Can preconcentration of cassiterite from its pegmatite ore reduce processing costs and improve operational sustainability?

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Abstract

Different concentration techniques were evaluated for preconcentration of a mineral ore at a coarser size to avoid energy and resource wastage. Specifically, the aim was to reduce milling costs and energy required in the beneficiation of tin. In this study, cassiterite (0.17% Sn) was the mineral of interest in a pegmatite ore body associated with quartz (SiO_2 > 60%) and alumina (Al_2O_3 > 20%). Three concentration techniques, namely dense media, shaking table, and flotation, coupled with characterization analysis, were used to assess the concentration response. The results confirmed that particle size and mineral liberation impact the separation process. High recovery and grade were obtained with gravity concentration methods (dense media and shaking table) for coarser (+300 to +212 µm) and intermediate (+150 to +53 µm) particle sizes. Lower recovery and grade were identified for much finer sizes (–53 to –38 µm). Flotation produced a high-grade product at a relatively low recovery and appeared to be only applicable to finer grains. Separation efficiency based on Schulz's equation measured a segregation performance of 74.8% for dense medium separation and 60.7% for the shaking table for the coarse and intermediate particle sizes. Flotation only achieved a separation of 30%–40%. The results suggest that use of dense media separation as a rough preconcentration method prior to further grinding, and the utilization of a more advanced concentration technique for mineral recovery and upgrade, constitute a successful approach to improve process economics.

Keywords

cassiterite, concentration, separation, efficiency

Introduction

Tin ore accounts for a small amount as a single mineral form, but a larger share in the form of key mineral or associated components. Tin (Sn) is listed as a critical and important metal. Critical metals are considered both important to society and vulnerable to supply disruption (Whitworth et al., 2022). Tin is mainly used as a plating agent for steel cans in the food industry, piping, and in glass manufacturing.

There are more than 50 types of tin-bearing minerals on earth. Of these, cassiterite (SnO_2) is the only natural mineral from which tin is usually economically extracted (Su et al., 2017). The mineral is identified in placer deposits, granite-related tin deposits, including vein type, and stanniferous pegmatites. The latter has been reported as economically favorable at a Sn grade of approximately 0.14% (Maritz and Uludag, 2019). Extraction and beneficiation of Sn from such deposits involves several stages to achieve production of metal as the final product. The beneficiation of SnO_2 usually uses its density because the mineral has a relatively higher density than that of co-existing minerals within the ore body assemblage. Traditionally, the mineral concentration flowsheets consist of gravity separation units, including jigs, spirals, and tables, for the primary concentration stage. The final stage typically utilizes enhanced gravity concentrators or flotation cells (Angadi et al., 2015). Thus, the process involves multi-stage grinding and multi-stage beneficiation.

Most minerals of interest are finely disseminated and intimately associated with gangue, so the need arises to unlock or liberate the mineral before separation can be undertaken. This is achieved by comminution, during which the particle size of the ore is progressively reduced until the liberated particles of mineral can be adequately separated (Wills and Napier-Munn, 2006). This is an energy-intensive process, which, in the mineral raw material sector, accounts for 50% or more of the energy consumption of a mine site. Cassiterite is characterized by extreme brittleness compared with other minerals usually present (Parapari, 2021). This attribute should be considered during size reduction prior to concentration. In production practice, –38 µm cassiterite exhibits very poor gravity separation and effective treatment of –10 µm cassiterite presents difficulties (Fu et al., 2023). It appears advantageous to recover cassiterite grains, where possible, at the earliest possible stage and at their largest size to avoid softness that is difficult to handle and recover (Ibrahim et al., 2022). Therefore, upgrading of mineral concentration of ores before fine comminution can decrease the capital expenditure and energy consumption of mining operations and

simplify environmental permitting (Leon et al., 2020). In this sense, targeted mineral preconcentration would, to some extent, discard gangue-related minerals at a coarse size to save on the energy required to further process barren waste material. In addition, mineral preconcentration could avoid generation of fines. This study explored the feasibility of preconcentration of SnO_2 from a pegmatite ore body at an early stage of the concentration process to discard/eliminate barren waste minerals, with the purpose of saving costs and improving operational sustainability of the process.

It is hypothesized that it will be possible to remove a large portion of the waste mineral at the front end of the process and that this will lead to substantial cost saving. The investigation employed characterization techniques, mainly bulk chemistry (X-ray fluorescence; XRF), to assess the liberation response of the mineral of interest (SnO_2) and the efficiency of different concentration methods, i.e., gravity separation (dense medium separation (DMS) and shaking table) and flotation.

Material and Methods

Materials and mineral validation

A 20 kg ore sample was obtained from the Uis deposit in the Erongo region of Namibia, which is currently operated by Andrada Mining. Homogenization was performed in accordance with the soil sampling protocol of the U.S. Environmental Protection Agency.

The sample was run-of-mine material that had passed through an initial crushing operation and was collected from the silo feeding the mill for further size reduction at an estimated average grain size of > 300 µm (Figure 1(b)) and relative density of 2.662 g cm^{-3}. The sample was characterized for its bulk chemistry and mineral composition using XRF and X-ray diffraction (XRD) analyses, respectively. Its composition revealed a Sn content of 0.17%, with Si and Al as the major elements, at 30% and 12%, respectively (Figure 1(c) and (d)). Mineral quantification showed the presence of cassiterite (SnO_2), quartz (SiO_2), albite, and magnetite (Fe_3O_4) at proportions of 0.27%, 75%, 24%, and 7.1%, respectively (Figure 1(e)). The tin content varied with screen size (Figure 1(f)). The as-received sample microscopically displayed the presence of free cassiterite grains and relatively coarse silicate mineral (Figure 1(g)), suggesting the possibility of tin-related minerals preconcentration, thus allowing silicate to be discarded at a coarse size. Attributes of the mineral phases are given in Table I.

Methods

A stepwise method was used during this investigation, with the aim of discarding barren material at a coarser size. A wet comminution (liberation) exercise was conducted at various resolved times (5, 10, 15, 20 min), using a laboratory rod mill at a volumetric charge of 40%, comprising rods and sample in a mass ratio of 0.25. The equipment was set at a rotation speed of 60 rpm. The as-received sample and each milled product were screened using sizes of 300, 212, 150, 106, 75, 53, and 38 µm. Three classes of particle sizes (coarse (+300 to +200 µm), intermediate (+150 to +53 µm), and fines (−53 to −38 µm)) were obtained. This group size segregation intended to process particles of relatively similar size and allowed the performance of the concentration process to be adequately assessed.

Differences in mineral properties of the ore assemblage were used to concentrate the SnO_2. Two gravity separation techniques were used: DMS and a Wilfley shaking table. Both methods make use of the relative density (RD) difference between the various

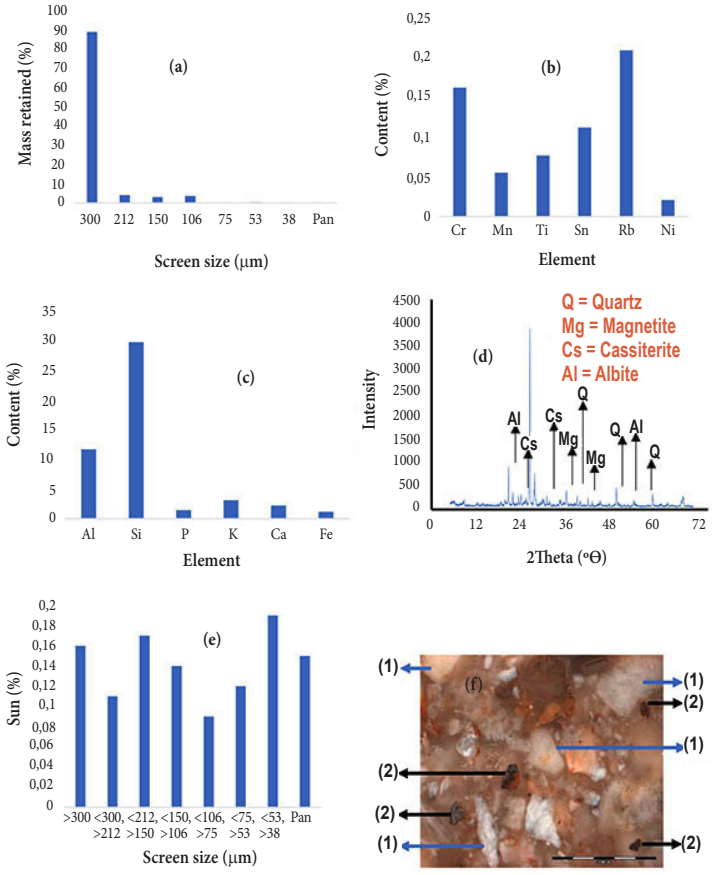

Figure 1—Sn ore initial characterization. (a) Particle size of as-received ore, (b) trace and (c) major elements, (d) mineral content, (e) Sn distribution by screen size, and (f) optical microscopy characterization ((1): silicate, (2): cassiterite)

Table I

Physical attributes and quantification of mineral phases

Phase	Albite	Cassiterite	Magnetite	Quartz	Muscovite	Tantalite	Fluorapatite
RD	2.63	6.98–7.1	5.1–5.18	2.65	2.76–3	8.0	3.1–3.2
Mhos	6–6.5	6–7	5.5–6.5	7	2–2.5	6–6.5	5
(%)	15.6	0.27	6.17	74.8	0.45	0.31	2.4

RD: relative density., Mhos hardness (%): percentage content

minerals in the ore. DMS was conducted using tetrabromoethane ($C_2H_2Br_4$ (2.97 g cm^{-3})) dissolved in acetone to obtain a final cut density of 2.75 g cm^{-3}. The process isolated the sample into two fractioned products, referred to as sinks (heavy, RD > 3 g cm^{-3}) and floats (light, RD < 2.7–2.65 g cm^{-3}), using a 1 L conical separatory funnel at density cut of 2.97 g cm^{-3}. The shaking table was set at an amplitude of 35 Hz with an inclination angle of 6° at a water flowrate of 500 L/h and feed rate of 50 kg/h. Three products were obtained from the shaking table: concentrates (heavy), tails (light), and middlings (mix of tails and residual concentrate).

Flotation was conducted using a laboratory cell (Denver) with oleic acid (collector, 600 g/t), sodium silicate (depressant, 300 g/t), Dewforth 40 (frother,t 20g/t), and H_2SO_4 as the pH regulator (maintained at pH 4.5–4.8). The device was agitated at 1126 rpm. The feed comprised a wide array of particle sizes (Figure 1(b) to highlight the importance of a specific size suitable to positively respond to the concentration process.

The efficiency of separation for all three techniques was evaluated. Bulk elemental chemistry was determined using an XRF powder method (Rigaku-ZSX Primus II, SQX analysis software, operating at 4 kW, 60 kV, and 150 mA). Mineral contents were determined from XRD analysis (Rigaku Ultima IV, PDXL analysis software), using a Cu Kα radiation source at 30 kV and 25 mA. Data were recorded over the range 5° ≤ 2θ ≤ 95°. The powdered samples were scanned at 0.5°/min with a step of 0.01°. Density was measured using a gas-displacement pycnometer (ACCUPY II 1345, Micromeritics).

Results and discussion

Effect of time on comminution and SnO_2 liberation

Figure 2 summarizes the comminution characteristics of the SnO_2 sample at selected retention times. Figure 2(a) shows the particle size distribution; Figure 2(b) summarizes the comminution extent of the ore assemblage at 5, 10, 15, and 20 min. A decrease in content of larger particles (+300 µm) from 88% to 40% after 20 min milling was observed. The content of finer particles (< 38 µm) did not significantly increase (Figure 2(b)). This could suggest that further grinding had little effect on comminution characteristics. However, the presence of fines particle (< 38 µm) had a significant effect on the redistribution of intermediate sizes quantities, including the 150, 106, 75, and 53 µm size fractions. Figure 2(a) indicates that the longest retained time (20 min) had the largest content of intermediate sizes, with the same relative content of fines. This result agreed with that of Yianatos et al. (2005), who reported a decrease in the content of the 212 µm size fraction by 3%, due to the presence of fines. The finer the particles produced in comminution, the greater is the amount of energy required to effect breakage (Stamboliadis, 2013).

Dense media concentrate characterization

Table II summarizes the separation results obtained from gravity concentration using the dense media with tetrabromoethane at a density cut of 2.75 g cm^{-3}. The heavy portion (sink) is characterized with regards to recovery, Sn grade, and density of the various size fractions. In all cases, the results displayed the possibility for preconcentration. Although possible to preconcentrate the as-received sample, poor recovery (68%) and low Sn grade are reported, suggesting that rapid mineral liberation should be required (Table II). The comminution time-of-grind dictated the concentration achieved. A short grind time was characterized by coarse particles and high concentrate grade, accompanied by a low recovery. Grinding conducted for 5, 10, 15, and 20 min gave recoveries of 72%, 74%, 74%, and 73% at Sn grades of 0.63%, 0.79%, 0.81%, and 0.66%, respectively (Table II (+300 to +212 µm)). Improved SnO_2 concentration was identified for the intermediate size (+150 to +53 µm) with recoveries of 78%, 88%, 79%, 83%, and 80% at Sn grades of 1.6%, 2.9%, 5.3%, 6.1%, and 5%, for the as-received sample and resolved grind times, respectively. This increase in Sn grade could be attributed to size reduction that resulted in SnO_2 liberation and redistribution among the particle sizes, ranging from −150 to +53 µm. Low concentrate recoveries are observed for relatively fine grains (−53 to −38 µm), which could be attributed to loss of SnO_2 fines due to the brittle nature of the mineral.

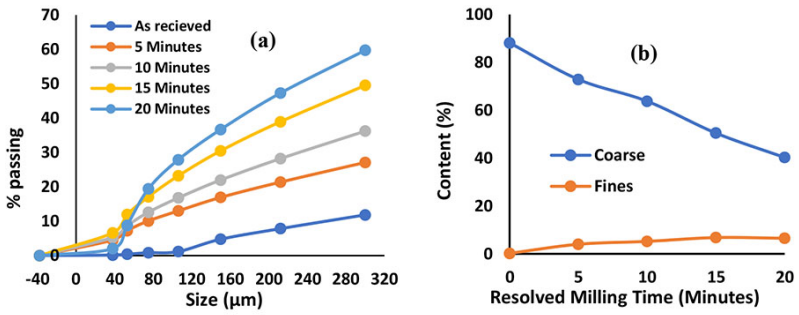

Figure 2—Tin ore comminution. (a) Particle size distribution and (b) coarse and fine particle size evolution as a function of time

Table II

Dense media preconcentration characteristics

Size fractions			Recovery (%)	Grade (%)	RD (g cm^{-3})
Coarse (+300, +212 µm)	Sink	As received	87.71	0.26	2.7486
		5	90.05	0.63	2.7746
		10	87.01	0.79	2.7674
		15	88.13	0.81	2.7578
		20	86.93	0.76	2.7581
Intermediate (+150, +53 µm)	Sink	As received	87.94	0.98	2.874
		5	89.70	1.13	2.8615
		10	81.85	1.95	2.8701
		15	84.4	2.11	2.8694
		20	82.97	1.97	2.8734
Fines (−53, −38 µm)	Sink	As received	44.45	3.01	3.1471
		5	48.73	4.84	3.0978
		10	31.46	6.94	3.1145
		15	37.85	7.01	3.1465
		20	35.73	5.21	3.1934

Table III

Shaking table preconcentration characterization

Size fractions			Recovery (%)	Grade (%)	RD (g cm^{-3})
Coarse (+300, +212 µm)	Sink	As received	63.76	0.28	2.6641
		5	77.94	0.48	2.6931
		10	69.01	0.81	2.6971
		15	71.2	0.83	2.6974
		20	68.17	0.78	2.6971
Intermediate (+150, +53 µm)	Sink	As received	57.79	0.41	2.6982
		5	67.9	0.88	2.7021
		10	72.09	0.94	2.7160
		15	71.91	1.04	2.7113
		20	68.22	0.98	2.7977
Fines (−53, −38 µm)	Sink	As received	53.07	0.92	2.8742
		5	52.58	1.04	2.9024
		10	39.65	1.49	2.9017
		15	42.61	1.53	2.9015
		20	39.07	1.31	2.9214

These results agree with those of Angadi et al. (2017), who reported that Sn can be selectively upgraded using heavy-liquid separation of coarse particles at a density cut of 2.75 g cm^{-3} over silicate minerals, including quartz (2.4–2.65 g cm^{-3}) and albite (2.61–2.64 g cm^{-3}). For instance, for 15 min retention time, the Sn content of the coarse products (+300, +212 µm) was upgraded from 0.14% in the feed to 0.81%, at a recovery of 88.4%. The intermediate size fraction was upgraded to 6.1% SnO_2 at a recovery of 83%.

The efficiency of gravity separation appeared to decrease due to the presence of smaller particles, as reported by He et al. (2021). Lu et al. (2021) reported that gravity separation is a poor choice for particle sizes < 40 µm, suggesting that all particles < 38 µm were not recovered and caused the low recoveries recorded for the fines. The observed upgrade further suggests that between 11.5% and 15.6% of the barren mineral can be removed at coarser and intermediate size distributions by selective size distribution. The fine particles would require a more advanced concentration technique for successful beneficiation.

Tabling concentrate characterization

Table III summarizes the concentration results for the various size fractions obtained during the tabling process and the respective Sn

grades. The tabling results from all feed fractions exhibited lower recovery than those of DMS using tetrabromoethane. This could be due to the presence and formation of a third "middling" product stream obtained from the shaking table concentration process.

Recovery appeared to decrease with decreasing particle size. At the coarser size, high recovery was observed for the short milling retention time (5 min); recovery decreased with an increase in milling time. The grade was inversely proportional to the recovery and increased with milling retention time. Although the tabling recoveries were lower than those of DMS, this technique produced a relatively higher-grade concentrate.

Lower recoveries were observed for the intermediate size fractions (+150 to +53 µm) and fines (–53 to –38 µm). In addition, these two size fractions contained lower SnO_2 grades than the respective values reported from the DMS process. The lower grade and recovery observed for the tabling process compared with DMS could be attributed to the wide array of particle sizes, mainly the intermediate group, which varied from +150 µm to +53 µm. The fine particle size fraction showed high losses due to the presence of a small ultrafine particle size fraction (–38 µm). In addition, the decrease could be attributed to the presence of the additional middling product stream.

Flotation concentrate characterization

Table IV summarizes the concentration results obtained for the flotation process in terms of the recovery and Sn grade for the two size fractions. The results show that particle size has an important role during the mineral upgrade. This method was ineffective at a coarse particle size, which is probably due to the resultant poor grain hydrophobicity. Literature further classifies oleic acid as a poor collector (Jin et al., 2021), which could have led to the low (< 40%) recovery.

Similar to the coarse size fraction, the intermediate and fine fractions were characterized by low recoveries accompanied by a fairly high Sn grade. However, the recovery and grade improvements were identified as related to the milling retention time, which was linked to the liberation of the targeted mineral (SnO_2) and material redistribution between particle sizes that were able to float easily. In this case, the poor recovery and grade reported for the sample retained for 5 min milling could be associated with the presence of relatively coarse grains (+300 µm) because most (68%) of the milled sample was within that particle size range (Figure 2(a). The improved recovery and grade observed for the longer retention time could indicate efficiency of flotation concentration. The obtained results support those of Zhang et

Table IV

Preconcentration by flotation

Size fractions			Recovery (%)	Grade (%)
Intermediate (+300, +212 µm)	Sink	As received	11.45	2.51
		5	17.92	2.76
		10	25.67	3.45
		15	38.45	3.47
		20	36.92	3.31
Fine (+300, +212 µm)	Sink	As received	29.12	3.78
		5	33.78	4.07
		10	37.67	5.09
		15	40.41	7.81
		20	40.34	7.43

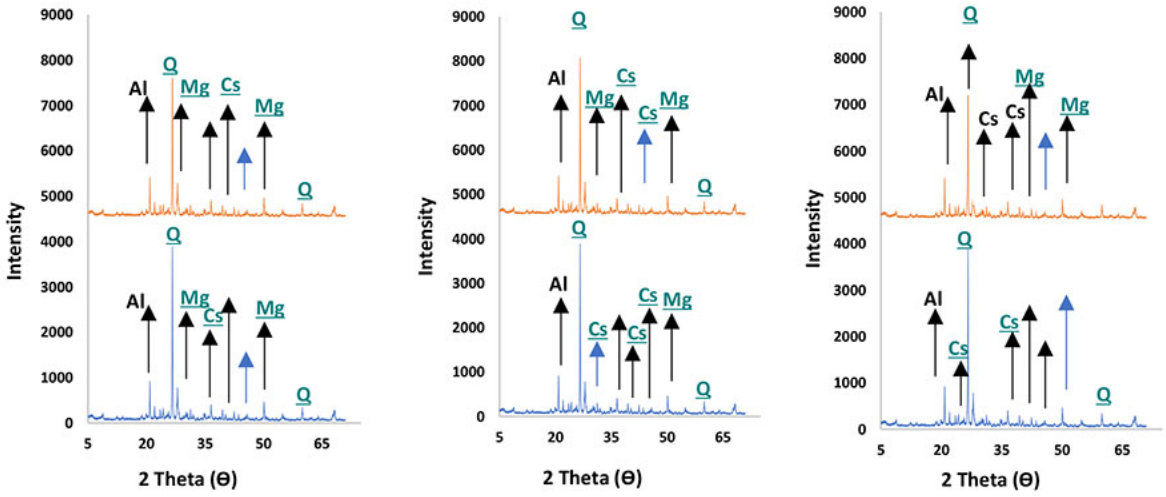

Figure 3—Mineral characterization comparison of feed and concentrate products for (a) dense media separation, (b) shaking table and (c) flotation. Q: quartz, Al: albite, Cs: cassiterite, Mg: magnetite

Table V

Mineral content variation (15 min retained milling time) by concentration process

Process	Albite	Cassiterite	Magnetite	Quartz	Muscovite	Tantalite	Fluorapatite
DMS	18.00	2.21	7.25	60.35	8.00	2.67	1.52
Tabling	22.00	1.43	7.06	56.00	11.00	2.00	0.51
Flotation	26.72	4.13	4.59	47.00	14.00	1.45	2.11

Table VI

Concentration process separation efficiency

Process	Separation efficiency (%)			
	Milling retention (min)	DMS	Tabling	Flotation
Coarse	5	71.80	56.08	
	10	73.51	57.94	
	15	74.13	59.89	
	20	70.93	54.73	
Intermediate	5	79.74	57.12	16.22
	10	75.67	62.07	24.09
	15	79.04	62.16	36.63
	20	77.57	55.54	35.52
Fines	5	48.73	45.53	31.99
	10	31.45	31.98	36.76
	15	37.85	37.55	39.80
	20	35.12	35.16	39.6

al. (2021), who reported that flotation was the most suitable concentration method for mineral sizes of −75+38 µm. Flotation process efficiency decreases for fine (−20 µm) and ultrafine (−10 µm) particle sizes (Leistner et al., 2016). These authors recommended an agglomeration operation prior to concentration using flotation.

Separation efficiency and X-ray diffraction characterization

Figure 3 summarizes the mineral contents of the concentrate obtained from each technique, and quantitatively and qualitatively compares the mineral composition of each concentrate product with its corresponding feed sample (Table V). Figure 3 and Table V display the mineral composition of the 15 min milled sample subjected to DMS, shaking table, and flotation separation, respectively. In all cases, the products showed an increase in Sn grade, indicated by XRF results (Tables II to IV) and a corresponding decrease in gangue minerals (silicates/quartz).

These recovery and grade data can be used to further assess and evaluate the concentration process. Separation efficiency (Table VI), which is a measure of the metallurgical efficiency, is defined as the recovery difference between the valuable and gangue minerals to the concentrate (Can et al., 2019). This concept is best described by Equation (1) (Schulz, 1970):

$$SE = R\frac{Cm(C-f)}{C(Cm-f)} \quad [1]$$

where SE is separation efficiency (%), R is recovery of the valuable constituent/metal/mineral in the concentrate (%), c_m is the assayed element grade in the mineral being concentrated (%), C is assayed element grade in the concentrate (%), and f is assayed element grade in the feed (%).

At the coarser and intermediate sizes, both concentration techniques (DMS and shaking table) can successfully be applied for mineral preconcentration. DMS appeared to have far better efficiency than the tabling operation and is recommended as the main preconcentration step. For instance, the coarser size after 15 min milled retention time showed separation efficiencies of 74.13% and 59.89%, respectively, for DMS and the shaking table. For the intermediate sizes at the same retention time (15 min), separation efficiencies of 82.7% and 68.5%, respectively, were obtained. A decrease in separation was observed for the fines, with both techniques yielding < 40%, suggesting that particle size impedes the performance of these techniques.

Flotation, in contrast, showed the worst separation of only 37% for the intermediate fraction at a grade of 3% (15 min milled retention time); the fines had much higher grade (8%) at a separation efficiency of 40%. The poor efficiency of this method could suggest that an adequate size distribution for mineral liberation should be determined prior to the concentration process, because this technique recovered a concentrate grade of 8%. In addition, fines would require some sort of agglomeration prior to flotation to enhance the process and avoid Sn losses.

Conclusions

The preconcentration of SnO_2 from a pegmatite ore body was investigated. Three concentration techniques, namely dense media, shaking table, and flotation, were assessed and the results compared. The results indicated that particle size distribution

dictates selection of the concentration process. DMS was effective at concentrating the coarse (+300 and +212 µm) and intermediate (+150 to +53 µm) particle sizes. The calculated Schultz separation efficiencies were 74.1% and 82.7% for the coarse and intermediate particle size fractions, respectively. By installing an additional DMS stage prior to further size reduction, the operation could allow discard of 10% to 15% of gangue-related minerals, which would contribute to energy saving and improved process sustainability. Flotation gave low recovery and could only be successfully applied to concentrate the fine sizes. The results suggest the use of DMS as the preconcentration technique for both coarse and intermediate particle sizes prior to further grinding, and application of flotation to the fine particle sizes to provide a mineral upgrade procedure to optimize the process in terms of comminution efficiency.

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