

Recovery of 'Nsutite' from Tailings Material of Ghana Manganese Company (GMC) Limited Mine, Nsuta*

M. Ali and R. K. Amankwah

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Abstract

An investigation was conducted for the enrichment of manganese oxide tailings generated by a spiral concentration plant at Ghana Manganese Company (GMC) Limited, Nsuta. The work involved mineralogical and particle size characterisation, gravity and magnetic separation techniques. Analytical methods used include; X-Ray Fluorescence and Atomic Absorption Spectrophotometry. The material composed of 9.01% Al_2O_3 , 27.43 % SiO_2 , 19.28% MnO_2 and 9.86% Fe_2O_3 , while other oxides ranged between 1.55 and 0.04%. By treating the distinct size fractions, it was discovered that the most efficient method was heavy medium separation at a bath density of 2.96 having an overall recovery of 51.18%. Enrichment ratios of 1.28, 0.98 and 0.91 were obtained in order of heavy medium separation, magnetic separation and shaking table. The highest grade obtained was 38.25% Mn which qualifies the concentrate as medium grade material in the manganese market.

1 Introduction

The name 'Nsutite' (γ - MnO_2); referred to as better-grade manganese oxide mineral type, was named after Nsuta, where extraction from the Nsuta-Dagwin manganese deposits began almost a century ago. According to Kesse (1985), the deposits mainly contain oxide (chiefly pyrolusite and psilomelane) at the upper deposition layer and carbonate (rhodochrosite) at the primary layer. The oxide material is suitable for use in both chemical and dry-cell battery manufacturing sectors (Christie, 2010). Records indicate that since 1916 when exploitation began, over 27 million tonnes of high grade (52% Mn), low grade (48-50% Mn) and other grades (46% Mn and 42-45% Mn) manganese oxides were produced from Nsuta mine for the mineral market (Kesse, 1985).

Although the Nsuta mine has produced oxide ore for many years, such production is no longer justifiable due to the oxide ore depletion (two decades ago). As such, production is now fully on the manganese carbonate (averaging 34.16% Mn (Kesse, 1985)) which is close to metallurgical grade of 38 - 55% Mn (Christie, 2010). Converting the manganese carbonate to oxide will however require extra treatment(s) such as calcination (Amankwah *et al.*, 1999; 2005) or leaching (Sharma, 1992). In another development, a study conducted by O'Shaughnessy *et al.*, (2004) concluded that Nsuta manganese carbonate is still acceptable in its nature for manganese ferroalloy production.

When oxides were being processed, concentration was done using jigs. The tailings generated by jigging were further enriched by the use of sorting spirals (Anon, 1999). These newly generated tailings

can be considered as a source for manganese oxide re-enrichment. Moreover, it is the best approach, as the scientific world is still struggling with how to arrive at large scale secondary manganese sourcing by recycling from steel, beverage cans and dry cell batteries (Gandhi, 2010).

It is in this light that comparative beneficiations were undertaken in order to identify the most suitable method for the high grade manganese oxide ("Nsutite") re-enrichment from the generated secondary oxide tailings without further comminution.

2 Experimental Procedures

Samples of manganese oxide tailings were obtained from the Ghana Manganese Company Limited, Nsuta, tailings dump. A quantity of about 400 kg was collected at random points from the various tailings heaps and treated at the Department of Mineral Engineering, University of Mines and Technology (UMaT), Tarkwa.

The sample materials were air dried for four days, at the rate of eight hours/day to remove moisture. The materials were then mixed thoroughly and split by coning and quartering. Two opposite quarters were separated; re-mixed and further coning and quartering were carried out until a manageable representative sample was obtained. The latter was further divided into two equal portions A and B using a Riffler. Portion A was taken for the characterisation and comparative beneficiations, while B was kept as a backup.

1 kg of portion A was used as feed to a set of ten (ASTM E11) sieves selected based on $4\sqrt{2}$ relationship in order to make possible much closer sizing

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(Anon., 1977; Wills and Napier-Munn, 2006). The topmost sieve was 10 mesh (2000 μm) and the finest was 80 mesh (180 μm). The sieves were loaded with the materials and shaken with Retch sieve shaker for ten (10) minutes after which the oversize material on each sieve was weighed. This was followed by subsequent sieving to generate sufficient feed materials from each fraction for the comparative beneficiations.

A Denver Shaking Table, model S.N. W 79107 set up horizontally was fed with 25% solid (600 g) slurry of each particle fraction at a time. The shaking table's reciprocation motion was at a rate of 1400 rpm, and the tangential flow of water through jets was at the rate of 210.10 cm^3s^{-1} . Two directions of materials flow achieved were the concordance flow of the heavy minerals and the discordant in respect of light minerals.

A Davies Dry Disc Magnetic Separator, Model No. 13008/350/4.5V fed with 300 g of each fraction was used for the magnetic separation. However, due to the limitation of the machine's discharge pipes, fractions coarser than 20 mesh could not be treated. The vibratory feeder conveyed materials to temporary magnetic disc rotating with respect to horizontal axis. The disc picked the paramagnetic materials and discharges them into three collectors while the diamagnetics were discharged into the fourth collector. The paramagnetic were reprocessed to optimise concentration.

In the case of the heavy medium separation, 1,1,2,2,-tetrabromoethane ($\text{CHBr}_2\cdot\text{CHBr}_2$) was selected. This is because; its specific gravity of 2.96 falls between that of the manganese oxide (4.75) and the major gangue mineral; quartz (2.65). 25 g of each particle fraction was fed into 500 ml beaker containing 250 ml of the heavy medium and the sink and the float materials were collected, rinsed with carbon tetrachloride (CCl_4), washed and filtered. The products of these separations were then dried and weighed.

An X-ray fluorescence spectrophotometer, Spectro XLAB 2000 was used in determining the major oxides content of the representative sample. Manganese in the samples was determined by pulverising to minus 90 μm , followed by aqua-regia digestion of 0.5 g subsamples. Each mixture was allowed to cool and then filtered. The filtrate was topped to 100 ml with distilled water and analysed using the Atomic Absorption Spectrophotometer (AAS) Varian AA 240FS Fast Sequential.

3 Analytical Methods

The concentration process was evaluated quantitatively using conventional parameters such as ratio of concentration, recovery and enrichment ratio as shown in Equations 1 to 3 (Wills and Napier-Munn,

2006).

$$\text{Ratio of concentration} = \frac{\text{Weight of feed}}{\text{Weight of concentrate}} \quad (1)$$

$$\text{Enrichment ratio} = \frac{\text{Assay value of concentrate}}{\text{Assay value of feed}} \quad (2)$$

$$\text{Recovery} = \frac{\text{Concentrate weight} \times \text{concentrate grade}}{\text{Feed weight} \times \text{feed grade}} \times 100 \quad (3)$$

4 Results and Discussions

4.1 Assay of the Head Sample

The quantitative estimation of oxides in the sample by X-ray Fluorescence (XRF) is presented in Table 1. The results show that there was 9.01% Al_2O_3 , 27.43 % SiO_2 , 23.64% MnO_2 (14.93% Mn) and 9.86% Fe_2O_3 , while the other oxides ranged between 1.55 and 0.04%. In contrast, the Nsuta ore was generally rich with average value of 83.28% (i.e. 52.61% Mn) (Kesse, 1985). Silica was the main gangue mineral at 27.43%. The Loss on Ignition (LOI) value of 25.97% significantly acknowledged the presence of volatile matter.

Table 1 Composition of the Representative Sample

Oxide	% Composition
Na_2O	1.55
MgO	1.24
Al_2O_3	9.01
SiO_2	27.43
P_2O_5	0.17
SO_3	0.04
Cl	0.00
K_2O	0.67
CaO	0.19
TiO_2	0.23
MnO_2	23.64
Fe_2O_3	9.86
L.O.I.	25.97

4.2 Particle Size Analysis

Fig. 1 shows the percentage composition by weight of the representative sample in order of; 850 μm (25.54%), 1400 μm (14.85 %), 650 μm (13.97 %) and 500 μm (10.59 %), while the remaining fractions had less than 10%. The results pointed that 74.1% of the bulk material consist of 2000 μm to 500 μm fractions while the remaining 25.9% composed of 425 μm to <180 μm fractions.

Table 2 depicts the Atomic Adsorption Spectrometry (AAS) assay values of the respective fractions ranging from a maximum of 24.89% Mn at 180 μm fraction to the lowest of 18.39 % Mn at - 180 μm .

The variation in assay could be attributed to the degree of liberation and concentration of the particle sizes at discrete fractions.

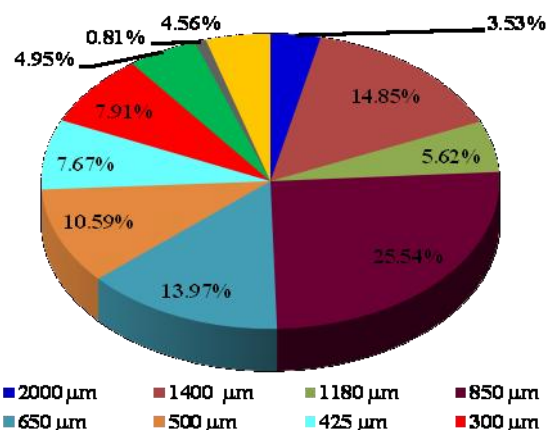


Fig. 1 Tailings Representative Sample Characterisation by Weight

A cursory examination showed that Mn content is greater than 20% from 2000 µm to 650 µm fractions and a drop to < 20% Mn from 500 µm to 300 µm. Then similar rise in values with 212 µm and 180 µm and finally a drop at the -180 µm was noted.

The foregoing assay values are considered as the baseline vis-a-vis the enriched values.

Table 2 Assay of the Representative Sample Sieved Fractions

Fraction (µm)	Mn Content (%)
2000	22.34
1400	21.64
1180	20.76
850	20.84
650	20.44
500	19.63
425	19.73
300	19.63
212	21.14
180	24.89
<180	18.39

4.3 Enrichment Tests

The percentage of the original material that reported in the concentrate after the enrichment tests are presented in Table 3. As indicated, the mass of concentrate that reported when the shaking table was used were greater than 80% by weight in all fractions except in <180 µm. In this case particles were mostly in discordant flow, because they were very fine in size; thereby reducing the efficiency of separation according to Gupta and Yan (2006).

In the case of Heavy Medium Separation, higher values were recorded with the coarser fractions (up to 58.48%), intermediate values with the finer fractions (maximum of 40.60%) and lower values with the intermediate size fractions (maximum of 33.96%). In addition, the non existence of result for <180 µm fraction was due to the very slow settling

rate in obtaining sink particles while running the test. This signifies the inability of the very fine particles to overcome the viscosity of the heavy medium (Jain, 2008).

The difference in performance between these gravity methods of concentration is related to the determined concentration criteria of 2.3 and 5.8 for water and tetrabromoethane media respectively. According to Anon (2008), the latter quotient signifies easy beneficiation while for the former, separation may be possible.

The efficiency of the magnetic separation is more biased towards finer fractions in which a maximum value of 50.05% was obtained buttressed Khan *et al.*, (2004) and Rao *et al.*, (1988) position. Also, Wills and Napier-Munn (2006) related the situation to increase in remanance and permeability with decrease in particle size. Furthermore, Jain (2008) said that many minerals response to magnetic separation depend on the distinctiveness of their reaction at a given mesh.

Table 4 indicates the variation in % Mn values of each fraction with respect to enrichment methods used. The heavy medium separation values were exceptionally higher than their counterparts; except in the case of 212 µm fractions.

Table 3 Percentage of Material reporting as Concentrate

Size (µm)	Concentrate Weight		
	Shaking Table	HMS	Magnetic Separation
2000	81.99	58.45	-
1400	97.27	55.52	-
1180	94.06	43.48	-
850	95.97	29.92	30.11
650	99.14	31.48	36.67
500	98.79	28.36	20.32
425	89.94	33.96	40.32
300	90.59	40.20	50.05
212	88.71	40.60	20.17
180	85.16	28.12	35.23
<180	31.51	-	46.92

Table 4 Mn Content in each Fraction after Enrichment Tests

Fraction (µm)	Manganese Content (%)		
	Shaking Table	HMS	Magnetic Separation
2000	24.07	34.96	-
1400	22.99	25.32	-
1180	22.63	38.25	-
850	16.51	24.96	17.71
650	15.52	26.56	19.10
500	18.26	25.72	22.03
425	21.06	23.20	17.72
300	20.38	22.72	19.31
212	16.53	24.54	25.51
180	16.23	24.1	13.98
<180	15.37	-	25.51

The low manganese content of concentrates obtained by magnetic separation can be attributed to contamination since the magnetic susceptibility of pyrolusite (1.0078 to 1.0088), haematite (1.008 to 1.024) and rutile (1.003 to 1.0053), and their middlings could have all been collected as paramagnetic. This was demonstrated by the effect of dissolved impurity of iron on ferruginous sphalerite (marmatite), in which the tiny inclusions of ferrous materials considerably influenced response to a magnetic field (Jain, 2008). However, it is possible that better liberation of manganese oxide was achieved with the fine fractions of 212 μm and 180 μm .

4.4 Assessments

Table 5 shows the results of the ratio of concentration after each concentration process. According to Wills and Napier-Munn (2006) the ratio measures the efficiency of concentration process and is closely related to the marketable content of the concentrate.

Table 5 Ratios of Concentration

Fraction (μm)	Shaking Table	HMS	Magnetic Separation
2000	1.22	1.71	-
1400	1.03	1.80	-
1180	1.06	2.30	-
850	1.04	3.34	3.32
650	1.01	3.16	2.73
500	1.01	3.53	4.92
425	1.11	2.94	2.48
300	1.10	2.49	2.00
212	1.13	2.46	4.96
180	1.17	3.56	2.84
<180	3.17	-	2.31

It can be seen that most of the values of shaking table were low except in the <180 μm fraction (3.17). The magnetic separation has higher ratios on two fractions (500 and 212 μm) compared with corresponding values from the other two tests. The heavy medium separation has higher ratios in eight fractions than its counterpart techniques.

These translated to collection of heavy particles closer in weight to the feed except in the finest fraction in the case of shaking table. Therefore, it has indicated that shaking table has virtually not done much to improve upon the feed content, because, "high ratio of concentration translates to increase in grade of concentrate" (Wills and Napier-Munn, 2006).

Table 6 is a representation of the enrichment ratios of the three different techniques employed vis-à-vis the head sample values. Heavy medium separation had ratios from a maximum of 1.84 to a minimum of 0.97. Enrichment ratios that are less than 1, indicate that the assay value of the feed is higher than that of the concentrate. The magnetic separation having maximum of 1.39 and minimum of 0.84 only suc-

ceeded in enriching three fractions. Shaking table's maximum of 1.09 and minimum of 0.65 had values that are slightly greater than the baseline in five fractions and less in six fractions.

Table 6 Enrichment Ratio

Fraction (μm)	Shaking Table	HMS	Magnetic Separation
2000	1.08	1.56	-
1400	1.06	1.17	-
1180	1.09	1.84	-
850	0.79	1.20	0.84
650	0.76	1.30	0.95
500	0.93	1.31	1.12
425	1.07	1.18	0.90
300	1.04	1.16	0.98
212	0.78	1.16	1.21
180	0.65	0.97	0.56
<180	0.84	-	1.39

Table 7 Recovery of Manganese Concentrate

Fraction (μm)	Shaking Table	HMS	Magnetic Separation
2000	88.34	91.41	-
1400	103.34	64.41	-
1180	102.53	80.15	-
850	76.03	35.89	25.59
650	75.28	40.82	34.23
500	91.90	37.07	22.80
425	96.00	39.96	36.22
300	94.05	46.44	49.23
212	69.37	47.07	24.34
180	55.53	27.17	19.79
<180	26.34	-	65.09

Finally, 2000 μm , 1400 μm , 1180 μm , 850 μm , 650 μm , 500 μm , 425 μm and 300 μm fractions were more enriched out of eleven fractions with HMS.

Table 7 represents the manganese distribution (recovery) which describes the amount of metal in the concentrate of each fraction in relation to that of the feed of the tailings;

Shaking table indicated higher recovery, followed by heavy medium separation and magnetic separation. For the concentrate obtained from the shaking table, some recovery values were very high and above 90%. The minimum of 21.17% was obtained for heavy medium separation. The magnetic separation values of 65.09% maximum and 19.79% minimum are the low values recorded.

Analysis of the recovery values obtained for the shaking table buttresses the assertion by Wills and Napier-Munn (2006) that recovery is independent of the ratio of concentration. Because, in the earlier determination of the latter, shaking table has the lowest value in comparison with the other two techniques. Therefore, its recovery values could be attributed to the higher weight collected in each fraction and hence, in multiplying with their respective assay yielded a greater metal content. Moreover, the relationship between recovery and grade (assay) is an inverse one (Wills and Napier-Munn, 2006).

Overall, Table 8 gives a summary of the results for each recovery process.

Table 8 Summary of the Results for each Recovery Process.

Method	Ratio of concentration	Enrichment ratio	Recovery, %
Shaking table	1.15	0.91	79.9
Magnetic separation	2.86	0.98	33.8
Heavy media separation	2.56	1.28	51.2

5 Conclusions

Comparative investigations were conducted with the aim of concentrating high grade manganese oxide mineral from the oxide tailing materials of the sorting spiral concentration plant at Ghana Manganese Company Limited, Nsuta. Mineralogical characterisation indicated that the manganese oxide content was 23.64% and the main gangue mineral was silica with composition 27.43 %. Other minerals included corundum, haematite and rutile. The particle size analysis of the material gave +850 µm as the discretely major constituent having 25.54% retained.

The recovery techniques tested were tabling, magnetic separation and heavy media separation. There was very good enrichment when heavy media of density 2.96 was used and the average ratio of concentration, enrichment ratio and recovery were 2.56, 1.28 and 51.2% respectively. The manganese content of the concentrates from the heavy media separation of between 38.3% and 22.7% Mn are suitable for ferroalloy industry. Thus there could be a business case for further processing of the manganese tailings.

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Authors



M. Ali is the current Head of Department Mineral Resources Engineering, Kaduna Polytechnic, Kaduna, Nigeria. He obtained HND in Mining Engineering (1991), PgD Mines Management (2001) and MTech. Mineral Processing Engineering (2003) from INPL, Nancy, France, PgD in Education (2010) from Ahmadu Bello University, Zaria, Nigeria and MPhil. Mineral Engineering (2011) from University of Mines and Technology, Tarkwa, Ghana. His current research interest include flotation and tailings enrichment. He is a member of Nigerian Mining and Geosciences Society, Nigerian Society of Mining Engineers.



Richard K Amankwah is an Associate Professor and Head of the Mineral Engineering Department, University of Mines and Technology, Tarkwa Ghana. He had his education at the Kwame Nkrumah University of Mines and Technology, Kumasi, Ghana, Technical University of Clausthal, Germany, and Queen's University at Kingston, Canada where he obtained a PhD in 2005. He is a mineral processing engineer, teacher and researcher, with ample knowledge of and hands-on experience in gold beneficiation, biohydrometallurgy, environmental biotechnology, ore testing, microwave processing and geometallurgy. His other research areas are mine-to-mill interface, education and eradication of Buruli ulcer and artisanal/small scale gold mining. He is a member of the Ghana Institution of Engineers, Society for Mining, Metallurgy and Exploration Inc and Chief Executive Officer of Process Innovations, a mineral consulting firm.