/h, mechanical processing could be profitable.

In the given scenario (Table 27, current pricing) profits of mechanical processing are slightly negative. There would be an annual loss of about 420 USD. Investment costs for equipment would be 62,500 USD, but might be reduced to 60,500 USD if the spiral concentrator would be neglected. Further cost reductions would be achieved by the reduction of water costs. By establishing a connection to the local water grid or a more advanced rain-water recovery and storing system, annual costs for water could be reduced and therefore the operation could be made profitable. In the peak-price scenario the artisanal operation is slightly profitable with an annual profit of 648 USD. The mechanical operation on the other hand would be profitable with an annual profit of 15,484 USD for the same pricing scenario. Even though the calculated profits are not very high, it has to be noted that applying mechanical processing would lead to significant higher income (or less loss), in two of the three pricing scenarios.

The results of the profitability calculations seem not to be liable for an ongoing mining operation. It has to be noted that the visited and sampled mine 3 (wolframite) has had already reduced its activities due to the recent wolframite pricing and little profit which has been generated, prior to this study. However, it might be the artisanal processing test was below average performance or the sampled ROM was of exceptional low quality. Further data can be found in the corresponding in Table 26 and Table 27.

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[16] Assumption that 50 % of the required water is collected from rain and is therefore free of charge. The other 50 % have to be bought at a price of 150 RWF per gallon, as stated by the mine management.

Table 26: Profitability analysis of artisanal operation of mine 3

Mine 3/Wolframite – Artisanal	**Unit**	**low**	**current**	**high**
Export Price Wolframite	USD/kg	6.59	9.72	16.00
Wolframite Price/kg for Mining Company	USD/kg	1.30	1.91	3.15
Feed per year	t	2,080	2,080	2,080
Wolframite Concentrate per year	kg	10,931	10,931	10,931
Total Revenue	**USD**	**14,190.59**	**20,930.59**	**34,453.64**
Royalties	USD	567.62	837.22	1,378.15
Net Proceeds	**USD**	**13,622.97**	**20,093.36**	**33,075.49**
Total Personnel Costs	USD	18,480.00	18,480.00	18,480.00
Total Energy Costs	USD	-	-	-
Total Water Costs	USD	13,176.80	13,176.80	13,176.80
Total Maintenance Costs		-	-	-
Total Fixed Costs		492.09	492.09	492.09
Total Costs	**USD**	**32,148.89**	**32,148.89**	**32,148.89**
Operating Profit	**USD**	**- 18,525.92**	**- 12,055.53**	**926.60**
Depreciation	USD	-	-	-
Profit b Tax	**USD**	**- 18,525.92**	**- 12,055.53**	**926.60**
Tax	USD	-	-	277.98
Net Income / Loss	**USD**	**- 18,525.92**	**- 12,055.53**	**648.62**
Revenue per t feed	USD/ton	6.82	10.06	16.56
Revenue per kg conc.	USD/kg	1.30	1.91	3.15
Costs per t feed	USD/ton	15.46	15.46	15.46
Costs per kg conc.	USD/kg	2.94	2.94	2.94
Profit per t feed	USD/ton	- 8.91	- 5.80	0.31
Profit per kg conc.	USD/kg	- 1.69	- 1.10	0.06
Return	**USD**			
Interest rate	%	15.00	15.00	15.00
NPV over 10 years	**USD**	**- 92,977**	**- 60,504**	**3,255**
Total Invest	USD	-	-	-

Table 27: Profitability analysis of mechanical operation of mine 3

Mine 3/Wolframite – Mechanical	**Unit**	**low**	**current**	**high**
Export Price Wolframite	USD/kg	6.59	9.72	16.00
Wolframite Price/kg for Mining Company	USD/kg	2.69	3.96	6.52
Feed per year	t	2,080	2,080	2,080
Wolframite Concentrate per year	kg	9,170	9,170	9,170
Total Revenue	**USD**	**24,641.87**	**36,345.83**	**59,828.52**
Royalties	USD	985.67	1,453.83	2,393.14
Net Proceeds	**USD**	**23,656.20**	**34,891.99**	**57,435.38**
Total Personnel Costs	USD	6,468.00	6,468.00	6,468.00
Total Energy Costs	USD	-	-	-
Total Water Costs	USD	13,176.80	13,176.80	13,176.80
Total Maintenance Costs		6,250.00	6,250.00	6,250.00
Total Fixed Costs		492.09	492.09	492.09
Total Costs	**USD**	**26,386.89**	**26,386.89**	**26,386.89**
Operating Profit	**USD**	**-2,730.69**	**8,505.10**	**31,048.49**
Depreciation	USD	8,928.57	8,928.57	8,928.57
Profit b Tax	**USD**	**-11,659.27**	**423.47**	**22,119.92**
Tax	USD	-	-	6,635.98
Net Income / Loss	**USD**	**-11,659.27**	**423.47**	**15,483.94**
Revenue per t feed	USD/ton	11.85	17.47	28.76
Revenue per kg conc.	USD/kg	2.69	3.96	6.52
Costs per t feed	USD/ton	12.69	2.88	12.69
Costs per kg conc.	USD/kg	2.88	- 0.20	2.88
Profit per t feed	USD/ton	- 5.61		7.44
Profit per kg conc.	USD/kg	- 1.27	- 0.05	1.69
Return	**USD**			
Interest rate	%	15.00	15.00	15.00
NPV over 10 years	**USD**	-68,052.55	-13,852.79	65,873.78
Total Invest	USD	62,500.00	62,500.00	62,500.00

9. Conclusions

The review of the processing tests of this study indicates that the efficiency of both artisanal and (basic) mechanical mineral processing is strongly influenced by the inherent mineralogical properties of the processed ore. As such, processing options at a given mine site should be evaluated and optimized on a case-by-case base. For example, both artisanal and mechanical processing worked fine with the well-liberated, medium-sized ore representing the ROM at mine 1 of this study. In contrast, at mine 2, the convergence of target minerals in the fine-grained fraction results in low recovery rates and concentrate grades for both of the performed processing approaches (mechanical and artisanal). In this case, the main factor to improve processing efficiency refers to the properties of the processed material which therefore needs to be assessed in detail to adjust the processing scheme. A suitable assessment of ore properties demands for a proper laboratory which is capable of identifying the elemental and mineralogical composition of the material which is to be processed. Further, to evaluate a suitable mechanical processing scheme, an experimental processing plant with different equipment and options for combination of such is necessary.

The highest recovery rate achieved in the test series was about 73 % (mechanical processing of pegmatite-hosted cassiterite ore, Mine 1). As the recovery rate in practical application cannot be 100 % but is limited by the investment in processing in proportion to the resulting increase in recovery, this recovery rate can be assumed to be a good practice. However, most of the performed tests showed low recovery rates between about 11 % and 25 %. With this, the main share of target minerals remained in the waste material for the artisanal and mechanical processing tests of mixed cassiterite-coltan and wolframite ore from Mine 2 and Mine 3, respectively.

It should be noted that a very basic mechanical processing approach was used for the tests in this study. The recovery rate of mechanical processing could likely be increased beyond what was achieved here through improving calibration of sorting equipment and additionally adapted techniques. This would further impact on the economic evaluations presented as case studies for the present report. For example, application of a jigging machine for pre-concentration of coarse target minerals is likely to be a suitable technique. It could be used before the grain size fractions >2 mm are comminuted and recover coarse material before the possibility of over-grinding by comminution occurs to this fraction. Unfortunately, no jig was available for testing.

Nevertheless, while lacking of options for proper calibration of process water flow and inclination of the shaking tables, mechanical processing still proved to work more efficiently than traditional artisanal processing for smaller grain sizes between 0.63 µm and 500 µm. Further, it was found that ASM operations tend to neglect target mineral concentrations in coarse grained material. It was observed that artisanal miners sort coarse-grained fractions only by hand, in most cases leading to a loss of target minerals. Comminution in order to liberate target minerals intergrown with gangue material in coarse particles was only applied in one ASM operation. From all sampled ASM mines, the latter operation also had the most efficient artisanal processing scheme in place in terms of recovery and concentrate grade.

In Table 28 a selection of characteristic features of artisanal and mechanical processing are listed. The main benefit of artisanal processing is its high flexibility. Since operations have very little investment costs and personnel costs are the main cost driver, these operations can be extended easily when demand (and mineral price) increases and can also be cut back or even put on hold when demand decreases.

As mechanical processing is in need of financial investment, a specific operating time and a particular degree of capacity utilization is necessary to ensure cost effectiveness. However, the processing tests showed that mechanical processing was more efficient in most of the cases.

Table 28: Comparison of benefits of artisanal and mechanical processing techniques

Characteristics of	
artisanal processing	mechanical processing
• Highly flexible (e.g. in response to metal price fluctuations) • Low investment costs • Personnel costs are the main cost driver • Straightforward technique • Poor monitoring of performance • Low to no assessment of target mineral distribution	• Higher throughput achievable • Consistent • Adjustable • Higher efficiency in small grain sizes than artisanal processing • Need for monitoring of performance to asses cost-efficiency

Further benefits of mechanical processing are the consistency and the calibration of processes. Water supply, amplitude and frequency of each shaking table can be adjusted in order to optimally fit to the material that is processed.[17] When operated correctly shaking tables usually outperform manual processing methods and therefore increase recovery and/or grade of the concentrate. This allows for a higher recovery of the deposits target minerals and eventually to a higher profitability of the operation.

The use of magnetic separation as final processing step allows for production of two individual concentrates (cassiterite and coltan) instead of one mixed Sn-Ta concentrate. This again leads to higher concentrate grades as well as higher achievable prices for the mining company or cooperative when selling concentrates locally. It should be noted, though, that concentrate upgrading to export grade does take place in Rwanda in any case. The main question is where this upgrading takes place. In the current practice of the mineral value chain in Rwanda, ASM operations are producing pre-concentrates which are upgraded to export grade and blended by the local buyer, usually comptoirs or exporters based out of Kigali. As such, improving concentrate grades already at the mine level, rather than at the export stage, would represent a shift in national value addition along the supply chain.

In terms of reducing losses of target minerals by artisanal processing, For ASM operators it might be reasonable to sell higher amounts of low-grade pre-concentrate with a high recovery rate.[18] Further

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[17] Mechanical equipment rented for this study comprised of aged equipment. New equipment presently available for purchasing should have more options for making detailed calibration.

[18] e.g. chapter 5.1 artisanal processing of mine 1 showing a recovery rate of 84 % after the first panning stage with a heavy fraction consisting of 11 % SnO2

processing could then be performed centralized in a monitored and optimized processing plant.

Another option for reducing losses of target minerals has been found to be reprocessing of artisanal waste material. The reprocessing of artisanal tailings for mine 1 has been found to be highly efficient. Even though recovery rate and concentrate grade of this approach showed to be below average in the test series, the target minerals would not have been recovered otherwise. Therefore an increase in concentrate mass of 20 % has been achieved by reprocessing waste material in the corresponding test.

A negative factor observed during this study was the lack of monitoring processing performance of ASM operations and individual processing steps. In contrast, a mechanical processing solution is in need of performance monitoring in order to proof cost-efficiency and return of investment. As such, performance monitoring should improve automatically as a result of investment in mechanization. To increase performance in general the availability of relevant data is crucial. As for now, due to the low investment costs of ASM operations, little knowledge of the exploited deposit and properties of the produced ores is available, hindering the process of planning improvement in mining and processing performance.

Table 29, shows all processing test results. Mine 1 (cassiterite) performed well and cost efficient with both, artisanal and mechanical processing measures. In terms of recovery and therefore profit, mechanical processing outperformed the artisanal approach for mine 1.

Table 29: Summarized results of the performed processing tests

Feed #	Processing	Ore Type	Feed Grade	Concentrate Grade	Mass Recovery of target mineral	Profit/ loss [USD/a]
Mine 1	Artisanal	Cassiterite	2.17% SnO_2	68.51% SnO_2	60.10% SnO_2	113,000
	Mechanical			71.19% SnO_2	72.84% SnO_2	127,000
Mine 2	Artisanal	Mixed Sn-Ta	0.46% SnO_2 0.20% Ta_2O_5	41.26% SnO_2 9.90% Ta_2O_5	24.50% SnO_2 13.40% Ta_2O_5	65,000
	Mechanical			36.32% SnO_2 10.20% Ta_2O_5	16.95% SnO_2 10.97% Ta_2O_5	46,000
Mine 3	Artisanal	Wolframite	0.50% WO_3	21.41% WO_3	22.32% WO_3	-12,000
	Mechanical			34.07% WO_3	26.02% WO_3	-400

Profits and losses in Table 29 refer to calculations at the current (2015) mineral pricing scenario.

Mine 2 (mixed cassiterite-coltan) represented the only case where model calculations indicate a more profitable scenario for application of traditional artisanal, rather than mechanical processing techniques. As mentioned before, is most likely that the results found in the corresponding mechanical processing tests can further be improved sharply by using properly adjustable mechanical separation equipment.

The operation of mine 3 (wolframite) showed to not be profitable with the collected data under the current pricing scenario in any of the tested cases. These negative findings could be caused by misleading information collected during field work.

The profitability analysis showed, that the mechanical process is more profitable than the artisanal way, for long term operation (>10 years of operation) of mine 3. It furthermore has to be stated that in the current artisanal operation only a fraction of the assumed feed rate is processed meaning losses would be much lower than indicated. Additionally, the service of casual workers and the selective mining processes are providing a high flexibility level in terms of reacting to changing ore grades or market prices. Those factors were not taken into account in the profitability analysis and could result to profit in the actual operation.

For all mines the profitability analysis showed an increase in income over a long time period (> 10 years) of operations with mechanical processing. Additionally, mechanical processing potentially decreases personnel costs as far as low-skilled labor is concerned (although replacing these jobs with fewer, higher-skilled but better-paid positions might partly offset this effect). Personnel costs have been found to be the main drivers of operational expenditures for artisanal operations. Further, ore throughputs of 250-500 kg/h were recorded when teams of 5-7 persons treated a processing feed using artisanal methods. In contrast, one single shaking table is able to handle at least 500-1000 Kg/h (depending on the grain size of the feed) while it only needs 1-2 persons to operate it. Therefore, mechanical processing is most likely to be applied beneficially for total operational profit, especially when upscaling the mining activities. On the other hand, mechanization is not necessarily an economic investment for small, isolated mines that fail to generate a regular ore throughput as appropriate for the processing equipment to be employed.

10. Recommendations

Tailings and Waste Management to Secure Future Assets

In demonstrating the efficiency of artisanal processing techniques at different mines in Rwanda, the study indicates that significant amounts of 3T minerals of economic interest can be inferred to be left in tailings and waste of artisanal processing operations. Whether or not these target minerals are currently recoverable by additional processing, the waste materials and tailings should be considered as a potential asset. For reducing the risk for lost profits due to inefficient processing measure, tailings of artisanal processing should be stored properly. A proper waste management concept would allow reprocessing of prospective materials at a later stage, while decreasing environmental impacts such as erosion, siltation of surface waters or heavy metal contamination of some water bodies.

Reprocessing of Artisanal Waste and Tailings

In Rwanda's long history of 3T mining, there have been a number of attempts to realize the value of old mine tailings through reprocessing. In some instances, such efforts have been economically successful and have even been applied at an industrial scale. In other cases, sophisticated processing plants were installed without economic and operational success. This study corroborates these experiences in demonstrating the variability of run of mine ore and its implications for mineral processing. It is therefore highly recommended to test the tailings of ASM operations for residual contents of cassiterite, coltan and wolframite. As demonstrated in the examples of the reprocessing of artisanal tailings of mine 1, the reprocessing can significantly increase the total recovery of the processed ROM. Before applying these processes, however, a scoping study should confirm that the tailings material at a given site has an acceptable grade, the valuable content occurs in a recoverable grain-size range, and the overall ore quantity scales with the required feed for the processing equipment. Mechanical processing is favorable for tailings reprocessing as it generally allows for a higher throughput and additionally treats the material with sorting properties that have not been applied before.

Infrastructure for Assessment of Ore Properties and Optimized Processing

Before exploiting a deposit with advanced mining and processing techniques it is necessary to obtain detailed information on the mineralogical properties of the ore and the size and geometry of the ore body. During preparation of the field work for this study, the team was challenged to find available and appropriately equipped processing plants and analytical equipment suitable for controlling concentrate grades. Before investing in a processing plant, different mechanical processing schemes should be tested to develop an optimized processing concept with regards to the site-specific properties of the material to be processed. As long as there is no testing facility with suitable equipment as well as know-how (qualified staff), only medium scale operations with high investment potential and highly skilled employees will be able to accurately assess the technical and economic feasibility of mechanical processing.

Comminution of Large Ore Particles to Improve Recovery

When performing artisanal processing, coarse feed fractions should be comminuted in any case. Even if the visual inspection with the naked eye seems to indicate a relatively low potential, it is likely that target minerals of economic interest are lodged in intergrown coarse particles. It is difficult or impossible to distinguish intergrown target minerals without crushing of coarse-grained particles, unless a hand lens or a microscope are available for experienced users. Comminution, even if done manually, will reduce the risks of losing unliberated target minerals in those fractions. As shown in test series for mine 3, large quantities of target minerals have been lost in the coarse grain-size fractions. The corresponding ASM operation did not prove to be financially viable. This finding was caused partly by the fact that most of the target minerals were lost to the coarse grain-size fractions rejected during artisanal processing.

Support for Assessment of Potential Upscaling

ASM operations are known to be underfunded. Further, skills and knowledge about operational investments, monitoring of process efficiency and quality management tend to be low. When stipulating improvement of processing practices and efficiency of production in ASM operations this instance needs to be considered. Taking into account the share of profit along the value chain[19], all participating actors tend to profit analog from decreased target mineral losses or increased production capacity. Individual support to ASM operators in evaluating the potential of implementing mechanical processing in their operations should therefore be considered to serve the self-interest for involved companies along the value chain. Also, mechanical processing can be assumed to be the right approach when planning on upscaling operations. Whether such equipment is then added to the artisanal mine site or provided in centralized processing facilities needs to be tested more specifically in respect to the involved mining operations..

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[19] Cook, R. & Mitchell, P. (2014): Evaluation of Mining Revenue Streams and Due Diligence Implementation Costs along Mineral Supply Chains in Rwanda. Contract Study for BGR & RNRA; ISBN 978-3-943566-18-5; available online at:http://www.bgr.bund.de/EN/Themen/Min_rohstoffe/CTC/Downloads/rpt_mining_revenues_rwanda_en.pdf?__blob=publicationFile&v=4

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Figures

Tables

Annex

Annex

All analytical data can be received upon request.

1. Sample Characterization Mine 1:

The sample of Mine 1 was taken from a heavily weathered pegmatitic deposit. In Figure 1 the grain size distribution of the feed sample is shown. On the x-axis the grain size is indicated in logarithmic scale. On the y-axis the cumulative percentage of the total mass that is passing the screen at each grain size is shown. This will be consistent for all following grain size distributions as well. The d_{50} is at about 2 mm. Almost 12 % of the mass are below 63 µm and about 3.5 % are coarser than 20 mm.

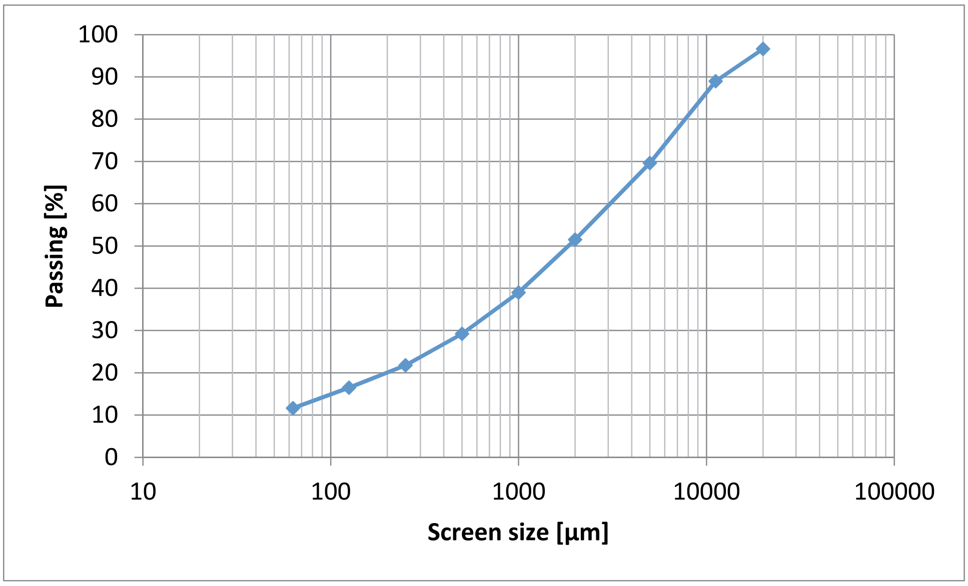

Figure 1: Grain size distribution Mine 1

In Figure 2 a microscopic picture of the grain size fraction 63-125 µm of the feed is shown. The picture was taken with transmitted light to give an impression of the amount of mica in the feed. All clear parts of the picture are mica or quartz. They can be distinguished from each other by their crystal structure. Quartz can be identified by its conchoidal fracture with cubic shapes (red circle in Figure 2), while mica shows a flaky shape with even fractures. The black parts are other minerals, which cannot be distinguished from each other in transmitted light. Next to the clear mica and quartz particles, cassiterite and some iron minerals like hematite and magnetite are present in the feed.

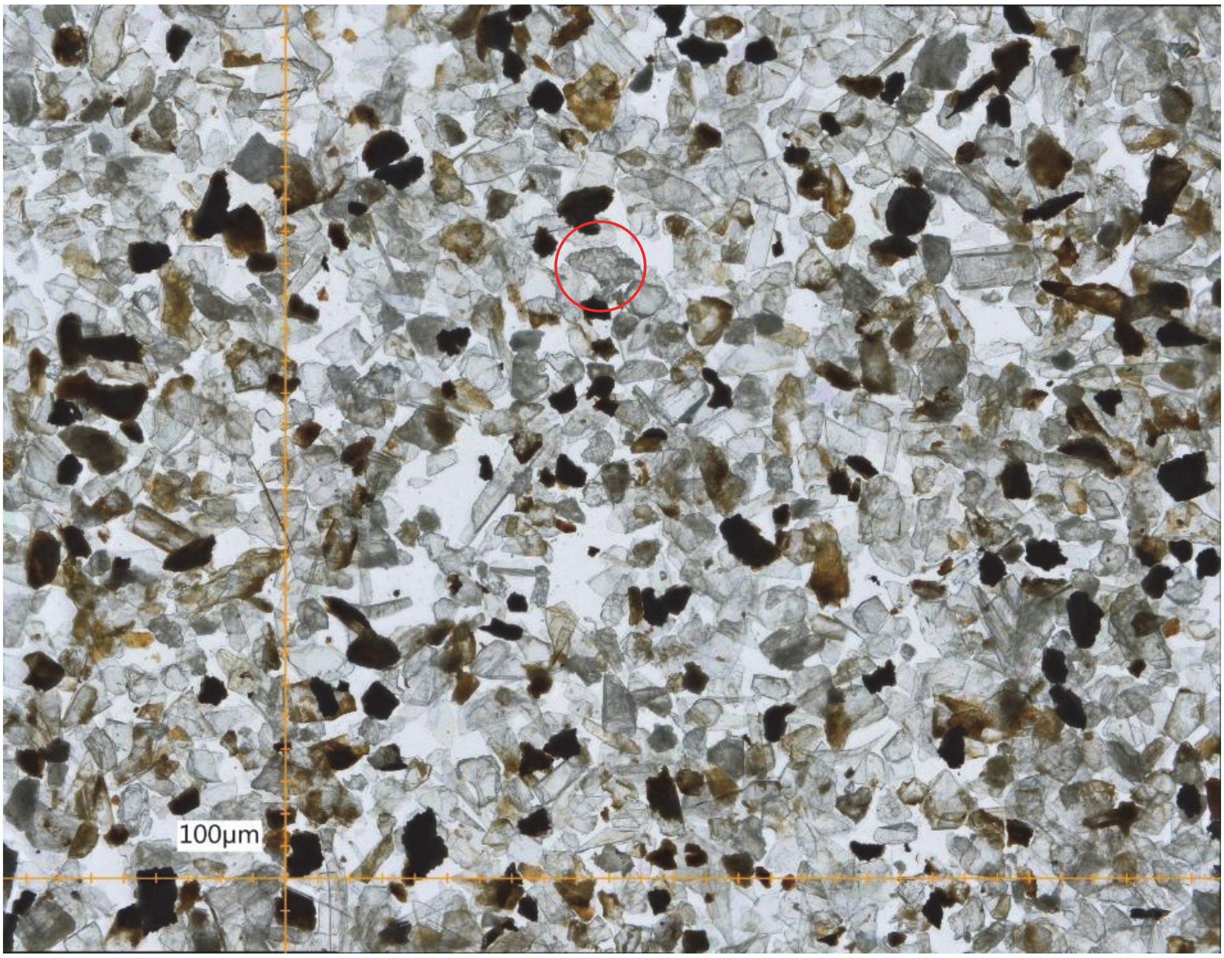

Figure 2: Transmitted light microscopic picture, Fraction 63-125 µm Mine 1

The cassiterite (SnO_2) grade of Mine 1 was determined to be 2.17 % by chemical analysis. Further, elements in the feed are about 4.5 % iron (Fe) which is present mainly in magnetite (Fe_3O_4) and hematite (Fe_2O_3) complexes. About 15 % aluminum (Al) is present in the feed, which in combination with about 6 % potassium (K) most likely is part of the mica. Mica was also found to be the dominant mineral in the feed. About 20 % silicon (Si) are split between mica and quartz as well as some minor amounts of feldspar. The contained SnO_2 is distributed over the grain size fractions as shown in Figure 3. The blue columns indicate the percentage of cassiterite in each fraction with regard to the total contained cassiterite and are related to the primary y-axis. The red dots show the cassiterite grade in each fraction and are related to the secondary y-axis. This scheme will be consistent for the following value material distributions.

About 70 % of the SnO_2 is contained in the grain size fractions between 1 mm and 11.2 mm. Another 14 % is contained above 11.2 mm and about 8 % SnO_2 are in the fraction between 0.5 and 1 mm. This leaves about 8 % Sn in the fractions below 0.5 mm.

Evaluation of the Sn-grade in each fraction shows that the fraction <63 µm has the lowest grade of 0.2 %. The SnO_2-grade basically increases with increasing grain sizes between 63 µm and 11.2 mm where a grade of about 38 % was measured. In the fraction between 11.2 and 20 mm it drops back to 2.1 %. The maximum SnO_2-grade was analyzed in the fraction above 20 mm, showing 4.2 % SnO_2.

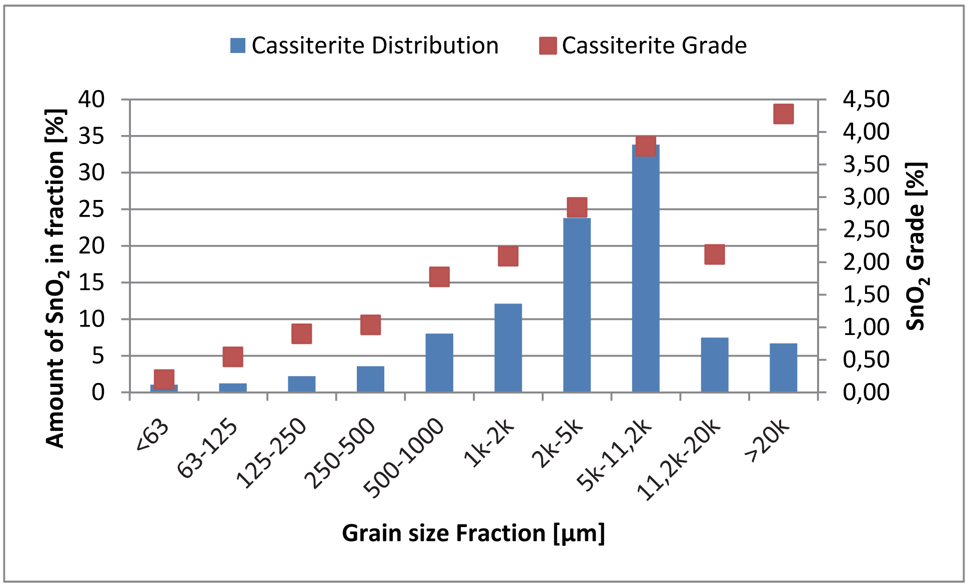

Figure 3: Cassiterite distribution over grain size fractions and grade of each grain size fraction of Mine 1

A visual analysis shows that in the fractions above 5 mm some interlocking of the cassiterite with barren rock exists. Anyways, major parts of the particles are liberated throughout all grain size fractions, setting up good conditions for density separation. The fact that most value material is contained in rather coarse grain sizes also is a good base for density separation. Particles smaller than 125 µm are especially difficult to recover by means of density sorting.

2. Sample Characterization Mine 2:

The sample of Mine 2 was taken from a pegmatitic deposit with overlaying shale. The pegmatites in the deposit are intensively weathered. All feldspar has turned into kaolin. Only quartz has remained intact, due to its hardness. The grain size distribution of the sample in Figure 4 shows that the d_{50} is at about 250 µm. Almost 30 % of the mass are below 63 µm and only about 2.5 % are coarser than 20 mm. Compared to the samples Mine 1 and Feed 2, the grain size is rather low.

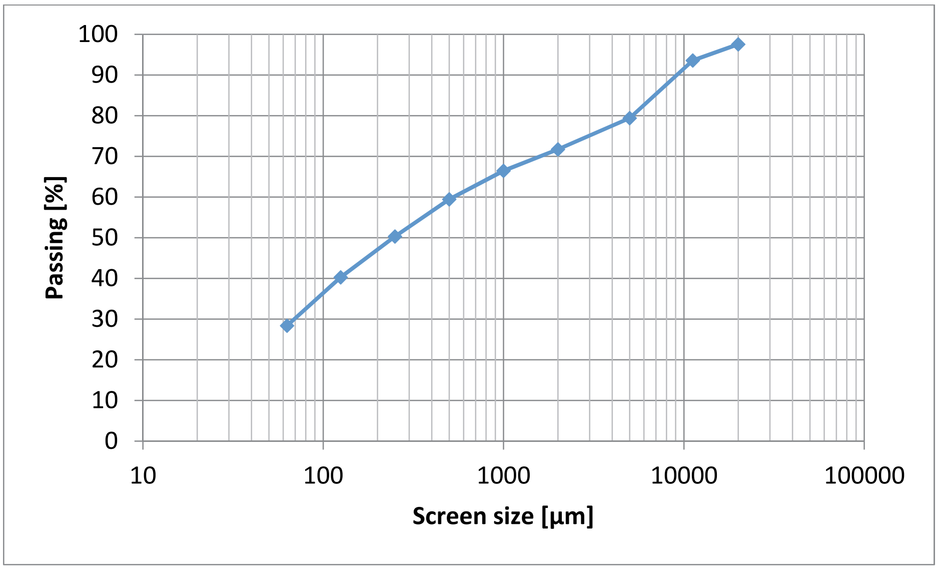

Figure 4: Grain size distribution Mine 2

The transmitted light microscopic picture in Figure 5 shows the grain size fraction between 125 µm and 250 µm of Mine 2. It is visible that, almost all particles are transparent. The transparent particles are mostly quartz and kaolinite. Further, minerals in the sample are several iron minerals. Magnetite and hematite complexes were identified as well as limonite and goethite. The target minerals are cassiterite and coltan.

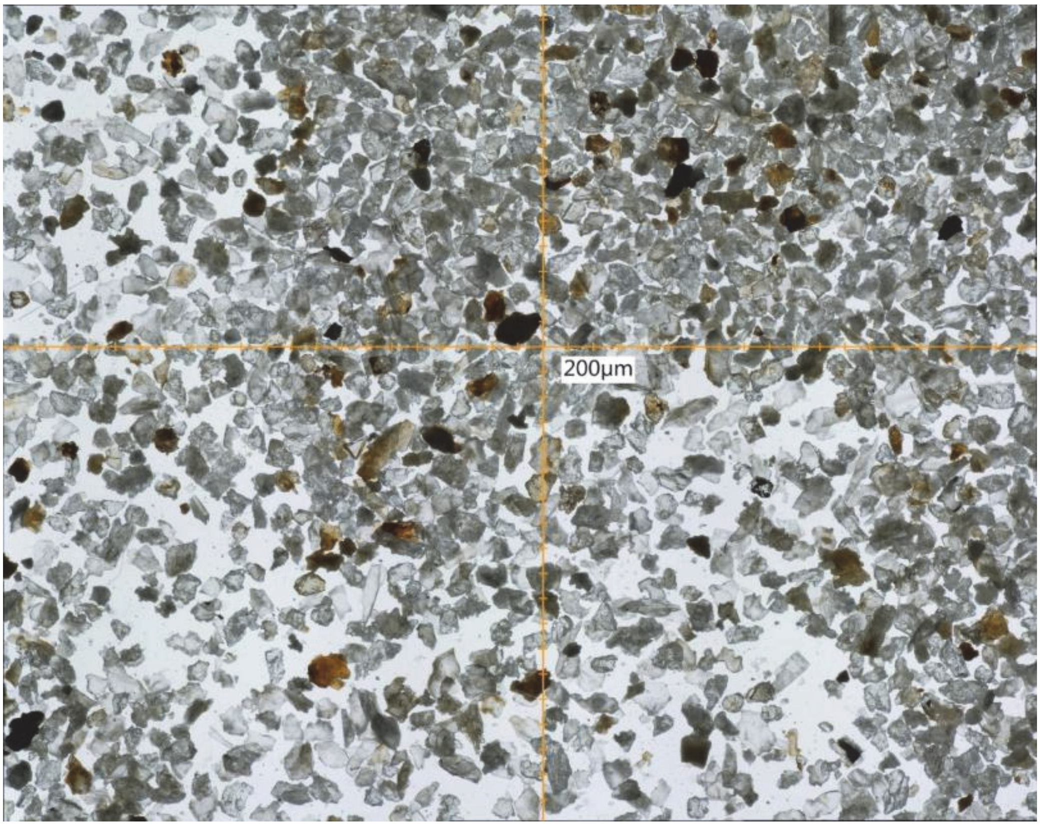

Figure 5: Transmitted light microscopic picture, Fraction 125-250 µm Mine 2

The tantalite distribution over the grain size fractions of the feed in Figure 6 reveals that over 60 %

of all contained tantalite is in the finest fraction below 63 µm. Further 10 % are contained in the fraction 63-125 µm. This is crucial information as it will be almost impossible to selectively recover those particles by simple means of density sorting. Further 20 % are contained in the fractions between 125 µm and 2000 µm. Only about 5 % of the tantalite is coarser than 2 mm.

The tantalite grade in each fraction looks similar to the distribution: Below 63 µm the grade is about 0.44 %. The fractions between 63 µm and 2000 µm have grades between 0.09 % and 0.23 %- Above 2 mm the grades vary between 0.01 % and 0.05 %.

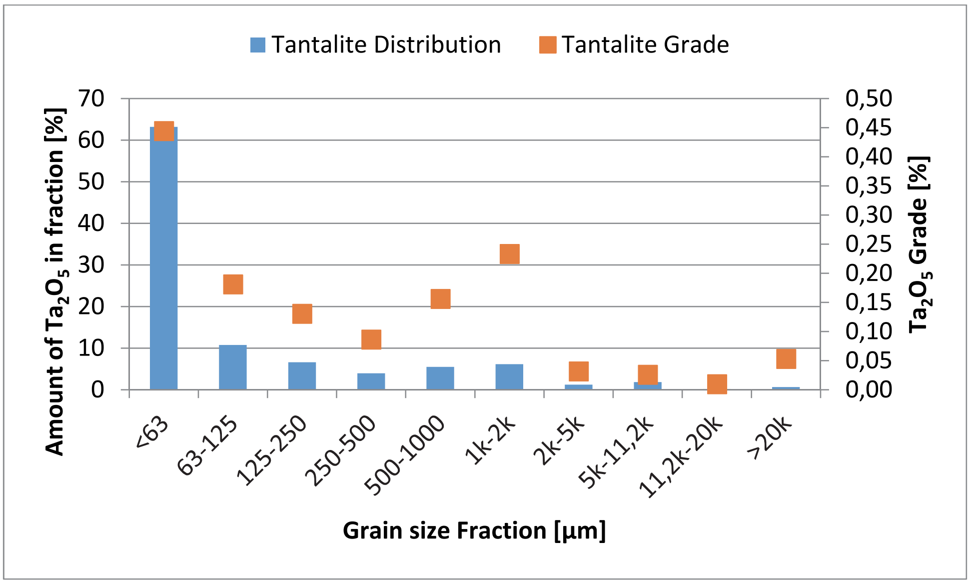

Figure 6: Tantalite distribution over grain size fractions and grade of each grain size fraction of Mine 2

In Figure 7 the cassiterite distribution over the grain size fractions and the cassiterite grades in each fraction are shown. In the fraction below 63 µm 60 % of the total cassiterite is contained. In the fractions between 63 µm and 1000 µm 4.5 % and 6.7 % are contained in each fraction. Further 13 % are contained in the fraction between 1 mm and 2 mm. The remaining 4 % are above 2 mm.

With above 0.9 % the fraction between 1 mm and 2 mm has the highest cassiterite grade of all fractions followed by the fraction below 62 µm with about 0.75 %. The grades of the fractions between 63 mm and 1000 µm vary from 0.14 % to 0.33 %. Above 2 mm all fractions have cassiterite grades below 0.1 %.

The visual evaluation reveals that in the fractions above 500 µm several cassiterite and coltan particles are still interlocked in quartz matrixes. In order to further concentrate those particles, comminution will be necessary. Below 500 µm most particles seem to be liberated, however not many value particles could be found in the feed.

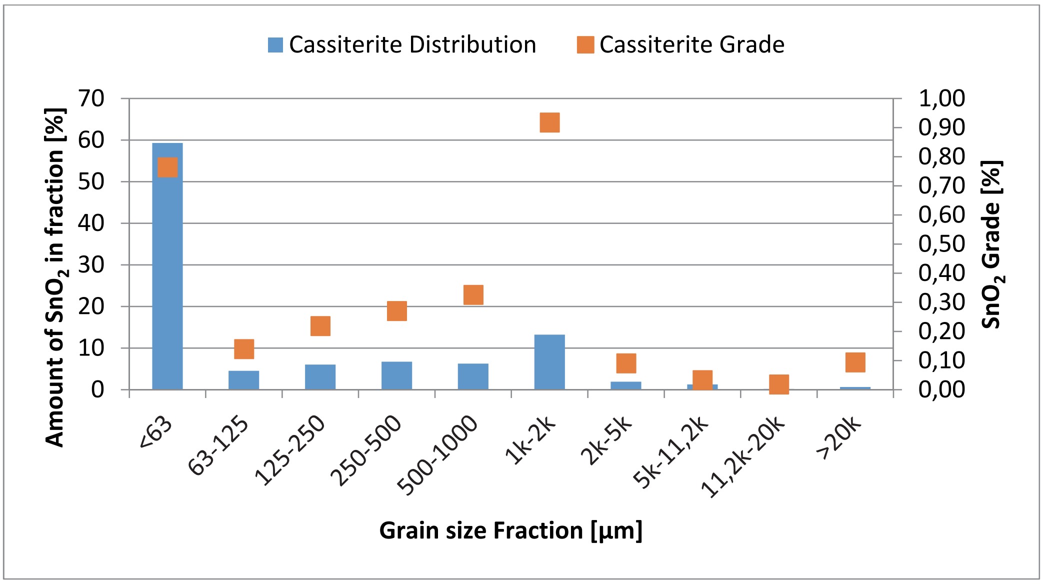

Figure 7: Cassiterite distribution over grain size fractions and grade of each grain size fraction of Mine 2

3. Sample Characterization Mine 3:

Mine 3 was collected from various stockpiles. The corresponding deposit consists of quartz veins, out of which medium to coarse grained wolframite is mined. Figure 8 shows the grain size distribution of the feed. It is noticeable that about 25 % of the mass are below 63 µm and about 15 % are above 20 mm. The d_{50} is at about 3 mm which shows that Mine 3 is generally coarser than the other three feed samples.

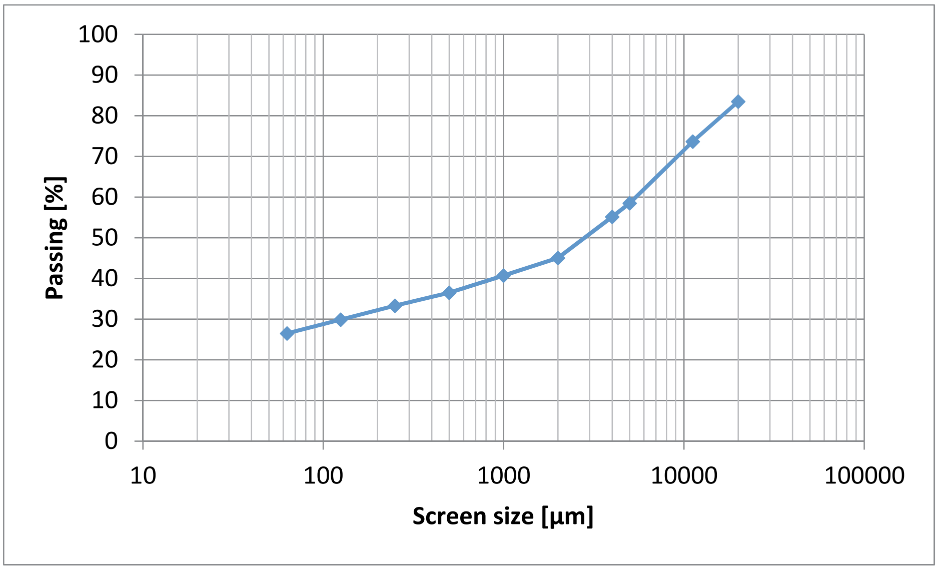

Figure 8: Grain size distribution Mine 3

Figure 9 shows the reflected light microscopic picture of the grain size fraction between 500 µm and 1000 µm of Mine 3. Transparent and dark particles are about evenly distributed. The transparent particles are quartz. The darker particles are different minerals. A lot of graphitic minerals are present which can be identified by their soft structure and their black streak. Ferberite ($FeWO_4$) seems to be the main wolframite mineral contained. It can be distinguished from hübnerite ($MnWO_4$), which is the other main wolframite mineral, by its chemical formula and the color of the streak. Since almost no manganese is present in the feed and the streak color of the tungsten rich particles is brownish it is likely that ferberite is the main tungsten mineral. Besides the mentioned minerals, some limonite covers several particles and goethite particles were identified.

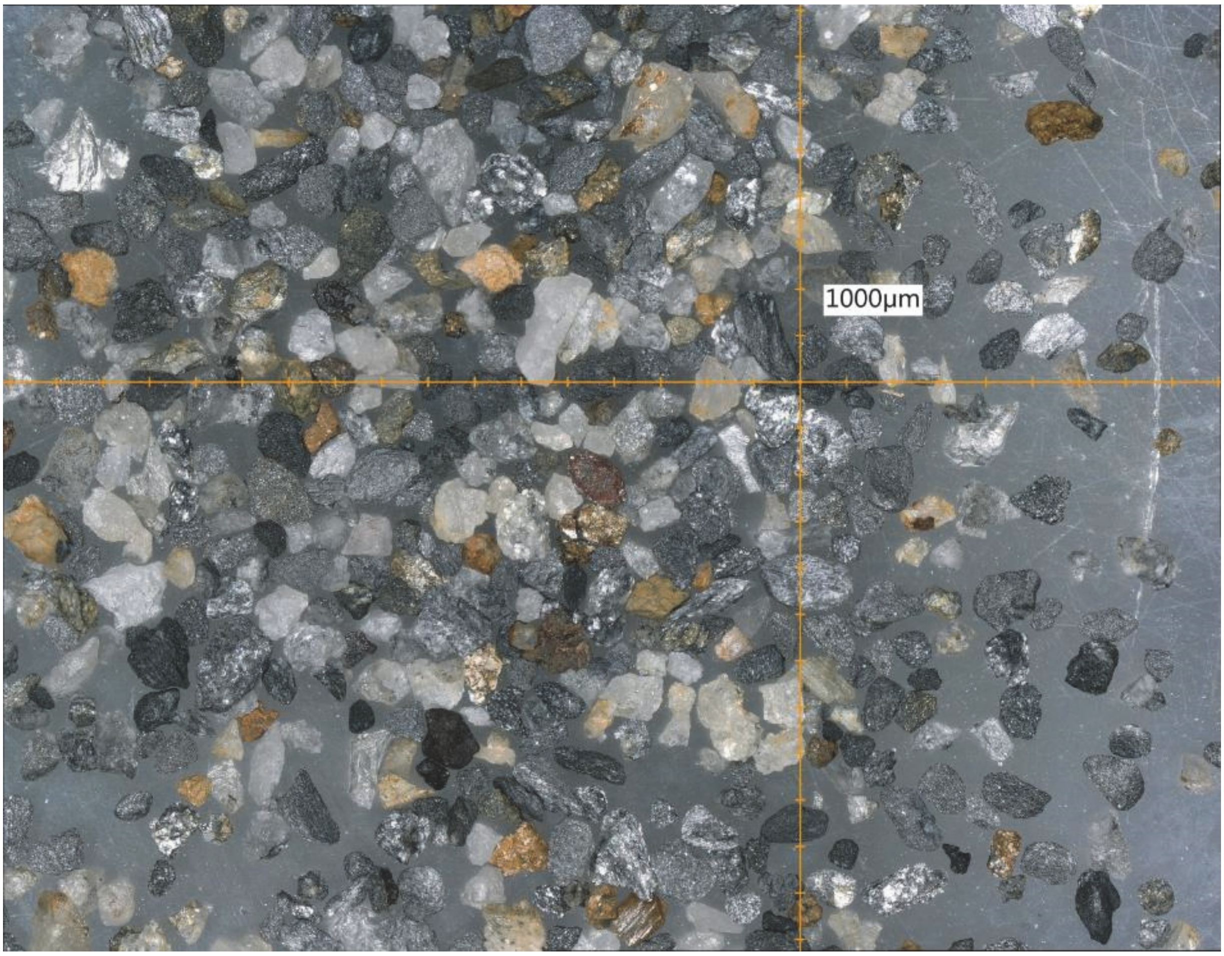

Figure 9: Reflected light microscopic picture of the grain size fraction between 500 µm and 1000 µm of Mine 3

The overall WO_3 grade was determined to be about 5 %. The Fe grade of the feed was about 1.9 % and most probably splits between ferberite, goethite and limonite. Arsenic (As) is present with 0.03 %. The presence of 0.63 % titanium (Ti) suggests that ilmenite is also contained in the feed. Silicon accounts for 26 % of the feed and aluminum and potassium are present with grades of 12 % and 4.8 % respectively.

In Figure 10 the wolframite distribution over all grain size fractions and the wolframite grade in each fraction is shown. About 10 % WO_3 are contained in the fraction below 63 µm. The fractions from 63 µm to 2000 µm combined contain about 15 % of the total WO_3. From 2-5 mm further 15 % are contained. The remaining 60 % are contained in the fractions above 5 mm.

The highest WO_3 grade (9.6 % WO_3) was found in the fraction 5-11.2 mm. Above 20 mm the grade was about 8.5 %. It is noticeable that in the fraction between those two high grade fractions from 11.2-20 mm the lowest WO_3 grade (1 % WO3)of all fractions was measured. The origin of this anomaly could not be identified. In the fractions between 250 µm and 5000 µm the grade varies between 4.5 % and 6.9 %. Between 63 µm and 250 µm grades of about 3 % WO_3 were determined. The WO_3 grade below 63 µm is about 1.8 %.

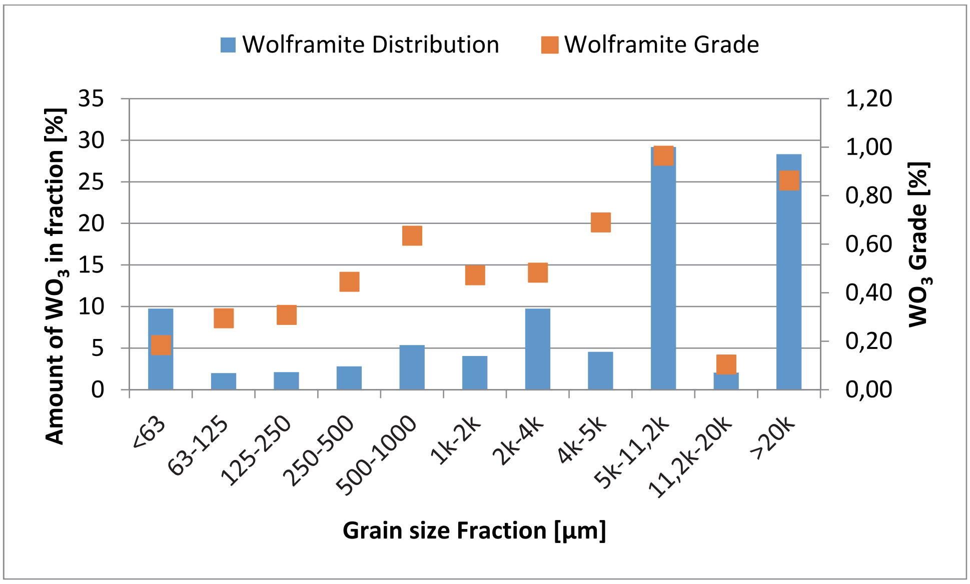

Figure 10: Wolframite distribution over grain size fractions and grade of each grain size fraction of Mine 2

Due to the overall darkness of the particles it is hardly possible to visually determine the extent of liberation of the wolframite particles. Especially in the finer fractions below 500 µm particles seem to be widely liberated. Above 500 µm quartz still seems to be interlocked with wolframite.

4. Processing flow sheets with analytical results of the products

The following flow sheets are showing the processing schemes including the results of chemical analysis. Unfortunately not all products could be recovered, leaving some of the flow sheets incomplete or absent.

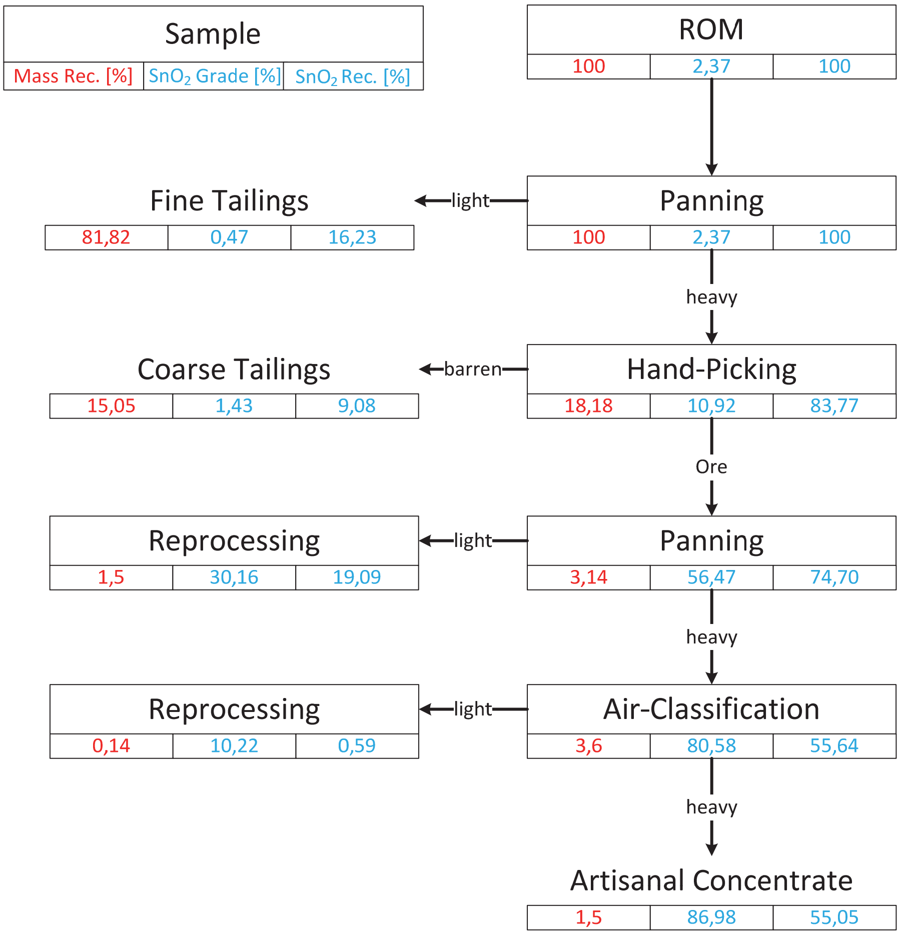

Figure 11: Flow-sheet of artisanal processing scheme of mine 1. The numbers underneath the process names indicate mass recovery (red number on the left) in percent, SnO_2 grade (blue number in the middle) in percent and SnO_2 recovery (blue number on the right) in percent. Reprocessing of Panning stage two indicates mechanical reprocessing.

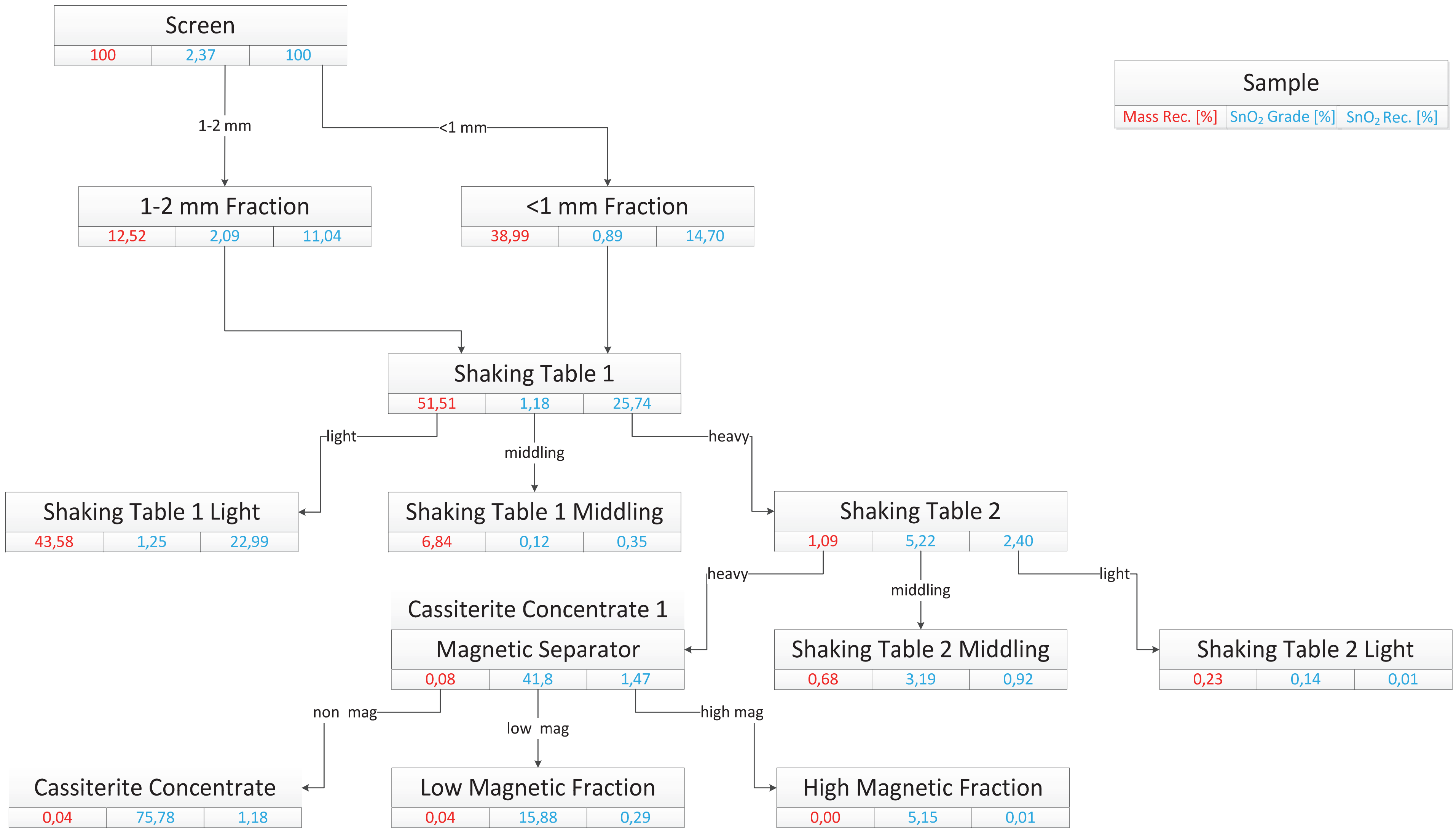

Figure 12: Flow-sheet of mechanical processing tests of feed fraction <2 mm of Mine 1

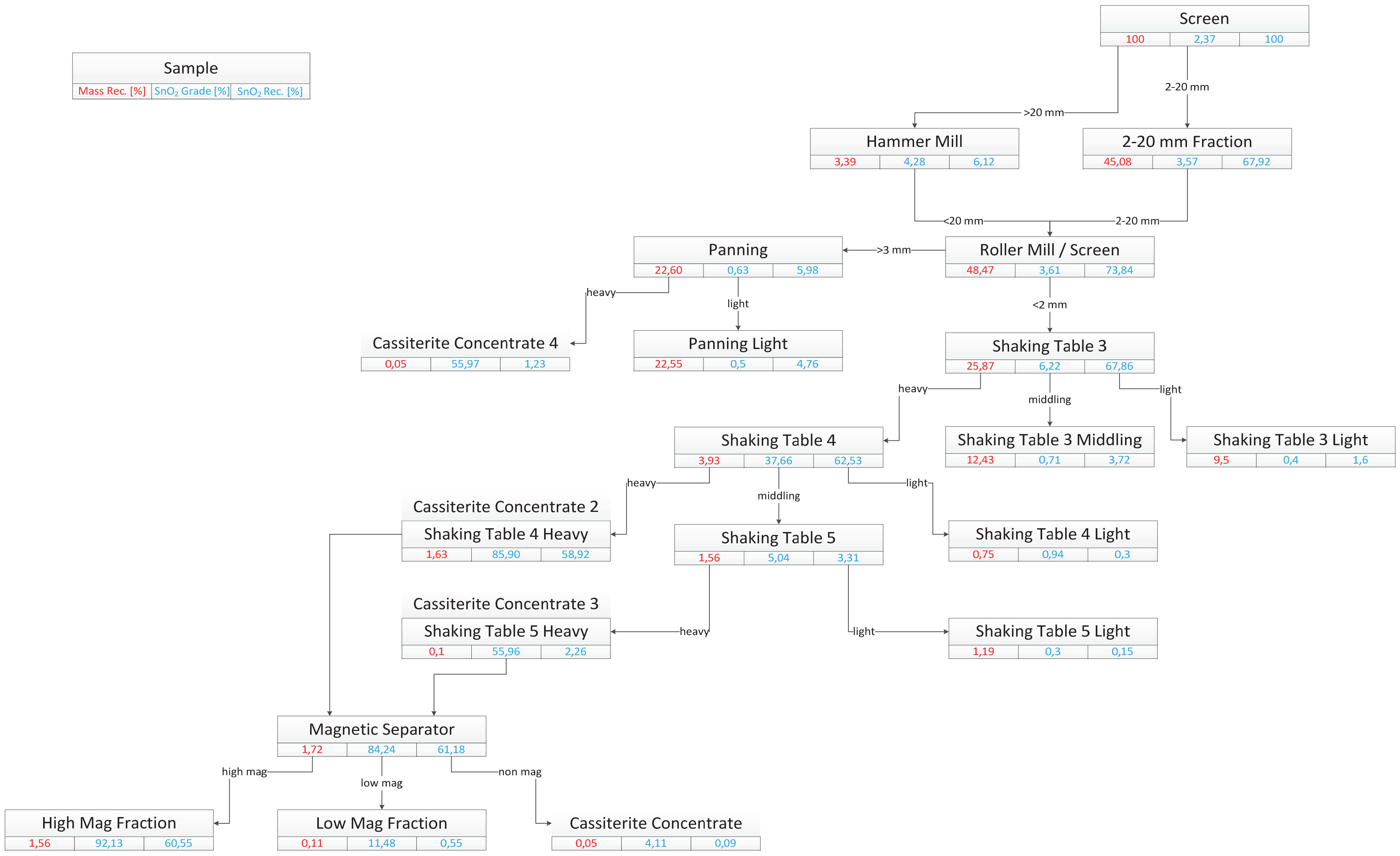

Figure 13: Flow-sheet of mechanical processing tests of feed fraction >2 mm of Mine 1

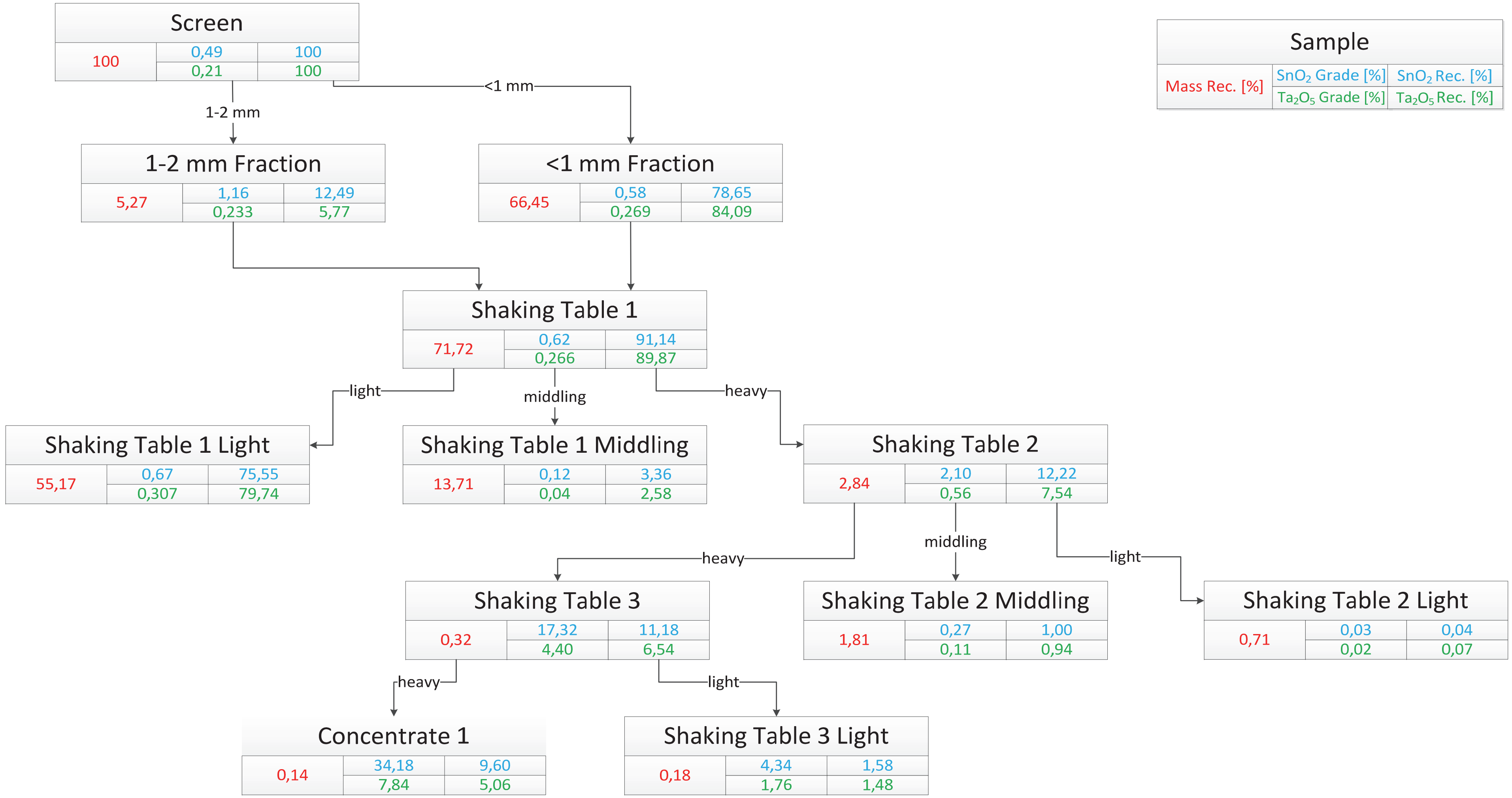

Figure 14: Flow-sheet of mechanical processing tests of feed fraction <2 mm of mine 2

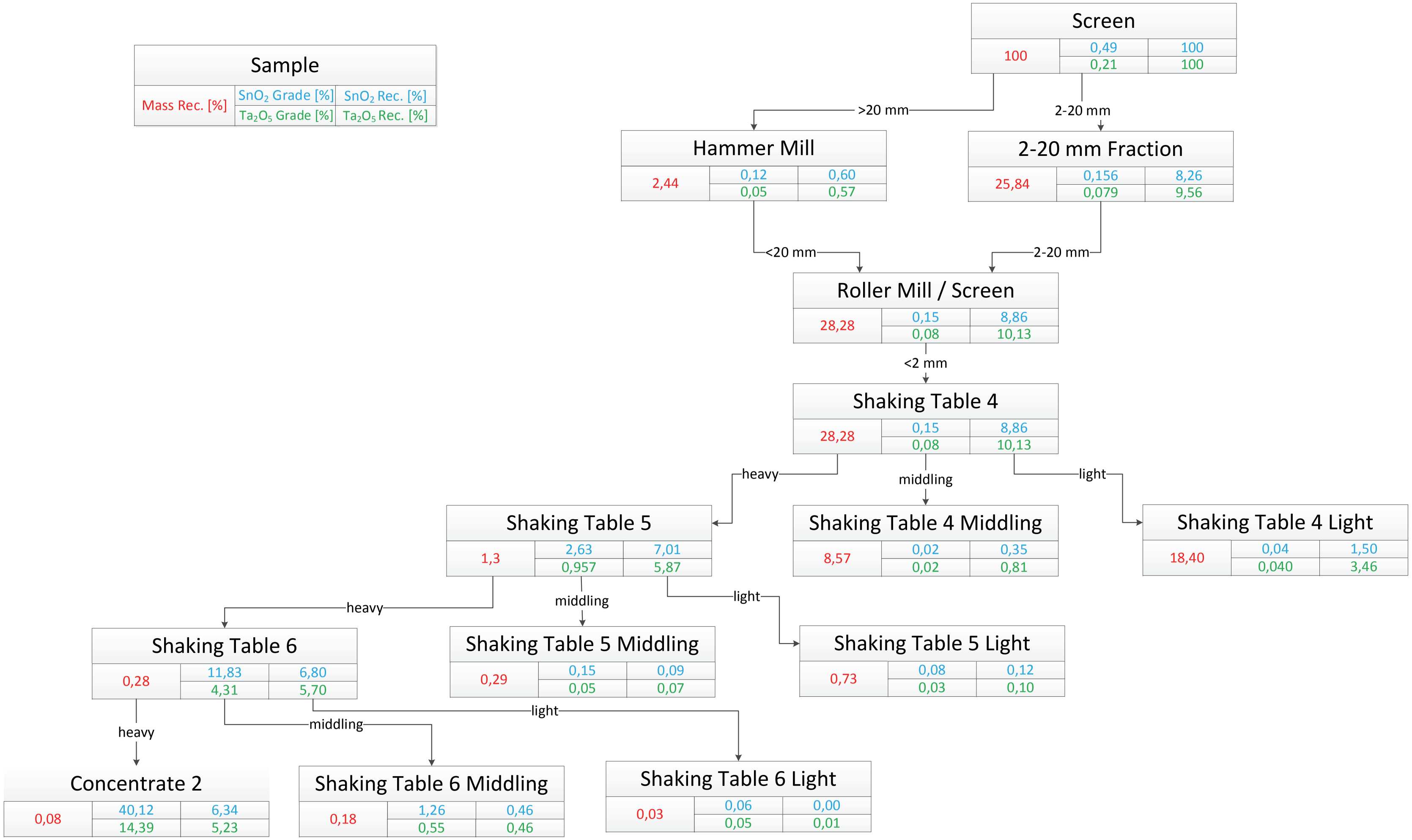

Figure 15: Flow-sheet of mechanical processing tests of feed fraction >2 mm of mine 2

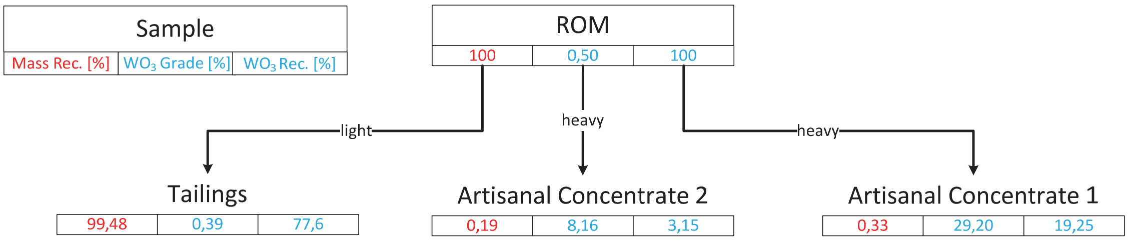

Figure 16: Individual concentrate grades and recoveries of artisanal processing Concentrates 1 and 2 of mine 3

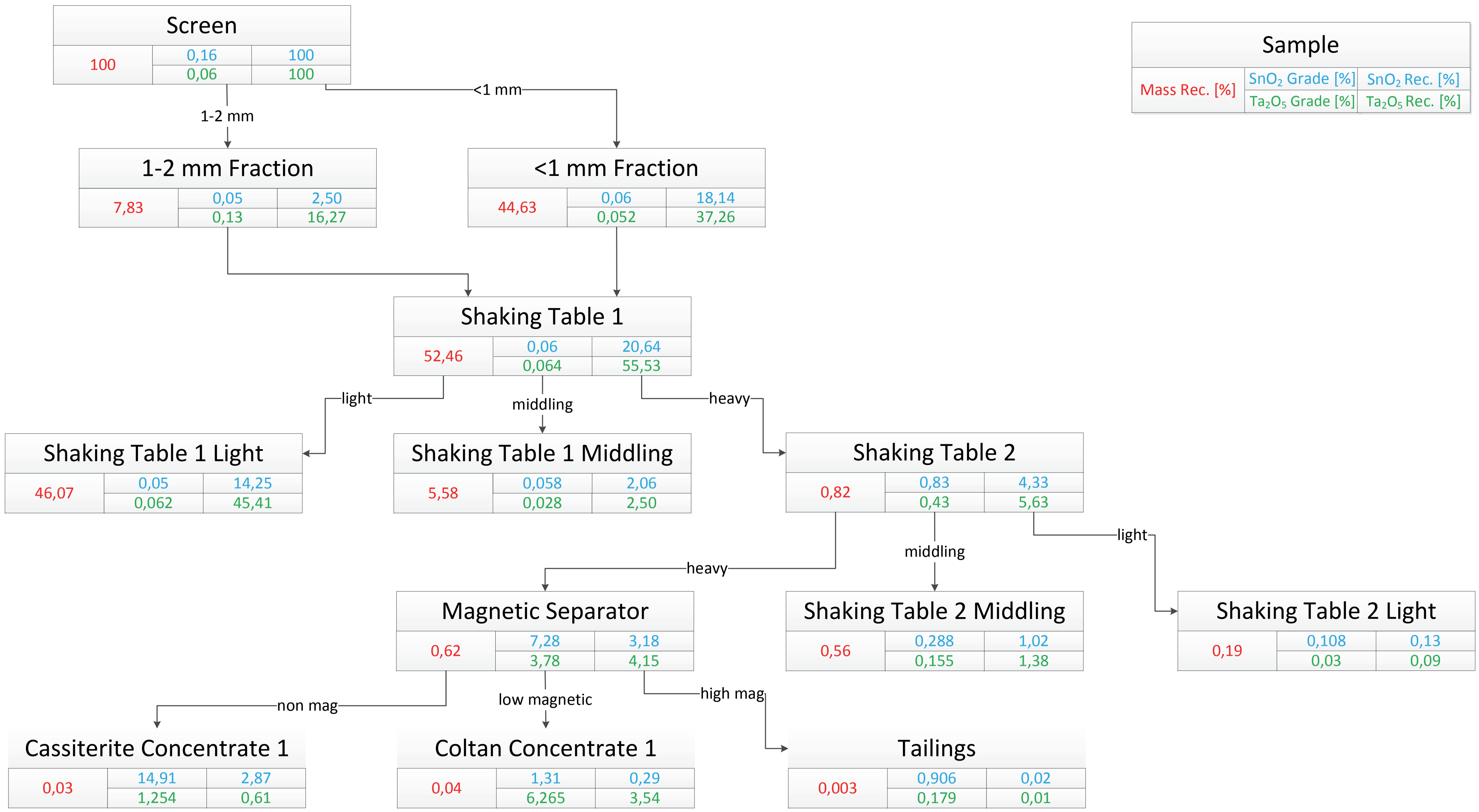

Figure 17: Flow-sheet of mechanical processing tests of feed fraction <2 mm of mine 4 (not evaluated in the study)

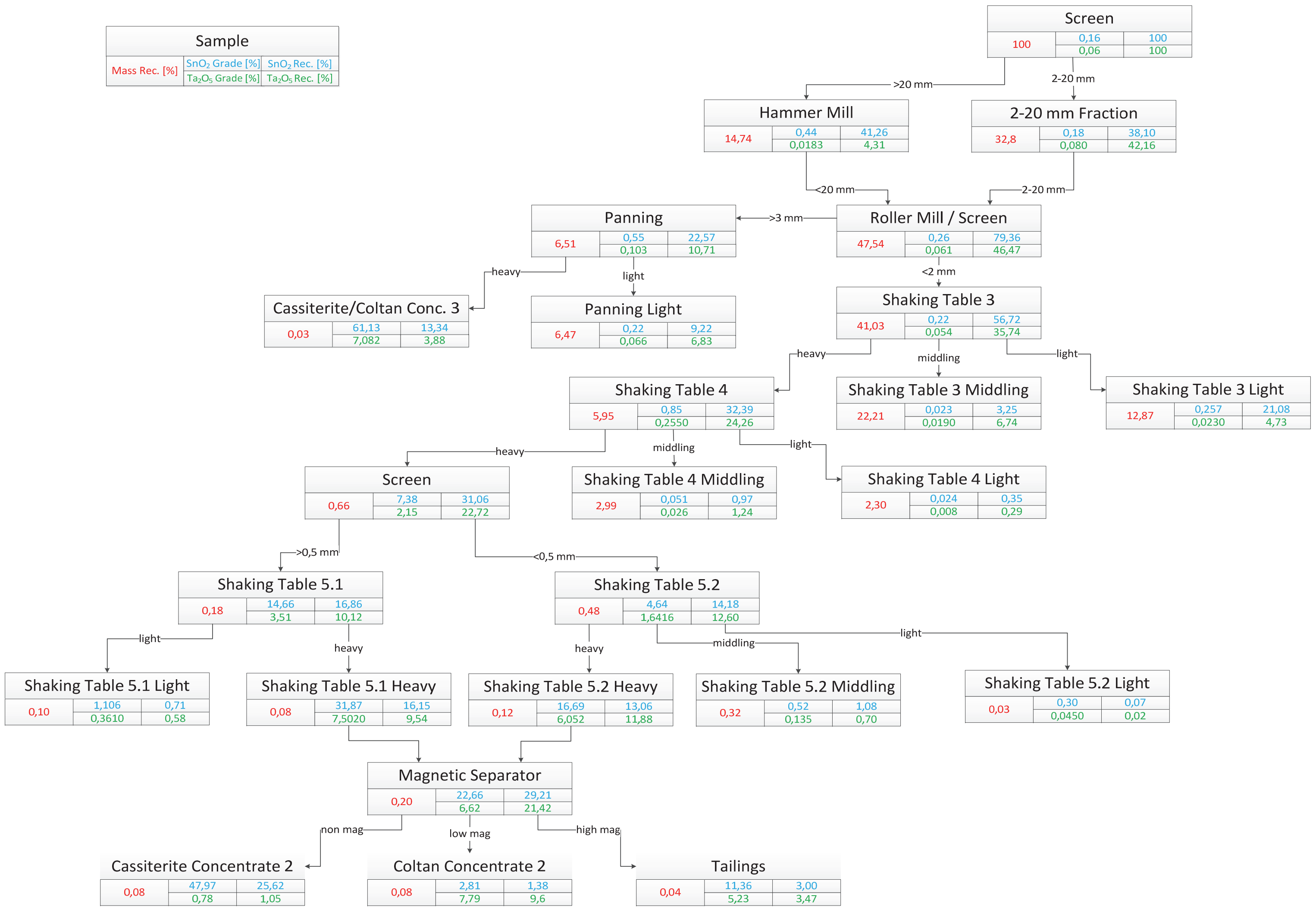

Figure 18: Flow-sheet of mechanical processing tests of feed fraction >2 mm of mine 4 (not evaluated in the study)

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