JSMPM

JOURNALS OF SUSTAINABLE MATERIALS PROCESSING AND MANAGEMENT
e-ISSN: 2821-2843

Vol. 6 No. 1 (2026) 100-111
https://publisher.uthm.edu.my/ojs/index.php/jsmpm

Work Index and Mineral Liberation Determination of a Complex Copper Ore: A Preliminary Step Towards Designing Sustainable Mineral Processing Flowsheets

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DOI: https://doi.org/10.30880/jsmpm.2026.06.01.009

Article Info

Received: 5 March 2026
Accepted: 21 May 2026
Available online: 28 June 2026

Keywords

Work index, comminution, copper ore, grinding, mineral liberation, separation

Abstract

Work index provides critical information about an ore's resistance to comminution, mineral liberation characteristics, and energy and time required for the comminution process. However, evaluating work index of ores is overlooked in many mineral processing operations and failing to assess work index of ores results in poor liberation of minerals, excessive energy losses and poor overall separation of valuable minerals. The aim of this study was to determine work index of Azara (Nigeria) complex copper ore and evaluate the liberation characteristics of the copper minerals. The evaluated work index of Azara complex copper ore was 13.90 kWh/t. Mineral liberation results from Mineral Liberation Analyser (MLA) showed 73.63% of malachite, 70.3% of chrysocolla, 52.66% of chalcocite and 24.25% brochantite particles were liberated at a grinding time of 90 minutes. With increasing of the grinding time, the liberation of minerals increased. At a grinding time of 180 minutes, the liberation of malachite, chrysocolla, chalcocite and brochantite improved to 86.72%, 76.66%, 74.14% and 53.27%, respectively. These findings provide new insights into the grinding conditions of Azara complex copper ore, liberation characteristics of the copper minerals and guidance for development of efficient and environmentally viable crushing/grinding circuits.

1. Introduction

Comminution is the most energy-intensive and costly operation in mineral processing operations [1–3]. Several previous studies [4–7], reported that comminution accounts for approximately 30–50% of the total energy consumption in mineral processing plants. This range can even reach up to 70% in hard rock minerals [8]. A report in 2024 by Association for British Mining Engineering and Consultancy (ABMEC) [9] showed that the global mining industry consumes 12 exajoule (1.2× 10^{16} kilojoule) annually, which is 3.5% of total energy consumed globally. From this total energy consumed by mineral processing plants, around 46%, i.e. 5.5 exajoule consumed during comminution. That means comminution alone accounts for about 1% of the total global energy consumed annually [9]. It is also important to emphasize that the majority of energy consumed during comminution is not used for actual particle size reduction. Instead, it absorbed by the grinding machines and dissipates in the form of heat and sound, and only 1% of the energy consumed during comminution is used for actual particle size reduction

[8,10]. To overcome such problems, it is important to design an efficient crushing (grinding) circuits with appropriate type and size of grinding machine, and estimate the grinding time and costs by evaluating the work index [7,11–13]. Furthermore, determination of work index provides valuable information on the optimum grinding condition of the minerals, which supports the design of downstream mineral processing flowsheets such as flotation and extraction [14,15]. Inadequate grinding results in poor mineral liberation, which significantly hinders the recovery of valuable minerals during beneficiation and extraction processes [2,13,16]. Conversely, over grinding not only leads to excessive energy consumption (making the process less cost-effective) but also reduces the recovery of valuable minerals due to the formation of ultrafine particles that are difficult to separate during the beneficiation process, especially during flotation [17–21]. Therefore, determination of work index of ores is crucial step for designing and optimizing mineral processing operations [22,23].

Work index is quantitatively defined as the energy, measured in kilowatt hour per ton (kWh/t), needed to grind a mineral from an infinitely large particle size to a final product size at which 80% of the material passes through a specified sieve size [6,24]. Work index provides important information about the ore's resistance to comminution (crushing and grinding), optimum grinding conditions, and the energy and time required for the comminution process [22,25,26].

In spite of its importance, work index tests are often overlooked in many mineral processing operations. Failing to assess an ore's work index results in poor liberation of minerals, excessive energy losses and poor overall separation (extraction) of valuable minerals [10]. Recently, the evaluation of mineral work indices has received greater attention compared to previous studies. However, W_i determinations for complex copper ores remain limited. Mama and Kebede [32] investigated the W_i of spodumene from Kenticha deposit in Ethiopia and reported an average value of 11.39 kWh/t. Abdelhaffez [23] studied the W_i of gold ore from Mahd Ad Dahab, Saudi Arabia and reported values in the range of 18.2-20.4 kWh/t. Furthermore, Nzeh et al. [10] examined the W_i of columbite from Jos, Nigerai and obtained a value of 2.34 kWh/t. Copper is mainly extracted from sulphide copper ores; however, high-grade sulphide copper ore has becoming increasingly depleted and the interest in extracting of copper from mixed sulphide-oxide and complex copper is growing. Due to their complex and heterogeneous nature, such ores require a preliminary investigation of W_i and mineral liberation characteristics to design efficient and sustainable mineral processing flowsheets.

The study area for this research is Azara, located in Awe Local Government, Nasarawa State, in the North Central part of Nigeria. It is situated approximately 276 kilometres to the southeast of Abuja, the capital city of Nigeria. The geological formation of Azara is part of the Benue Trough (Middle Benue Trough), which was formed by the rifting of the Central West African basement during the Cretaceous period. The Nigerian Geological Survey Agency (NGSA) has been conducted preliminary studies at the Azara region and confirmed the presence of copper in the area. The ore occurs with association with other minerals such as lead, zinc, barium and iron bearing minerals [27,28]. However, work index of the ore and its liberation characteristics of the copper ores that are needed for developing an efficient processing flowsheet have not yet been studied.

In this study, a copper ore sample from Azara, North Central Nigeria, was analysed to determine its work index, which helps to design efficient copper processing flowsheets. Moreover, the elemental concentration and distribution across the various sieve size fractions was studied. Furthermore, the liberation characteristics of copper minerals were studied using Mineral Liberation Analyser (MLA).

2. Materials and Methods

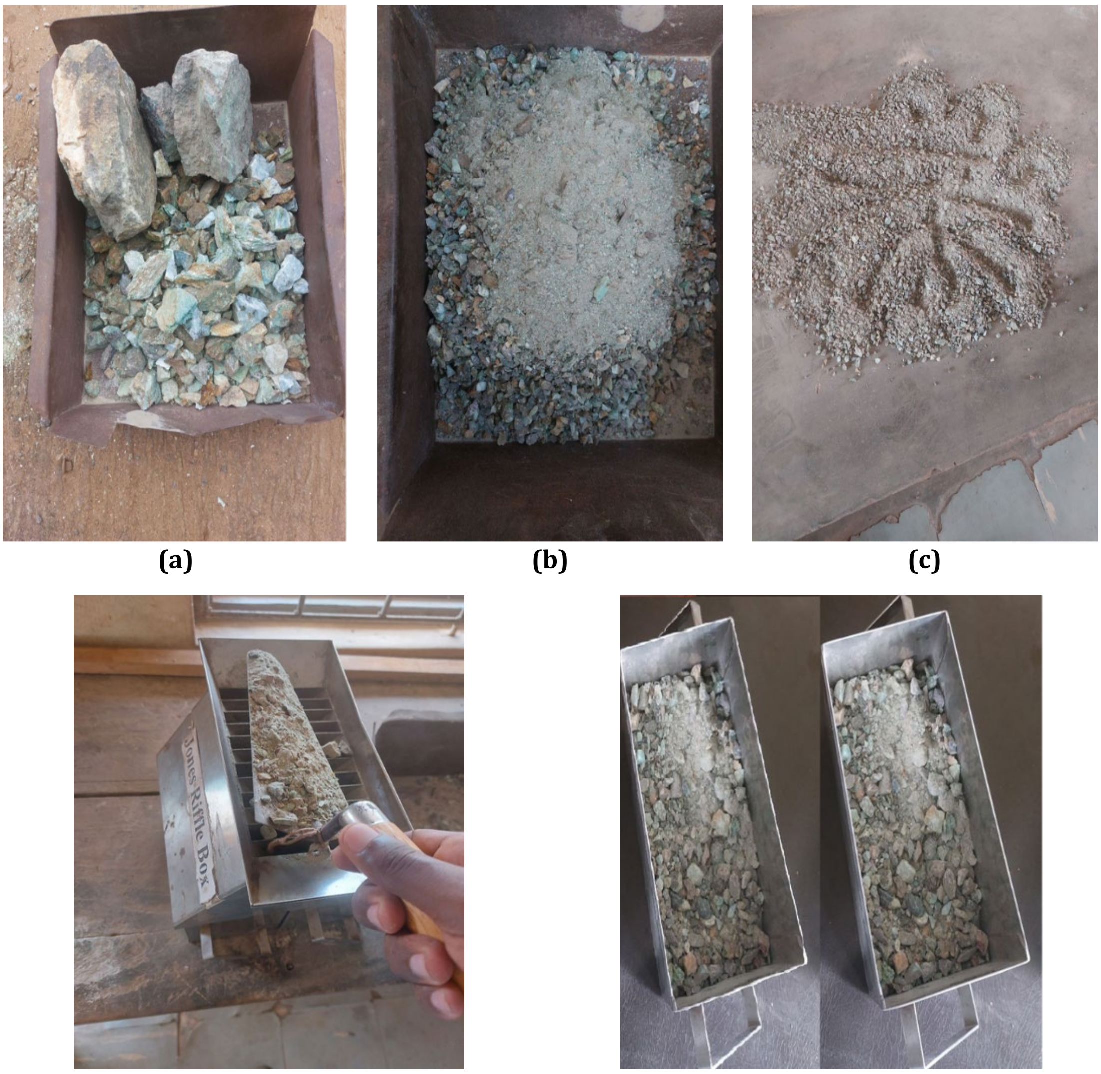
2.1 Materials

A 20 kg sample of copper ore was collected from Azara. In addition to the Azara copper ore (test ore), 600 g granite reference material with a known work index (15.12 kWh/t) was collected from Jos, Nigeria [29]. This reference material was used to determine the work index of the test ore. According to the Modified Bond work index, a reference ore with a known work index is required to determine the work index of the test ore [10,30].

2.2 Methods

2.2.1 Sample Preparation for Work Index

The 20 kg collected sample was first crushed using a Denver jaw crusher (5" × 6", England), followed by Fritsch laboratory jaw crusher (Pulverisette 1) to attain a particle size acceptable for milling. After crushing, the entire sample was homogenized to ensure a uniform distribution the minerals. Subsequently, a 200 g representative sample was taken from the homogenized Azara copper ore (test ore). Out of it, 100 g was ground in a ball mill (Denver ball mill) using steel balls (ø50 mm) for 30 minutes to get grounded ore for sieve analysis. Both the 100 g feed and the resulting product were subjected to sieve analysis using automatic Denver sieve shaker. The sieves were arranged in a stack with aperture sizes ranging from 1000 µm to 63 µm in the following sequence: 1000, 710, 500, 355, 250, 180, 125, 90 and 63 µm. The sieving process was carried out at 240 rpm for 15 minutes. The mass retained on each sieve was recorded for both the feed and the product using an electronic weight balance (DAWOOD balance, DA30002). The test was repeated three times, and the average values were reported. The same procedure was followed for the reference material. During the grinding of both the test ore and the reference ore, the energy input and rotational speed of the ball mill were kept constant. The tests were conducted at Federal University of Technology, Akure, Nigeria. The as-received sample and the sample preparation for the work index determination is shown in Fig. 1.



(a) **(b)** **(c)**

(d) **(e)**

Fig. 1 *Azara copper sample photos (a) As received; (b) Crushed; (c) Homogenized; (d) Sampling (splitting) using John rifle sampling; (e) Split sample; and (f) Grinding using ball mill*

2.2.2 Sample Preparation for Mineral Liberation Characterization

A 500 g of representative portion of the homogenized sample (Section 2.2.1) was taken and split into four sub-samples. The four sub-representative samples were ground using different grinding time. The samples were designed as AZ180, AZ150, AZ120 and AZ90 for 180, 150, 120, and 90 minutes grinding time, respectively.

2.3 Particle Size Distribution and Work Index Determination

The particle size distribution (PSD) of the samples, which are critical for determination of the work index, was analysed using the data obtained from the automatic Denver sieve shaker. Each weight retained in the sieves were weighted in electronic weight balance and the weight percentage, cumulative retained and cumulative passing were calculated accordingly. Subsequently, the work index of the Azara copper ore (test ore) was calculated using

Berry and Bruce (Bond's modified work index) formula. The Berry and Bruce work index equation is given in Equation 1 [10,31].

$$W_{it} = W_{ir}[\left(\frac{10}{\sqrt{P_{r80}}} - \frac{10}{\sqrt{F_{r80}}}\right) \div (\frac{10}{\sqrt{P_{t80}}} - \frac{10}{\sqrt{F_{t80}}})] \quad (1)$$

Where: W_{it} and W_{ir} represent the work index of the test ore and reference ore, respectively, expressed in kilowatt-hours per tonne (kWh/t); P_{r80} and F_{r80} are the 80% passing product and feed sizes of the reference ore (µm), respectively; and P_{t80} and F_{t80} are the 80% passing product and feed sizes of the test ore (µm), respectively [10,31].

The amount of energy required to reduce from F_{t80} to P_{t80} can be estimated by the Bond's energy equation, given in Equation 2 [4,31].

$$W = W_{it}[\left(\frac{10}{\sqrt{P_{t80}}} - \frac{10}{\sqrt{F_{t80}}}\right) \quad (2)$$

P_{r80}, F_{r80}, P_{t80}, and F_{t80} were calculated using the Gate-Gaudin-Schuhmann formula in Equation 5 derived from Equation 3 [10].

$$\text{Log P} = \alpha \log \frac{X}{K} + \log 100 \quad (3)$$

Where;

$$\alpha = \frac{[logP\ (X_2) - P\ (X_1)}{\log(X_2) - (X_1)} \quad (4)$$

$$Size_2 = \frac{(\%\ passing\ size_2)^2}{(\%\ passing\ size_1)^2} \times size_1 \quad (5)$$

Where P is the cumulative percent passing, X is the particle size, K is the theoretical maximum particle size obtained by extrapolation, and α is a distribution coefficient constant [10,32].

Finally, representative samples from all the size fractions of the test ore (Azara copper ore) were analysed using X-ray fluorescence (XRF) to determine the elemental concentration in each sieve size fractions. The composition and concentration of the sieve size fractions were analysed using X-ray fluorescence (XRF). For each analysis, a 1.5 g of representative sample was placed into the XRF plastic sample holders and measurements were carried out. Each sample measurement repeated three times, and the average values were reported.

2.4 Mineral Liberation Characteristics

The mineral liberation characteristics were conducted using Mineral Liberation Analyser (MLA), which integrates Field Emission Scanning Electron Microscope (FE-SEM) equipped with two Energy Dispersive X-ray (EDX) detectors and controlled by MLA Suite Software version 3.1. A 1.5 g representative sample of AZ180, AZ150, AZ120 and AZ90 was taken and blended with powdered graphite to uniformly distribute the minerals. Then the samples were embedded in epoxy resin and form a mount with diameter of 25 mm. The formed mounts for each sample were then cured for 24 hours in air. The mounts were then vertically sectioned into 5 pellets and rotated 90°. Then re-embedded in new epoxy resin and cured in air again. Afterwards, samples were ground and polished to produce a flat surface and then coated with carbon to make the surface of the samples electrically conducting. MLA measurements were conducted at 25 kv, 4.78 µm, 10 nA and 20 µm of accelerating voltage, spot size, probe current and pixel resolution, respectively. Particles were segmented into individual grains based on the backscattered electron intensity (BSE) and phase identification was performed by matching the acquired spectra to a predefined mineral phase database.

3. Results and Discussion

3.1 Particle Size Distribution (PSD) and Work Index of the Ore

The particle size distribution (PSD) results of feed and product of the test ore (Azara copper ore) are presented in Tables 1 and 2, respectively. PSD results of feed and product of the reference ore are presented in Table 3 and 4, respectively. Fig. 2 shows the PSD curves for the feed of the test ore, product of the test ore, feed of the reference ore and product of the reference ores, respectively. The key parameters required for the determination of the work index (w_i); F_{t80}, P_{t80}, F_{r80}, and P_{r80} were evaluated using the Gate–Gaudin–Schuhmann equation (Equation 5).

$$F_{t80} = \frac{(80\%)^2}{(80.69\%)^2} \times 1000\ \mu m = 982.97\ \mu m$$

$$P_{t80} = \frac{(80\%)^2}{(85.51\%)^2} \times 500\ \mu m = 437.64\ \mu m$$

$$F_{r80} = \frac{(80\%)^2}{(71.89\%)^2} \times 1000\ \mu m = 1238.35\ \mu m$$

$$P_{r80} = \frac{(80\%)^2}{(76.97\%)^2} \times 710\ \mu m = 540.14\ \mu m$$

Using the calculated values of F_{t80}, P_{t80}, F_{r80}, and P_{r80}, the work index of Azara copper ore was evaluated using Equation 1. The work index of the reference ore (known work index) is 15.13 kWh/t. Therefore, the work index of the test ore (W_{it}) was evaluated as:

$$W_{it} = 15.13\ kWh/t\ [(\frac{10}{\sqrt{540.14\ \mu m}} - \frac{10}{\sqrt{1238.35\ \mu m}}) \div (\frac{10}{\sqrt{437.64 \mu m}} - \frac{10}{\sqrt{982.97\ \mu m}})] = 13.90\ kWh/t$$

The minimum estimated amount of energy needed to reduce the feed of the test ore to the product size was calculated using Equation 2.

$$W = 13.90\ kWh/t\ (\frac{10}{\sqrt{437.64}} - \frac{10}{\sqrt{982.97}}) = 2.21\ kWh/t$$

The evaluated work index (w_i) of the Azara copper ore was found to be 13.90 kWh/t. Based on standard classifications that relate work index values to material hardness, this result falls within the range of medium-hard materials [11,33]. The minimum estimated energy required to reduce the feed of the test ore (F_{t80}) to the product size (P_{t80}) was 2.21 kWh/t, representing the minimum energy needed to grind one tonne of copper ore from 80% passing feed size to 80% passing product size.

The results presented in Tables 1–4 and Figures 3a-d show the particle size distribution (PSD) analyses for both the test ore (Azara copper ore) and the reference ore. Due to the presence of rocky and relatively hard particles in the Azara copper ore feed, a cumulative 19.31% of oversize mineral particles was retained on the 1000 µm sieve, while 80.69% of undersized particles passed through it. Similarly, on the 710 µm sieve, 29.33% of the sample particles were retained with 70.67% passing. On the 500 µm sieve, 43.21% particles were retained with 56.79% passing undersized particles. In the same trend, on the 355, 250, 180, 125 and 90 µm sieves, 56.47%, 69.65%, 82.15%, 92.31%, and 95.69% of the particles were retained, respectively, while 43.53%, 30.35%, 17.85%, 7.69%, and 4.31% of finer particles passed through those sieves accordingly. It was further observed that 98.80% of the ore particles were retained on the 63 µm sieve, the finest sieve used; while only 1.20% of fine particles passed through and were collected in the pan. These observations are represented in detail in Table 1 and illustrated graphically in Fig. 2a.

Table 1 *Particle size distribution of feed test ore i.e. feed to the ball mill (Ft)*

Particle size (µm)	Normal Aperture Size (µm)	Weight Retained (g)	% Weight Retained	% Cumulative Weight Retained	% Cumulative Weight Passing
+1000	1000	18.95	19.31	19.31	80.69
-1000+710	710	9.83	10.02	29.33	70.67
-710+500	500	13.62	13.88	43.21	56.79
-500+355	355	13.02	13.27	56.47	43.53
-355+250	250	12.93	13.18	69.65	30.35
-250+180	180	12.26	12.49	82.15	17.85
-180+125	125	9.97	10.16	92.31	7.69
-125+90	90	3.32	3.38	95.69	4.31
-90+63	63	3.05	3.11	98.80	1.20
-63	Pan (-63)	1.18	1.20	100	0.00
Total	---	98.13	100	---	---

The PSD tables and corresponding curves for the test and reference ores (Tables 1–4 and Fig. 2) exhibited a normal particle size distribution pattern, consistent with established comminution theories [10]. As indicated in Tables 1, 80.69% of the Azara copper ore particles generally fall within the size range of 63 to 1000 µm (-1000 +63 µm). While 19.31% of the particle were over 1000 µm. Figures 3a-b show the cumulative weight percentages of passing (undersized) and retained (oversized) particles plotted against sieve sizes of the test ore. The results revealed that 80% of the mineral particles passed at a sieve size of 982.97 µm for the feed and 437.64 µm for the product, corresponding to sieve ranges of -1000 +710 µm and -500 +355 µm, respectively. This indicates that the Azara copper ore sample consists predominantly coarse mineral particles and the sample should undergo further grinding for downstream operation. Likewise, from the PSD data and curves presented in Table 3 and Table 4, Fig. 2c and Fig.2d, the reference ore samples showed 80% passing sieve sizes of 1238.35 µm for the feed and 540.14 µm for the product. The 80% passing feed and product corresponds to sieve size ranges of +1000 and -710+500 µm, respectively.

Table 2 *Particle size distribution of product test ore i.e. product from the ball mill (Pt)*

Particle size (µm)	Normal Aperture Size (µm)	Weight Retained (g)	% Weight Retained	% Cumulative Weight Retained	% Cumulative Weight Passing
+1000	1000	4.38	4.47	4.47	95.53
-1000+710	710	3.92	4.00	8.47	91.53
-710+500	500	5.91	6.03	14.49	85.51
-500+355	355	21.82	22.24	36.74	63.26
-355+250	250	19.98	20.37	57.11	42.89
-250+180	180	17.86	18.21	75.31	24.69
-180+125	125	10.09	10.29	85.60	14.40
-125+90	90	6.68	6.81	92.41	7.59
-90+63	63	4.49	4.58	96.99	3.01
-63	Pan (-63)	2.96	3.02	100	0.00
Total	---	98.09	100	---	---

Table 3 *Particle size distribution of the feed reference ore i.e. feed to the ball mill (Fr)*

Particle size (µm)	Normal Aperture Size (µm)	Weight Retained (g)	% Weight Retained	% Cumulative Weight Retained	% Cumulative Weight Passing
+1000	1000	27.65	28.11	28.11	71.89
-1000+710	710	9.81	9.97	38.08	61.92
-710+500	500	13.52	13.75	51.83	48.17
-500+355	355	12.95	13.17	65.00	35.00
-355+250	250	11.26	11.45	76.45	23.55
-250+180	180	9.73	9.89	86.34	13.66
-180+125	125	7.55	7.68	94.02	5.98
-125+90	90	2.49	2.53	96.55	3.45
-90+63	63	2.31	2.35	98.90	1.10
-63	Pan (-63)	1.08	1.10	100	0.00
Total	---	98.35	100	---	---

The PSD of the four samples (F_t, P_t, F_r and P_r) show distinct differences in both feed characteristics and milling response. The two feed samples (F_t and F_r) demonstrated coarse distributions. However, F_r is notably coarser, with a higher proportion of material in the +1000 µm fraction (28.11%) compared to F_t (19.31%). After milling, both feed materials show a shift toward finer particle sizes, although the extent of size reduction varies between samples. P_t shows a more pronounced reduction in particle size compared to P_r, with a higher accumulation of fine material. For instance, P_t contains a greater proportion of material passing finer size classes (below 180 µm region) compared to P_r, indicating more efficient grinding of the F_t sample. In contrast, P_r retains relatively higher proportions of coarse and intermediate particles after milling, particularly in the 355–250 µm size range. This suggests a higher resistance to size reduction in the F_r sample. Similar trends have been reported by Nzeh et al. [10].

Table 4 *Particle size distribution of the product reference ore i.e. product from the ball mill (Pr)*

Particle size (µm)	Normal Aperture Size (µm)	Weight Retained (g)	% Weight Retained	% Cumulative Weight Retained	% Cumulative Weight Passing
+1000	1000	7.15	7.25	7.25	92.75
-1000+710	710	6.86	6.95	14.20	85.80
-710+500	500	8.71	8.83	23.03	76.97
-500+355	355	21.31	21.60	44.63	55.37
-355+250	250	19.54	19.81	64.44	35.56
-250+180	180	15.68	15.89	80.33	19.67
-180+125	125	9.94	10.08	90.40	9.60
-125+90	90	4.94	5.01	95.41	4.59
-90+63	63	2.78	2.82	98.23	1.77
-63	Pan (-63)	1.75	1.77	100	0.00
Total	---	98.66	100	---	---

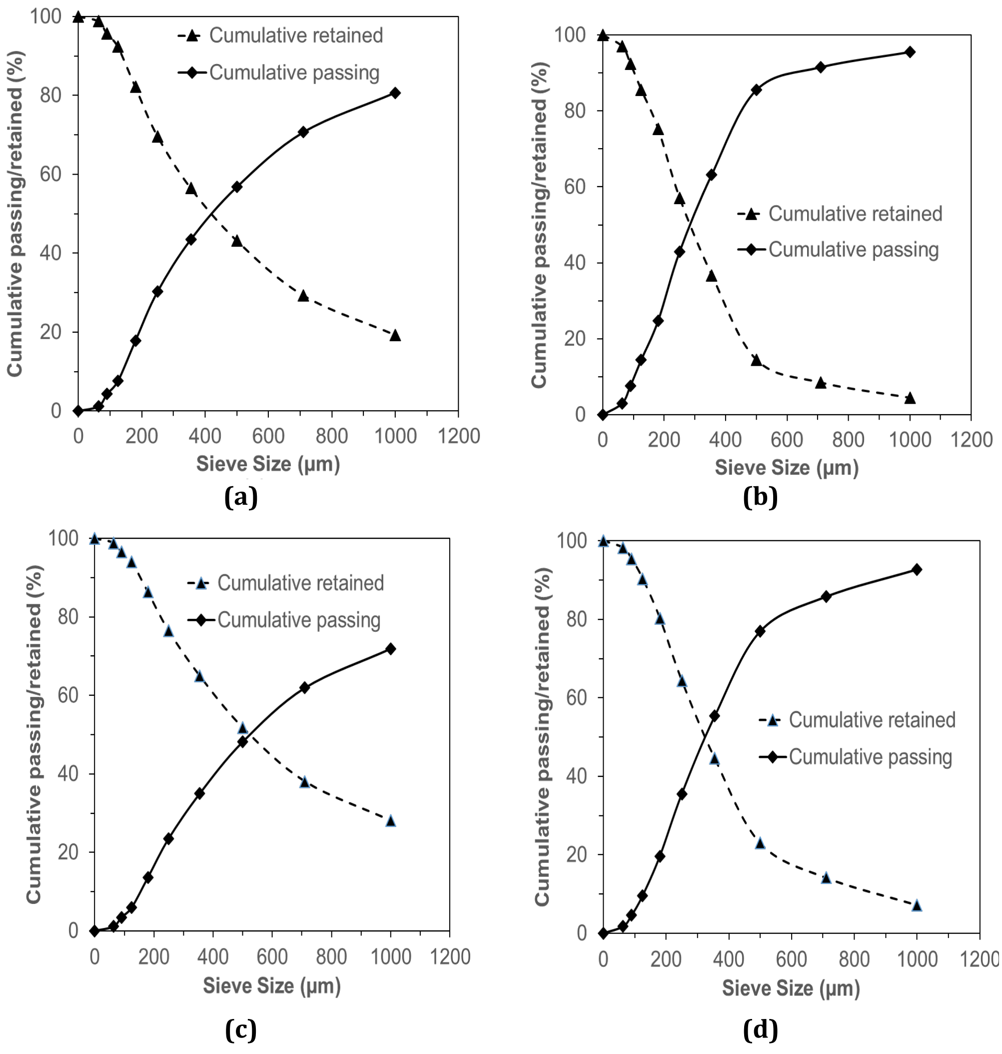


Fig. 2 *Particle size distribution curves: (a) Azara copper ore feed (F_t); (b) Azara copper product (P_t); (c) Reference ore feed (F_r); and (d) Reference ore product (P_r)*

3.2 Elemental Concentration of Sieve Size Fractions

The result for the chemical/elemental analysis for the sieve size fractions is given in Table 5. The distribution of the elemental composition of the test ore across the sieve size fractions shows variations in the concentration of copper and other elements according to their particle sizes. Copper exhibited an increase in its concentration with decreasing particle size, rising from 1.06% in the coarsest fraction (above 1000 μm) to 4.18% in the finest fraction (below 63 μm). This progressive improvement of copper in the finer fractions suggests that the copper-bearing minerals are finely disseminated within the ore matrix and become increasingly liberated as particle size decreases. Such behavior is an indication of disseminated sulfide ores, where minerals like chalcocite and chalcopyrite are closely associated with the gangue and require adequate size reduction to achieve liberation.

On the other hand, the major gangue-forming elements, for instance silicon, calcium, and aluminum exhibit decreasing concentrations in the finer sieve size fractions. This indicates that these minerals are more dominant in the coarser particles. The high grade of copper, iron, and sulfur toward finer sizes indicates close mineralogical association of these elements. Lead also follows a similar trend, suggesting possible presence of lead-bearing sulfides, such as galena, which may be associated with the same sulfide mineral assemblage. This distribution pattern has important implications for beneficiation, indicating that effective grinding and particle-size control will enhance liberation and improve the recovery of copper minerals during flotation or bioleaching operations.

Table 5 *Elemental composition and grade of the Azara copper ore at various sieve size fractions*

Sieve Size (μm)	Normal Aperture Size (μm)	Element (wt.%)								
		Al	Ca	Cu	Fe	K	Pb	S	Si	Ti
+1000	1000	8.75	7.38	1.06	1.94	1.27	0.32	0.2	23.41	0.13
-1000+710	710	8.43	7.11	1.17	2.18	1.32	0.49	0.24	23.68	0.18
-710+500	500	7.78	7.02	0.94	2.62	1.17	0.28	0.16	24.24	0.14
-500+355	355	8.22	7.14	1.22	2.65	1.19	0.29	0.18	23.86	0.16
-355+250	250	8.21	7.48	1.69	2.72	1.15	0.45	0.21	23.53	0.14
-250+180	180	7.12	7.17	1.97	2.45	1.34	0.35	0.21	22.72	0.13
-180+125	125	6.41	7.1	2.06	2.71	1.61	0.46	0.25	22.99	0.14
-125+90	90	6.01	7.16	2.37	2.93	1.51	0.53	0.25	23.65	0.15
-90+63	63	6.75	7.77	3.25	3.39	1.23	0.78	0.34	23.39	0.15
-63	Pan (-63)	6.24	7.79	4.18	3.87	1.1	1.16	0.45	22.65	0.15

3.3 Mineral Liberation Characteristics

Analysis of mineral liberation is an important parameter to design efficient beneficiation and extraction flowsheets. Properly studied mineral liberations helps to maximize the grade and recovery. The effect of mineral liberation on the beneficiation of minerals and extraction has been studied by several researchers. For example, a study by Shi el al.[34] Showed that well studied mineral liberation supports efficient copper-cobalt sulphide ore flotation circuit design. According to their study, a significant improvement of grade of copper from 1.28% to 9.27% and recovery of 93.69% was observed after the detailed mineral liberation characteristics.

The mineral liberation results of the four major copper-bearing minerals of Azara copper ore at different grinding time is presented in Fig. 3. The liberation of the minerals was determined based on the proportion of the grains containing copper mineral phases within the analysed area. The mineral particles were classified into four liberation classes according to their percentage of surface exposure (x) as: $x \leq 30\%$ (locked), $30\% \leq x < 60\%$ (low middling), $60\% \leq x < 90\%$ (high middling) and $x \geq 90\%$ (liberated). For sample AZ90, malachite exhibited the highest degree of liberation with 73.63% of the malachite containing particles were liberated, 12.41% high middling, 4.88% low middling and 9.07% locked. Chrysocolla also showed high liberation characteristics. 70.3% of chrysocolla containing particles were liberated, 4.12% high middling, 3.96% low middling and 21.62% locked. Chalcocite and brochantite exhibited low liberation degree compared with malachite and chrysocolla. Only 52.66% of the chalcocite particles were liberated, 24.25% high middling, 12.28% low middling and 10.81% locked. For brochantite, only 20.29% of brochantite particles were liberated followed by 39.22% high middling, 16.13% low middling and 24.36% locked. The liberation of the minerals increased with the increasing of the grinding time, which is consistent with the previous study reported by Bahrami *et al*, (2020). The degree of liberation (liberated particles) reached 86.72%, 76.66%, 74.14% and 53.27% for malachite, chrysocolla, chalcocite and brochantite respectively at the grinding time of 180 minutes (sample AZ180). The full mineral characteristics of the copper ore samples is presented in Fig. 3.

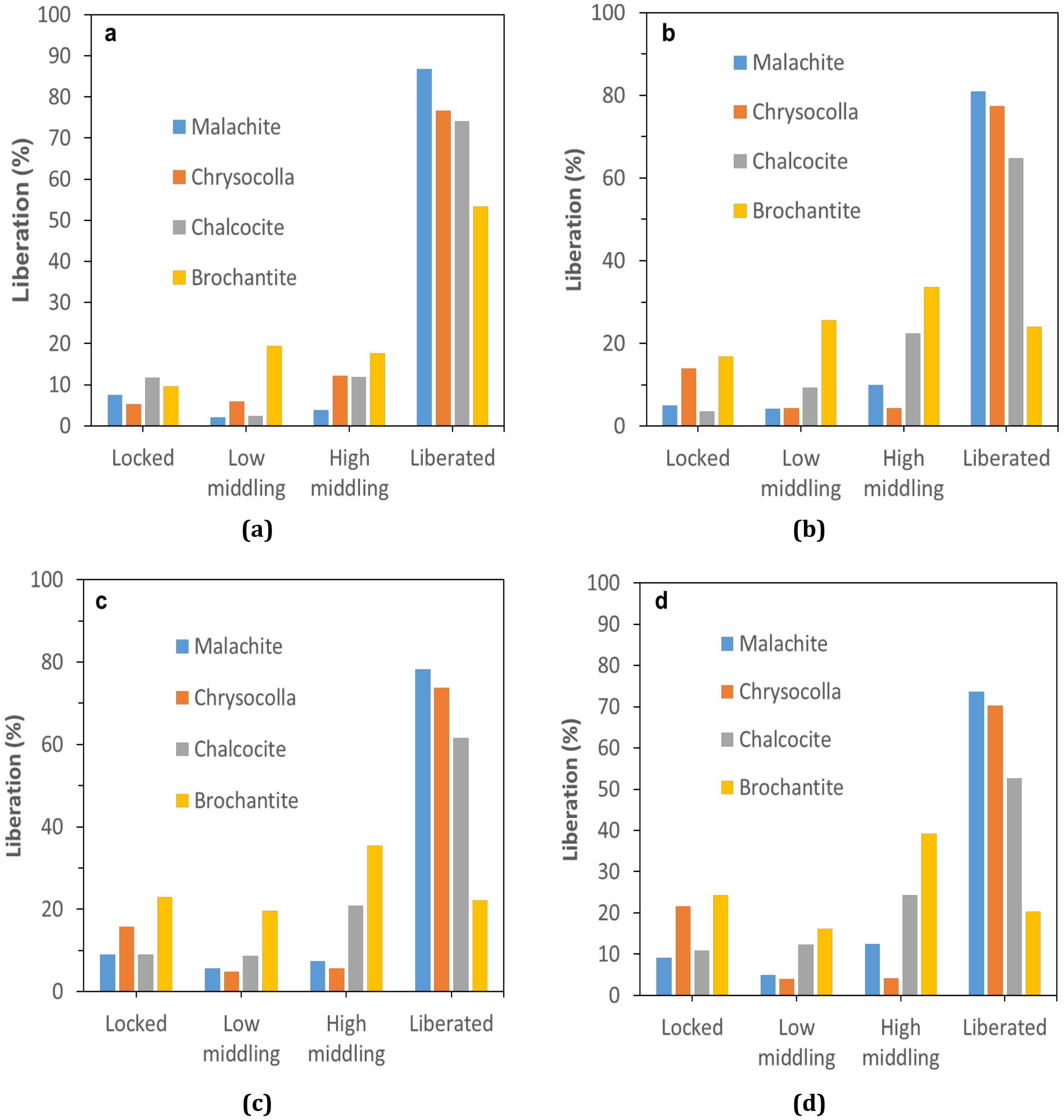


Fig. 3 *Mineral liberation characteristics of copper minerals from Azara copper ore at different grinding time: (a) 180 minutes; (b) 150 minutes; (c) 120 minutes; and (d) 90 minutes*

4. Conclusion

A complex copper ore collected from Azara, Nigeria was evaluated its work index. The obtained work index of the ore was 13.90 kWh/t, which falls in the range of medium hard minerals. The minimum estimated amount of energy required to grind the feed of the test ore to the product size was 2.21 kWh/t. This indicates a minimum of 2.21 kWh energy is required for size-reducing of one ton of copper mineral from 80% passing 982.97 µm sieve size to a particle size allowing 80% passing 437.64 µm sieve size.

The results showed that the degree of liberation of minerals increases with increasing of the grinding time. For the sample with 90 minutes grinding time (AZ90), 73.63% of malachite, 70.3% chrysocolla, 52.66% of chalcocite and 24.25% brochantite particles were liberated. The degree of liberation increases with increasing of the grinding time. In the 180 minutes grinding time (sample AZ180), the liberation degree of malachite, chrysocolla, chalcocite and brochantite reached 86.72%, 76.66%, 74.14% and 53.27%, respectively. Among the copper minerals, malachite was with the highest degree of liberation, while brochantite was with the lowest degree of liberation. These findings help to design optimum grinding conditions, which maximize the grade and recovery of valuable materials from complex ores.

Acknowledgement

The authors would like to acknowledge Dr. Ola-Omole of FUTA for facilitating access to the crushing and grinding machines.

Conflict of Interest

Authors declare that there is no conflict of interests regarding the publication of the paper.

Author Contribution

Elias Tadesse Fisha*; Conceptualization, design, draft manuscript preparation, data collection and analysis, interpretation of results and review.* ***Daniel Toyin Oloruntoba****; supervision, design, draft manuscript preparation and review.* ***Oladunni Oyelola Alabi****, supervision, design, draft manuscript preparation and review.* ***Oluwatoyin Adenike Olaseinde****; supervision, design and review.*

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