NIGERIAN MINING JOURNAL

ISSN 1117 - 4307

Volume 18 | Number 1 | November 2020

A Publication of

NIGERIAN SOCIETY OF MINING ENGINEERS

NIGERIAN MINING JOURNAL

Volume18 - Number 1 - November 2020

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NIGERIAN MINING JOURNAL

ISSN 1117-4307

Volume 18	Number 1	November 2020

A PUBLICATION OF THE NIGERIAN SOCIETY OF MINING ENGINEERS

Editor-in-Chief

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Publisher

Nigerian Society of Mining Engineers (NSME)
NSME Secretariat, Bukuru,
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NIGERIAN
MINING JOURNAL

ISSN 11174307

A Publication of
Nigerian Society of Mining Engineers

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Manuscripts: Manuscripts submitted for publication must represent original contributions and should not have been proposed for publication elsewhere. The papers should be based on original research, innovations and field experience in mineral exploration, mining, mineral processing, extractive metallurgy and equipment maintenance, relevant to the minerals industry. The manuscript must be prepared preferably in Microsoft Word environment, and should be submitted by email to *nigerianminingjournal@gmail.com*

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Unpublished work:
Arubisan, A.P. (2013). *Amelioration of grain size and its effect on iron ore concentrate in Itakpe, West Central Nigeria*. Unpublished M.Tech. Thesis, Federal University of Technology, Minna.

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Oyeladun, O.A.W, Abubakar, S. & Adewuyi, E.A.A. (2012). Application of Igoli mercury-free gold extraction for recovery of Wamba gold. *Proceedings of Nigerian Society of Mining Engineers,* Ilorin, 2012, 67-70.

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Optimization of Loading and Haulage of Materials in Selected Quarries in Jos Using Arena Simulation Software

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Abstract
This work made use of computer simulation software (Arena) for the optimization of the loading and haulage operation in three (3) quarries in Jos, Plateau State Nigeria. The loading and haulage process of the quarries were studied and the data generated were inputted in the computer simulating software (ARENA) used for the optimization; however, two models were used in the optimization process named the As-Is and To-Be models where As-Is model is the field generated data and the To-Be model is the simulation data from the software. The results generated from the As-Is model shows average unproductive time of each quarry to be 38.12%, 20.55% and 7.26%, average cycle time of operation are 38.83 min, 24.64 min and 7.26 min and the fuel consumption per cycle of operation 22.21, 12.01 and 5.32 litres respectively. After simulation of data, the results generated from the To-Be model revealed that the average unproductive time of each quarry to be 31.18%, 19.33% and 5.29%, average cycle time of operation are 28.48 min, 21.06 min and 15.12 min while the fuel consumption per cycle of operation was 19.01, 11.01, 4.24 litres respectively. The results can be compared to standard requirements for optimization of a quarry loading and haulage operations. The results can serve as guide for intending investors in decision making. It can also be used as supporting document when designing quarry layout.

Keywords: loading and haulage of material, cycle time of operation, computer simulation models, Arena

Introduction

Optimization is the process of obtaining the best result under given circumstances. In design, construction and maintenance of any engineering system, engineers have to take many technological and managerial decisions at several stages. The ultimate goal of all such decisions is either to minimize the effort required or to maximize the desired benefit. A number of optimization methods have been developed for solving different types of optimization problems.

Loading operation involves filling the dump trucks with rock fragments and haulage is the transportation from the mine face to either the stockpile or the crushing plant (Aladejare, 2008).

In surface mining, equipment selection problem involves in a way (about 90% of equipment capital costs and more than 70% of operating costs in open pit mines is for loading and haulage of materials) choosing a fleet of trucks and loaders that have the capacity to move the materials specified in the mine plan within a stipulated period (May, 2012). The optimization problem is to select these fleets in such a way that the overall cost of materials handling is reduce.

Generally, the scale of operations involves the purchase of single mining equipment that may cost several millions of dollars, but as productivity increases over time, the cost of operation outweighs the purchasing cost of the equipment. The equipment selection problem is often affected with a cascade of interdependent variables and parameters. For example, the cost of using a piece of equipment depends on its utilization, the availability and age of the equipment (Salama *et al*, 2015). The ultimate goal of mining operation is to provide the required amount of raw material needed by the community at reduced costs.

The aspect of the mining operation, which has the most influence on profit, is the cost of materials handling. This paper focuses on developing an oriented algorithm that can be used during the calculation of shovel productivity as well as fleet production management for a quarry.

The mining industry is currently faced with constantly increasing capital and operating costs, this has necessitated the need to reduce the operational costs where possible. Loading and Haulage costs are normal items to consider, as it can represent up to 50% of the total mining operating cost. (Bureau of mines, 2011).

Study Area

Jolex Construction Company, Eighteenth Engineering Construction and Afrimines Nigeria Limited are three (3) quarries located in Jos and Bassa Local Government area of Plateau State. Jos and Bassa are two local government area located in Plateau State, Nigeria between latitude 8^{o} and 10^{o}N, Longitude 7^{o} and 11^{o} E respectively (Plateau State Ministry for lands, survey and town planning). Jolex Construction Company is located in Mista Ali Bassa Local Government area of Plateau State while, Eighteenth Engineering Construction and Afrimines Nigeria limited are located in Bisichi and Shen both in Jos Local Government area of Plateau State.

The geology of the rock in Jolex Construction Company ranges between medium to small grain biotite granite, while that of Eighteenth Engineering Construction and Afrimines Nigeria Limited ranges between medium to small grain hornblende granite.

Cycle Time

This is defined as the time spent by any equipment to complete one cycle of operation. For a truck, cycle time includes the time to spot and load, travel to the dump site; manoeuvre, spotting, dump and return to the loading point, also predictable delays, unpredictable and wait times are included in the cycle (Lineberry, 1985).

Storage of loading times, travel time, queues and unloading times in a database are required in order to define the sequence of operation with the theories presented. Ercelebi and Bascetin (2009) based the theories on the proper calculation of trucks per shovel with the aim of decreasing the cost of material movement. Finally, different equations are presented whose parameters should be extracted from a database. Equation 1 shows the components of a typical cycle time in an open pit mine.

$$Cycle\,Time = lt + dt + qts + qtd + lht + eht \quad \ldots Equ.1$$

Where:

lt = load time
dt = dump time
qts = queuing time at the shovel
qtd = queuing time at the dump
lht = load haul time
eht = empty haul time

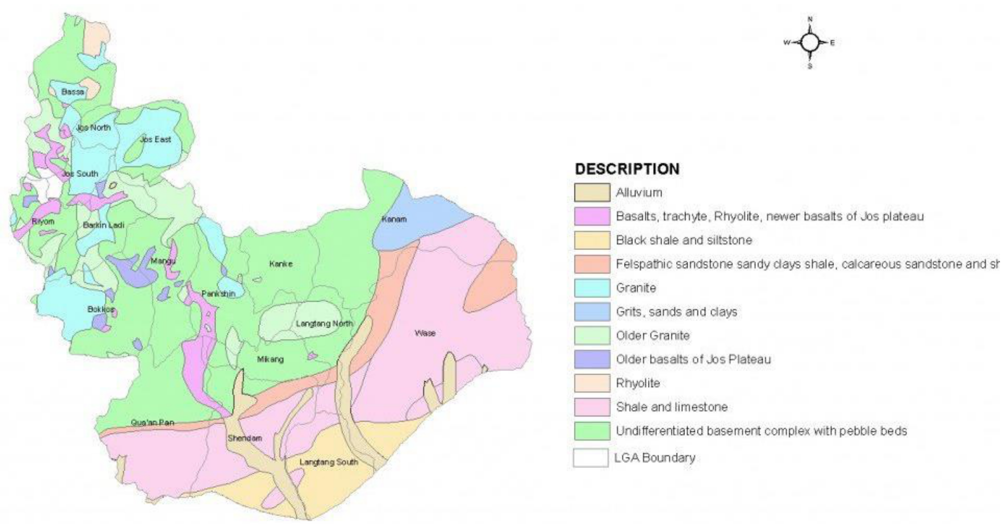

Figure 1: Geological map of Plateau State (http://www.researchgate.net)

Modeling Using Simulation Techniques

Mining operations are studied by experimentation. Simulations are used to represent real life operations in a model. This enables testing of different scenarios and their results can be analyzed for decision making. Kelton *et al.* (2007) defined that - Simulation refers to a broad collection of methods and applications to mimic the behaviour of real systems, usually on a computer with appropriate software.

For testing the efficiency of haulage and transportation processes, it is possible to use simulation software to recreate different configurations and scenarios, identifying variables that need to be adjusted. In doing this simulation, different algorithms are used; Yifei (2012) presents an algorithm to determine the best allocation of trucks to meet a particular grade with a stable production.

I. *As-Is Model:* Models are important to estimate how the behavior of operations in mining will be, understanding the operation, identifying problems and looking for the solutions. For loading and haulage of material in a quarry, An As-Is model is a prerequisite to understand how processes are executed in the current system. An As-Is model helps analysts as a significant model for understanding and for the improvement of the processes (Lodhi *et al*: 2010).

II. To-Be Model: The To-Be model is the result generated after incorporating the improvements found on the As-Is model. The To-Be model has the same structure of an As-Is model, therefore any adjustment can be applied directly without any modification in the original configuration (Tan et al: 2012). The application of the As-Is and To-Be modeling is currently part of the simulation of operations in mining.

Materials and Methods

In the quarries analyzed, hydraulic shovels are used for loading and excavation, and dump trucks for haulage of fragmented granite to the jaw crusher. Several "standard" loading and haulage operations have been monitored. Data were recorded in prepared forms, to be treated with data processing programs. The following have been carefully determined:

Spot at Loading: The time required for a truck, as soon as it arrives near the vicinity of the loader, to maneuver into a stopping position for loading.

Loading Time: The total time for the loader to load the bowl of the truck to its required payload.

Full Haul: The total travelling time for a loaded truck to reach the dump site from the loading site.

Spot at Dump: The time required for a truck, as soon as it arrives near the vicinity of the crusher, to maneuver into a stopping position for dumping.

Dumping Time: The total time spend by truck in dumping fragmented granite in crusher.

Empty Haul: Refers to the total travelling time to for an empty truck to reach the load site from the dump site.

Wait at Load: the total time an empty truck has to wait in line before it can maneuver into a position for loading (when two or more trucks are used).

Wait at Dump: The total time a loaded truck has to wait before it can maneuver into a position to dump (when two or more trucks are used).

Delays: The total time spend by a truck when traffic or crusher maintenance occurs in one cycle.

Productive Time: The total time the dump trucks are operating i.e. spot at load, loading time, full haul, spot at dump, dumping time and empty haul.

Unproductive Time: The total time the dump trucks are not operating i.e. wait at load, wait at dump and delays.

The information on the technical features of machineries has been obtained from manufacturer's technical literature: bucket capacity; installed power of the shovel; dump truck capacity; dump truck installed power which is enumerated in Table 1.

Table 1: Ranges of important features of the machines engaged in the sample sites.

Equipment	Power (kW)	Weight (tons)	Capacity (m^3)
Hydraulic excavator	80 - 230	19 - 50	1 – 1.5
Dump truck	166 - 447	20 - 38	10 - 34

It can be seen that the ranges of the machines employed in the sample sites are well within the general range. We can conclude that the sample sites are quite typical, as far as the fleet engaged is concerned. Note that for the purpose of this research, this plot will represent the tabulation of data from the field.

Table 2: Names of Quarry Used as Sample Site

Quarry 1	Jolex Construction Company
Quarry 2	Eighteenth Engineering Construction
Quarry 3	Afrimines Nigeria Limited

The distance between the loading/ dumping zone and the number of machineries involve in the operation which contributes to the increase of cycle time (due to queues) of operation. It is therefore paramount to highlight the distance between the loading/ dumping zone and the number of machinery involve in each quarry as shown in Table 3

Table 3: Number of machinery, distance between the pit and the crusher, and the maximum possible queue.

EQUIPMENT	QUARRY 1	QUARRY 2	QUARRY 3
Distance of haul road (KM)	0.67	0.54	0.46
Number of Hydraulic Excavator	1	2	1
Number of Dump Trucks	3	2	1
Maximum Queues	2	1	0

Development of the Model

A simulator software (Arena) is utilized to produce a model of the haulage process, which includes the transportation of material from the mine faces to different destinations and prepared to adapt to varying mine layouts. The model is an abstraction of a real quarry operations which is represented as logic blocks by the software. Some blocks were grouped to simplify the model so as to be able to get a simulation. The sub-activities for the Operations are represented in Figures 2-7:

A. Arrival of Trucks

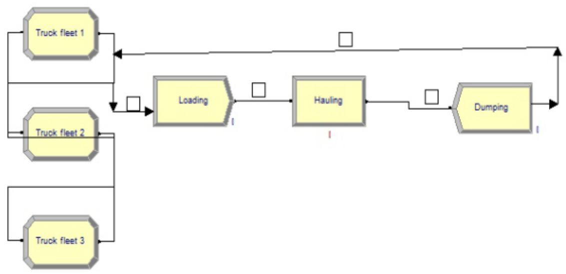

Figure 2: Sub-model sequence for simulating arrival of trucks.

B. Load Cycle in Loading Zones

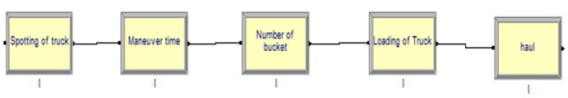

Figure 3: Sub-model for simulating loading zones.

C. Haulage of Material to Dumping Zone

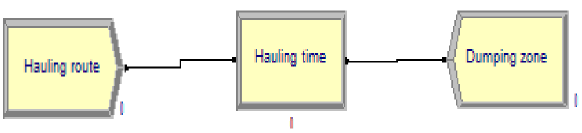

Figure 4: Sub-model sequence for simulating haulage of material.

D. Discharge of Material

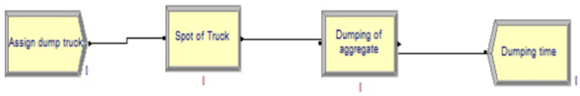

Figure 5: Sub-model sequence for simulating material discharge.

E. Return to Load Zone

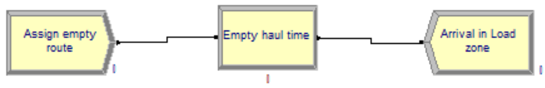

Figure 6: Sub-model sequence for returning to load zone.

F. Delays

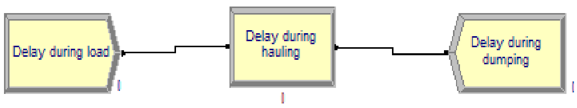

Figure 7: Sub-model sequence for delays.

Optimization of Quarry Profitability

In other to show how optimization of loading and haulage of material can increase quarry profitability, data was collected for diesel consumption (Litres) for each machine and the fragmented granite hauled (Tons) in the process. The machines fuel consumption was calculated based on their consumption per hour while the hauled materials were obtained from the tonnage per cycle. Their consumption and material hauled will be tabulated base on the As-Is and To-Be model cycle time of operation obtained from Table 3. Equations 2, 3, 4, 5, and 6 are used to calculate profitability prediction.

Calculation for productivity of a quarry;

Dump truck fuel consumption:

$$\text{cycle time} = \frac{\sum \text{truck circle time (spot time load + full haul + spot time dump + empty haul)}}{60} \quad \text{... Equ. 2}$$

$$\text{fuel consumption per cycle} = \text{cycle time} \times \text{truck fuel consumption per Hr} \quad \text{... Equ. 3}$$

Hydraulic excavator fuel consumption:

$$cycle\ time = \frac{\sum loading\ time}{60} \quad ...Equ. 4$$

$$\text{Fuel consumption per cycle} = \text{cycle time} \times \text{excavator fuel consumtion per Hr} \quad \text{... Equ. 5}$$

Material hauled per cycle:

$$\sum truck\ capacity\ (tons) \quad \times No\ of\ Trucks \quad ...Equ. 6$$

RESULTS AND DISCUSSION

The evaluation was made for each quarry according to their quarry plan, mode of operations, the total number of trucks and excavators involve in the operation. In order to compare results, every case contains two models: As-Is and To-Be. They represent the quarries in their existing (current) status, and the optimized (future) status respectively. The results obtained were cycle times in minutes. Each cycle of activities was categorized in productive time, unproductive time, and total time, with their corresponding sub-activities, Tables 4-6 show the cycle times generated as As-Is model from data of quarries 1 to 3 respectively".

Table 4: Cycle times generated for As-Is model from Quarry 1 data

Activity	Sub Activity	Truck Fleet 1	%	Truck Fleet 2	%	Truck Fleet 3	%
Productive time	**Spot Time Load**	2.17	5.83	2.19	6.18	2.06	4.49
	Load Time	2.26	6.32	2.30	6.50	3.10	6.76
	Full Haul	9.04	25.7	8.74	24.7	10.36	22.59
	Spot Time Dump	1.20	3.41	1.05	2.97	1.64	3.38
	Dump Time	4.15	11.8	3.18	8.98	5.46	11.91
	Empty Haul	4.17	11.8	3.90	11.01	4.90	10.68
Unproductive Time	**Wait at Load**	2.9	8.23	3.20	9.03	6.32	13.78
	Wait to Dump	7.7	21.9	8.18	23.1	10.62	23.16
	Delays	1.64	4.66	1.67	4.71	1.40	3.05
Total	**Productive Time**	22.99	65.3	21.36	60.32	27.52	60.01
	Unproductive Time	12.24	34.7	14.05	39.68	18.34	39.99
	Cycle Time	35.23		35.41		45.86	

Table 5: Cycle times generated for As-Is model from Quarry 2 data

Activity	Sub Activity	Truck Fleet 1	%	Truck Fleet 2	%
Productive time	**Spot Time Load**	1.70	6.9	1.76	7.07
	Load Time	3.17	13.0	2.90	11.66
	Full Haul	6.36	26.07	6.70	26.93
	Spot Time Dump	2.30	9.43	2.10	8.44
	Dump Time	3.14	12.87	2.90	11.66
	Empty Haul	2.94	12.05	3.18	12.78
Unproductive Time	**Wait at Load**	-	-	-	-
	Wait to Dump	3.42	14.02	4.10	16.48
	Delays	1.36	5.58	1.24	4.98
Total	**Productive Time**	19.61	80.4	19.54	78.50
	Unproductive Time	4.78	19.6	5.34	21.50
	Cycle Time	24.39		24.88	

Table 6: Cycle times generated for As-Is model from Quarry 3 data

Activity	Sub Activity	Truck Fleet 1	%
Productive time	**Spot Time Load**	1.92	11.42
	Load Time	3.20	19.05
	Full Haul	5.06	30.12
	Spot Time Dump	1.10	6.55
	Dump Time	2.20	13.1
	Empty Haul	2.30	13.69
Unproductive Time	**Wait at Load**	-	-
	Wait to Dump	-	-
	Delays	1.22	7.26
Total	**Productive Time**	15.78	93.93
	Unproductive Time	1.22	7.26
	Cycle Time	16.80	

The goal of this research is to monitor the behaviour of the cycle time during the operation for optimization. Figures 8–10 show the chart of different sub-activities of the total cycle per truck in quarries 1-3. The difference between each sub-activity is displayed, which allows comparing the performance of each truck during the cycle time. Optimization opportunity was identified, but Quarries 1 and 2 have similar sub activity but different from

Quarry 3, because of one truck to one loader present in Quarry 3 —Wait to Dump. It means that every truck spends several minutes before dumping. It can be seen that the time

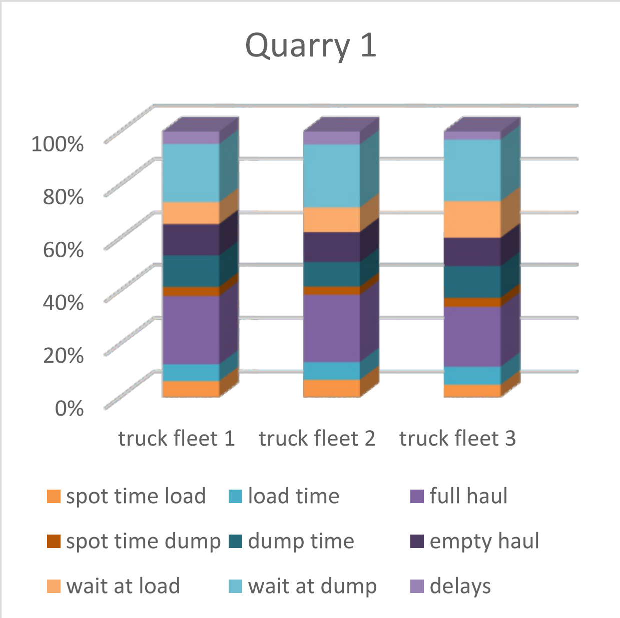

Figure 8: Total Cycle Times Per Truck in As-Is model

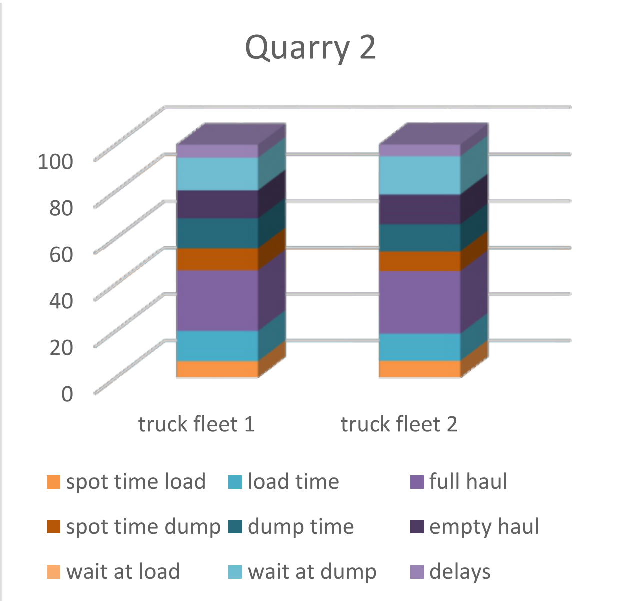

Figure 9: Total Cycle Times Per Truck in As-Is Model

spent in this sub-activity compared to —Full Haul was virtually the same for Quarry 1, 2 and 3.

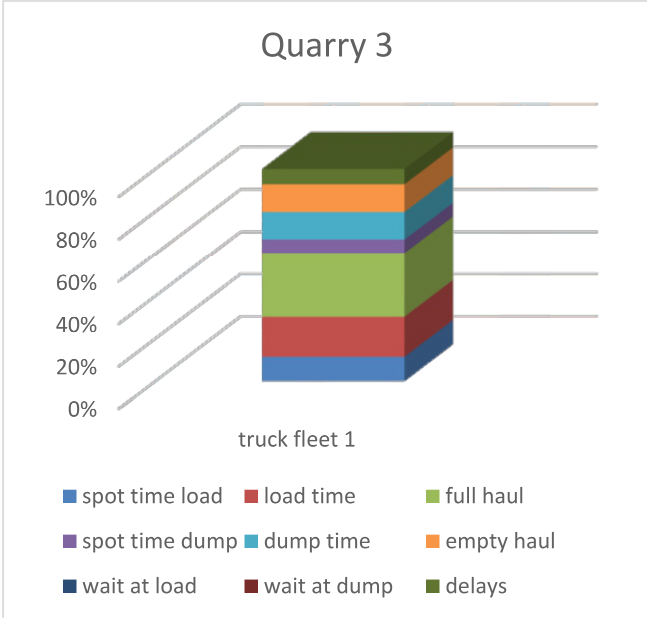

Figure 10: Total Cycle Times Per Truck in As-Is model

The Stimulation methodology was applied when trucks were hauling fragments to the crusher. Quarry 1, the —Wait to Dump time for the fleet was 22.5% of the total time. With this value it was confirmed that trucks wait too long before discharging boulders to the crusher. The next alert was detected in —Wait at Load with 10.34% over the total cycle time. As regard this, the variation of individual truck time versus the truck fleet is not significant, thereby approximately matching the fleet average.

Quarry 2, the —Wait to Dump time for the fleet was down to 15.25% of the total time accounting for the reduction in trucks availability. With this value it was confirmed that trucks waiting to discharging boulders to the crusher was minimized. The next alert was detected in —wait at load with 0% over the total cycle time. Therefore, the variation of time from individual trucks versus the truck fleet was significantly much, since there was no queue in the loading area (one loader to one truck).

Quarry 3, the —full haul time for the fleet was 30.12% of the total time. With this value, it was confirmed that trucks spend too much time taking boulders to the crusher, therefore deserve serious attention when considering optimization.

The comparison of the different time for the sub-activities was compared to the productive and unproductive times, the results is as follows. The As-Is model indicated that the unproductive time for Quarries 1, 2 and 3 are 36.3%, 36.7% and 20% of the total cycle time respectively, allowing identifying potential opportunities for improving the haulage. Tables 7-9 show profitability prediction for As-Is and To-Be for quarries 1 to 3

Table 7: Profitability prediction for As-Is and To-Be - Quarry 1

Equipment	Measured Values	
Hydraulic Excavator (Liters/Hr)	22	
Dump Trucks (Liters/Hr)	Truck fleet 1	23
	Truck fleet 2	23
	Truck fleet 3	23
Hauled Material (Tons/cycle)	Truck fleet 1	16.63
	Truck fleet 2	14.12
	Truck fleet 3	12.40

Table 8: Profitability prediction for As-Is and To-Be - Quarry 2

Equipment	Measured Values	
Hydraulic Excavator (Liters)	Excavator 1	20
	Excavator 2	20
Dump Trucks (Liters)	Truck fleet 1	24
	Truck fleet 2	24
Hauled Material (Tons)	Truck fleet 1	11.25
	Truck fleet 2	11.25

Table 9: Profitability prediction for As-Is and To-Be - Quarry 3

Equipment	Measured Values
Hydraulic Excavator (Liters)	22
Dump Trucks (Liters)	24
Hauled Material (Tons)	11.25

Table 10: Cost of material handling of each Quarry generated from As-Is model

Name of Quarry	Material hauled (Tons)	Cost of Hauling (Naira)
QUARRY 1	43.15	5,627.5
QUARRY 2	22.50	3,002.5
QUARRY 3	11.25	1,330.5

The current price of diesel in Nigeria is #250 per litre which infers that the total cost of moving material from blasting pit (material handling) to the crusher per circle of operation is as shown in Table 11.

The next step was finding the solutions based on the experience and predefined situations. An analysis of the different weaknesses was made to find the root cause, and the solutions found were adapted to the reality of the operations. The actions taken for remediation of these weaknesses are presented in detail:

a) Improvement for spot time load and loading time.
b) Improvement for haul full time and empty road time.
c) Improvement for waiting dump time.
d) Defining maximum number of queue

Table 11: Cycle Time Generated by To-Be Model for Quarry 1

Activity	Sub Activity	Truck Fleet 1	%	Truck Fleet 2	%	Truck Fleet 3	%
Productive time	**Spot Time Load**	1.87	6.96	1.89	7.06	1.82	5.72
	Load Time	2.10	7.82	2.15	8.04	2.75	8.64
	Full Haul	7.80	29.03	7.30	27.30	8.10	25.44
	Spot Time Dump	1.20	4.47	1.05	3.93	1.40	4.40
	Dump Time	2.95	10.98	2.50	9.35	3.40	10.68
	Empty Haul	3.40	12.65	3.10	11.59	3.95	12.41
Unproductive Time	**Wait at Load**	2.40	8.93	2.80	10.47	4.20	13.19
	Wait to Dump	4.10	15.26	4.90	18.32	5.20	16.33
	Delays	1.05	3.91	1.05	3.93	1.02	3.20
Total	**Productive Time**	19.32	71.90	17.99	67.28	21.42	67.27
	Unproductive Time	7.55	28.10	8.75	32.72	10.42	32.73
	Cycle Time	26.87		26.74		31.84	

Table 12: CYCLE TIME GENERATED BY TO-BE MODEL for QUARRY 2

Activity	Sub Activity	Truck Fleet 1	%	Truck Fleet 2	%
Productive time	Spot Time Load	1.40	6.69	1.46	6.89
	Load Time	2.80	13.37	2.65	12.51
	Full Haul	5.10	24.36	5.16	24.36
	Spot Time Dump	1.90	9.07	1.85	8.73
	Dump Time	2.80	13.37	2.74	12.94
	Empty Haul	2.94	14.04	3.18	15.01
Unproductive Time	Wait at Load	-	-	-	-
	Wait to Dump	2.90	13.85	3.20	15.11
	Delays	1.10	5.25	1.94	4.44
Total	Productive Time	16.94	80.90	17.04	80.45
	Unproductive Time	4.00	19.10	4.14	19.55
	Cycle Time	20.94		21.18	

Table 13: CYCLE TIME GENERATED BY TO-BE MODEL for QUARRY 3

Activity	Sub Activity	Truck Fleet 1	%
Productive time	**Spot Time Load**	1.40	9.26
	Load Time	2.90	19.18
	Full Haul	4.84	32.01
	Spot Time Dump	0.90	5.95
	Dump Time	1.98	13.10
	Empty Haul	2.30	15.21
Unproductive Time	**Wait at Load**	-	-
	Wait to Dump	-	-
	Delays	0.80	5.29
Total	**Productive Time**	14.32	94.71
	Unproductive Time	0.80	5.29
	Cycle Time	15.12	

In other to compare the As-Is and To-Be models, the quarry with more than one loader to one dump truck was calculate as average to represent one column in the chart. Figure 11, 12 and 13 show comparisons of the different cycle times between As-Is and To-Be models where there is an improvement in reducing unproductive time, productive time, and total cycle time for each section respectively. The average for the quarry 1 and 2 was taken for As-Is and To-Be models

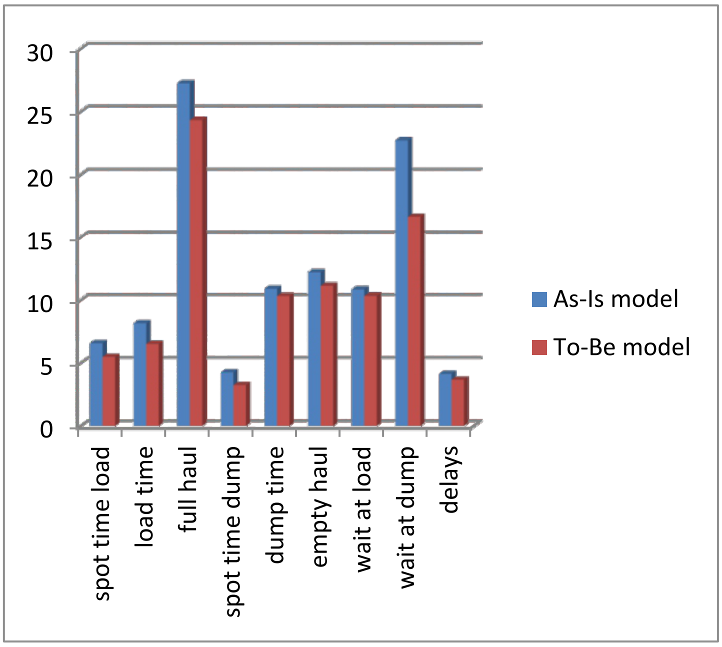

Figure 11: Cycle Time in Quarry 1- As-Is vs. To-Be Model

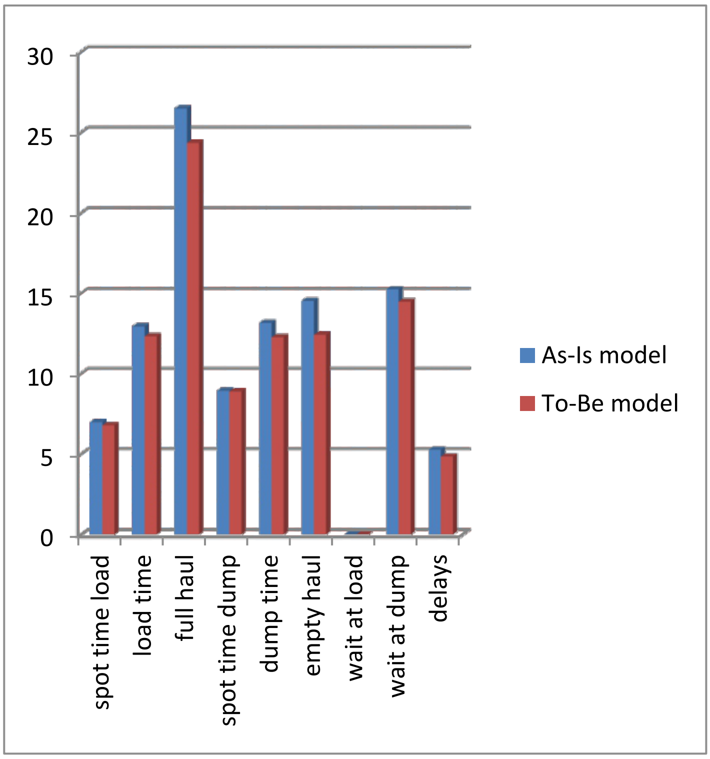

Figure 12: Cycle Time in Quarry 2- As-Is vs To-Be Model

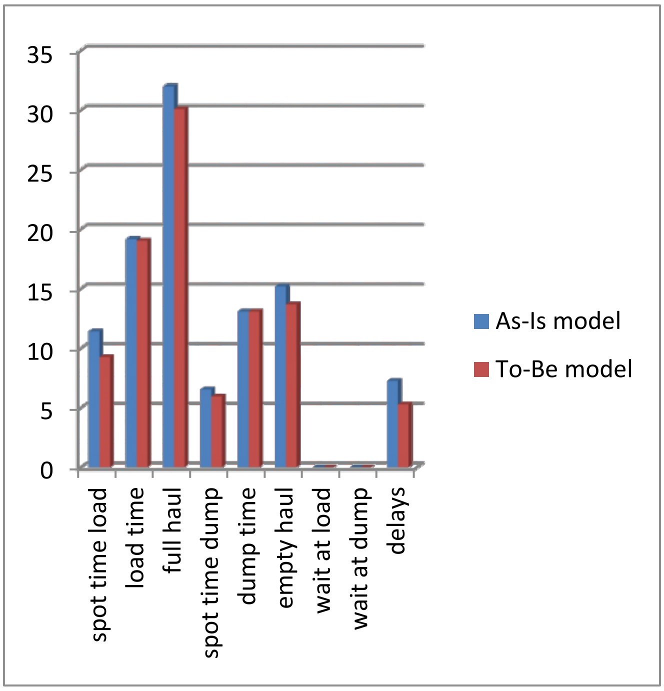

Figure 13: Cycle Time in Quarry 3- As-Is vs. To-Be Model

After completing the above corrections, several alternative scenarios are proposed in order to correct the behavior of production, utilization, and queues by varying the number of trucks and loader.

With the execution of the To-Be model it can therefore be said that the sub-activities with optimization potential are wait to load, wait to dump and full haul which affect the productive, unproductive and cycle times. From the above analysis it can be concluded that the one loader and one truck is the best optimization scenario for loading and haulage of material in a quarry since the only unproductive time present is the delays, which can be eradicated with good spotting and quarry trafficking.

Quarry Productivity Under Peak Operation

After running the simulation of the cycle time of operation of the Quarries, the optimization of profitability was achieved. As shown in the Table 14;

Table 14: Cost of Material Handling of Each Quarry Generated from To-Be Model

Name of Quarry	Material hauled (Tons)	Cost Of Hauling (Naira)
QUARRY 1	43.15	4,752.5
QUARRY 2	22.5	2,752.5
QUARRY 3	11.25	1,059.4

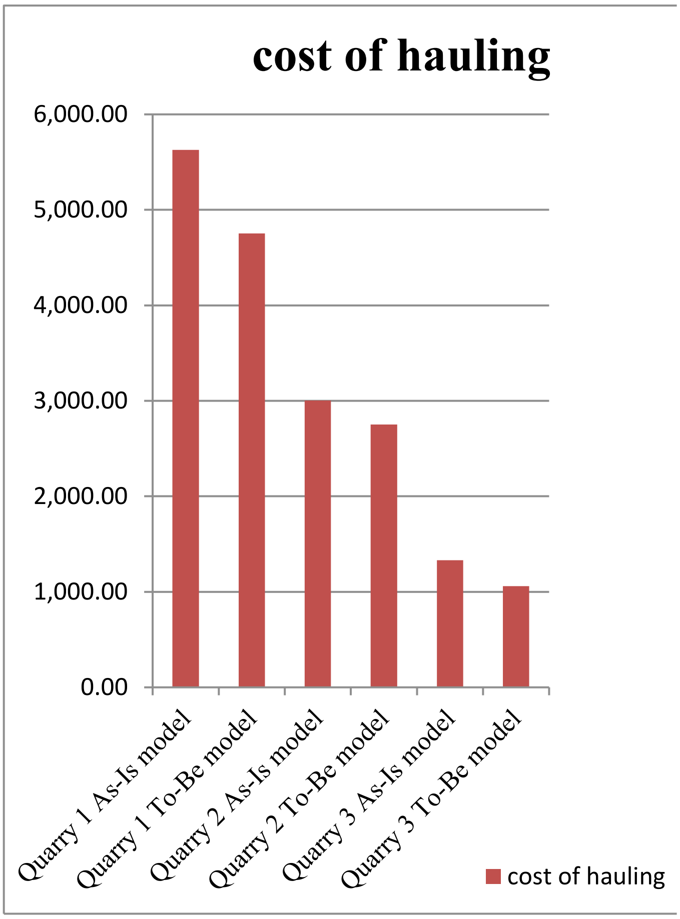

FIGURE 19: COST OF MATERIAL OF AS-IS VS TO-BE MODELS

From the above analysis it has been confirmed that profitability of a quarry loading and hauling can be maximized through optimization.

Conclusion

This research studied the loading and hauling operation in three (3) quarries for optimization. Arena simulation software was used for the optimization process. before the optimization, the unproductive time of quarries 1-3 were 38.12%, 20.55% and 7.26%, average cycle time of operation 38.83min, 24.64min and 7.26min and the fuel consumption per cycle of operation 22.21, 12.01 and 5.32 litres respectively. After simulation of data, the results generated from the To-Be model shows that the average unproductive time of each quarry to be 31.18%, 19.33% and 5.29%, average cycle time of operation 28.48min, 21.06min and 15.12min while the fuel consumption per cycle of operation was 19.01, 11.01, 4.24 litres respectively. The results can be compared to standard requirements for optimization of a quarry loading and haulage operations. The results from this investigation can guide intending investors in their decision making. After using computer simulation software to analyze discreet event in the quarry, it has been statistically proven that unproductive time can be brought to its minimum; it can therefore be concluded by saying that the effective utilization of a quarry loading and hauling system is dependent on loading time, haul full time and dumping time. Also, computer simulation software is said to be a very useful to quarry decision makers because it allows the alternation of loading and hauling process until optimization is achieved. It can also be noted that poor quarry planning such as bad positioning of crusher, haul road maintenance and maneuvering contribute significantly to the increase in operation cycle time and subsequently reduce productivity and profitability.

Acknowledgements
The authors wish to acknowledge the management of Jolex Construction Company Mista Ali, Eighteenth Engineering Construction Shen and Afrimines Nigeria Limited for the permission to use their quarry as a case study. Also the contribution of Engr. Clement Jonah and Engr. Monday Sunday Quarry managers of Jolex Construction Company and Eighteenth Engineering Construction company is highly appreciated.

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Groundwater Prospecting and Lithology Description in Kreigani Town, Southern Nigeria, Using Vertical Electrical Sounding

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Abstract
Electrical resistivity survey was carried out for groundwater potentials and subsurface lithology in Kreigani town, Ogba/Egbema/Ndoni L.G.A of Rivers State. Vertical electrical sounding method with Schlumberger array using ABEM SAS 1000 Terrameter was employed to delineate the variation in the lithologic distribution and depths to aquiferous formations in the area. Five (5) vertical electrical sounding were carried out with a maximum spread of 300m. The VES raw data obtained from the field was improved with the use of WINRESIST computer software. VES 1,2 and 3 revealed five (5) lithologies of curve type AAA beginning with top soil, followed by fine sand and the third is medium grained sand. The fourth and fifth layers observed in VES 1,2 and 3 are aquiferous, composed of coarse sand with resistivities between 568.3Ωm and 1541Ωm and depths between 45.2m to infinity. VES 4 showed six (6) lithologies of curve type QQHA while VES 5 also showed six (6) lithologies of curve type AAKH. The first to third layers from the fourth and fifth VES are top soil, followed by fine sand and medium grain sand. The fifth layers in VES 4 and 5 are aquiferous, composed of coarse sand with resistivties of 1026Ωm and 1081Ωm and thickness of 25.9m and 30.5m respectively. The 6th layers are coarse sand to infinity depth in both VES 4 and 5. A recommendation of an average depth of 46m is to be drilled for all boreholes in the study area to yield potable and less vulnerable groundwater.

Keywords: Geoelectric layer. Resistivity survey. Lithology. Groundwater. Borehole.

Background of Study

The improvement of technology has made the quest for water for all purpose in life to drift from ordinary search for surface water to the prospecting for steady and reliable subsurface water, groundwater, from boreholes. In Nigeria presently, boreholes have rescued the citizenry from acute shortage of water.

Groundwater occurrence in an aquifer is characterized by some certain parameters (such as aquifer composition and water quality) which are determined by geophysical methods such as electrical resistivity method, seismic method, magnetic method, gravity method among others. The use of electrical resistivity method in geophysical exploration for groundwater in a sedimentary environment has proven reliable (Emenike, 2001). Records show that the depths of aquifers differ from place to place because of variation in lithologic occurrence and distribution (Okwueze, 1996). Kreigani town in Ogba/Egbema/Ndoni local government area of Rivers State have witnessed increase in oil exploration activities. This is likely to have effect on groundwater in the area by the infiltration of contaminants from the surface of the earth to the water table. In order to determine if these activities (such as oil spillage, gas flaring, effluent discharge etc.) have impacted groundwater in the areas in terms of pollution, this research was carried out. Electrical resistivity method was used for this study because of its relatively low cost compared to other geophysical methods. Electrical resistivity method is one of the most useful techniques in searching for underground water.

tributaries and creeks. Five major geomorphic units have been recognized in the Niger Delta (Akpokodje 2001) namely:

a) Dry flatland and plains.
b) Sombreo Warn deltaic plains with abundant fresh water back swamps.
c) Fresh water swamps, meander belts and alluvial swamps.
d) Salt water or mangrove swamps.
e) Active/abandoned coastal ridges.

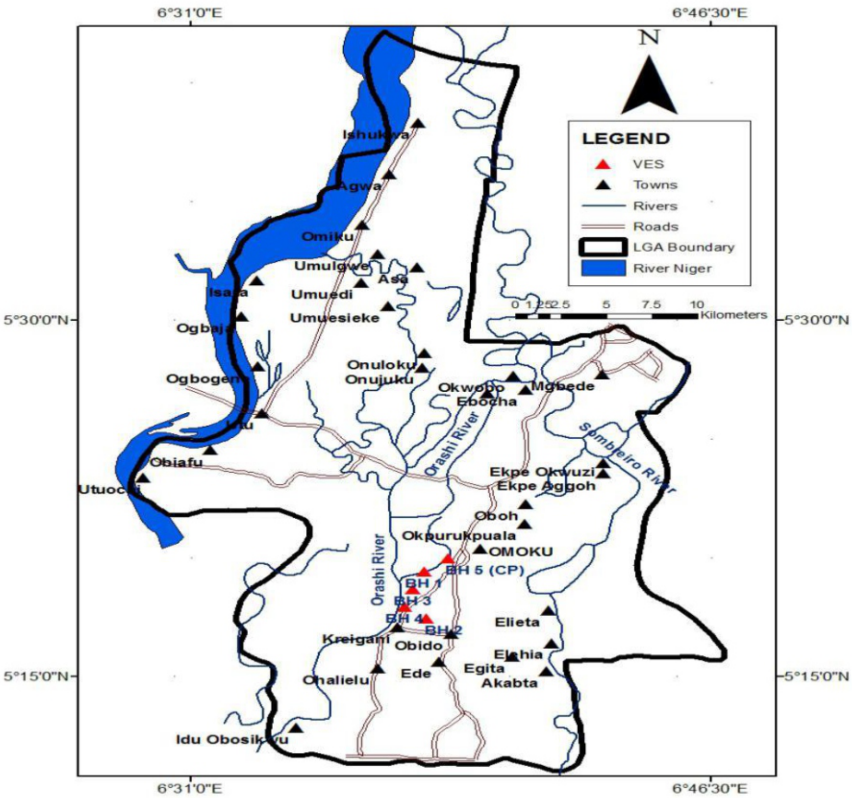

Figure 2: Map of Study Location showing VES lines

Method of Study

Vertical Electrical Sounding (VES) technique of electrical resistivity method in geophysical exploration was employed for this study using the Schlumberger electrode configuration. Five (5) of VES points were carried out cutting across the study area.

The equipment used was Abem Terrameter SAS 1000C. While sounding, the four (4) electrodes were arranged along a straight line with the potential electrodes placed between the current electrodes and all four were on a straight line, disposed with respect to the center of the configuration at each spreading. The current electrode spacing was constant on either side by AB/2 (horizontal distance between the two electrodes).

Electrical current (I) was applied to A and B electrodes and potential was measured between M and N electrodes. The geometric factor (K) was obtained for four electrode arrays of AMNB configuration as:

$K = [(AB/2)2 - (MN/2)2/MN]^{\pi}$ ..Equ 1 (geometric factor)

where $\frac{AB}{2} =$ half current electrode spacing

$MN/2 =$ half potential electrode spacing.

The VES array consists of the series of the electrode combinations AMNB with gradually increasing distances among the electrodes for consequent combinations.

The depth of sounding increases with distance between A and B electrodes. The K factor for the combinations calculated using Equation 1 above. From this field work, five (5) VES sounding were done on the field. The plots obtained from the resistivity data were examined and their character noted in terms of the pattern of resistivity with depths of VES or laterally in profile data. The areas of high resistivity were noted, described and attributed to the presence or absence of conducting bodies below the surface at the point or area of the observation of the anomalies.

For VES, the types of curve (Fig.3) obtained were noted in terms of:

A - Type: Continuous increase of resistivity with depth i.e. p1< p2 < p3

Q - Type: Continuous decrease of resistivity with depth i.e. p1 > p2 > p3

H- Type: 3 layers in which p1 > p2 < p3

K - Type: 3 layers in which p1 < p2 <p3

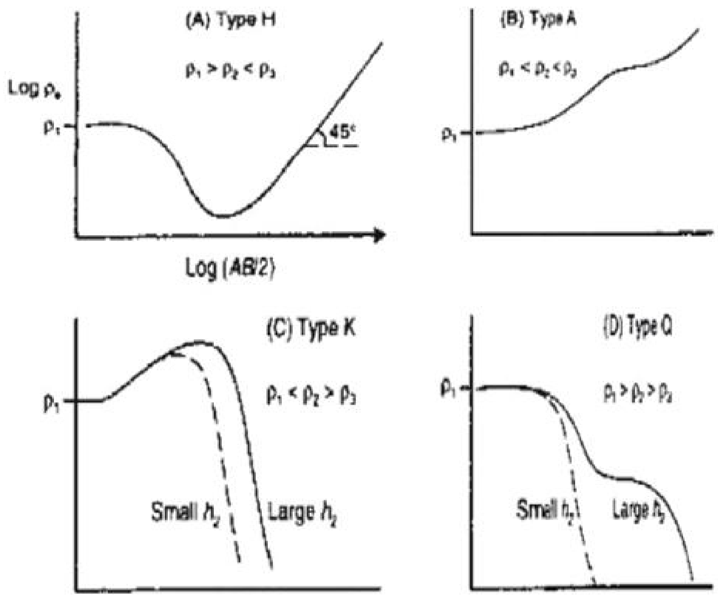

Figure 3: Graphical representation of VES curve types, (USEPA, archive, 2016)

Results and Discussions

The geo-electrical data obtained from VES 1 (Fig.4) revealed that the depth delineated is composed five (5) lithologic secession (Fig 9) of curve type AAA with topsoil to a depth of 1.8m, 2nd layer; fine sand with strata thickness of 3.2m, 3rd layer; medium sand with thickness of 11.4m, 4th layer; coarse sand with thickness of 29.5m, 5th layer, coarse sand to an infinity depth. The 4th layer is observed to be the first aquifer with resistivity of 568.3Ωm and depth of 45.6m (Table1). VES 2 revealed five (5) lithologies of curve type AAA (Fig. 5) with topsoil to a depth of 1.8m, 2nd layer; fine sand with thickness of 3m, 3rd layer; medium sand with thickness of 12m, 4th layer; coarse sand with thickness of 28m, 5th layer coarse sand to infinity depth. The 4th layer is observed to be the first aquifer with resistivity of 9871Ωm at the depth of 45.2m (Table 2). VES 3 revealed five (5) lithologies of curve type AAA (Fig. 6) with topsoil to a depth of 2.6m, 2nd layer; fine sand with thickness of 3.3m, 3rd layer; medium sand with thickness of 9.5m, 4th layer; coarse sand with thickness

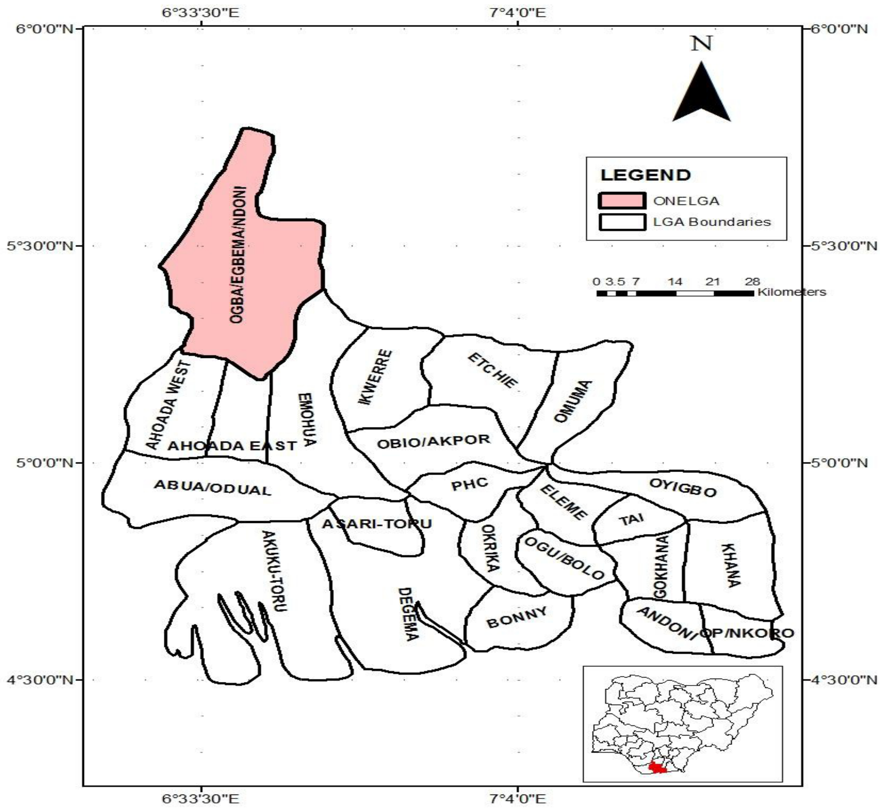

Figure 1: Map of Rivers State showing Ogba/Egbema/Ndoni L.G.A (Kekwaru and Nwankwoala, 2018)

Study Location

The study area is Kreigani Towns, located in Ogba/Egbema/Ndoni L.G.A., Rivers State (Figure 1). The area lies within Latitude 05^0 $18^{'}$ $58^{''}$ – 05^0 $19^{'}$ $24^{''}$ N and Longitude 006^0 $38^{'}$ $16^{''}$ – 006^0 $37^{'}$ $57^{''}$ E with elevation of 6m above sea level. The area is accessible through Port Harcourt – Omoku Road (Figure 2).

Brief Geology

Geologically, the study area lies within the Niger Delta and generally has a dry flatland surface and shallow water table level. The water table depth decreases seawards, varying from about 5m (inland) to 0.5m at the coast.

The shallow water table level in the Niger Delta is as a result of the high rainfall, general swampiness and flat topography of the area. These features of the Niger Delta also account for the higher moisture content of soil in the region, affecting the geotechnical properties of its soils negatively and causing serious drainage problem in the region.

The three main depositional environments of most deltaic environment (marine, mixed and continental) are observed in the Niger Delta and termed the Akata, Agbada and Benin Formations. This have been confirmed by studies carried out by Nedeco (1959); Allen (1956); Reyment (1965); Short and Stauble (1967); Maron (1969); Weber (1971); Burke (1972); Kogbe (1989) and Etu-Efeotor (1997).

Geomorphologically, the Niger Delta has a flat land topography. Its terrain is drained and crisscrossed by numerous rivers, of 31.3m, 5th layer coarse sand to infinity depth. The 4th layer is observed to be the first aquifer with resistivity of 1541Ωm at the depth of 46.7m (Table 3). VES 4 revealed six (6) lithologies of curve type QQHA (Fig. 7) with topsoil to a depth of 1.7m, 2nd layer; medium sand with thickness of 2.2m, 3rd layer; medium sand with thickness of 4.5m, 4th layer; fine sand with thickness of 11.7m, 5th layer coarse sand with thickness of 25.9m, 6th layer; coarse sand to infinity depth. The 5th layer is observed to be the first aquifer with resistivity of 1026Ωm at the depth of 46.3m (Table 4). VES 5 revealed six (6) lithologies of curve type AAKH (Fig. 8) with topsoil to a depth of 1.7m, 2nd layer; fine sand with thickness of 2.2m, 3rd layer; medium sand with thickness of 3.9m, 4th layer; medium sand with thickness of 8m, 5th layer coarse sand with thickness of 30.5m, 6th layer; coarse sand to infinity depth. The 5th layer is observed to be the first aquifer with resistivity of 1081Ωm at the depth of 46.3m (Table 5).

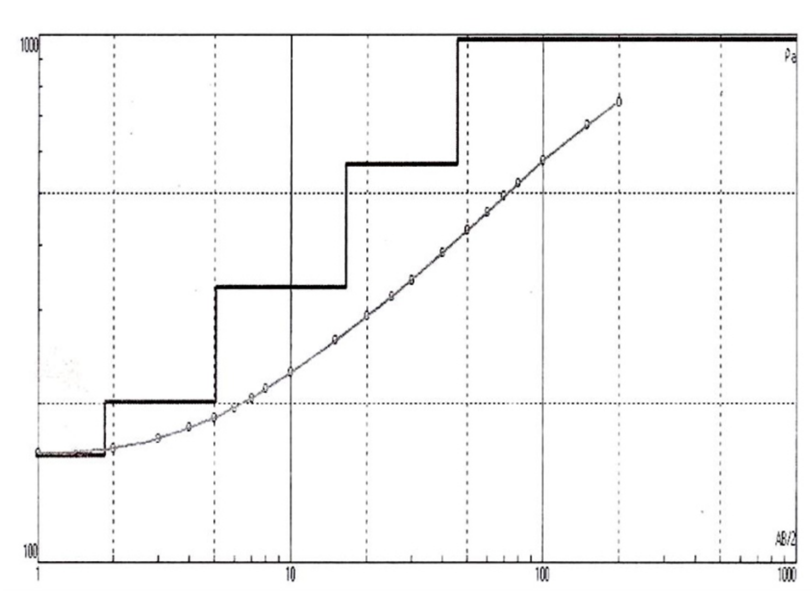

Figure 4: VES curve for Station 1.

Table 1: Geoelectric Parameters and Layer Interpretation for VES 1.

Layer	Resistivity (Ωm)	Thickness (m)	Depth (m)	Geological Interpretation
1	159.40	1.84	1.84	Top soil
2	201.20	3.24	5.07	Sand
3	332.30	11.44	16.51	Sand
4	568.30	29.15	45.66	Sand pressure aquifer
5	979.00	Infinity	Infinity	Aquifer

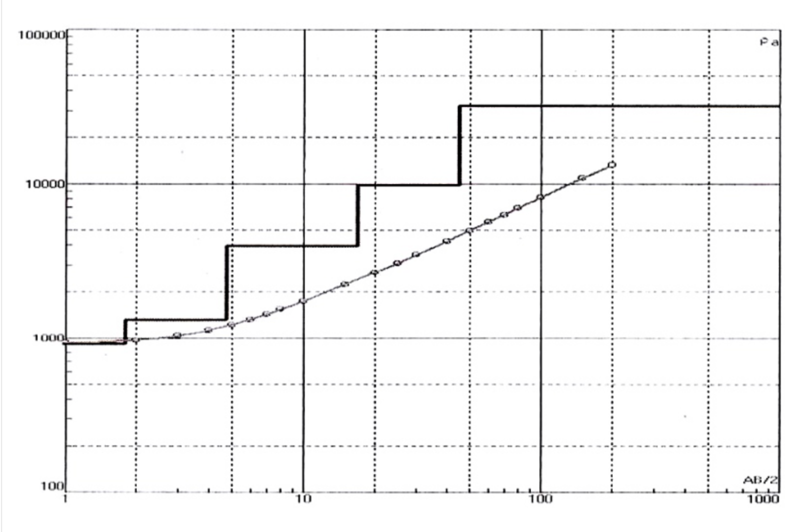

Figure 5: VES curve for Station 2

Table 2**:** Geoelectric Parameters and Layer Interpretation for VES 2

Layer	**Resistivity (Ωm)**	**Thickness (m)**	**Depth (m)**	**Geological Interpretation**
1	913.00	1.79	1.79	Top soil
2	1314.00	2.97	4.76	Sand
3	3971.00	12.07	16.83	Sand
4	9871.00	28.39	45.22	Sand (pressure aquifer
5	32247.00	Infinity	Infinity	Sand (aquifer)

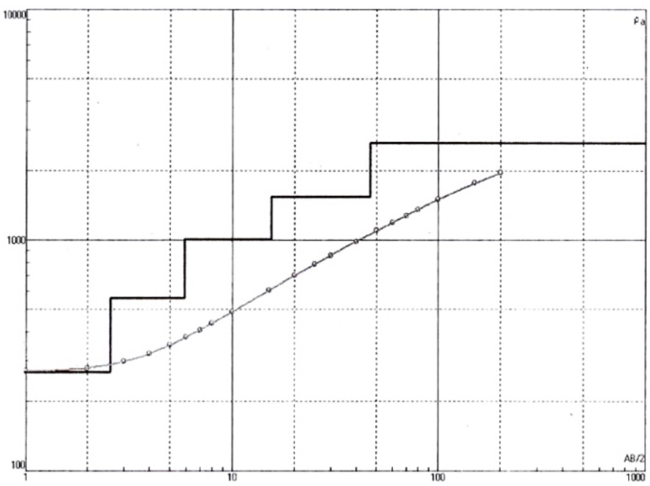

Figure 6: VES curve for Station 3

Table 3: Geoelectric Parameters and Layer Interpretation for VES 3

Layer	Resistivity (Ωm)	Thickness (m)	Depth (m)	Geological Interpretation
1	268.60	2.59	2.59	Top soil
2	557.70	3.30	5.89	Sand (dry)
3	1005.00	9.53	15.42	Sand
4	1541.00	31.27	46.69	Sand (pressure aquifer)
5	2641.00	Infinity	Infinity	Sand (Aquifer)

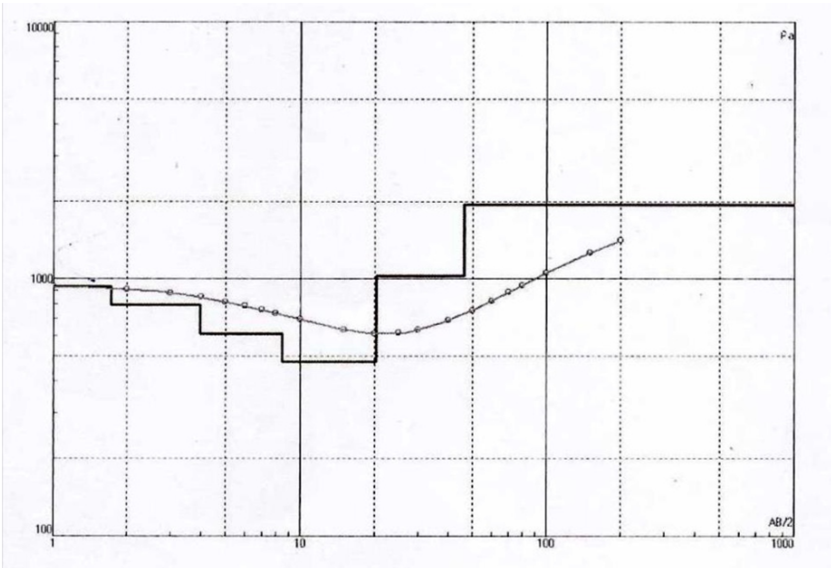

Figure 7: VES curve for Station 4

Table 4: Geoelectric Parameters and Layer Interpretation for VES 4

Layer	Resistivity (Ωm)	Thickness (m)	Depth (m)	Geological Interpretation
1	933.10	1.72	1.72	Top soil
2	791.60	2.24	3.96	Sand
3	610.50	4.52	8.48	Sand
4	475.00	11.86	20.34	Sand and clay
5	1026.00	25.94	46.28	Sand (pressure aquifer)
6	1948.00	Infinity	Infinity	Sand (Aquifer)

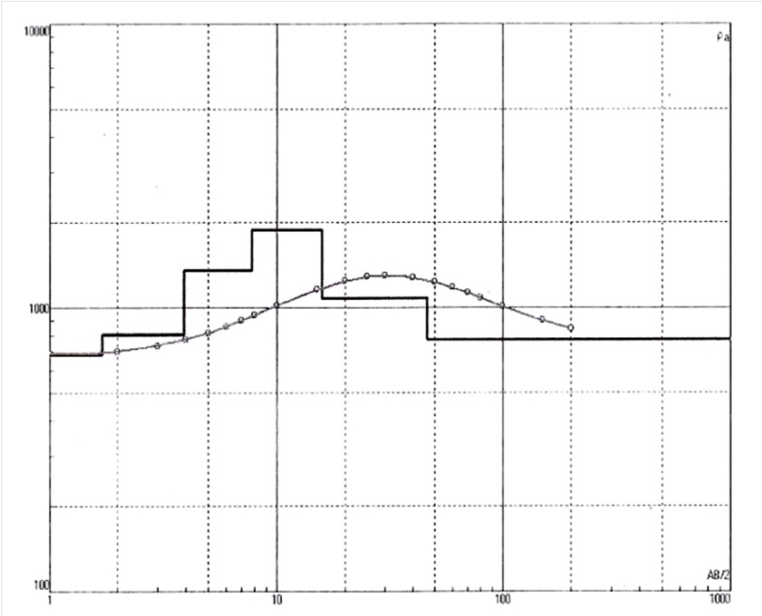

Fig.8: VES curve for Station 5

Table 5: Geoelectric Parameters and Layer Interpretation for VES 5

Layer	Resistivity (Ωm)	Thickness (m)	Depth (m)	Geological Interpretation
1	683.30	1.71	1.71	Top soil
2	805.40	2.21	3.92	Sand
3	1354.00	3.86	7.78	Sand
4	1881.00	8.04	15.82	Sand
5	1081.00	30.46	46.28	Sand (pressure aquifer)
6	771.30	Infinity	Infinity	Sand

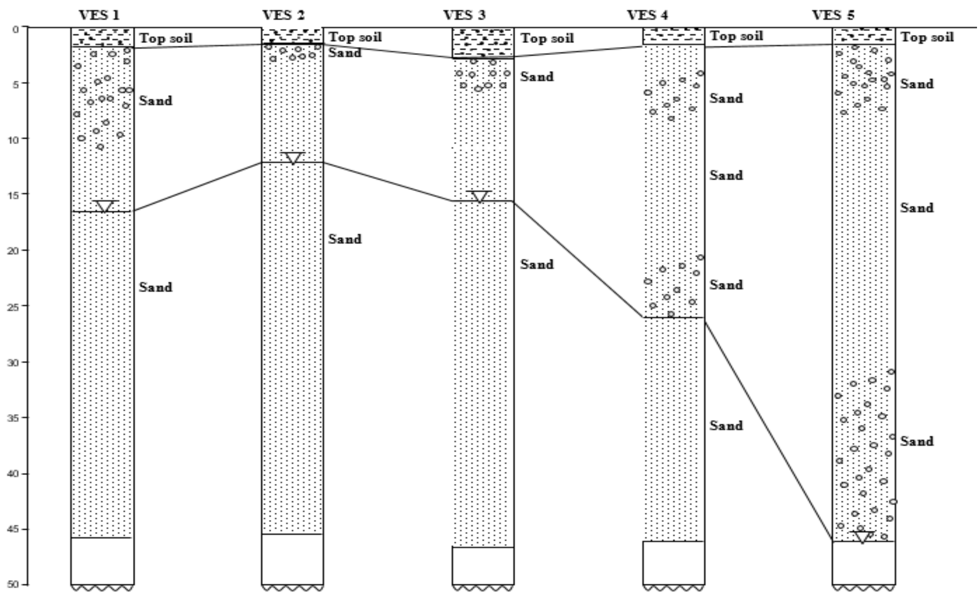

Fig. 9: Lithologic correlation of the study area

Conclusion

The electrical resistivity survey carried out in the study area revealed that the lithologies (Fig 9) are a typical representation of Benin formation in the Niger Delta with lithologies ranging from fine sand to medium sand and coarse sand. The coarse sand formation in all the locations delineated has the capacity to transmit water sufficiently and therefore represents good aquifers. A recommendation of an average depth 46m is to be drilled for all boreholes in the study area to sustainably yield potable and less vulnerable groundwater

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Chemical and Physical Properties of Selected Clay Deposits in Kogi State, Nigeria

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ABSTRACT
Clay deposits at Ojodu, Ugunoda, Ahoko, Natako and Omavi were found in commercial quantity in Kogi State, North Central Nigeria, and were investigated for their economic importance. X-ray diffractive test, Atterberg's limits test and linear shrinkage test were carried out on the representative samples. The chemical analysis test indicates the presence of fluxes in the five deposits, while their plasticity index ranges from 4.27 to 15.68% and their linear shrinkage values ranges from 3.57 to 8.57% which is within the internationally accepted standard values of 7 to 10% for alumino-silicates, kaolinite and fire clays. Their silica content of >50% shows that they are kaolinitic in nature. The test results indicate that these clay deposits are suitable for applications in colour and glazes, paints, kaolinite clay modifier, table ware, ceramic wares production.
Keywords: Clay deposit, Chemical and physical analysis, Economic, International standard, Quality, Ceramic wares.

Introduction

The significance of solid mineral resources has been of great value to man. Clay minerals appear not to be the most valuable among the minerals of the earth surface, yet they affect life on earth in far reaching ways. Nigeria in sub-Saharan Africa (surface area: 923,768 km^2) is a country with considerable wealth in natural resources, with a record of over 30 minerals of proven reserves, Obaje, (2009).

Noteworthy, clay minerals constitute over 50% of the non-metallic, earthy and naturally-occurring resources abounding throughout Nigeria's sedimentary basins and on the basement. In Olokode and Aiyedun (2011), it was observed that extensive investigation has been carried out on the liquid mineral endowment of the country, while little has been done to solid mineral endowment of which clay is prominent and as a result, adoption of solid mineral on industrial scale is scanty.

The most abundant, ubiquitous and accessible material on the earth crust is clay, Rado, (1988). Clay deposit is spread across the six geo- political zones of the country, Adegoke, (1980). Clays have their origin in natural processes, mostly complex weathering, transport, and deposition by sedimentation within geological periods, Nosbusch and Mitchell, (1988).

Fatunsin, (1992) said that the abundance of the clay minerals in Nigeria supports its rich and historic traditional pottery industry that dates from the Stone Age.

This study had considered the industrial potentialities in addition to the properties of clay mineral from selected locations in Kogi State of Nigeria. The qualities of clay found determine its application and suitability for ceramic products such as in bricks, ceramic wares, and refractory.

LOCATIONS OF THE STUDY AREA

The study areas are Ojodu, Ugunoda, Ahoko, Natako and Omavi. All are located within Kogi State, North Central Nigeria. Table 1 shows the coordinates of the study area and the colour of the clays as obtained.

Table 1: Locations of the Study Area

Samples	Source	Identification	Coordinate	Colour
Clay	Ojodu	L A	Long. - 07^0 24’45.4’’ Lat. - 006^0 56’15.3’’	Whitish
Clay	Ugunoda	L B	Long. - 07^0 24’58.0’’ Lat. - 006^0 48’01.4’’	Brownish
Clay	Ahoko	L C	Long - 08^0 18’05.4’’ Lat. - 006^0 51’26.2’’	Reddish\ferru-ginous
Clay	Nataco	L D	Lat. - 006^0 51’26.2’’ Lat. - 006^0 45’22.2’’	Pale-whitish
Clay	Omavi	L E	Long. - 07^0 38’16.7’’ Lat. - 006^0 11’10.8’’	Greyish-brown

Materials and Methods

100g of the five representative sample was measured placed inside a regulated temperature controlled at 100°C oven (Binder EDH04-69907) for 2 ½ hrs. This changed the clay substance from its fragile and smooth nature/texture to a more rough and coarse form. The clay sample size was later reduced or pulverized to about $6000\mu m \pm 200\,\mu m$, using the “pfaff milling machine-PAL-M100R”. 10g of the pulverized sample was then measured into a small hollowed steel ring and a pressure of up to 200 kilo Newton’s was applied to the sample while placing the steel ring on a clean hard surface with the aid of Herzoa MA-13961-1-1 (pelletizing machine). The sample from Ahoko was analyzed using two different program (clay and laterite). This was because the clay result shows high content of Fe_2O_5 (above the curve limit of the calibration graph). Hence, this placed the sample within the haematite family and the reason for analyzing it under the laterite program. The samples from each location was mixed together at the laboratory before being subjected to several tests at the Civil Engineering Laboratory and Minerals Processing Laboratory, Kogi State Polytechnic, Lokoja, Kogi State.

Chemical Analysis

The pulverized and palletized samples were put in AKL-9900 series of X-ray machine for the determination of its chemical composition.

Physical Analysis of the Samples

The physical analyses performed on the samples are: Atterberg’s limits test and linear shrinkage test.

Atterberg’s Limits

These test methods covered the determination of the liquid limit, plastic limit, and the plasticity index as stipulated by ASTM standards.

Liquid Limit

The liquid limit for each water content sample was determined using Equation 1:

$$LL^n = W^n . \left(\frac{N}{25}\right)^{0.121} \qquad .... Equ\ 1$$

Where: LL^n is the one-point limit for given trial, %;

N is the number of blows causing closure of the groove for given trial; and

W^n is the water content for given trial,%.

Plastic Limit

The water content of the sample contained in the container was determined in accordance with Test Method ASTM (2003b) D2216. The average of the water content (trial Plastic Limits) was computed and rounded to the nearest whole number. This value was the Plastic Limit, PL.

Plasticity Index

The Plasticity Index (PI) was calculated using Equation 2; by subtracting Plastic Limit (PL) from Liquid Limit (LL).

$$PI = LL - PL \qquad ... Equ. 2$$

Where: PI is the Plastic Index;

LL is the Liquid Limit; and

PL is the Plastic Limit

Moisture Content

The moisture content of soil samples was determined in according with ASTM (2002) D 854 – 00 standard procedures. The moisture content was determined using Equation 3.

$$Moisture\ Content = \frac{Mass\ of\ Water}{Mass\ of\ Dried\ Sample}\ x\ 100 \qquad ... Equ\ 3$$

Linear Shrinkage

This linear shrinkage of soil samples was determined in according with ASTM (2002) D 854 – 00 standard procedures of the linear shrinkage (LS) was calculated using Equation 4:

$$Linear\ Shrinkage = 1 - \frac{L_f}{L_o}\ x\ 100 \qquad ... Equ. 4$$

Where:

L_f is length after drying (cm); and

L_o is length before drying (cm)

Results

Chemical Analyses Result

The result of the chemical analysis (XRD Test) in percentage is shown in Table 2.

Table 2: XRD of the Studied Soils

ID	LA	LB	LC		LD	LE
Parameters			Clay	Laterite		
SiO_2	70.37	69.55	54.25	41.31	71.80	63.99
Al_2O_3	11.66	10.26	11.29	14.96	10.91	14.84
Fe_2O_3	1.36	4.67	19.44	42.60	0.23	6.19
CaO	0.15	0.24	0.19	0.03	0.56	0.63
MgO	0.06	0.33	0.10	0.00	0.36	0.68
K_2O	0.10	0.17	0.89	0.80	0.19	1.76
Na	0.00	0.08	0.00	0.00	0.41	0.65
SO_3	0.01	0.02	0.14	0.20	0.02	0.02

The summary of the Atterberg's Limits Test is shown in Table 3 below:

Table 3: Summary of Atterberg's Limits of the Studied Soils

Sample	Plastic limit (%)	Liquid limit (%)	Plasticity index (%)
LA	17.53	21.80	4.27
LB	22.94	36.65	13.71
LC	25.91	41.50	15.59
LD	23.23	35.20	11.97
LE	24.62	40.30	15.68

The Linear Shrinkage test of the five (5) locations is represented in Table 4.

Table 4: Linear Shrinkage of the analyzed Soils

Sample	Length Before Drying (cm)	Length After Drying (cm)	Shrinkage (%)
LA	14.00	13.50	3.57
LB	14.00	13.00	7.14
LC	14.00	12.90	7.86
LD	14.00	13.10	6.43
LE	14.00	12.80	8.57

The result obtained was compared to the recommended standard as suggested by Chester (1973) and Grimshaw (1971) for refractory/ceramic production as shown in Table 5.

Table 5: Physico-chemical Propertiesof the Studied Soils compared with (Chester 1973; Grimshaw 1971).

ID	LA	LB	LC		LD	LE	Chester	Grimshaw
Parameters			**Clay**	**Laterite**				
SiO_2	70.37	69.55	54.25	41.31	71.80	63.99	46-62	40-60
Al_2O_3	11.66	10.26	11.29	14.96	10.91	14.84	25-39	25-45
Fe_2O_3	1.36	4.67	19.44	42.60	0.23	6.19	0.4-2.7	1-5
CaO	0.15	0.24	0.19	0.03	0.56	0.63	0.2-1.0	<2.0
MgO	0.06	0.33	0.10	0.00	0.36	0.68	0.2-1.0	2-5
K_2O	0.10	0.17	0.89	0.80	0.19	1.76	0.3-3.0	<2.0
Na	0.00	0.08	0.00	0.00	0.41	0.65	0.3-3.0	<2.0
SO_3	0.01	0.02	0.14	0.20	0.02	0.02		
Al_2O_3-SiO_2 Ratio	0.17	0.15	0.21	0.36	0.15	0.23	0.54-0.63	0.62-0.75
LL	**21.80**	**36.65**	**41.50**		**35.20**	**40.30**		
PL	**17.53**	**22.94**	**25.91**		**23.23**	**24.62**		
PI	**4.27**	**13.71**	**15.59**		**11.97**	**15.68**		**10-60**
LS	**3.57**	**7.14**	**7.86**		**6.43**	**8.57**	**7-10**	

Discussions

The test results indicate the presence of some fluxes such as CaO in the five clay deposits, MgO in Ojodu (in small quantity), Ugunoda, Natako and Omavi clay which is also a fluxe is present in all the deposits. The presence of these compounds in the deposits is an added advantage, since it will enhance production as fluxes. The plasticity index that ranges from 7 to 18 is good for brickmaking. This means that deposits in Ugunoda, Ahoko, and Natako are slightly plastic and Ojodu deposit is slightly above non-plastic clay. Clay with low plasticity index will be difficult to handle for brick-making (moulding).–However, Ojodu clay with low plasticity index can be improved upon by adding clay of higher plasticity index to it. This will also make it to be suitable for ceramics. The linear shrinkage of the five deposits ranges from 3.57 to 8.57 which are within an acceptable and tolerable range. Most insulating materials will begin to shrink at some definite temperatures. Usually the amount of shrinkage increases as the temperature of exposure becomes higher ASTM (2017) C 356 – 17, Apart from the Ojodu clay, the plastic index of the other clays fall within the recommended standard of 10-60% (Grimshaw, 1971) while the linear shrinkage of Ugunoda, Ahoko and Omavi clays fall within the range recommended which is 7-10% (Chester, 1973) and that of Natako is just a little lower (6.43%) which is not too bad. there may be instances that may require either the addition of sand (for clay with a high plastic index) or the addition of high plastic clay to the one with low plastic index in order for them to be very suitable for the production of ceramics and brick- making. The result obtained from the tests when compared with some recommended standard as indicated in Table 5 shows that the silica content is high in location LA, LB, LD and LE and should be reduced before use for refractory/ceramic production. The high content of Fe_2O_3 in Ahoko clay makes it to be the most preferred for cement production. Omavi and Ugunoda clay will be very good for brick-making.

Conclusions

The characteristics was identified through X-ray diffraction test (XRD), Atterberg's limits test and linear shrinkage test to ascertain their potentials for ceramic and other applications. The result confirms that the clays are mainly composed of kaolinite clay mineral.

Recommendations

From the results of the research, following recommendation can be drawn:

1. The high plastic clay must be added to Ojodu clay to improve it and make it suitable for ceramic and brick-making production. Also the iron content of the clays excluding that of Natako clay should be reduced in order to achieve fired brightness value in excess of 83% at 1180^{o}C.
2. The presence of fluxes and low alumina content in these deposits is an advantage; as there would be no need for high temperature which is the major factor that increases production cost.

Acknowledgement

The XRD test carried out on these samples was at the Geology laboratory of the Faculty of Physical Science, Department of Geology, University of Benin, Benin City, Edo State, Nigeria.

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Sirajo, Yaro, Thomas, Maina and Asuke (2020) Nigerian Mining Journal, (18) 31 - 42

Characterization of Bassa-Nge Iron Ore, Kogi State, Nigeria

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Abstract

The characterization of Bassa-Nge iron ore deposit in Kogi State, Nigeria was carried out. The samples were sourced from four different pits 1, 2, 3, and at 120m apart and at 5m depth respectively. The four samples were mixed together to form composite sample. The composite sample was crushed, pulverized and ground. The result of the chemical analysis using XRF revealed that the composite sample contained 54.78% Fe, 6.47% Al, 3.10% Si, 0.88% P, 0.16% Ti, and 0.5% Mg on the average. Mineralogical analysis using XRD showed that the ore contained the following minerals in major quantities: hematite, magnetite, aluminium phosphate while, titanite, manganese silicate, and silica minerals present in minor quantities. The SEM/EDS analysis results revealed that the iron bearing minerals are separated from other minerals contained in the ore matrix by smooth boundaries and allows easy liberated from other associated minerals by crushing and grinding. Therefore, Bassa-Nge iron ore can be beneficiated, exported and supplied to iron and steel industries in the country and beyond.

Keywords: Iron ore, Bassa-Nge, Characterization, XRF, XRD, SEM/EDS

INTRODUCTION

Recent characterization studies have become very important in mineral processing because they contribute to the understanding of the behaviour of ores during processing. These studies involve several integrated techniques, and their results can aid to improve process efficiency (Biswas, 2005). Characterization of a mineral ore is a very important step to perform before any processing takes place whereby quantity, grade or quality, densities, shape, and physical characteristics are determined to allow for appropriate application of technical and economic parameters to support production planning and evaluation of the economic viability of the deposits (Uwadiale, 1989, Sparks and Sirianni, 1974 and Cheng *et al.,* 1999)

This information assists in understanding the behaviour of the ore when subjected to the various production conditions, and also when it comes in contact with the raw materials used in iron production; metallurgical coke or gas and limestone. This is not only important in beneficiation but enables the simulation/modelling of a production system that can be used to predict the behaviour of the iron ore during production (Platts, 2010)

It is apparent that most of the known deposits contain low-grade ores with iron contents less than 40%. By contemporary growth of the world consumption of iron ores (about 10-15% per annum), the known resources of iron ores could run out within the next 55-60 years (Lester *et al.,* 2006, Guider 1981 and Dobbins *et al*., 1982). It is thus imperative to find new sources of iron ore to supplement the existing sources in order to meet the growing demand. Therefore, revealing and exploiting new deposits of iron ores, particularly of low and high-grades, is very imperative. Iron ore deposits have been known to occur in the Bassa-Nge area in the Eastern of Kogi State of Nigeria. However, the Bassa-Nge iron ore deposit is still unexploited and little study has been done on them. The deposits, located on the hill in Egeneje district, Bassa- Nge local government Area of Kogi State. Specific quantification for the exact tonnage has not been carried out yet but estimates put the ore reserves at 400 million of estimated iron ore tonnes reserves.

MATERIALS AND METHODS

Sample Collection

Samples of the iron ore were collected from various pits of deposit site located at Egeneji village, in Bassa-Nge Local Government Area of Kogi State. GPS was used to measure the exact location at which sample was taken. The iron ores deposit covers a distance of about 10.3 kilometres square and has an estimated reserve of 400 million tonnes. Grab method of sampling was adopted in collecting the sample. About eighty (80) kg of the sample was collected from four (4) four points at interval of 25-100m apart at 6m depth.

Sample Preparation

Sample preparation involves comminution which consists of crushing and grinding process. The lump sizes of the ore sample were reduced to the sizes that could be accepted by the crusher using sledge hammer. The sample was crushed using jaw crusher and pulverized using ball mill.

Sampling for the Purpose of Tests and Analyses

Coning and quartering sampling method was used to divide the pulverized sample into smaller portions that were used for other tests and analyses conducted.

Characterizations

Samples from different pits were prepared for X-ray Diffraction (XRD), X-ray Fluorescence (XRF), and Scanning Electron Microscopy (SEM)/Energy dispersive spectroscopy (EDS) respectively in accordance to the standard procedure.

Chemical Analysis of the Bassa-Nge iron ore using XRF

The samples were prepared as pressed powders. The ARL Perform’X Sequential XRF instrument with Uniquant software was used for the analyses. The software analysed for all elements in the periodic table between Na, (Z=11) and U, (Z=92) but only elements found above the detection limits were reported. The values were normalised, as no loss on ignition (LOI) was done to determine crystal water and oxidation state changes.

Mineralogical Analysis of the Bassa-Nge iron ore using XRD

The iron ore sample was crushed using jaw crushing machine, disc miller and finally cup miller ground to fine particle size of 100mesh (0.15 microns). The sample was smeared evenly on the sample made of aluminium material. The setting of machine was done between angle of 2^{o} to 60^{o}degree theta as the sample scanning range. The running rate (scanning speed) was set at 6 degrees per minutes. The holder is carefully placed on the loading point of the movable gonimeter arm that contain a clamp capable of gripping the sample firmly. The window indicating readiness after properly closed. The pronounced PEAKS OR DIFFRACTOGRAMMS displayed, express the minerals composition at the various angle of the degree theta. The experiment was performed by X'Pert HighScore Philips Analytical. The mineralogical components of the pulverized iron ore carried were out by X – ray diffraction technique on the samples from pits and composite respectively. The main minerals found in the samples of pits and composite were presented in Tables 4 – 8, while Figures 4 – 8 showed the X-ray diffraction pattern samples from pits (1-4) and composite. The X-ray diffraction technique (XRD) was extensively used for identification of various mineral phases especially the iron ores minerals for assessing the abundance of each phase in an iron ore sample. The XRD was carried out using PANalytical X-ray Diffractometer, MODEL D500 having automatic receiving slit, divergence slit, and graphite mono-chrometor assembly. Cu, Kα radiation operating at 40 kv and 20 nA was used for this purpose. A diffraction pattern recording the angle 2θ against the intensity was obtained over a range between 10˚ to 70˚ corresponding to d-values between 20 A and 1.34A. The scanning rate was 2˚ per minute with recorder full scale set in to 2 X 103 counts. Each mineral phase exhibits a characteristic reflection peak corresponding to its d-values. These of D values were matched from the various minerals identified**.** Furthermore, the variations in the peak intensities of different mineral phases in the ore sample indicate their relative abundance.

Microstructural Analysis of Bassa-Nge iron ore using SEM/EDS

Morphological, quantitative and qualitative analyses of the iron ore samples were carried out using SEM/EDS model JEOL 840. Small pieces from each hill-sample were put in bakelite and silica polished and then observed under the microscope at different magnifications of 50x, 100x, 200x, and 500x. The SEM studies for the mineral analysis of representative samples were conducted in stages. All the samples were carbon coated in order to make the minerals surface conductive. In the first stage, which was aimed at examining the minerals morphology and identifying their mode of occurrence, crushed carbon coated minerals were examined directly with scanning electron microscopy without polishing. The second stage of the microscopic study was aimed at identifying the mineral phases present within the samples. Qualitative chemical analysis of minerals was carried out on the iron ore samples to produce Backscattered images (BSI). The Quantitative analysis was carried out using the EDS analysis.

RESULTS AND DISCUSSION

The Result of Chemical Analysis Bassa-Nge Iron Ore Samples

The XRF analysis results of Bassa-Nge iron ore samples from different pits and composite, pits and the composite are presented in Table 1. From the results of the analysis, pit 3 in Table 1 contains the highest iron ore (haematite, Fe_2O_3, 85.66%) and iron (Fe, 59.96%) content when compared with the other samples from other pits 1, 2, and 4. This could be attributed to the different in mineralization process of the deposit, which may have favoured pit 3 to other pits. The silica (SiO_2, 13.48%) and silicon (Si, 6.29%) is the highest in pit 2 content of the ore deposit. The aluminium oxide was highest in pit 2 (20.07%Al_2O_3). The major elements present in the composite sample are as followings: iron content is Fe, 54.78%, Al, 6.47%, Si, 3.10%, phosphorus (0.88%) while magnesium, titanium, sodium and calcium minor elements present respectively.

The values obtained in table 3 are comparable to other major iron ore deposits for iron and steel productions within and outside the country (Magudu, 2007). The minor elements such as titanium, vanadium, magnesium can be used as an alloying element during the iron and steel making processes (Sirajo, 2008).

Table 1: Chemical Analysis Result of the Samples from different pits and composite

No. of Pits	1	2	3	4	Composite
Fe_2O_3	82.35	64.11	85.66	85.04	78.26
Al_2O_3	9.01	20.07	7.26	7.69	12.23
SiO_2	5.21	13.48	3.39	3.13	6.64
P_2O_5	2.11	0.76	2.73	3.13	2.01
MgO	0.40	*<0.01*	0.15	0.17	0.05
Na_2O	0.28	0.24	0.10	0.12	*<0.01*
TiO_2	0.15	0.74	0.12	0.13	0.27
CaO	0.08	0.04	0.09	0.13	0.07
V_2O_5	0.07	0.11	0.05	0.05	0.06
Ho_2O_3	0.05	*<0.01*	0.05	0.04	0.04
$Gd_2O_{.e}$	0.04	0.04	0.05	0.04	0.04
Co_3O_4	0.04	0.04	0.05	0.05	0.04
Dy_2O_3	0.04	*<0.01*	0.06	0.05	0.04
ZnO	0.04	0.02	0.04	0.04	0.03
Yb_2O_3	0.02	0.01	0.03	0.03	0.02
MnO	0.02	0.01	0.09	0.06	0.05
K_2O	0.02	0.09	0.02	0.01	0.03
SO_3	0.02	0.10	0.02	0.01	0.03
MoO_3	0.01	0.01	0.01	0.01	0.01
Cr_2O_3	0.01	0.06	0.01	0.01	0.02
SrO	0.01	0.01	0.02	0.02	0.01
Ag_2O	0.01	0.01	0.01	0.01	0.01
NiO	0.01	*<0.01*	0.01	0.01	0.01
ZrO_2	0.01	0.05	0.01	0.01	0.01

Mineralogical Composition of Various Pits and Composite Samples

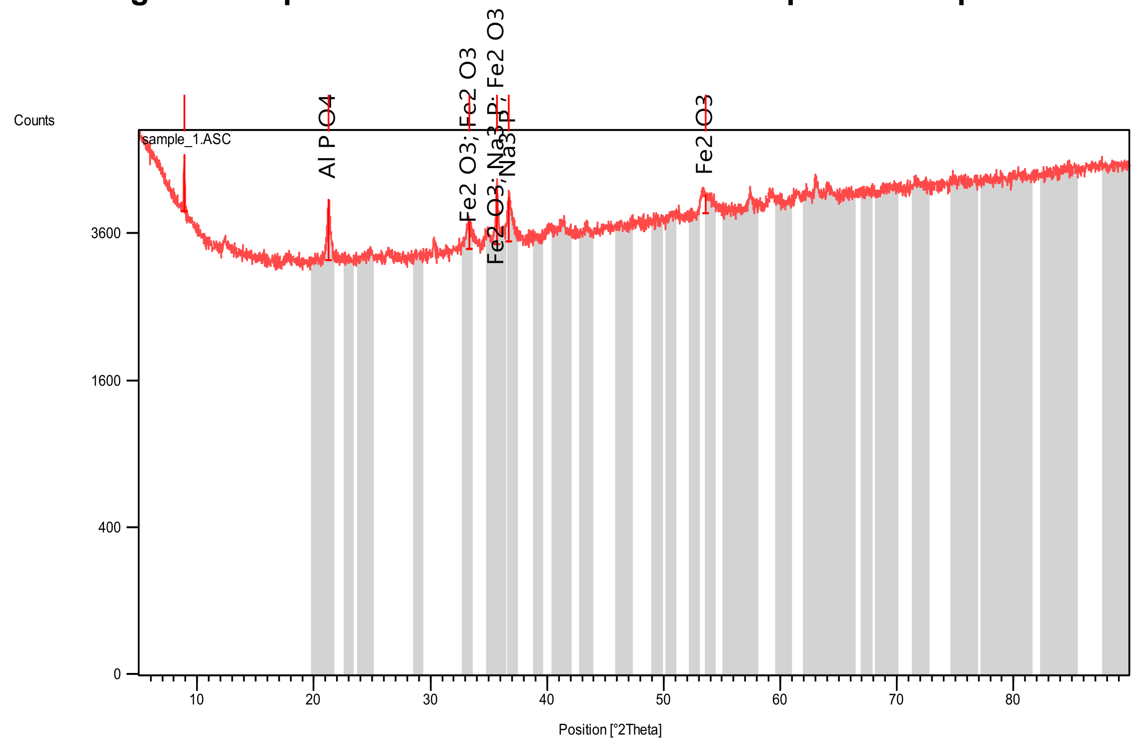


Figure 1: X-ray diffraction pattern sample from pit 1

Table 2: XRD Analysis Result for Sample from Pit 1

Ref. Code	Score	Compound Name	Scale Factor	Chemical Formula
73-0603	50	Hematite, syn	0.697	Fe_2O_3
31-0028	35	Aluminum Phosphate	0.501	$AlPO_4$
74-1164	17	Sodium Phosphide	0.328	Na_3P
72-0469	25	Hematite	0.459	Fe_2O_3

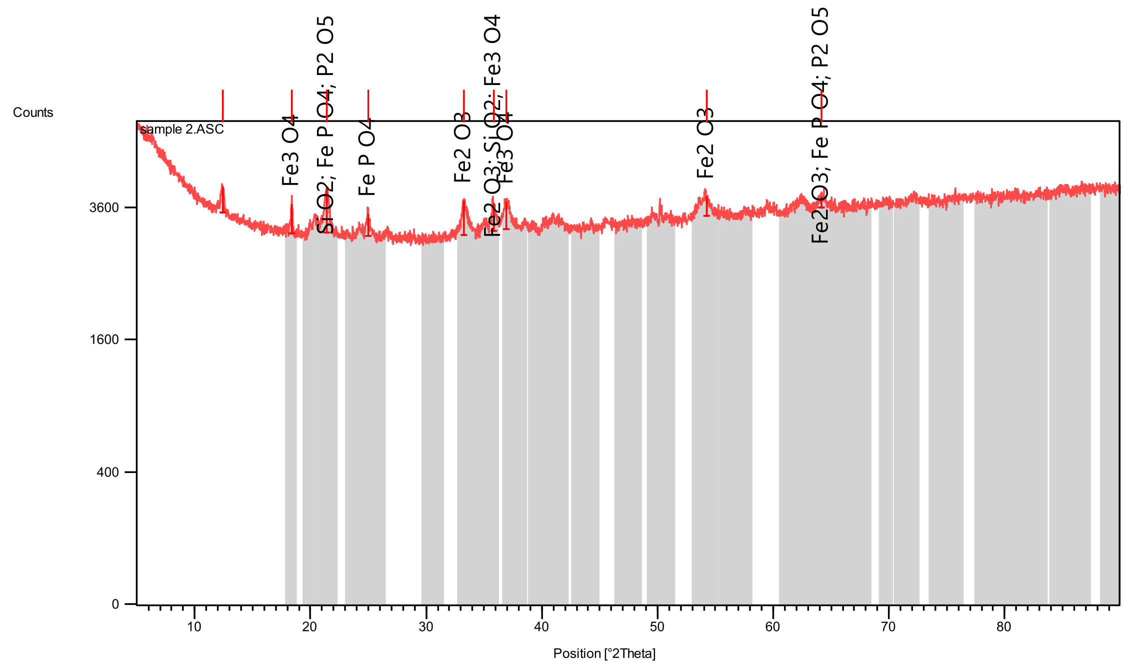


Figure 2: X-ray diffraction pattern sample from pit 2

Table 3: XRD Analysis Result for Sample from Pit 2

Ref. Code	Score	Compound Name	Scale Factor	Chemical Formula
85-0599	50	Hematite	0.692	Fe_2O_3
76-0931	49	Silicon Oxide	0.929	SiO_2
74-1910	31	Magnetite	0.405	Fe_3O_4
31-0647	22	Iron Phosphate	0.147	$FePO_4$
23-1301	12	Phosphorus Oxide	0.138	P_2O_5

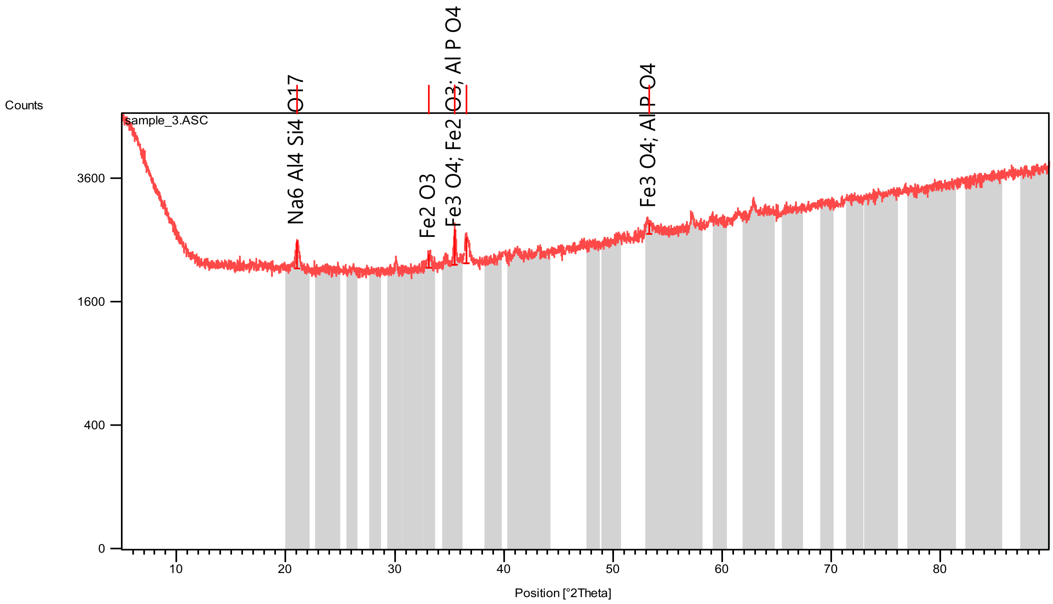


Figure 3: X-ray diffraction pattern sample from pit 3

Table 4: XRD Analysis Result for Sample from Pit 3

Ref. Code	Score	Compound Name	Scale Factor	Chemical Formula
03-0863	40	Magnetite	0.570	Fe_3O_4
76-2385	35	Sodium Aluminum Silicate	0.354	$Na_6Al_4Si_4\ O_{17}$
86-0550	28	Hematite, syn	0.129	Fe_2O_3
48-0652	9	Aluminum Phosphate	0.099	$AlPO_4$

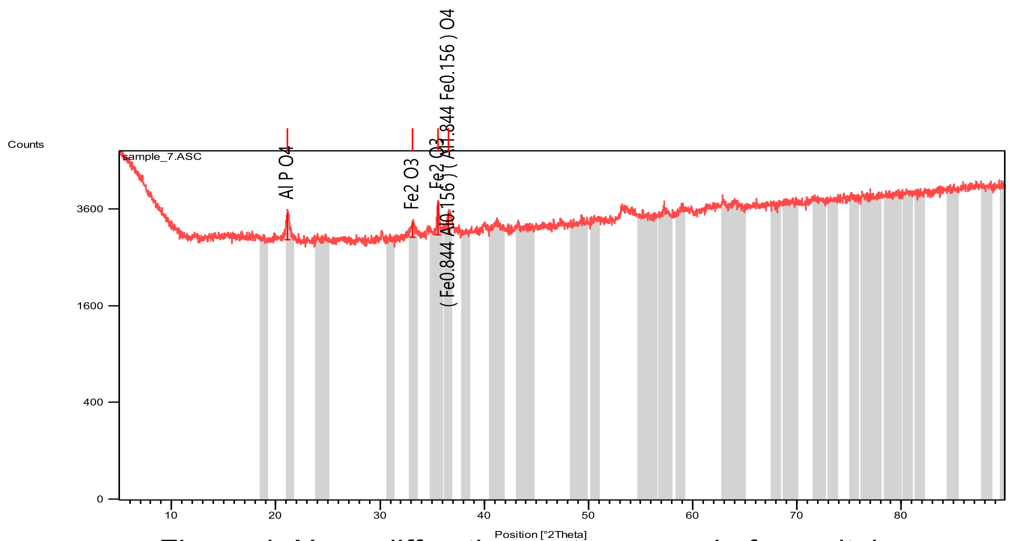


Figure 4: X-ray diffraction pattern sample from pit 4

Table 5: XRD Analysis Result for Sample from Pit 4

Ref. Code	Score	Compound Name	Scale Factor	Chemical Formula
73-0603	34	Hematite, syn	0.458	Fe_2O_3
31-0028	32	Aluminum Phosphate	0.368	$AlPO_4$
82-0589	26	Hercynite, syn	0.464	(Fe0.844 Al0.156) (Al1.844 Fe0.156) O_4

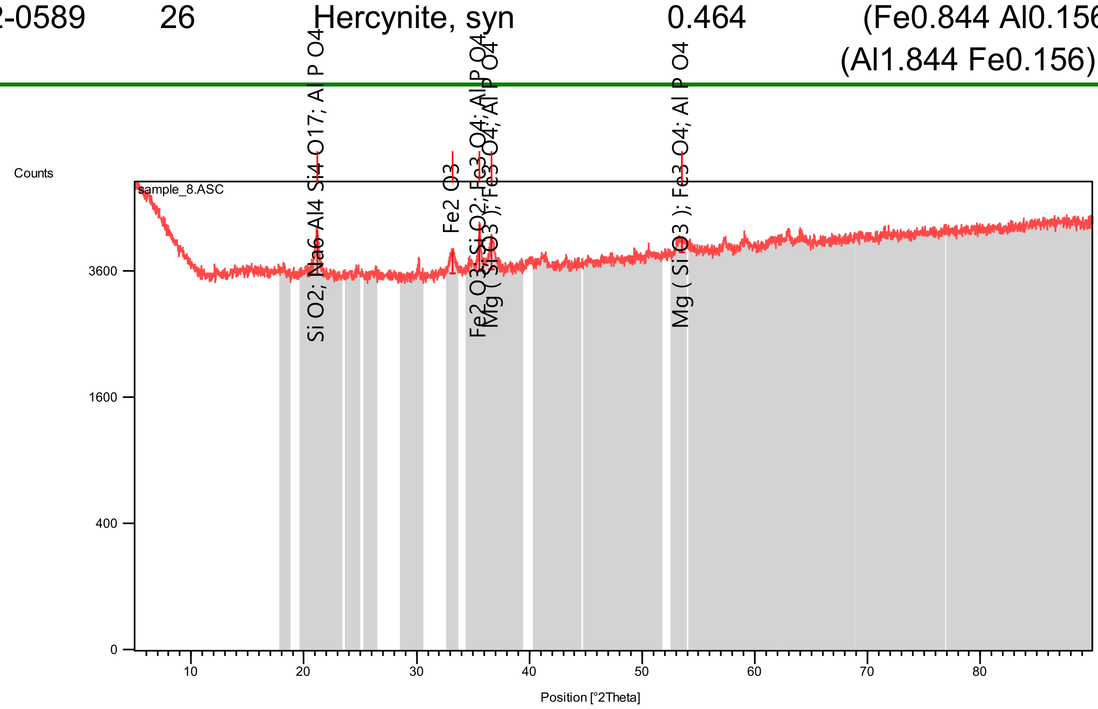


Figure 5: X-ray diffraction pattern sample from composite

Table 6: XRD Analysis Result for Sample from composite

Ref. Code	Score	Compound Name	Scale Factor	Chemical Formula
73-0603	45	Hematite, syn	0.486	Fe_2O_3
27-0605	30	cristobalite, high	0.147	SiO_2
76-2385	27	Sodium Aluminum Silicate	0.550	Na6Al4 Si_4O_{17}
86-0006	19	Magnesium Silicate	0.354	Mg (SiO_3)
89-0950	12	Magnetite	0.249	Fe_3O_4
50-0303	14	Aluminum Phosphate	0.229	$AlPO_4$

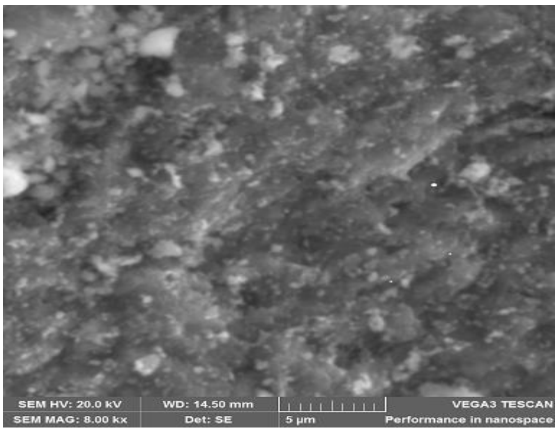
From the Tables 2 – 6 and Figures 1 – 5, the ore sample from the pit 1 contains haematite, aluminum phosphate, and sodium phosphide, pit 2 contains haematite, silicon oxide, magnetite, Iron phosphate and phosphorus oxide, pit 3 contains magnetite, sodium aluminum silicate, haematite, and aluminum phosphate and pit 4 also contain haematite, aluminum phosphate, and hercynite respectively.

However, the composite sample in Figure 5 and Table 6 have the following minerals: haematite, cristobalite, Sodium aluminum silicate, magnesium silicate, magnetite, and aluminum phosphate. From the composite sample results, the ore could be said to be predominantly haematite

(Fe_2O_3), aluminum phosphate ($AlPO_4$), magnetite (Fe_3O_4), phosphate (PO_4) while cristobalite (Fe0.844 Al0.156), magnesium silicate (Mg (SiO_3), sodium alumimium silicate $Na_6Al_4Si_4O_{17}$ and phosphorus pentoxide (P_2O_5) respectively. The mineralogical compositions of this iron ore deposit are similar and comparable to the findings of the previous deposits worked upon and being cited in the literatures (Salawu, 2015 and Ogwuegbu, 2011)

Scanning electron microscope/Energy Dispersive Spectroscopy (SEM/EDS)

Plates 1 – 5 present the results of SEM and EDS analyses of Bassa-Nge iron ore deposit in pits and composite samples. It can be observed from the EDS results that the Fe contents of the ore samples are generally high and indication that the Bassa-Nge iron ore deposit is rich in iron. The spectra of the major elements contained in the iron ore are iron, oxygen, aluminium, silicon, carbon, oxygen, phosphorus in all the pits and the composite investigated. The results of the EDS tally with those values obtained from the XRF analysis samples presented in Table 1. These also conform to the previous findings of other researchers (Folorunso *et al* 2014, Salawu, 2015 and Oloche *et al*., 2001)



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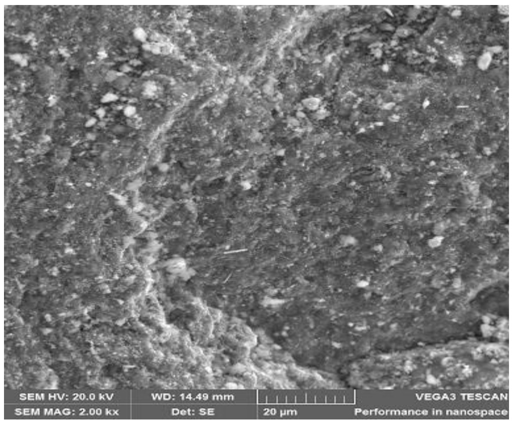

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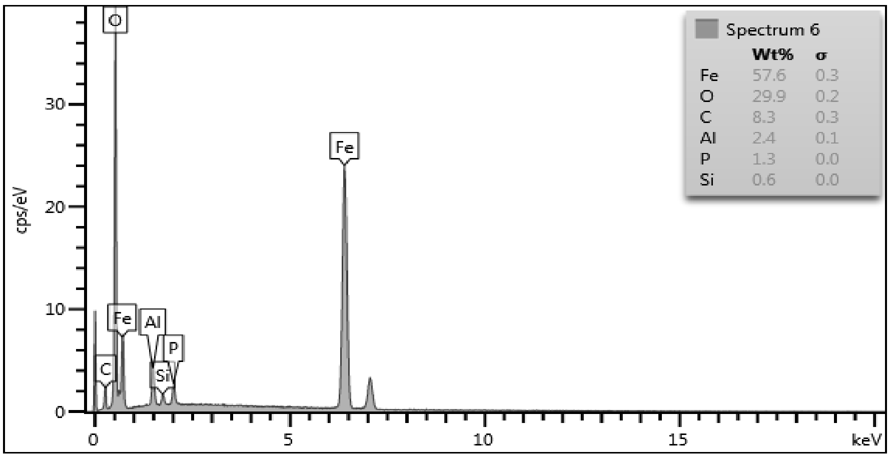

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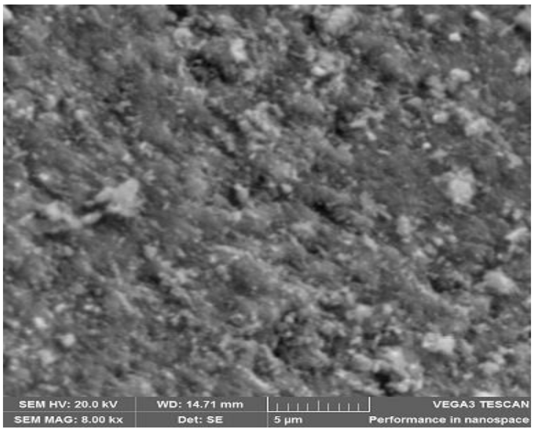
Plate 1: (a) Microstructure at 8kX (b) Microstructure at 2kX (c). Spectrum of minerals in graphic format in Pit 1



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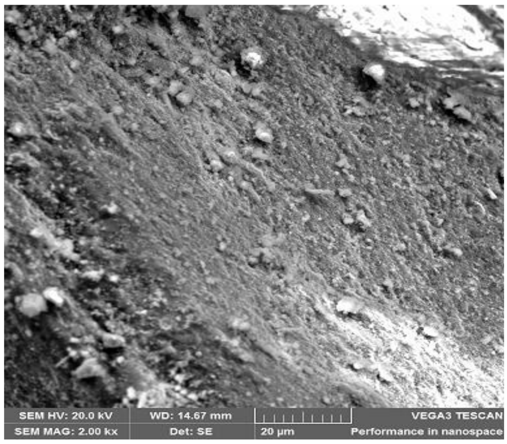

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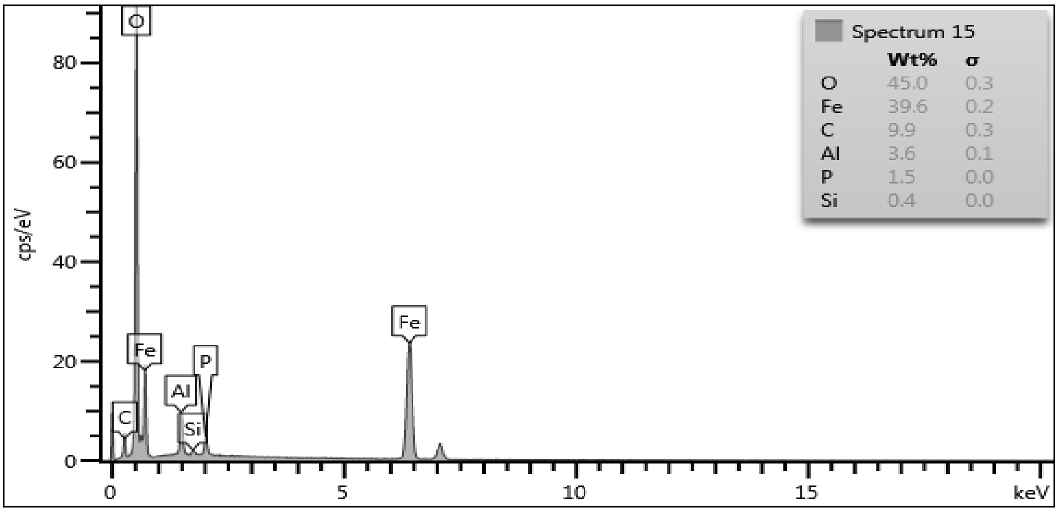

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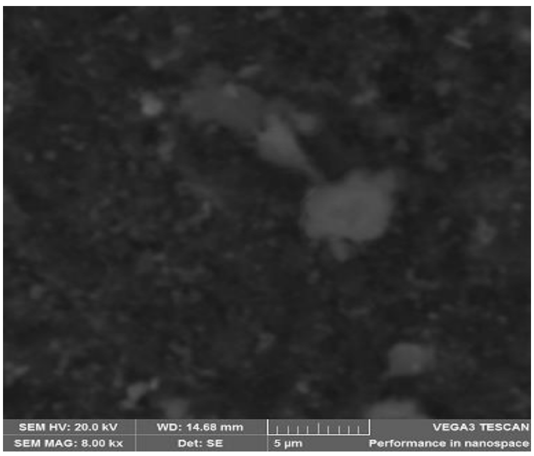
Plate 2: (a) Microstructure at 8kX (b) Microstructure at 2kX (c). Spectrum of minerals in graphic format in Pit 2



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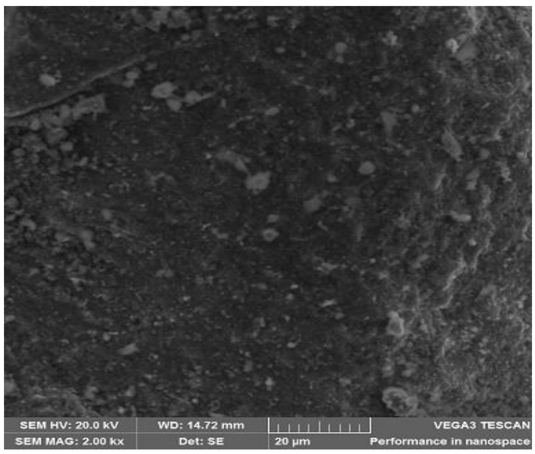

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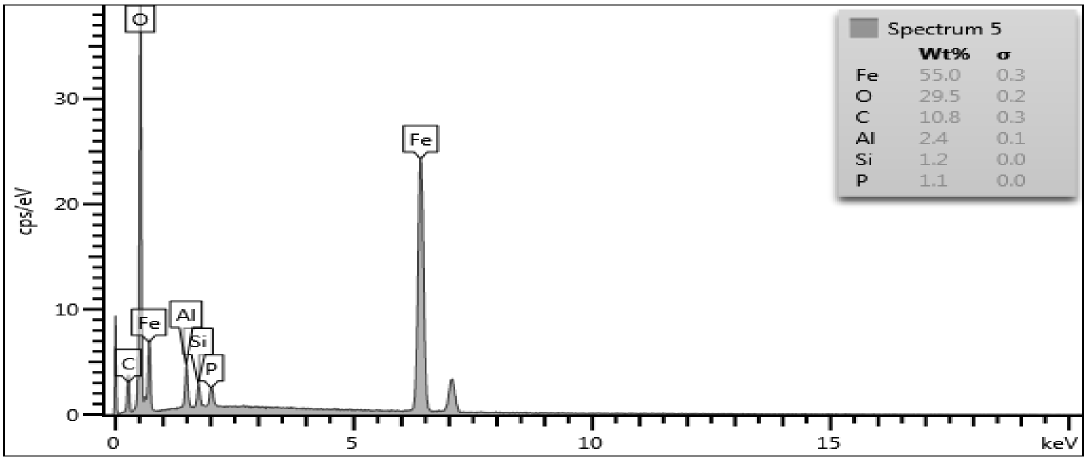

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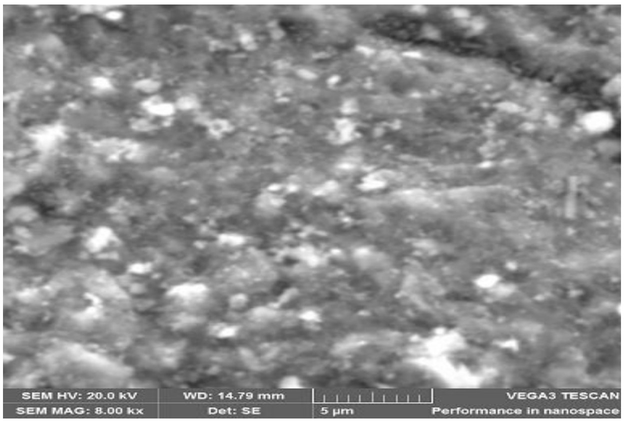
Plate 3: (a) Microstructure at 8kX (b) Microstructure at 2kX (c). Spectrum of minerals in graphic format in Pit 3



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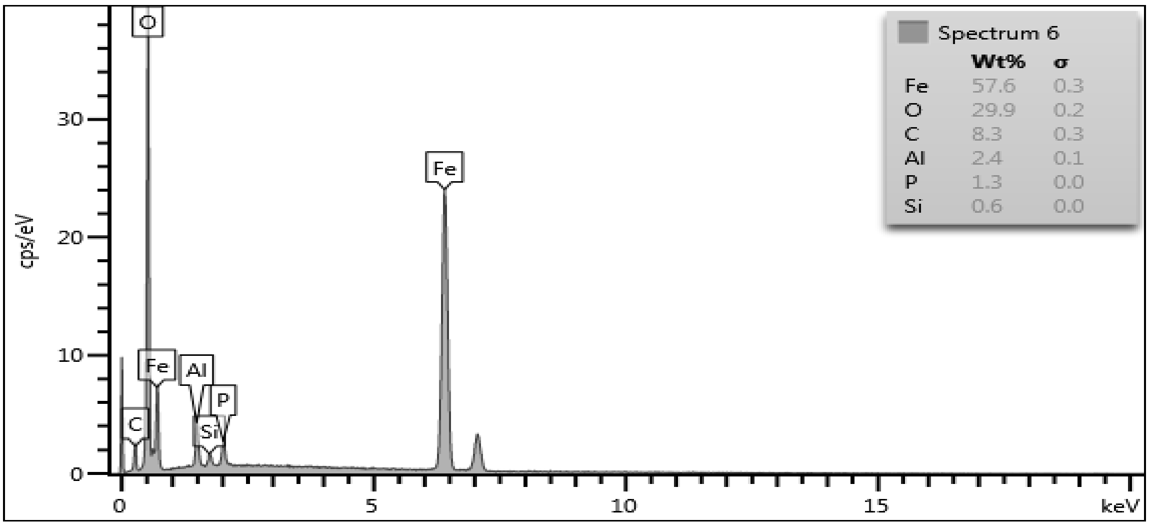

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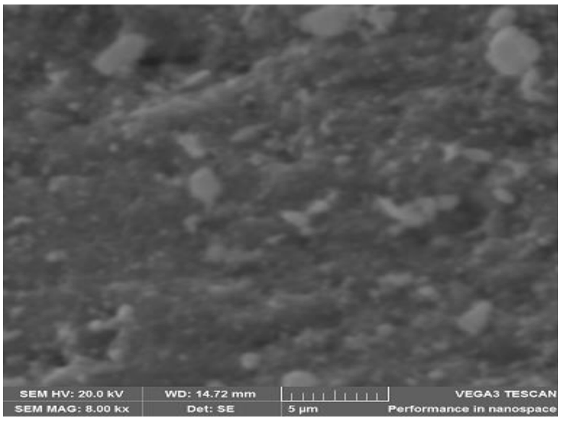
Plate 4: (a) Microstructure at 8kX (b) Microstructure at 2kX (c). Spectrum of minerals in graphic format in Pit 4



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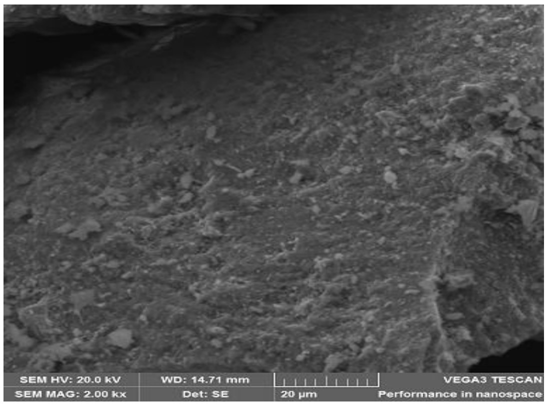

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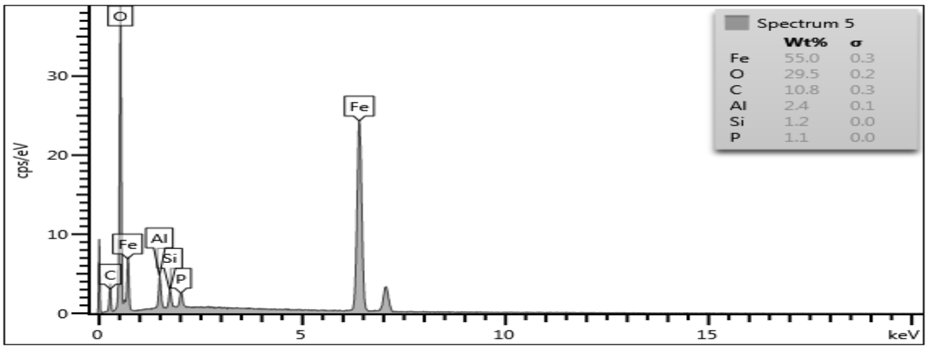

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Plate 5: (a) Microstructure at 8kX (b) Microstructure at 2kX (c). Spectrum of minerals in graphic format in composite sample

CONCLUSION

The following conclusions can be drawn from the results obtained:

i. the chemical analysis of the Bassa-Nge iron ore using XRF revealed that the sample contains: 54.78% Fe, 6.47% Al, 3.10% Si, 0.88% P, 0.16% Ti, and 0.5% Mg respectively.
ii. mineralogical analysis of the ore using XRD revealed that the iron ore is predominantly haematite, magnetite, iron phosphate and other associated minerals like aluminite, silica, phosphorus, zincite and titanium oxide which can be separated during mineral processing.
iii. The SEM analysis results revealed that the iron minerals can be separated from other minerals contained in the ore by smooth boundaries that may make it easy to liberate from other associated minerals.

Thus, making Bassa-Nge iron ore another potential iron ore deposit that can be explored, exploited and exported for both local iron and steel productions.

Acknowledgements

The authors are thankful to the staff of the Department of Geology, University of Pretoria, South Africa and others who had contributed to this work. May God rewards us together Amin.

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Cost Modelling and Estimation of Drilling and Blasting Parameters affecting Quarry face and Fragmentations: A Case of Jolex Construction Company, Bassa, Plateau State

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Abstract
This research is based on the estimation and modelling of operational cost at varying drilling and blasting parameters that may likely cause elevated floors, poor fragmentation, and formation of toes at Jolex Construction Company Quarry and recommends best practices in order to provide good working pit floors and to eliminate or reduce the cost of secondary breakage. The methods employed include: drilling performance analysis using statistical tools, modification of secondary breakage cost estimation model using two different existing models, and estimation of the cost of secondary drilling and blasting using the designed modification model and other two existing models; sensitivity analysis. From the analysis, it was observed that the blast designed parameters and blasting practices (desk design) were acceptable to produce good fragmentation and good working pit and bench floors but the actual drilling parameters deviated from the designed parameters by about 26%, 19%, 26% and 2% in hole depth, burden, spacing and hole inclination respectively. It was concluded that the cause of the ineffective fragmentation leading to high cost of secondary breakage and uneven pit floors was due to operational errors during drilling. It is therefore recommended that in order to reduce excessive deviations in the drilling parameters, periodic training of operators must be conducted, supervision of drilling and blasting operations must be enhanced, combination of field experience and theoretical knowledge of drill and blast geometry should be encouraged, and inclinometers should be used during drilling activities to ensure the accuracy and precision of all blast holes.

Key words: fragmentation, sensitivity analysis, drilling, blasting

Introduction

Researches have demonstrated abundantly that there are several challenges often associated with mining operation as a business. Examples of those challenges are the operators and owner's inadequate knowledge and innovation about its development and management. They also include its planning and design, operation and maintenance, financing and productivity, machinery and efficiency, revolutions and other economic related factors. To these challenges may be added the fact that most attempts on the solution as a means of overcoming most of the above challenges, dwell mainly on issues of profit maximization, minimizing operating and production cost, improve efficiency and productivity, wealth creation and mineral economics with little attention given to the source, optimization, amelioration, or elimination of all these challenges from the prospecting stage to the post closure.

Quarries principally produce sand gravel and crushed rock for construction and these materials are usually described as "aggregates" (Anthony. et al, 2018). Materials produced by quarries are numerous, such as: gypsum, salt, potash, coal, chemical grade limestone, common clays, china clay, kaolin, ball clays, and silica sand etc. All these materials are very important for sustainable development of every nation (Tose, 2006).

Growth in business is often measured by increased profitability. However, how that growth comes about can be the result of technological advances, improved operating practices, smarter deployment of resources, appropriate planning and design - the list is endless. The common theme that flows through these incremental gains is invariably innovation, something in which the mineral extractives and quarrying sector excels (Robin, 2019). Mining operation is a business, and the ultimate aim is to make profit by minimizing overall operation cost, and increase the overall efficiency. However, all these are possible by application of knowledge and innovation in all the areas of site operations.

Drilling and blasting operations are among the most important unit operations in any hard-rock mine. Fragmentation is an important factor to qualify a blast because it has a clear impact on productivity of mining equipment and on the overall profit of the mine. (Iversion. et al, 2013) Mining companies often encounter challenges in their drilling and blasting performance. Despite great efforts to improve on the drilling and blasting practices at the pit, the results of the blasting and degree of fragmentation are not satisfactory. (Sharma, 2011)

The pit floor, after mucking the fragmented materials, usually becomes uneven (i.e. undulating, with many toes). This condition has cost implications on the mine as it leads to re-drilling and blasting of the bumps on the pit floor, grading to achieve the required pit floor levels, increased wear and tear of equipment, and reduction in the life of truck tyres, reduced equipment performance, and low productivity as well as a low profit margin (Eshun et al, 2016). It is given that unleveled bench floor will form after drilling and blasting operation. The degree of undulation and the rate at which bench floor and slope deform are dependent on the geology, mining method and mine design. Unlevelled bench floor causes tire wearing, reduces efficiency, increasing operational cycle time and decreases profitability. Thus, elevated mine floor is a serious hindrance to mining operation, especially in surface mining methods such as quarrying for aggregate stone. (Afum. et al, 2015)

A valuable contribution of a study of this nature to mining operation is often not acknowledged, because the operation typically exploits what might otherwise be considered as an economic resource such as surveying equipment, statistical skill and so on. It is a simple amelioration that is capable of saving excess operating and maintenance cost, such as cost of secondary blasting, equipment maintenance, excessive fuel consumption, replacement cost, services and several other related issues that may arise. The objective of this study is aim at investigating the drill and blast performance to determine the causes of uneven quarry floor and its effects on the overall operational cost. In a nutshell, the idea of innovation should perhaps motivate mining industry to shed some of their concessions, which could then be worked on by both small and large scale mining operation.

Description of the Study Area

The study area, Jolex Construction Company Quarry, is located at Tippo Kwana, Mista Ali village, Bassa local government area, which is located at the north of Plateau State, Nigeria. It lies between latitudes 10^0 2’ 23’’N and 10^0 2’ 45’’N and longitudes 8^0 51’ 10’’E and 8^0 51’ 30’’E (Naraguta 1:50,000 sheet 168 NW) in Bassa Local Government Area, which is situated in the Northern Fringe of Jos, the

Plateau State capital. The main access route to the area is the Jos-Miango major road. . The area has a distinct and rugged topography with hills of different heights. The drainage pattern is mainly dendritic. The major rock type in the study area is granite overlaid by laterite. (Bala et al., 2015)

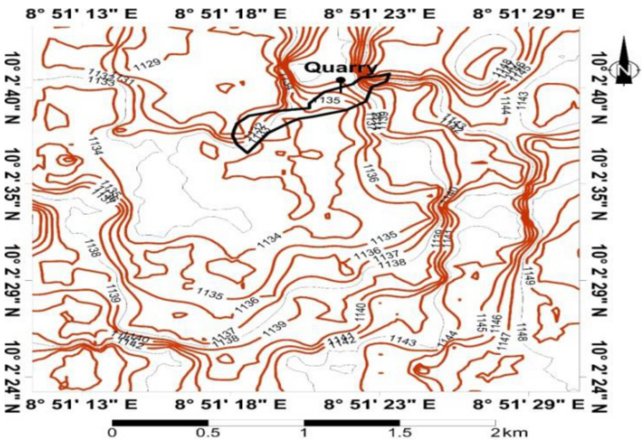

Figure 1. Contour Map of the Study Area

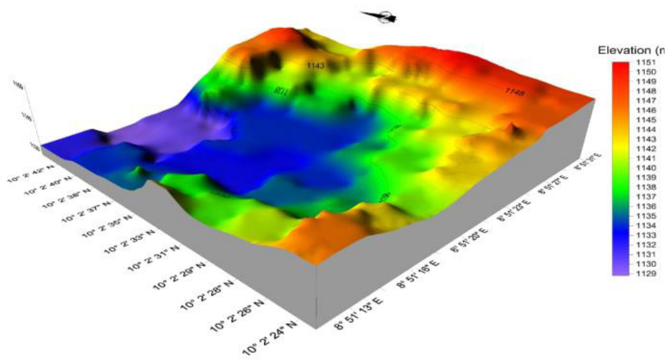

Figure 2. Digital Elevation Model (DEM) of the Study Area

Figure 1 shows the contour map of the study area where aggregate stone is being mined. Figure 2 shows the Digital Elevation Model (DEM) of the study area

Materials and Methods

Materials

Field study or measurement of the study area was carried out using the following working tools and/or resources:

i. Global Positioning System (GPS) device: To obtain coordinate and elevation within the study area.
ii. Measuring Tape: To take measurement of various blast design parameters in the quarry.
iii. Ranging Pole: To determine the depth and inclination of each drilled hole both on primary blasting and on boulders during secondary blasting operations.
iv. Field Notebook: For keeping records of both primary and secondary data.
v. (Phone) camera: To take pictures during research within the study area.

Other resources are:

a. Drill and blast data from field studies and desk blast design at JCC quarry; and
b. Microsoft-Excel software was used for the analysis of drilling performance and sensitivity analysis.

Methods

Main methods employed in the research include:

i. Evaluation of drill and blast performance;
ii. Model design modification, verification and comparison with two existing models; and
iii. Sensitivity analysis at Jolex Construction Company Quarry.

Drill and Blast Performance Evaluation

Various literature sources were continuously reviewed to add knowledge and understanding to the problem statement throughout the investigation. Preliminary research related to the theme of the study includes conventional data-gathering techniques like interviews with quarry manager, other employee of the mine, and group-based discussions with the quarry members of staff. These insights added qualitative value and support to a mainly quantitative investigation. Documents, records, and desk design data that were also retrieved from the information relevant to the project scope were made available with permission from the general manager of the operation. Another method of investigation was based on a time study where the records of overall drilling and blasting cost over 12 eight-hour shifts simultaneously. Side-by-side on-side observations were recorded. This gave an opportunity to record multiple 'day-in the-life' of observations whereby half-hourly events could be logged.

These methods can also be categorized as visual observations alone over a period of 96 hours. Quantitative data obtained were used to estimate the cost of secondary breakage and degree of blast design deviation. During the field study, blasting operations followed immediately after the drilling process. The blast holes were measured again before the charging process commences. Commenced blast holes were primed, and the explosives were charged into each blast hole according to the designed charging plan. The blast holes were stemmed. The shots were connected together and fired after total evacuation of personnel and property from the shots had been ensured. A single blast constituting of 130 blast holes were used for the studies at the study area. Prior to drilling, during drilling and post drilling operation, the following steps are involved in drill and blast performance evaluation.

Estimation of Blast Design Percentage Deviation

The degree of deviation table below shows the values of all the percentage deviations. This is achieved by dividing the total number of blast design parameters deviated by the total number of the blast design parameters. The fraction obtained is then multiplied by 100% as shown in the equation below:

Using the equation below;

$$\frac{A_i n}{T_o} \times 100 = \text{Ai}\,\% \qquad ..\text{Equ. 1}$$

Where A_i = Degree of blast design parameter deviated.

$A_i n$ = Number of observed blast design parameters deviated.

T_o = Total number of blast holes.

T (%) = Total sum of percentage deviation.

$A_i\%$ = Percentage deviation.

Estimation of Hole Inclination Percentage Deviation

The practical steps involved in determination of hole inclination deviation during field study are:

i. Each hole inclination is measured by insertion of ranging pole into it and placement of another vertical ranging pole rod beside it from the hole surface.

ii. Application of trigonometric expression to determine either right-inclination deviation or left-inclination deviation. This is shown in the figure 3;

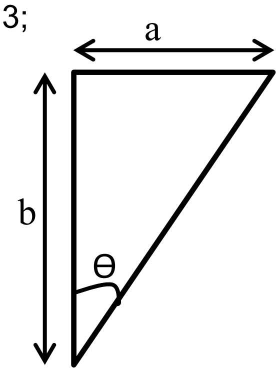

Figure 3: Set-Up for Inclination Measurement

The percentage deviation is achieved by dividing the total number of deviated holes by the total number of blast holes by the total number of blast holes. The value obtained is then multiplied by 100% as shown in the equation 3.2:

Using the equation;

$$\frac{n_i}{T_o} \text{x}100 = \ \Theta\text{i}\% \qquad ...\text{Equ. 2}$$

Where, Θ_i = Degree of blast hole deviated by inclination.

n_i = Number of observed inclination deviated from blast design.
T_o = Total number of blast holes.
T(%) = Total sum of percentage deviation.
Θ_i% = Right or left percentage deviation.

$$\Theta = \tan^{-1}\left(\frac{a}{b}\right) \quad \ldots \text{Equ. 3}$$

Therefore:

$$90 - \Theta_R = \Theta_{INC} \quad \ldots \text{Equ. 4}$$

OR $90 - \Theta_L = \Theta_{INC}$ ... Equ. 5

Where,
Θ = Hole deviated inclination.
Θ_R = Right deviated inclination.
Θ_L = Left deviated inclination.
Θ_{INC} = The true blast hole inclination measured during the field study.

Hence; Percentage deviation will now be expressed using the equation below:
Inclination percentage deviation

$$= \left(\frac{\text{Number of blast hole deviated}}{\text{Total number of blast hole}}\right) \times 100\% \quad \ldots \text{Equ. 6}$$

Models Used in Drilling and Blasting Model Modification

The cost estimation model developed here includes the performance targets such as the total cost of drilling and blasting operation. There are a number of models that can be used to estimate drilling and blasting cost. However, the two major existing models used in this analysis are:

- Ghanizadeh's model; and
- Jimeno's model.

1. The Ghanizadeh et al. (2017), model developed by a group of four scientists in 2018. The model is one of the most recent models and is a result of latest discovery and modification in mining.

A drawback of this model is that it needs verification and calibration by other means and the geological data of the case study needs to put into consideration. This is due to the nature of the rock being blasted. The rock properties change significantly from site to site, even within the same site these properties may change over relatively short distances. According to Ghanizadeh (2017), other points to note concerning the model include:

i. All blasting costs were modeled in comfar technical and economic analysis software to calculate the cost per cubic meter of the broken rock, and
ii. 87% of the blasting operation costs depend on the cost of ANFO and drilling costs.

2. The Jimeno drilling and blasting cost estimation model uses a modified version of the Ghanizadeh model to predict the cost of both drilling and blasting operation. The model uses depreciation cost, interest rate and royalty cost, insurance cost, maintenance and repair cost, labour cost, fuel or energy cost, drilling bits and rods cost, drilling, productivity, and explosive parameters to predict the entire drilling and blasting cost. Unlike Ghanizadeh model, this model is a singler hole model (Kecojevic, et al. 2014). i.e, it assumes the same parameters (blast pattern) for the entire blast volume. However, in practical situations, each input parameter will have same variations associated with it. For example, rock mass properties such as joint spacing and strength can vary due to drilling inaccuracy.

a. Ghanizadel's Model

Ghanizadel's model is given as:

$$C_1 + C_2 = 87\%\, BC \quad \ldots \text{Equ. 7}$$

$$BC = \frac{1}{0.87}(C_1 + C_2) \quad \ldots \text{Equ. 8}$$

$$BC = 1.15\,(P_A \times S_C + P_D \times S_D) \quad \ldots Equ. 9$$

Where; C_1 = ANFO cost.
C_2 = Drilling cost.

BC = Blasting cost per cubic meter.
P_A = Price of ANFO per kilogram.
P_D = Price of drilling per meter.
SC = Specific charge (kg/m^3)

b. Jimeno's Model

The Jimeno's model is given as (Jimeno, et al):

$$C_T = \frac{C_A + C_I + C_M + C_O + C_E + C_L}{P_r} + C_B \quad \text{... Equ. 10}$$

Where; C_T = drilling cost per meter (₦/m)
C_A = Depreciation cost per hour (₦/h)
C_I = Interest rate and Insurance per hour (₦/h)
C_M = Maintenance and repair cost per hour (₦/h)
C_O = Labour cost per hour (₦/h)
C_E = Fuel or energy cost per hour (₦/h)
C_L = Cost of oil, grease and filters per hour (₦/h)
C_B = Cost of bits, rods, sleeve and shanks (₦/h) per hour.
P_r = Drilling productivity in (m/h)

$$C_{TB} = C_T + C_{ANFO} + C_H + C_L + C_A \quad \text{... } Equ. 11$$

Where; C_{TB} = Total cost of Blasting (₦)
C_T = Transportation cost (₦)
C_{ANFO} = Cost of ANFO (₦)
C_H = Cost of high explosive (₦)
C_L = Labour cost (₦)
C_A = Cost of explosive accessories (₦).

Cost Estimation of Secondary Drilling and Blasting Operation.

Secondary fragmentation usually through drilling and blasting techniques are employed at the study area. One-month data was collected to estimate the cost of drilling and blasting boulders, toes and elevated floors resulting from inappropriate drill and blast operations in the quarry. The data was captured and processed using Ghanizadeh's drilling and blasting cost evaluation model, and the modified model for cost estimation for both drilling and blasting elevation developed through this research. The results were compared, and the causes of variations and similarity from one model to another explained. The result will be interpreted and its effect on the quarry productivity and production rate determined. This is considered as cost saving methodology.

Sensitivity Analysis.

The drilling and blasting parameters in blast design were varied and the cost of drilling and blasting was determined for each parameter used. The sensitivity analysis was performed to determine the variation in cost of drilling and blasting at varied parameters on the base case of Net Present Value (NPV). The hole diameter and bench height was flexed within the range of ±50% and a sensitivity analysis was computed using Microsoft-Excel to determine its effect on the overall cost, while keeping the other parameters constant as used by Saliu et al., (2017).

From the graph plotted, mathematical expression will be established, to show the relationship between the two variables at any point on the graph. The idea of equation of a straight line is adopted to deduce the corresponding cost at any flexed parameter's magnitude. This mathematical equation of vector on a straight line is given below:

$$\frac{y - y_1}{x - x_1} = \frac{y_2 - y_1}{x_2 - x_1} = m \quad \text{... Equ. 12}$$

Where; y = dependent variables
x = independent variables
m = slope or gradient

Results and Discussion

The geometric drill parameters which include hole depth, hole inclination, burden and spacing were assessed in each blast. The deviation of the measured parameters to the designed parameters were analysed for all the 130 blast holes. The drill parameters as listed in Table 1 are used in the study. The absolute deviations for the hole depth, burden and spacing, less than or equal to 0.3m from the designed were considered as acceptable error, absolute deviations between 0.3 and 0.5m from the designed need improvement, while deviation greater than or equal to 0.5m are considered as unacceptable error. In line with the design of the quarry, the absolute deviations for the hole inclination, less than or equal to 5^0 from the design were considered as acceptable error, absolute deviations between 5^0 and 10^0 from the designed need improvement, whole deviation greater than or equal to 10^0 are considered as unacceptable error.

Table 1: Designed Drilling and Blasting Parameters

Parameter	Value
Hole diameter (mm)	105
Burden (m)	2
Spacing (m)	2
Hole inclination (^{0})	90
Bench height (m)	4
Sub – drill (m)	0.4
Number of drill holes	130
Hole depth (m)	3.8
Stemming (m)	1.1
Drilling productivity (m/h)	15
Boulders hole depths (m)	0.52
Blasting pattern	Staggered
Number of rows	12
Powder factor	0.72

Table 2: Explosive Properties Data

Properties	**Explosive Type/Detonation Method**
Column charge	ANFO
Bottom charge	Gelatin
Detonators	Electric method

Table 3: Rock-mass Characteristics

Properties	Value
Rock type	Laterite, Granite
Rock mass	Generally layered
Compressive strength (MPa)	120
Mohr’s scale of hardness	6 – 7
Specific gravity	2.65

The compliance of the blast hole depth to the design and the deviations were assessed as part of the study. Figures 4 and 5 show the analysis of the blast hole depth deviations in the quarry. The results are summarized in the Table 4:

Table 4: Percentage Hole Depth Deviation

Degree of hole depth deviation	Number of holes deviated	Percentage
Under-drilled above 0.5m	17	13%
Under-drilled between 0.5m and 0.3m	9	7%
Under-drilled between 0.3m and 0m	47	36%
Over-drilled between 0m and 0.3m	49	38%
Over-drilled between 0.3m and 0.5m	5	4%
Over-drilled above 0.5m	3	2%
Total	130	100%

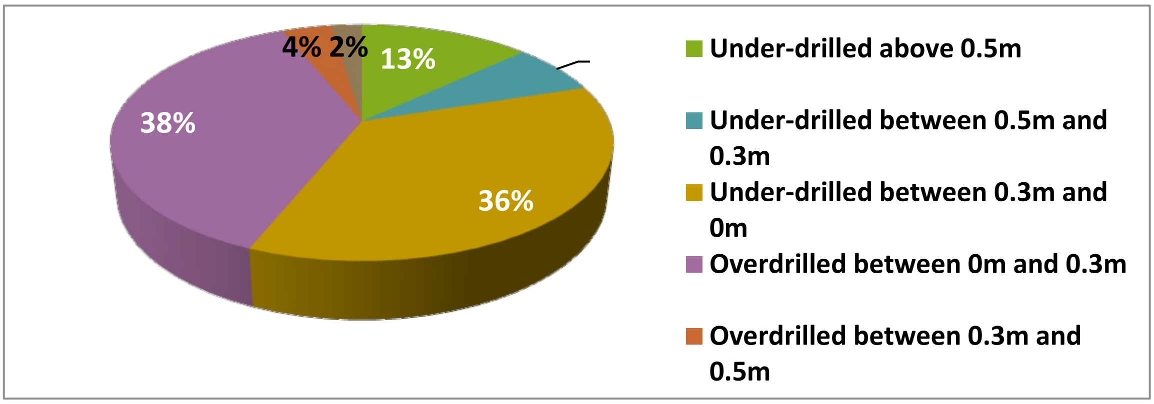

Figure 4: Hole Depth Deviations (m)

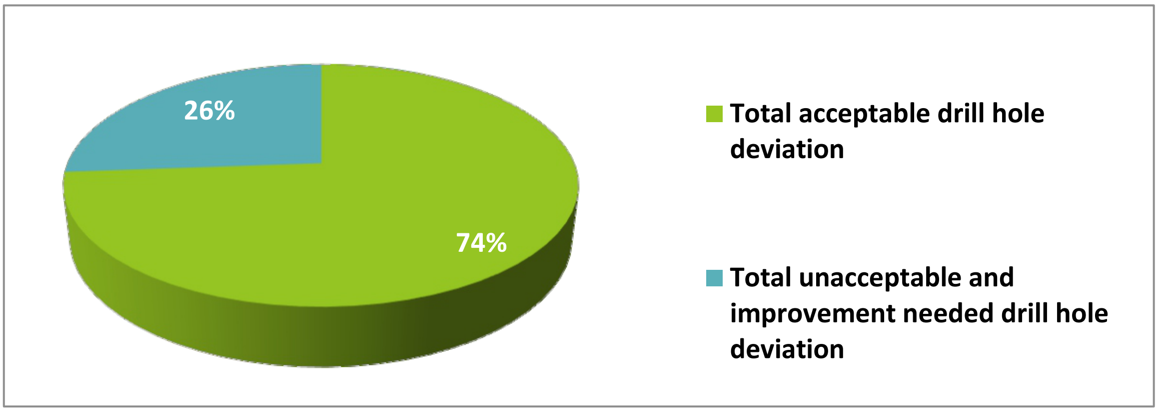

Figure 5: Hole Depth Deviations (%)

From Table 4 and Figure 4, the blast hole depth compliance analysis on the 130 production blast holes shows that about 11% of the blast hole depths deviated between ±0.3m and ±0.5m (i.e. 7% under-drilled and 4% over-drilled). This needs improvement per the technical requirement of the quarry. However, about 15% of the blast holes have depth deviation in excess of ±0.5m from the designed (i.e. 13% under-drilled and 2% over-drilled). This indicates unacceptable levels per the technical requirement of the Quarry. From Figure 5, about 26% of the total blasts studied had their hole depths not meeting the acceptable levels of the quarry. These under-drilled and over-drilled blast hole depths are potential causes of elevated floors (or mine floor undulation) and toes respectively on the bench floor after blasting and mucking operations. Secondary drilling of these toes or elevated floors were regular activities observed in the pits.

The compliance of the spacing distances from the designed and the allowable deviations were assessed as part of this study. Figures 6 and 7 show the deviation analysis of the spacing distance in the quarry. The result is shown in Table 5.

Table 5: Percentage Spacing Distance Deviation

Degree of spacing distance deviation	Number of spacing deviated	Percentage
Under-spacing above 0.5m	0	0%
Under-spacing between 0.5m and 0.3m	1	1%
Under-spacing between 0.3m and 0m	15	12%
Over-spacing between 0m and 0.3m	79	62%
Over-spacing between 0.3m and 0.5m	24	19%
Over-spacing above 0.5m	8	6%
Total	128	100%

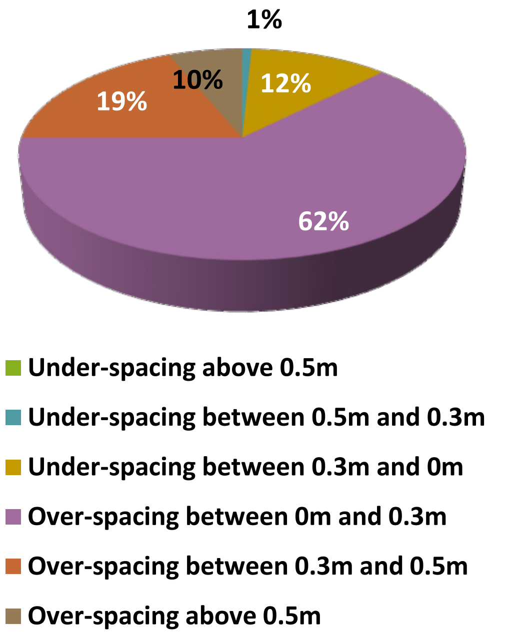

Figure 6: Spacing Distance Deviation (m)

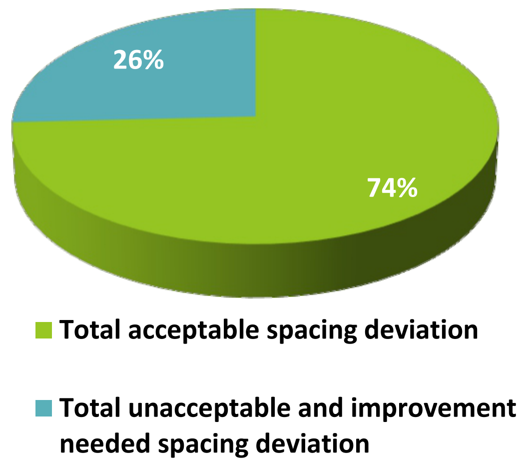

Figure 7: Spacing Distance Deviation (%)

From Table 5 and Figure 6, about 20% of the holes had spacing deviation between ±0.3m and ±0.5m (i.e. 1% under-spacing and 19% over-spacing). Per the technical requirement of the quarry, improvement is necessary to correct this anomaly in spacing deviation. Only 6% of the measured spacing deviations were in excess of ±0.5m from the designed values (i.e. 0% under-spacing and 6% over-spacing). Similarly, about 26% of the spacing measurements deviated from the planned spacing. These deviations will inevitably affect the fragmentation of the blast, thus, producing significant boulders and hence secondary breakages activities in the quarry. This confirms the numerous boulders associated with the blast fragments observed during the field study.

The compliance of the burden distances from the designed and the allowable deviation were assessed as part of this study. Figure 8 and 9 show the deviation analysis of the burden distance in the quarry. The result is shown in Table 6:

Table 6: Percentage Burden Distance Deviation

Degree of burden distance deviation	Number of burden deviated	Percentage
Under-burden above 0.5m	0	0%
Under-burden between 0.5m and 0.3m	0	0%
Under-burden between 0.3m and 0m	35	27%
Over-burden between 0m and 0.3m	70%	54%
Over-burden between 0.3m and 0.5m	13	10%
Over-burden above 0.5m	12	9%
Total	130	100%

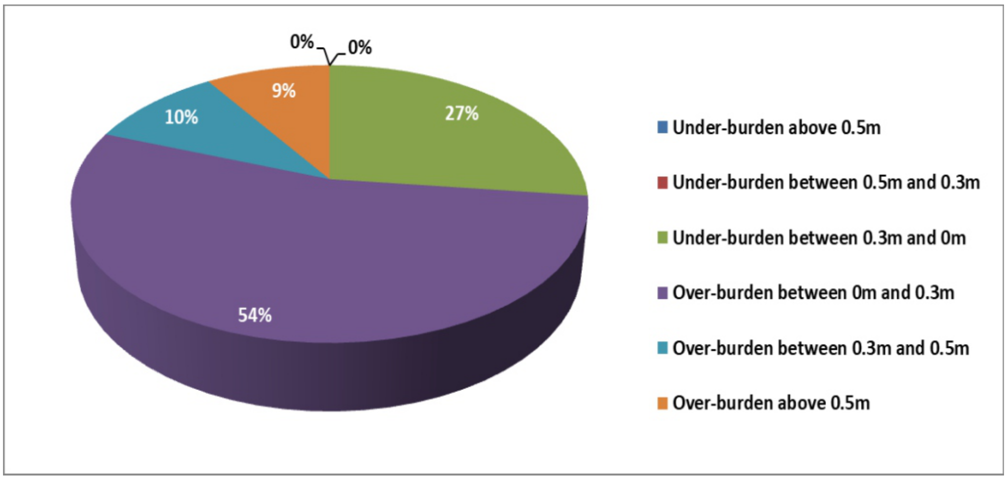

Figure 8: Burden Distance Deviation (m)

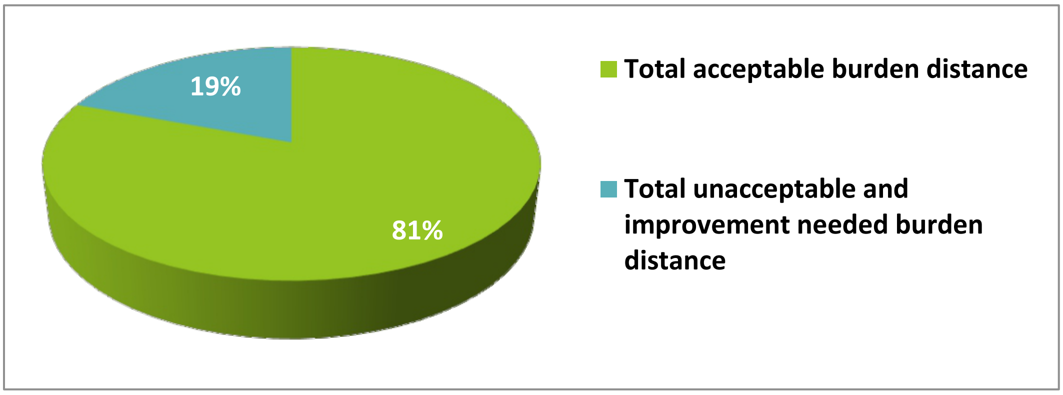

Figure 9: Burden Distance Deviation (%)

Table 6, Figures 8 and 9 show the deviations analysis of the burden distances measured as part of the study. From Figure 8, about 10% of the holes had burden deviation between ±0.3m and ±0.5m (i.e. 0% under-burden and 10% over-burden) per the quarry's technical requirement, there is the need for improvement in the burden distances during primary blasting about 9% of the blast holes had burden deviations in excess of ±0.5m from the designed value (i.e. 0% under-burden but 9% over-burden). Again, about 19% of the blast deviated from the planned burden distance.

The compliance of the blast hole inclination to the design and the deviation s were assessed as part of the study. Figures 10 and 11 show the analysis of the blast hole inclination deviations in the quarry.

Table 7: Percentage Hole Inclination Deviation

Degree of hole inclination deviation	Number of holes deviated	Percentage
Right-inclination above 10^0	0	0%
Right-inclination between 5^0 and 10^0	1	1%
Right-inclination between 5^0 and 0^0	56	43%
Left-inclination between 0^0 and 5^0	72	55%
Left inclination between 5^0 and 10^0	1	1%
Left-inclination above	0	0%
Total	130	100%

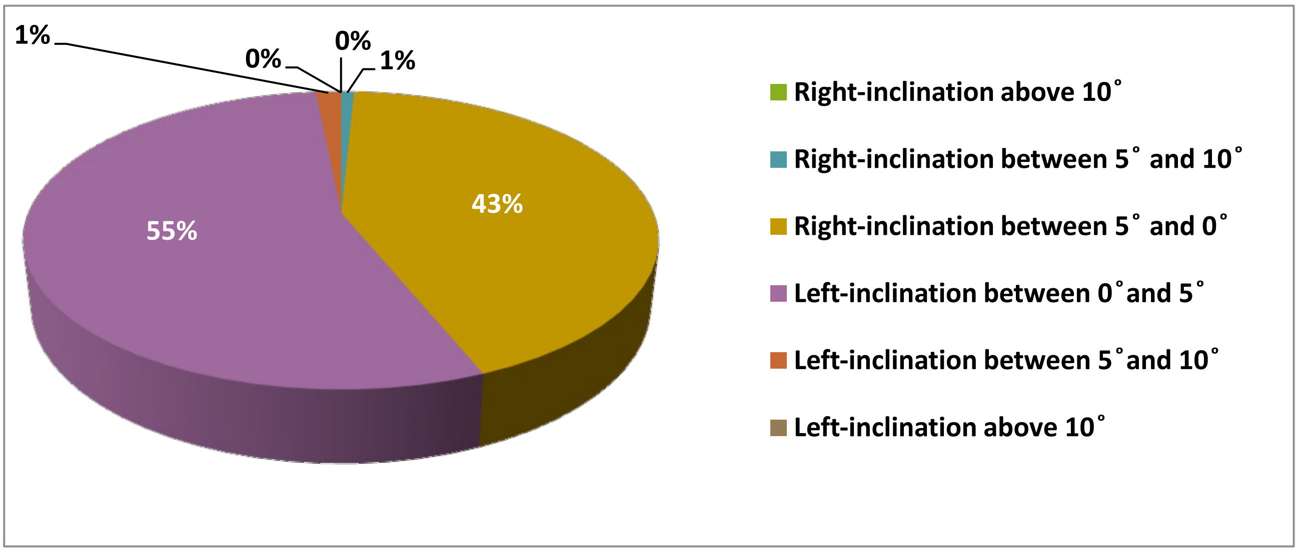

Figure 10: Hole Inclination Deviation (^{0})

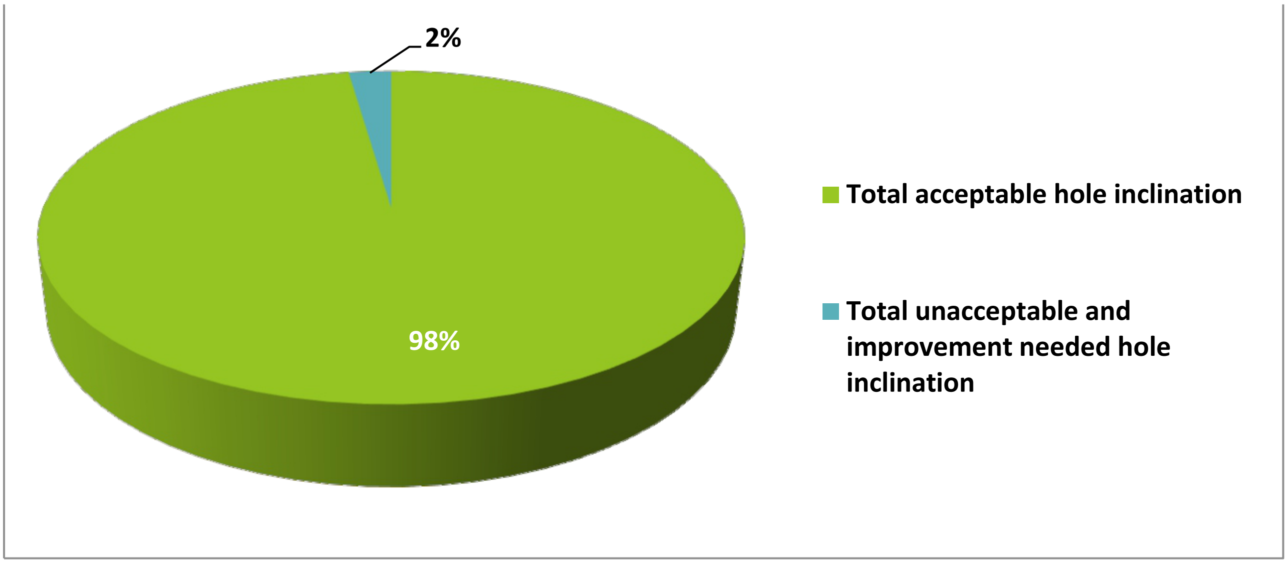

Figure 11: Hole Inclination Deviation (%)

Form Table 7, and Figure 11, the blast hole inclination compliance analysis on the 130 production blast holes shows that about 2% of the blast hole inclinations deviated between 5^0 and 10^0 either to the right or left (i.e. 1% right-inclination deviation and 1% left-inclination deviation). This needs improvement for the technical requirement of the quarry.

However, none of the blast holes had inclination deviations in excess of 10^0 either to the right or left from the designed value. Again, exactly 2% of the blast holes considered for the blast deviated from the planned inclination angle. This indicates minor unacceptable levels per the technical requirement of the quarry. Therefore, only 2% of the total blasts studied had their hole inclination s not meeting the acceptable levels of the quarry. From the analysis above, one can easily suggest the magnitude of errors is negligible, but these iota errors also have contributions or potentials to cause elevated floors and toes problems after blasting and mucking operation. These in turn result to additional cost of secondary drilling and blasting for their corrections.

Ghanizadeh and Jimeno's models are the two existing model combined with the designed modification model applied to evaluate the cost of drilling and blasting of secondary breakage. Three approximate results (values) were obtained, and the average value calculated. The following are the observed reasons for the result variation, before taking the average estimated cost in Naira (Currency unit):

i. Each model is designed at different study area, where different factors are put into consideration.
ii. Lease or hiring cost varies from one study area to another (e.g. from Ghana to Nigeria).
iii. Some equipment are hired, owned or available at a cheaper price relative from one study area to another.
iv. Tax or tariff varied from one country (Location/Study area) to another in magnitude.
v. The cost of hiring is measured per hour in one quarry, while other quarry hired almost nothing (Thus eliminating the hiring cost in other place). Therefore, the estimation is relative.

The uniqueness of the designed model as compared with the other two existing models is obvious. Since the cost estimation model for breakage developed through this research uses a modified version of both Jimeno and Ghanizadeh's models to predict the cost of both drilling and blasting in the study area. The model uses both drilling and blasting parameters to predict the cost. This model does not only use some obvious factors such as depreciation cost, interest and royalty cost, maintenance and repair cost, labour cost, fuel or energy cost, and some explosives parameters, but also put some variables into considerations such as: hiring cost of bits, rods, sleeves, shanks, wagon drill and compressor, and security (supervision and coordination) expenses, to predict the cost.

Unlike some models, this model is a single hole model, i.e. it assumes the same parameters (blast pattern) for the entire blast volume. It is worth noting that in practical situations, each input parameter would have some variations associated with it. For instance, the hole depth of blast holes may be varied due to some factors such as surveying result in reduce level (RL).

Drilling Cost Model Modification

$$\mathrm{CTD} = \left(\frac{C_{OPR}+C_{EOF}+C_{DEP}+C_{MAR}+C_{BRS}+C_{WAC}+C_{OTHER}}{P_r}\right) \quad \ldots \mathrm{Equ.}\ 12$$

Where: C_{TD} = Total cost of drilling (₦/m)
C_{OPR} = Operator/Labour cost (₦/h)
C_{EOF} = Fuel or Energy cost (₦/h)
C_{DEP} = Depreciation cost (e.g. Interest, Insurance and royalty, and Value) (₦/m)
C_{MAR} = Cost of Maintenance and Repair (Grease, oil, filter and services cost) (₦/h)
C_{BRS} = Hiring cost of Bits, Rods, Sleeve and Shanks (₦/h)
C_{WAC} = Hiring cost of wagon drill (₦/h)
C_{OTHER} = other costs such as security, supervision and coordination.
P_r = Drilling Productivity (m/h)

Blasting Cost Model Modification

$$C_{TB} = (C_{LAB} + C_{GEL} + C_{ANFO})n + C_{TRA} + C_{DAC} + C_{DPW} + C_{RIT} + C_{OTHER} \quad \ldots Equ\ 13$$

Where; C_{TB} = Total cost of blasting (₦/)
C_{LAB} = Labour cost (₦/hole)
C_{GEL} = Cost of High explosive (₦/hole)
C_{ANFO} = Cost of ANFO (optional) (₦/hole)
C_{TRA} = Cost of transportation (₦/)
C_{DAC} = Cost of electrical detonator and other explosive accessories
C_{DPW} = Cost of (Detonating source of power) (₦)
C_{RIT} = cost of Insurance, Tax and Royalties (₦)
C_{OTHER} = Other cost such as Anti-bomb squad charges and other security measures (₦)
n = Number of blast holes.

The cost estimation model for drilling and blasting operation developed above is based on the following assumptions:

i. All equipment operates at their 100% of its nominal capacity, while the drill and blast performance can be good, fair or poor.
ii. Drilling and blasting require less planning and poor design to produce poor performance but more effort, good precision and field design accuracy to produce good performance.
iii. There is presence of anti-bomb squad in every blasting operation.
iv. Drilling productivity decrease with poor maintenance and repair.
v. There is always approval before any blasting operation.

vi. The wage of each worker can be paid hourly, daily, weekly, monthly, yearly etc.
vii. The cost of insurance, royalty, tax, lease or rent and source of power varies with location (Such as country or study area).
viii. Both equipment and personnel can be hired for any period of time.
ix. Equipment needs service and appropriate inspections prior to any drilling and blasting operation.
x. The geometry of all blast hole is the same.

Table 8: Cost of Variables in Drilling and Blasting

Variable descriptions cost/unit	Value
Cost of high explosive (₦/kg)	950
Cost of ANFO (₦/kg)	780
Transportation cost (₦)	70000
Cost of detonating cord (₦)	160
Cost of firing cable (₦)	75000
Cost of electric detonator (₦)	800
Labour/operator cost (₦/h)	2500
Wagon drill fuel cost (₦/h)	625
Compressor fuel cost (₦/h)	2475
Cost of hiring compressor (₦/h)	5000
Cost of hiring wagon drill (₦/h)	5000
Cost of relay and delay (₦)	13650
Royalty and interest (₦/blast)	100000
Land owner lease cost (₦/day)	6944
Anti-Bomb squad cost (₦)	5000
Maintenance and repair (₦)	35000
Other cost (₦)	85000

Table 9: Secondary Drilling and Blasting Cost Using Modified Model

Operation (₦)	Value (₦)
Total drilling cost	87350
Total blasting cost	207680
Total	295 080

Table 10: Secondary Drilling and Blasting Cost Using Ganizadeh's Model

Operation (₦)	Value (₦)
Total drilling cost	75 400
Total blasting cost	183 900
Total	259 300

Table 11: Secondary Drilling and Blasting Cost using Jimeno's Model

Operation (₦)	Value **(₦)**
Total drilling cost	93750
Total blasting cost	216 850
Total	310 600

Table 12: Average Secondary Drilling and Blasting Cost

Operation (₦)	Value **(₦)**
Total drilling cost	85500
Total blasting cost	202810
Total	288310

The resulting boulders, elevated floors, and toes form primary blasting in the Jolex Construction Company Quarry were handled using the re-drilling and blasting method. In a period of one month, a total of 86 holes of boulders were drilled at an average depth of 0.52meters (1.72ft) per hole while that of elevated floor and toe drilling was 62 holes at an average depth of 2.2m per hole. With an average cost of drilling and blasting per meter of ₦16, 500, then for the period of one month, the quarry spent ₦2, 250, 600 in drilling and blasting boulders and elevated floors (or toes) respectively. This implies that, for the period of one month, the quarry lost approximately ₦2 988 480 or ₦1 494 240 per month resulting from poor performance of drilling and blasting.

Table 13: Cost of Drilling and Blasting at Various Hole Diameter

Hole diameter (mm)	Mean cost of drilling (₦)	Mean cost of blasting (₦)	Total cost (₦)
60	110 000	135 000	245 000
80	111 500	160 000	271 500
100	112 000	185 400	297 400
120	113 000	207 500	320 500
140	114 500	240 000	354 500
160	115 000	270 000	385 000
180	117 000	295 000	412 000

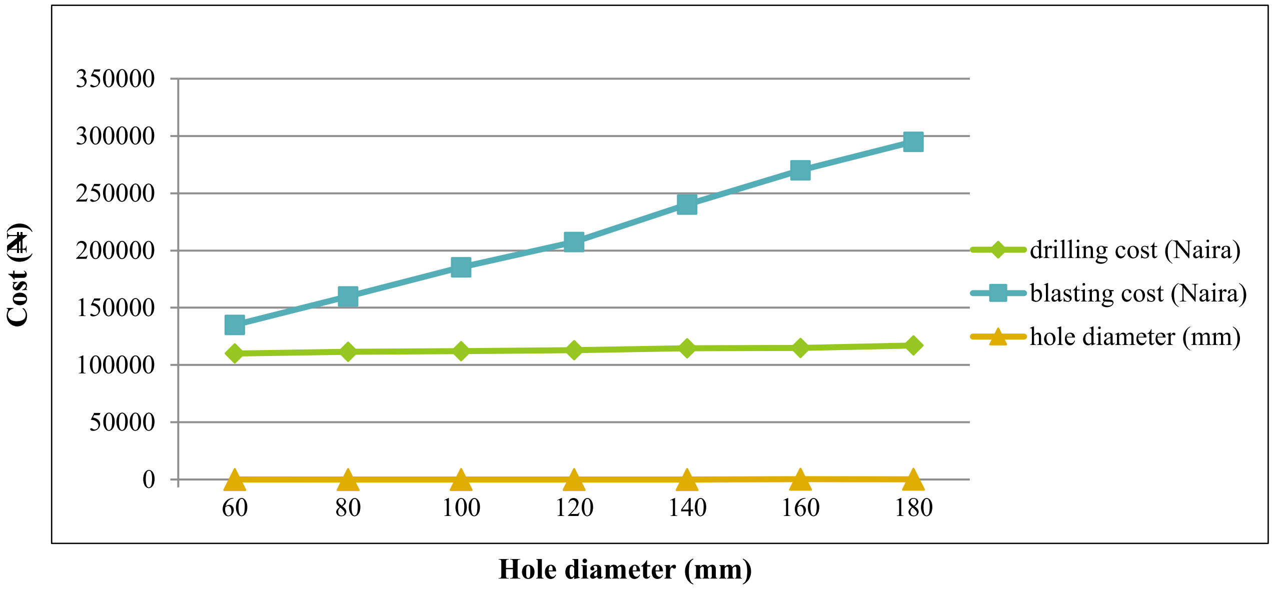

Figure 12: Relationship between Hole Diameter and Cost of Drilling and Blasting

Blasting cost approximate equation from the graph

$C_B = 1375D_h + 51500$ … Equ. 14
Where; C_B = cost of blasting (₦)
D_h = hole diameter (mm)

Drilling cost approximate equation from the graph

$C_D = 58.3D_h + 106500$ … Equ. 15
Where; C_D = cost of drilling **(₦)**
D_h = hole diameter (mm)

The practical burden used in each case was derived from Konya and Walter's formula multiplied by an "uncertainty factor" of 1.053 (or divided by 0.95). All other parameters were calculated based on the adjusted burden. The reason for this "uncertainty factor" is to account for the variation between what is obtainable from Konya and Walter's formula and what was practicable on the sites under consideration. The hole diameter was flexed within the range of ±50% and a sensitivity analysis was computed using Microsoft-Excel to determine its effect on the overall cost while keeping the other parameters constant. Table 13 and Figure 12 show that increasing hole diameter, when all other parameters are kept constant, corresponds to an increase in blasting cost and no significant change in drilling cost.

Again, approximately one hundred and thirty 105mm diameter holes were drilled and the diameter was flexed between the range ± 50%. The practical burden used was 2m. Thus, figure 13 shows the effect of change in hole diameter on the overall drilling and blasting cost.

Table 14: Cost of Drilling and Blasting at Various Bench Heights

Bench height (m)	Mean cost of drilling (₦)	Mean cost of blasting (₦)	Total cost (₦)
2.0	423, 250	796, 500	1, 219, 750
2.5	426, 600	856, 920	1, 283, 520
3.0	432, 850	908, 350	1, 332, 200
3.5	439, 300	963, 700	1, 403, 000
4.0	444, 600	1, 055, 460	1, 500, 060
4.5	449, 370	1, 103, 700	1, 553, 070
5.0	453, 640	1, 160, 200	1, 613, 840
5.5	458, 720	1, 216, 350	1, 675, 070
6.0	465, 150	1, 270, 600	1, 735, 750

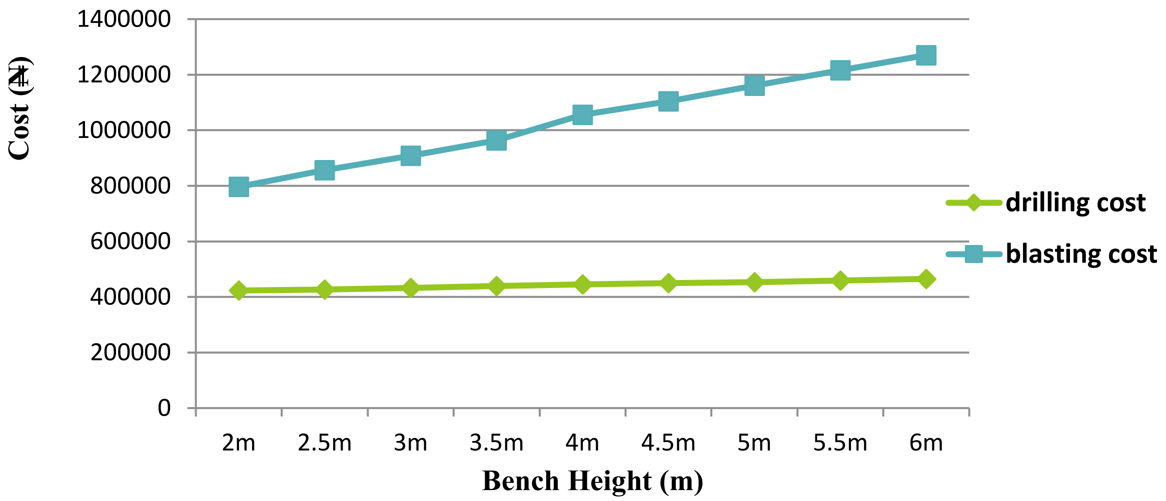

Figure 13: Relationship between Bench Height and Cost of Drilling and Blasting

Blasting cost approximate equation from the graph

$C_B = 118525B_h + 560000$ …Equ. 16

Where; C_B = blasting cost (₦)

B_h = bench height (m)

Drilling cost approximate equation from the graph

$C_D = 8225B_h + 410513$ … Equ. 17

Where; C_D = cost of drilling (₦)

B_h = bench height (m)

For every increase in bench height, there must be a corresponding increase in the depth of hole to be drilled and also the amount of hole to be drilled and also the amount of explosives required to fill these holes. Therefore, in light of the above statement, it can be inferred that increase in bench height causes an increase in both drilling and blasting cost as shown in Table 14 and Figure 13. The fragmentation of the blast became more satisfactory as the bench height increased which confirms the optimum bench height to burden ratio. Thus, Figure 13 shows that increase in bench height causes an increase in both drilling and blasting cost.

The assumption of the four models generated through sensitivity analysis is that the graph is a straight-line graph. Therefore, the vector equation of a straight line is used.

Conclusions

From the study and analysis, it could be concluded that deviation in the blast hole depth, burden and spacing were very high, except in the case of hole inclination, in which the degree of deviation seems negligible. These deviations are above the acceptable limits of the quarry, and they are respectively 26%, 19%, 26% and 2% for blast hole depth, burden, spacing and blast hole inclination deviations. These deviations could be attributed to operational errors on the field during the drilling process.

In addition, model design parameters are relative. Therefore, it is important to put several factors in to consideration when consider the variables. Again, the important of dimension homogeneity principles can never be over emphasized to obtain a reasonable outcome with high degree of precision and accuracy. Cost estimation models varied with respect to study are, hiring cost of equipment and personnel from one location to another, tax or tariff payment, and the working hours per shift. Thus, model design is a function of geographical description of the study area, which includes both social and economic activities.

In this study, the mathematical model developed to estimate the cost of secondary breakage is a modification of two existing models. Furthermore, the sensitivity analysis performed is to take into account uncertainty of dominant influence factors on the total cost of drilling and blasting. Data from the granite quarry were used throughout this study. From this analysis, when hole diameter is increased, and all other parameters are kept constant, there is a corresponding increase in blasting cost and no significant change in drilling cost provided the powder factor of 0.72kg/m^3 is not exceeded. Also, increase in bench height causes a corresponding increase in both drilling and blasting cost and that bench height limits the size of the charge diameter and the burden in a given blast design.

However, increase in burden or spacing decreases the number of holes to be drilled and consequently the amount of explosives needed to fill these holes which ultimately decreases the overall drilling and blasting cost. The good news is that, both drilling and blasting cost at any flexed parameters are predictable approximately using the established mathematical relationship from the graph plotted. However, all these input data have a major impact on setting appropriate drilling and blasting patterns. Therefore, further research is required to address and reflect the changes in drilling and blasting patterns within the same blasting zone.

Finally, from the above explanations, it can therefore be concluded that the huge deviations in the drill design geometric parameters (spacing, burden, hole depth and inclination) were the main factors that cause the quarry elevated floor (mine floor undulation), poor fragmentation of the blasts, resulting in boulders and toes in the pit. This subsequently leads to the excessive cost on secondary breakage to achieve the required mill throughout. This averagely costs the Quarry ₦1 494 240 per month.

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Geomechanical Evaluation for Economic Purposes of Migmatitic Rocks of Ayere, Southwestern Nigeria

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ABSTRACT

Migmatitic rock outcrops and are widely quarried in Southwestern Nigeria. The rarity of the studied rock is noteworthy. Five samples were collected at random from the study area. The samples' thin sections and photomicrographs were studied with the petrographic microscope and *Image J* respectively. Constituent minerals with their modal composition included quartz/40%, plagioclase/27%, hornblende/24%, and biotite/9%. The Schmidt hammer was used to capture, in line with ISRM procedure, a rebound hardness of 52.6. Results of other physico-mechanical tests carried out in line with ASTM methods include 0.051% for NMC, 0.70% for porosity, 2.78 for specific gravity. Slake durability is 97.46%, point load is 8.38MPa, AIV is 22.3%, ACV is 23.8%, direct UCS is 189.2MPa while rebound hardness and Point load gave 183MPa and 201.1MPa respectively. Inclusions of finer grains of zircon and quartz in larger grains of plagioclase and quartz were observed. Multiple stages of orogenesis, inferred from the presence of inclusions, is believed to have impacted beautiful patterns (aesthetics) to the rock. Conclusively, the patterns, the inclusions, high quartz composition, and the high values of the physico-mechanical parameters imply high strength and aesthetics translating to great economic value if exploited for dimension and aggregate stones production.

Keywords: Physico-mechanical, Migmatite, Aesthetic, Rebound hardness, Schmidt hammer

INTRODUCTION

Rock materials are useful for various engineering purposes, mainly because of their strength characteristics. The behavior of rocks mostly depends on their physical and mechanical characteristics. Therefore, there is a need to know the strength characteristics of the available rock materials to apply each to its appropriate use/s (Ademeso *et al*., 2008). Migmatite is a heterogeneous silicate rock with properties of both igneous and metamorphic rocks (Allen, 1992). Sawyer *et al*. (2011) described migmatite as partially melted rocks of the continental crust, which is made up of two components the neosomes and paleosomes. Neosomes crystallized from the melt (introduced or melted pre-existing), whereas the paleosomes are the unmelted rocks. In Southwestern Nigeria, migmatites are well exposed compared to other rocks and are widely employed and economically useful for construction works and ornamental purposes (Adebisi *et al.,* 2010 and Jethro *et al.,* 2014). General field disposition, petrographic attributes, physico-mechanical and geochemical characteristics of migmatites have been substantially researched (Olade and Elueze, 1979; Rahaman, 1976; Elueze, 2000). However, little or no study has been conducted on the rock in Ayere. The distinct banding and structures revealed in the Ayere migmatite are noteworthy, which stands it out from other migmatites. This research, therefore, seeks to evaluate the physico-mechanical parameters of migmatites from the Ayere area to determine the possible economic use.

Study Areas

The study area is Ayere in Kogi State, Nigeria. The outcrop located within latitudes 07°39'N and 07°43'N and longitudes 005°55'39 "E and 005°59'13"E with an altitude of about 363m above sea level (Figure 1). Ajigo *et al. (2019),* reported that Ayere is dominated by the Nigerian Basement complex rocks comprising of migmatite overlain by the north-south trending bands of metasedimentary schist and the metavolcanics.

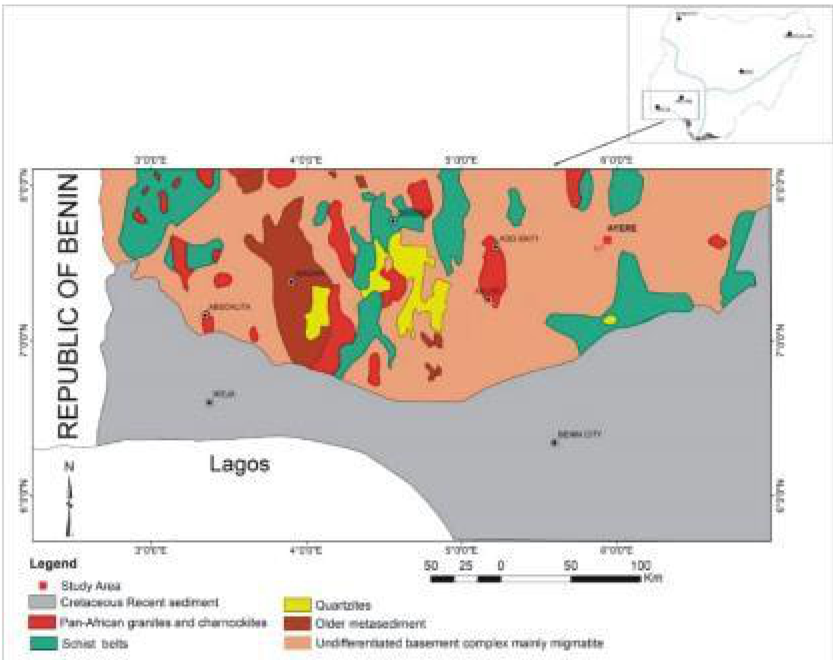


Figure 1: Geological Map of Southwestern Nigeria showing the study areas (Ademeso, 2011)

METHODOLOGY

Field observation and Insitu Test

Migmatitic rock from Ayere was studied for the physico-mechanical characteristics. The rock was observed for its field relations, texture, and structure. L-type Schmidt hammer having impact energy of 0.74Nm was used to measure the rebound hardness of the rock at the location with the coordinate taken with the aid of GPS. The test was carried out following ISRM (1978a) and ASTM (2001) methods. A special note was made of the positioning of the hammer. Ten sets of ten readings were acquired on the rock from the location. The rebound values were treated accordingly. The mean of the remaining values was referred to as the adjusted mean in this work. The adjusted mean was then used to estimate the indirect UCS with the aid of Deere and Miller (1996).

Sample Description and Laboratory work

Five representative rock samples of migmatitic were collected at random from the outcrop at Ayere and were labeled L1a-L1e. Global Positioning System (GPS) was used to record the coordinates of the points where samples were taken. Thin sections of the samples were prepared and studied under a polarizing microscope for their mineralogy, mineral association, texture, and structures. The photomicrographs of the different views of the thin sections were captured and analyzed with "Image J" version 1.40c. The moisture content, porosity, specific gravity, slake durability of the samples of the rock was determined following the standards suggested by ISRM (1981) and ASTM (1994). The UCS and Point load tests of the samples of the rock were determined following ISRM suggested methods for determining uniaxial compressive strength and point load strength (Brown, 1981). The ACV and AIV were determined in accordance with BS 812: part 110:1990 and BS 812: part 111: 1990. All the tests were carried out in engineering geology laboratory, Federal university of technology Akure.

RESULTS AND DISCUSSION

Mode of occurrence and Field Characteristics

The outcrop is massive, extensive, and reveals distinct banding (alternating light and dark bands) and structures (Figure 2). They are generally medium to coarse-grained. Structural features identified include quartzo-feldspathic veins (orientation 122-130°), joints (orientation 120-180°), pegmatitic intrusions (average width is about 34cm), foliation, symmetrical folds (Figure 2b). The strike and dip values determined on the outcrop average about 102°/71°E and 154°/50°E.

a b c d e

Figure 2: Field photograph of migmatitic rock at Ayere (a) symmetric fold, (b) alternating light and dark layered minerals, (c) pegmatitic intrusion (d) joints and (e) quartzo-feldspathic vein

Petrography

The minerals present in the studied migmatites as seen from thin sections under transmitted light (Figure 3) include plagioclase feldspar, biotite, hornblende, and quartz with the following respectively as modal composition: 27%, 9%, 24% and 40% (Table 1). Inclusions of finer grains of zircon and quartz in larger grains of plagioclase and quartz as well as the interlocking of the crystals of the major minerals were observed. Hornblende exhibits cleavage in two directions ($60^0/120^0$), brownish, and pleochroic from light green to light brown. Plagioclase grains display albite twinning. Biotite presents pleochroism as it changes from brown to dark brown under crossed-nicol and exhibits cleavage in one direction, some of the flake carry inclusions of zircon. Quartz is colorless, has an irregular shape, displays undulose extinction, and occurs as inclusions in larger grains of quartz and plagioclase.

Table 1: Mineralogical Composition of Ayere Migmatites

Minerals	Slide A	Slide B	Slide C	Slide D	Slide E	Average (%)
Quartz (%)	23	31	54	51	41	40
Biotite (%)	12	11	7	-	14	9
Plagioclase (%) Feldspar (%)	21	47	29	23	17	27
Microcline (%)	-	-	-	-	-	-
Hornblende (%)	44	11	10	27	28	24

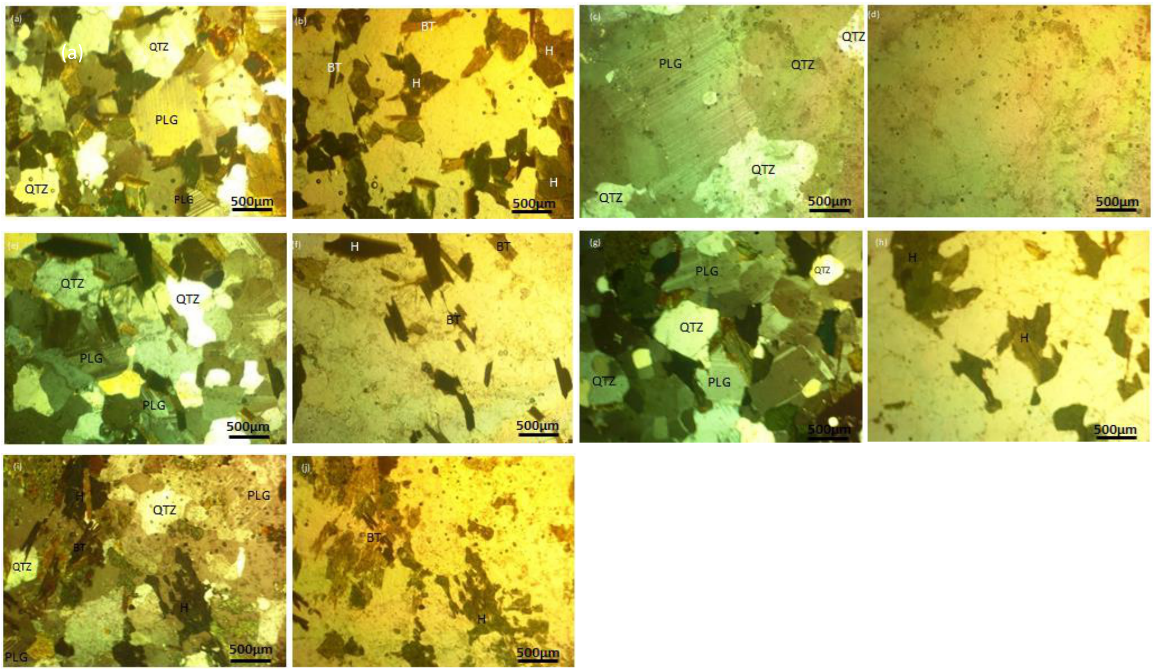


Figure 3: Photomicrographs of Ayere migmatitic rock. Figures 4a, c, e, g, and i were under cross nicol while Figures 6b, d, f, h, and j were under plane-polarized light with a magnification of X 40. (c) Migmatite showing Plagioclase displays albite twinning as well as inclusions of finer grains of zircon and quartz. Quartz shows undulose extinction. QTZ= Quartz, PLG=Plagioclase, BT= Biotite, H= Hornblende.

Schmidt Rebound Value

The mean range is 45.8 to 53.3, while that of adjusted mean is from 46.9 to 56.8. The average of the mean and adjusted mean are respectively 50.5 and 52.6, while the estimated UCS range from 135 to 270MPa, and the average of the estimated UCS is 193MPa (Table 2).

Laboratory Analysis

The values of the physical and strength properties of the migmatites are presented in Table 3, while their statistical summaries are shown in Table 4. Regression plots of the UCS with natural moisture content, porosity, and specific gravity, schmidt hammer value, percentage quartz composition, aggregate impact value, and aggregate crushing value are presented in Figures 4. The massive nature of the outcrop with few joints and petrographic characteristics of interlocking crystals could be indications of very high mechanical strength. The major minerals contained in the rocks as presented in Table 1 show that they are hard minerals that support their high mechanical strength. The modal analysis shows that quartz has the highest abundance, a factor suspected to have influenced the compressive strength of the rock, this is consistent with the findings of Plumb *et al* (1992) and Jethro *et al* (2014).

Table 2: Converted of rebound values of Ayere migmatitic rock to UCS after Deere and Miller (1966) Graph.

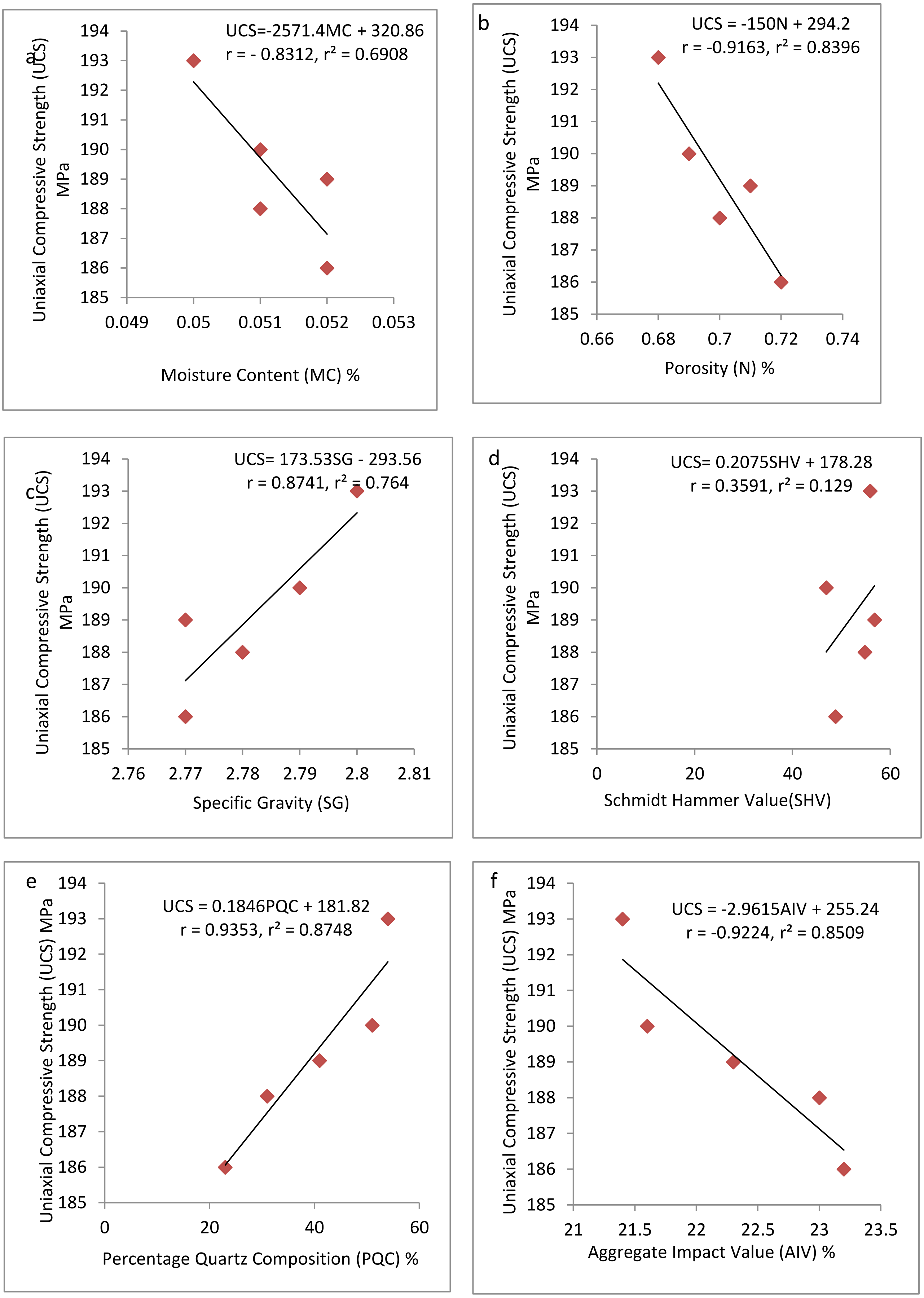
S/No	GPS	Rebound values	Mean	Adjusted Mean	Density (g/cc)	UCS (MPa)
1	N07°40'24.9"	42, 46, 58, 59, 33, 56, 59, 60, 62, 53	52.0	55.9	2.71	270
2	E5°59'13.3"	56, 40, 51, 46, 57, 45, 44, 52, 44, 46	48.1	46.9	2.71	135
3		48, 62, 43, 43, 59, 62, 60, 43, 60, 45	52.5	56.8	2.71	220
4		58, 53, 48, 54, 40, 60, 58, 55, 60, 47	53.3	54.8	2.71	200
5		47, 52, 55, 52, 54, 57, 44, 34, 27, 36	45.8	48.8	2.71	140
6		42, 46, 58, 59, 33, 56, 59, 60, 62, 53	52.0	55.9	2.71	270
7		56, 40, 51, 46, 57, 45, 44, 52, 44, 46	48.1	46.9	2.71	135
8		48, 62, 43, 43, 59, 62, 60, 43, 60, 45	52.5	56.8	2.71	220
9		58, 53, 48, 54, 40, 60, 58, 55, 60, 47	53.3	54.8	2.71	200
10		47, 52, 55, 52, 54, 57, 44, 34, 27, 36	45.8	48.8	2.71	140
The range of the UCS is 135 – 270MPa while the average is 193MPa						

Table 3: Physical and Strength Parameters of Ayere Migmatitic rock

Test	1	2	3	4	5
Moisture content (%)	0.050	0.051	0.052	0.051	0.052
Porosity (%)	0.68	0.69	0.71	0.70	0.72
Specific gravity	2.80	2.79	2.77	2.78	2.77
Slake durability value (%)	97.66	97.58	97.47	97.34	97.25
Point load test (MPa)	11.00	9.0	8.6	6.9	6.4
Uniaxial compressive test (MPa)	193	190	189	188	186
UCS from point load test (MPa)	264	216	206.4	165.6	153.6
Aggregate impact value (%)	21.4	21.6	22.3	23.0	23.2
Aggregate crush value (%)	23.2	23.7	23.8	23.9	24.4

Table 4. Statistical summary of the studied migmatite

Test	Range	Mean	Standard Deviation	Variance
Moisture content (%)	0.050-0.052	0.051	0.0008	0.0000007
Porosity (%)	0.68-0.72	0.7	0.0158	0.00025
Specific gravity	2.77-2.80	2.78	0.0130	0.00017
Slake durability value (%)	97.25-97.66	97.46	0.1681	0.02825
Point load test (MPa)	6.4-11	8.38	1.8308	3.352
Uniaxial compressive test (MPa)	186-193	189.2	2.5885	6.7
UCS from point load test (MPa)	153.6-264	201.1	43.9403	1930.75
Aggregate impact value (%)	21.4-23.2	22.3	0.8062	0.65
Aggregate crush value (%)	23.2-24.4	23.8	0.4301	0.185
Percentage Quartz composition (%)	23-54	40	13.11	172


a
UCS=-2571.4MC + 320.86
r = - 0.8312, r² = 0.6908
Uniaxial Compressive Strength (UCS) MPa
Moisture Content (MC) %
b
UCS = -150N + 294.2
r = -0.9163, r² = 0.8396
Uniaxial Compressive Strength (UCS) MPa
Porosity (N) %
c
UCS= 173.53SG - 293.56
r = 0.8741, r² = 0.764
Uniaxial Compressive Strength (UCS) MPa
Specific Gravity (SG)
d
UCS= 0.2075SHV + 178.28
r = 0.3591, r² = 0.129
Uniaxial Compressive Strength (UCS) MPa
Schmidt Hammer Value(SHV)
e
UCS = 0.1846PQC + 181.82
r = 0.9353, r² = 0.8748
Uniaxial Compressive Strength (UCS) MPa
Percentage Quartz Composition (PQC) %
f
UCS = -2.9615AIV + 255.24
r = -0.9224, r² = 0.8509
Uniaxial Compressive Strength (UCS) MPa
Aggregate Impact Value (AIV) %

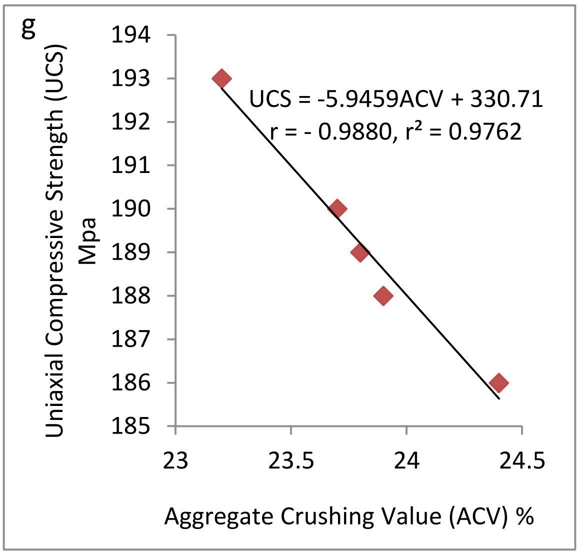


Figure 4: Plot of UCS with (a) Moisture Content, (b) Porosity, (c) Specific Gravity, (d) Schmidt Hammer value, (e) Percentage Quartz Composition, (F) Aggregate Impact Value, and (g) Aggregate Crushing Value.

There are inclusions of finer grains of zircon and quartz in the larger grains of plagioclase and quartz. This is an indication that the rock might have undergone more than one phase of stress/orogeny following the findings of Boesse and Ocan (1992) and Ademeso (2009).

Discussion

Table 4 shows that the rock possesses a very low value of natural moisture content and porosity with a high value of specific gravity, which agrees with high strength (Vasarhely and Van, 2006). The value of the slake durability of the analyzed rock fall in the range of high strength, according to ISRM, 1978, and 1981. These results show that the rock is suitable and useful as an aggregate for concrete and other construction purposes. The average Aggregate Impact Value (AIV) and Aggregate Crushing Value (ACV) values are 22.3% and 28.3%, respectively. The AIV result falls within the standard of "not greater than 25% set for rocks that qualify for use as aggregates in heavy-duty concrete finishes following British Standards (1990). The Aggregate crushing value (ACV) of the tested rock is less than 35%. It falls within the standard that signifies the rock as strong aggregates (Olarewaju, 1987) and therefore qualifies it to be useful as aggregates in heavy-duty concrete. The average values of the direct UCS and those estimated from Schmidt rebound hardness and point load are 189.2MPa, 193MPa, and 201.1MPa, respectively. It indicates that the rock possesses high uniaxial compressive strength and is useful as an engineering material following the strength classification of ISRM (1978). There is a significant relationship between the values of uniaxial compressive strength obtained when using the MTS servo machine and those obtained from the Deere and Miller (1966) chart and estimation from point load, as these values are very close which is consistent with the findings of Ademeso *et al.,* (2008). The results of regression plots show high negative correlations (r = – 0.8312,– 0.9163, – 0.9224, – 0.9880 respectively) between uniaxial compressive strength and moisture content, porosity, aggregate impact value and aggregate crushing value respectively; high positive correlations (r = 0.8741, 0.9353) and low positive correlation (r =

0.3591) between uniaxial compressive strength and specific gravity, percentage quartz composition and Schmidt hammer value respectively. Natural moisture content, porosity, AIV, and ACV of the studied rock decrease with increasing Uniaxial compressive and point load strength while specific gravity, Schmidt hammer value, and quartz content increase with increasing uniaxial compressive and point load strength.

Conclusion

Conclusively, the rock exhibited high of percentage quartz composition and possesses inclusions of strong minerals with the texture ranging from fine to coarse, which aligns with high strength as well as the existence of more than one stress/orogenic episodes. It further exhibited structural patterns that made them aesthetically relevant to use as dimension stone. Besides, it could also be useful as aggregates applicable to most civil engineering purposes.

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Assessing Fleet Performance in Haulage Operation at Bua Limestone Quarry, Okpella, South-Southern Nigeria

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Abstract

Fleet performance is a determining factor for effective fleet management in the quarrying industry. The study was carried out to assess fleet performance at BUA limestone quarry in Okpella, Edo State. To conduct this research, personal interviews, review of literature, field studies and personal observations were used. Findings from the research have indicated several values for the mechanical reliability, physical availability, utilisation factor and fleet efficiency for dozers, dumpers, wheel loaders and excavators respectively. The mechanical reliability for dozers, dumpers, wheel loaders and excavators was found to be 84.82%, 92.83%, 95.43%, and 94.67 respectively, while their physical availability was obtained to be 88.34%, 94.11%, 96.13% and 96.16% respectively. The utilisation factor of the fleet was found to be 70.66%, 81.34%, 87.05% and 85.70% while fleet efficiency for each of the machine was obtained to be 62.74%, 75.70%, 83.32% and 81.36% accordingly. The result of the study shows that the fleets are fairly utilized and the efficiency of the fleet is averagely put at 77.75%. The limestone production per annum was found to be 72% of the designed production target which is 5,791,886 tonnes instead of 8,000.000 tonnes per annum. It was recommended that the haulage way should be improved in other to increase speed of the haulage fleet for quick delivery of mine materials to the stockpile or crusher. The number of fleet should be increased especially the dumpers, to enable the company meet up with production target.

Keywords: BUA limestone quarry; haulage; fleet performance; haulage; cycle time; productivity

Introduction

Surface mining requires the utilisation of vehicles and other specialized equipment for successful operation and managing the fleets and their handlers is an essential mandate of the mine operators. Fleet in surface mining is expensive to maintain but is increasingly becoming more effective due to high productivity and reduced cycle time; yet, the entire performance could be threatened unless effective assessment of haulage equipment is matched with well-planned haulage system. In spite of this threat, productivity of haulage fleet is influenced by factors such as truck-shovel match and allocation, schedules of shift operation, design of haul roads and gradients of ramp, rolling resistance of haul road and size of mixed trucks in the fleet (Nel, *et al.,* 2011). Unplanned and improper fleet operation can also lead to break downs and accidents which cause injury and death of personnel, delays and disruptions in production, damage to equipment and eventual financial loss. However, improving productivity and production efficiency can reduce production and operating costs associated with fleet management (Pasch and Uludag, 2018). Large scale truck haulage systems in limestone quarrying are complex operations which require the synchronization of loading and hauling equipment. Analysis of this complex system necessitates breaking down and examining the detailed elements of cycle time of load-haul-dump equipment. Thus, increasing fleet distance between loading and dumping locations and varying the haul road gradient using technological advancement have a significant effect on production cycle time and dumper performance (Dey *et al,* 2017). Thus, according to Cheng (2019), excavator-truck performance can be enhanced when the former is positioned at a reasonable height above the latter by creating an elevated bench with the same height for it.

According to Adams and Bansah (2016), materials handing in surface mining involves loading and transporting of broken ore materials from pits where they are mined to the point of processing them using haulage equipment. Haulage is a major operation in limestone quarrying and is responsible for materials handling within the quarry. Cost of haulage constitutes about 50-75% of the total unit cost of mining (Nwosu, 1988; Burt and Chan, 2009). However, with respect to these higher costs, selection and application of type, size and number of equipment could significantly reduce the total production costs (Lashgari *et al*, 2010). Therefore, effective cycle time estimation and dump truck productivities are essential to designing the production rate. Truck cycle time is the time required to complete a single cycle by a mine truck including sorting and loading, haul loaded, turn and dump return empty, wait and delays as shown in Figure 1. Actual cycle time of a dump truck can be represented mathematically by the following equation:

$$T = t + t + t + t + t + t + t + td \qquad \ldots \text{Equ. } 1$$

Where T = Actual (total) cycle time taken of dump truck

t = time required to complete a cycle

td = time taken by the truck to dump materials

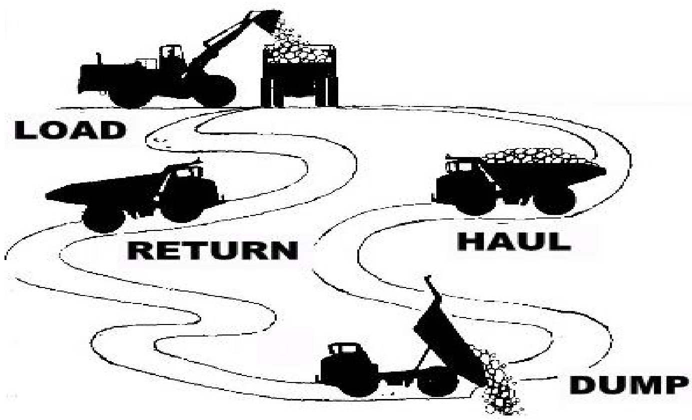

Figure 1 Cycle Time Illustration

According to Burt and Caccetta (2013), the general purpose of equipment selection involves a careful choice of compatible rather than homogenous items of equipment to carry out specified tasks. In many applications, the tasks are meant to move volumes of materials from one location to another. Thus, surface mining applications entails the selection of equipment to extract and haul mined materials, including both waste and ore, over the lifetime of the mining pit.

Accordingly, a major determinant for the efficiency or productivity of the selected equipment in mining and construction industries is the match factor, simply defined as the ratio of truck arrival times to loader service rates (Burt, 2008). The match factor, as defined by Fisonga and Mutambo (2017), can be calculated using the following equations:

FA = N x p x t/n x t = total number of trucks/number of loading units x loading cycle time/truck cycle time …Equ. 2.

where N = Total number of haulage units;
n = Total number of loading units;
T = Cycle time of each haulage unit;
t = Cycle time of each loading unit;
x = Number of haulage units per loading unit;
p = Number of buckets necessary to fill a truck.

According to Ercelebi and Bascetin (2009), optimizing shovel-truck system in surface mining requires careful determination of production over a given time period of interest (typically one shift) can be calculated by the number of loads that trucks take to the dump:

Production = time period of interest/average cycle time x N x truck capacity …Equ. 3.

Where N is the number of trucks in the system.

Production can also be estimated from:

Production = time period of interest x η_{shovel}x μ_{shovel} x truck capacity …Equ. 4.

x $\eta_{shovel}$is shovel utilization and $\mu_{shovel}$is shovel loading rate. Figure 2 shows the four phases of a shovel-truck system in surface mining.

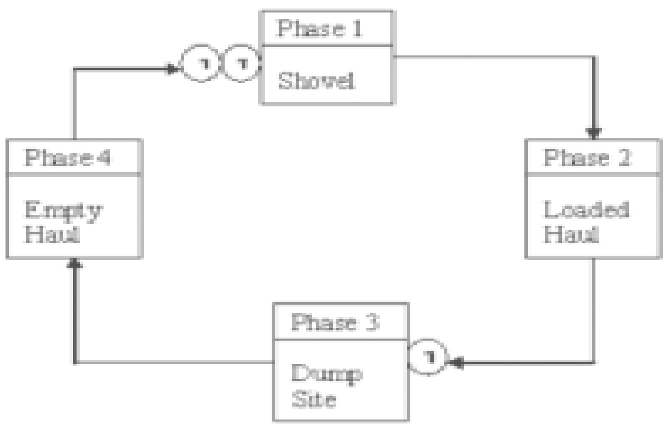

Figure 2: Phases of Shovel-Truck System (Ercelebi and Bascetin, 2009)

Fleet management in mining does not require one fits all concept due to the stochastic nature of fleet optimisation problems (Dembetembe and Mutambo, 2018). Although they can be modeled statistically, predicting them by means of statistic is often not feasible. Unscheduled down time for critical equipment; operator error or efficiency; adverse weather; on-site operational deviations from established procedure and equipment purchase budget limitations are some of the inhibiting factors which account for such realities from mine to mine. Hence, there is need to establish some particular site's needs and priorities to enable effective assessment of the system's capabilities which can match site requirements. To properly address the highlighted challenges associated with fleet performance in surface mining, this study, therefore, examines the level of efficiency of fleet used in limestone quarrying at BUA quarry in Okpella, Edo State, Nigeria, with a view to attaining optimum productivity and reducing cycle time losses to barest minimum.

Materials and Methods

An assessment of the current fleet performance was done through an analysis of data captured through observations of field operations as well as data from mine records on availability and utilization. Determination of fleet utilization and availability was done by careful estimation of the following parameters: spot and load time, travel haul time, delay period, turn and dump time, wait time and total cycle time. The various fleets used in the quarry were identified and their capacities noted, while their haulage routes and distances were studied to determine the root causes of delays and break downs during haulage.

A comparison was made between the principle of overall equipment effectiveness

(OEE) adopted by Elevli and Elevli (2010) and also adopted by Dembetembe and Mutambo (2018) and used in optimizing fleet performance at Nchanga open pit mine in Chingola vicinity of Zambian copper belt. Although the fleet optimization problem in these two cases is not exactly the same with that of BUA limestone quarry, the general terrain in terms of materials haulage in surface mining is very similar and assessment of their fleet performance can be done using the same parameters. Emphasis of the comparison focused on three primary considerations in equipment selection for materials handling namely: productivity, cost and compatibility to which Burt and Chan (2009) adduced.

Data Collection

Data collected included budgeted time, the incident and maintenance time, the stand by hour, the number of shift per month and the number of hour per shift, among other information. The following indicators were subjected to various forms of assessment through the collection of both raw data, field studies and personal observation: mechanical reliability of equipment, physical availability of equipment, machine utilization factor and fleet efficiency at BUA limestone quarry. Calculations based on raw data collected from the quarry for each of the parameters on weekly basis were used in arriving at the values for the respective indicators.

The indicators are mechanically expressed as follows:

Physical availability (%)

$$= \frac{\text{Running hour} + \text{standby hour}}{\text{budget hours}} \times 100 \ldots \text{Equ. 5.}$$

Use of availability (%)

$$= \frac{\text{Running hour}}{\text{Running hour } + \text{Standby hour}} \times 100 \ldots \text{Equ. 6.}$$

Utilisation factor %

$$= \frac{\text{Actual running hour}}{\text{Total hours}} \times 100 \qquad \ldots \text{Equ. 7.}$$

Efficiency factor %

$$= \text{Mechanical availability x utilization factor} \times 100 \qquad \ldots \text{Equ. 8}$$

Results and Discussion

Results from Tables 1 to 4 on the performance assessment of the fleet used in the BUA limestone quarry for six (6) weeks show that the fleet performance is fairly good. This implies that the working hours of the fleet are well utilized and they are performing well because the utilization and efficiency factor calculated none is less than 70%. Figure 3 shows the chart of the performances assessment of the fleet, which implies that the mechanical reliability and the physical availability of the fleet is substantially and having high percentage throughout the six (6) weeks of assessment, and the utilization and efficiency are moderate percentage which implies that about 30% of the hours is not utilized.

From Figure 3, the mechanical reliability of excavators and their physical ability at BUA limestone quarry are of higher frequencies all through the six weeks of the study than equipment utilization and efficiency. Furthermore, efficiency, compared with machine utilization, has the least percentage value.

From the fleet assessment, it was discovered that the mechanical reliability of the fleet was high except for most of the problem encountered during repair/incident hours. The

Physical availability shows that fleets are physically available. This in turn will lead to higher production to meet the company's proposed target of being one of the largest cement producing company. This high productivity is not only contributed by the mechanical and physical availability only but also on the record of hours the operators make use of the available machine (use of availability) even though most of the fleet are not (100%) efficient, but the 80% productive and 20% lost due to repair make it possible for the fleet to be able to work and produce more which makes the use of availability also high and make the efficiency of those fleet good which differentiates it from other availability factors.

The utilization factors of the fleet on the other hand is fairly good, which contributes and enhances good production rate, since the matching of the correct equipment size help fleet to work faster and boost the morale of the operators. The cycle time of operation was calculated as 8.68 minutes. It can vary due to the influence of the factors making it up like hauling time loading time, spotting time and dumping time etc. The cycle time was used to calculate the number of trips per shift which was 42.86 trips (about 43 trips) in every shift of six (6) hours. The tonnage per shift was also calculated to give 19,501.03 tonnes. The total predictable production per annum was calculated to be 5,791,886.1 tonnes. The comparison between the predictable production per annum (5,791,886.1 tonnes) and the planned production per annum (8,000,000 tonnes) was 72.39%. This comparison is averagely due to the fact that there is need to increase the numbers of fleet in the quarry. There is a good speculation that when additional fleets are employed, production will increase and probably the planned production per annum may be achieved.

Table 1 Total hours of the fleet in six (6) weeks

No of fleet	Budgeted hour	Running hour	Standby hour	Incident hour
Bulldozers	420hr	354hr	34hr	32hr
Dumpers	420hr	368hr	28hr	24hr
Wheel loaders	420hr	379hr	24hr	16hr
Excavators	420 hr.	382hr	22hr	18hr

Table 2 Bulldozer performances assessment on weekly basis

No. of Weeks	Mechanical Reliability (%)	Physical Availability (%)	Utilization (%)	Efficiency (%)
1	81.82	85.72	70.57	64.40
2	84.61	85.71	64.28	54.38
3	90.47	91.41	72.85	65.90
4	89.55	95.23	75.71	67.79
5	80.05	84.82	72.21	65.34
6	82.45	87.16	68,34	58.64
Average Percentage	84.82	88.34	70.66	62.74

Table 3 Dumper performance assessment on weekly basis

No of weeks	Mechanical Reliability (%)	Physical Availability (%)	Utilization (%)	Efficiency (%)
1	93.84	94.28	81.43	76.42
2	94.02	94.28	84.28	79.24
3	92.31	92.86	78.87	72.52
4	92.53	95.38	81.42	75.33
5	90.64	93.42	79.72	73.62
6	93.69	94.49	82.37	77.12
Average percentage	92.82	94.11	81.34	75.70

Table 4 Wheel loader performance assessment on weekly basis

No of weeks	Mechanical Reliability (%)	Physical Availability (%)	Utilization (%)	Efficiency (%)
1	96.92	97.14	87.14	84.46
2	94.11	94.28	85.71	80.66
3	92.42	92.85	80.00	73.93
4	97.05	97.06	91.41	88.73
5	95.36	98.21	90.62	87.52
6	96.74	97.26	87.45	84.62
Average Percentage	95.43	96.13	87.05	83.32

Table 5 Excavator performance assessment on weekly basis

No of weeks	Mechanical reliability (%)	Physical Availability (%)	Utilization (%)	Efficiency (%)
1	96.97	97.14	88.57	85.88
2	95.52	95.71	87.14	83.23
3	92.53	95.38	81.42	75.33
4	95.58	97.01	88.57	84.65
5	94.62	96.25	87.34	82.62
6	92.82	95.47	81.16	76.45
Average percentage	94.67	96.16	85.7	81.36

Table 6 Summary of major equipment, their capacities and their assessment at BUA cement quarry

Fleet	Model	Number available	Equipment capacities				Mechanical availability (%)	Physical availability (%)	Utilization factor (%)	Efficiency factor (%)
			Engine capacity (hp)(kw)	Fuel capacity (litres)	Bucket capacity (m^3)	Load capacity (tonnes)				
Dozers	D9R 3408C	2	330 (kw)	889	13.5	-	84.82	88.34	70.66	62.74
Wheel loaders	Cat 98.8H C18	3	414kw 555hp	712	6.4	8.3	95.43	96.13	87.05	83.32
Dumpers	Cat 775G C27	8	768(kw) 812 (hp)	210	50	65	92.83	94.11	81.34	75.70
Back hoe excavator	Cat 390D C18	3	523 (hp) 390 (kw)	1240	4.5	5.9	94.67	96.16	85.70	81.36

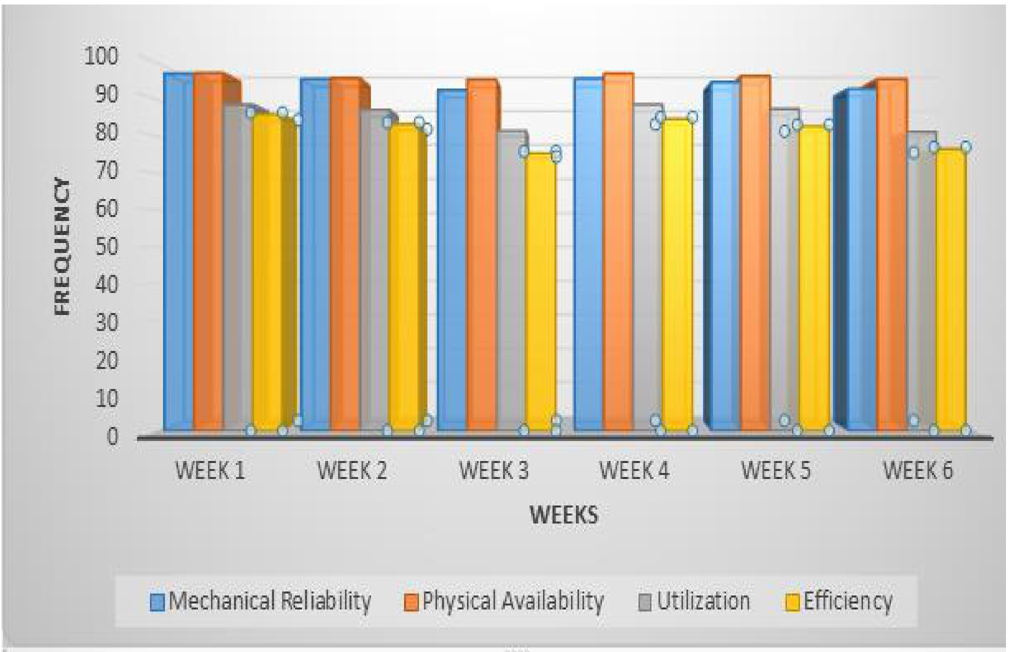

Figure 3: Excavator performance assessment

Conclusions and Recommendations

From the results of the study, it can be concluded that the utilisation of the fleet attained are: 81.34%, 70.66%, 87.05% and 83.32% for dumpers, bulldozers, wheel loaders, and excavators respectively and the efficiency attained are: 75.70%, 62.74%, 83.32%, and 81.36% for bumpers, bulldozers, wheel loaders and excavators respectively. Therefore, the fleets employed can be said to be performing optimally, while production has been stepped up to 5,791,886 tonnes. Sustenance of this feat would help the company to achieve their target in no distant time.

In view of the observations made in this research, the following recommendations are hereby suggested:

i. Prompt fleet maintenance should be given top priority to minimize equipment breakdowns, production interruptions and other avoidable problems.
ii. The number of fleet should be increased from ten (10) to fifteen (15), especially the dumpers, to enable the company meet up with production target.
iii. Constant monitoring of fleet is important apart from maintenance to check sudden anomalies and interruptions that could affect equipment performance.
iv. The haulage way should be improved in other to increase speed of the haulage fleet for quick delivery of materials to the stockpile or crusher.

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Characterization of Ngaski gold ore resource near Yauri, Kebbi State, Nigeria.

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Abstract
Mineralogical characterization of Ngaski gold ore resource was carried out using Heavy Liquid Separation (HLS), X-ray Diffraction and Electron Scanning Microscope Energy Dispersive Spectroscopy. This was done to establish the mineralogical composition of the ore needed to facilitate the development of a suitable flowsheet for beneficiating Ngaski gold ore resource. The characterization studies was carried out on blended Ngaski gold ore crushed and milled to 100% passing 75 μm and wet-screened at 25 μm to remove slimes. Both deslimed material and slime fraction were subjected to heavy liquid separation (HLS) at a liquid density of 2.96. More than 97% of the material reported to floats in both fractions. 0.71% of +25 μm fraction reported to the sink fraction while 2.87% of the -25 μm fraction reported to its corresponding sink fraction. Split aliquots of the sink fractions were analyzed by quantitative XRD. The results showed quartz (SiO_2) to be the predominant mineral followed by pyrite (FeS_2), pyrrhotite ($Fe_{1-X}S$), hematite (Fe_2O_3) with cerussite ($PbCO_3$), chromite ($(Fe,Mg)(Cr,Al)_2O_4$), k-felspar ($KAlSi_3O_8$) and epidote ($Ca_2(Al,Fe)Al_2O(SiO_4)(Si_2O_7)(OH)$) as the minor minerals. The +25μm fraction contained quartz and pyrite as the abundant minerals while the -25 μm contained quartz as the predominant mineral with pyrite being fairly abundant. Scanning electron microscope (SEM) Energy Dispersive Spectroscopy conducted on aliquot of the sink fractions from the +25 μm and -25 μm confirmed the presence of gold in Ngaski gold ore. The micrographs showed that most of the gold grains were liberated but contained quartz inclusion while some were associated with Fe-oxide, pyrite, epidote, chromite and ilmenite. The SEM-EDS also revealed the particle sizes of Ngaski gold to range from <10 to ~ 200 μm in +25 fraction, and <5 to ~25 μm in -25 μm fraction. The presence of liberated and exposed gold in the sink fractions as shown in the micrographs revealed the amenable of Ngaski gold ore to processing by gravity separation and leaching methods.

Key words: Characterization, Ngaski Gold ore, XRD, SEM, Heavy Liquid Separation.

INTRODUCTION

Mineralogical characterization of an ore deposit has dual importance of providing generic information about the source and physicochemical environment of formation of the deposit as well as play a critical role in optimum utilization of scarce natural resources. It helps in the selection of suitable recovery method and dictates the process flow sheet and plant flow sheet optimization (SGS Rocks to Results, 2014; Marsden and House, 2009; Lunt and Weeks, 2005; Xiao and Laplante, 2004;, Yaro, 1997; Cabri, 1981; Henley, 1983; and Petruk and Hughson, 1977).

The key to the exploitation of these gold resource lies on the nation's ability to fully identify and characterize the various gold resources in such a manner that would expose their potentials for economic exploitation. This is in line with the postulation that identification and characterization of minerals ~~is~~ of fundamental importance in the development and operation of mining and mineral processing systems (Hope et al., 2001).

Gold is an essential metal commodity traded internationally. Nigerian Gold is naturally found occurring in primary veins, alluvial and eluvial placers in the schist belts of the northwest and southwest of Nigeria (Woakes, Rahaman and Ajibade, 1987) with most of these resource sites still at various stages of exploration and skeletal exploitation by artisanal and small scale miners (Tychsen et al, 2011). Ngaski gold resource is located South West of Bin Yauri and falls partly within Zuru Schist Belt as shown in Figure 1.. The major rocks in the study area include amphibolite schist, amphibolite, granite gneiss, undifferentiated schist including phyllites, fine grained flaggy quartzite and quartzite schist (Ramadan and Abdal Fattah, 2010; Obaje, 2009).

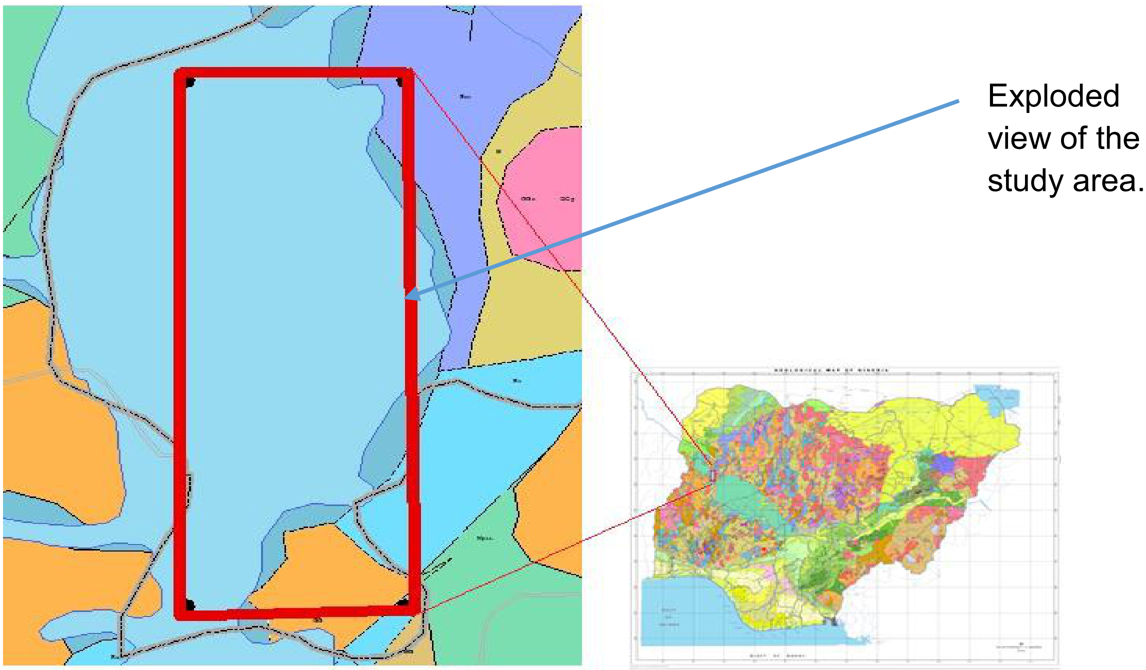

Figure 1: Geological map of the study area with Exploded view (Source: Nigerian Geological Survey Agency, 2004)

The prevalent illegal mining syndrome especially in Zamfara which led to massive lead poisoning that killed scores of people made up of mainly women and children involved in "scavenging or pig cropping" for gold are generally associated with poor characterization and lack of sufficient knowledge of ore composition. Knowledge of Nigerian gold ore composition would facilitate the development of appropriate extension programme for ASM operators leading to reduction in environmental and human pollution associated with ASM gold exploitation.

MATERIALS AND METHODS

Materials

150kg each of Run of Mine were obtained from artisanal miners that mine Libale and Masamale gold vein. The samples were crushed to -2mm and blended to produce test material used in the study.

Equipment

The equipment used comprised laboratory jaw crusher, cone crusher, ball mill, sieves, sieve shaker, electronic weighing scale (0.01g), Panalytical Xpert Pro Diffractormeter, Scanning Electron Microscope (SEM) - Evo-E430, Sample Splitter.

Experiments

Representative sample of Ngaski gold resource was subjected to the following laboratory test works after reduction to appropriate test weights using cone and quartering sampling techniques and riffling. The various test carried out are as follows.

Heavy liquid Separation

The sample was wet-screened at 25 μm, to remove the slimes, which were retained and dried. The deslimed fraction (+25μm) and slimes (-25μm) produced were thereafter subjected to Heavy Liquid Separation (HLS) using tetrabromoethane (TBE) at a liquid density of 2.96 with the two sets of floats, sinks and slimes weighed.

Mineralogical composition of Ngaski gold ore

This was determined using XRD and SEM-EDS. The XRD analysis was carried out on split aliquot of the Heavy Liquid Separation sink fractions using a Pananalytical X'Pert Pro Diffractometer, employing Co-Kα radiation. The resulting patterns were processed using HighScore Plus analytical software and the PanICSD database.

The SEM-EDS investigation was carried out on four resin mounted polished sections of +25μm and -25μm sinks produced from Heavy Liquid Separation tests carried out

RESULTS AND DISCUSSION

Heavy liquid Separation of the Ngaski gold ore

The results of the Heavy Liquid Separation are given in Table 1. More than 97% of the material reported to floats in both fractions. The +25 μm fraction produced less than 1 % sinks while the -25 μm fraction produced ~3% sinks.

Table 1. Heavy Liquid Separation

Fraction	Mass (g)		Sink		Floats	
	G	%	G	%	G	%
+25 μm sink	772.90	77.62	5.50	0.71	767.38	99.29
-25 μm sink	222.80	22.38	6.32	2.84	216.48	97.16
Total	**995.70**	**100.00**	**-**	**-**	**-**	**-**

X-ray Diffraction of the Ngaski gold ore

The XRD analyses are given in Table 2. The +25 μm fraction was indicated to contain abundant amounts of quartz and pyrite and lesser hematite. Minor amounts of goethite, chromite, mica and epidote were present along with trace amounts of pyrrhotite, galena and cerrusite. The -25 μm fraction was dominated by quartz with fairly abundant amounts of pyrite, and minor amounts of hematite, pyrrhotite, mica and epidote. Trace amounts of plagioclase, K-feldspar, galena, cerrusite, iron, chromite and goethite were also present.

Table 2: Mineralogical Composition of Ngaski Gold Ore using XRD

Mineral	Approximate Formula	+25 μm	-25 μm
Hematite	Fe_2O_3	10 - 20%	3 - 10%
Goethite	α- FeO.OH	3 - 10%	<3%
Chromite	$(Fe,Mg)(Cr,Al)_2O_4$	3 - 10%	<3%
Iron	Fe	-	<3%
Pyrite	FeS_2	20 - 50%	10 - 20%
Pyrrhotite	$Fe_{1-x}S$	<3%	3 - 10%
Galena	PbS	<3%	<3%
Cerussite	$PbCO_3$	<3%	<3%
Quartz	SiO_2	20 - 50%	>50%
Mica	$KAl_2(Si_3Al)O_{10}(OH,F)_2$	3 - 10%	3 - 10%
Epidote	$Ca_2(Al,Fe)Al_2O(SiO_4)(Si_2O_7)(OH)$	3 - 10%	3 - 10%
Plagioclase	$(Na,Ca)(Al,Si)_4O_8$	-	<3%
K-feldspar	$KAlSi_3O_8$	-	<3%

>50% Predominant, 20-50% Abundant, 10-20% Fairly abundant, 3-10% Minor and <3% Trace

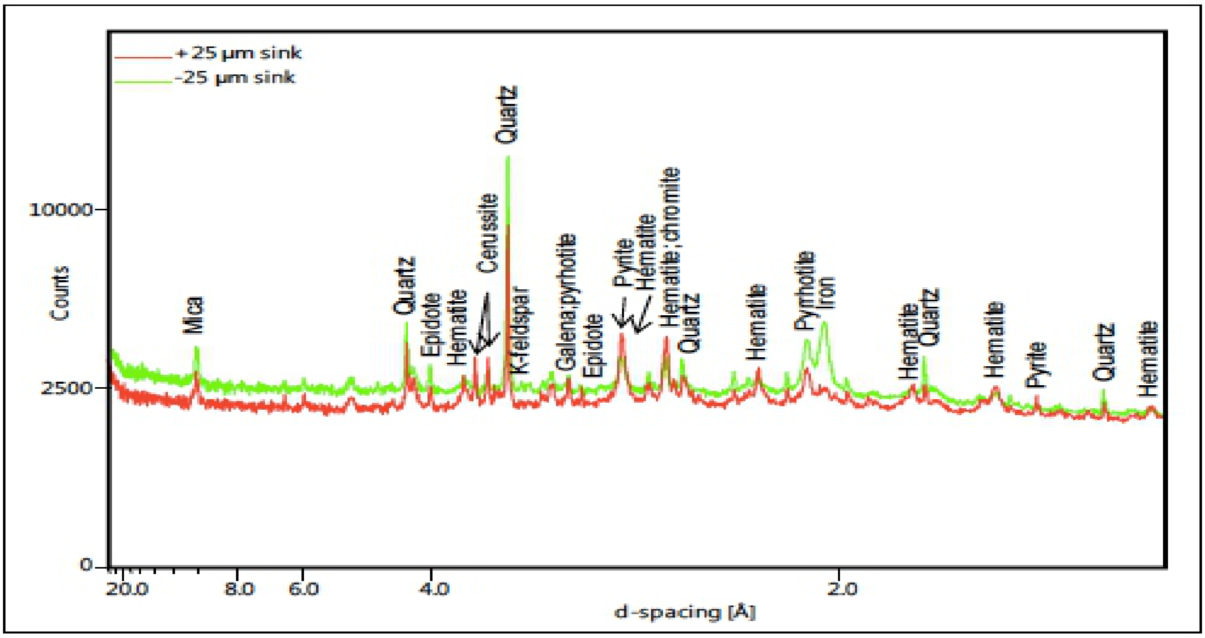

Figure 2: X-ray diffractogram of the Ngaski gold ore

Scanning Electron Microscope of the Ngaski gold ore

Scanning electron microscope backscattered electron (SEM-BSE) images and energy dispersive spectras (EDS) were as given in Plate I to VI and Figure 3 – 5 respectively. Gold grains were observed during SEM investigation. These grains varied in size from <10 to ~200 µm in size in the +25µm fraction, and <5 to ~ 25 µm in size in the -25 µm fraction. Most of the gold grains observed were liberated, but contained quartz inclusions. Some of the gold observed, was associated with Fe-oxide, pyrite, epidote, galena, chromite and ilmenite. Acanthite grains were also observed, and they were mostly associated with galena, but a few were observed to be associated with chalcopyrite and Fe-oxide. Other sulphide minerals observed in the samples include arsenopyrite, chalcopyrite, sphalerite and silver-bearing tetrahedrite. Zircon and monazite grains were observed in the +25 µm fraction (Figure 3 – 5).

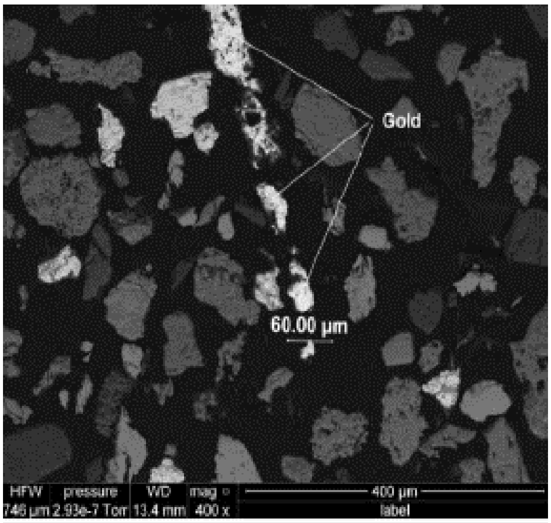

Plate I: SEM-BSE photomicrograph of liberated gold grains (A) from +25µm fraction

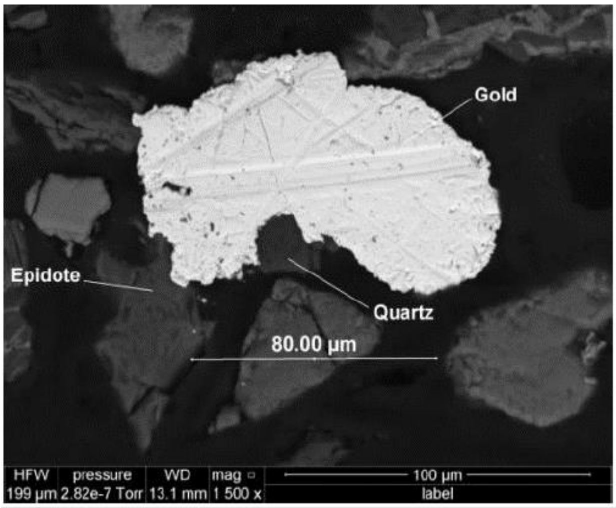

Plate II: SEM-BSE photomicrograph of large gold grain attached to epidote and quartz from +25µm fraction (D)

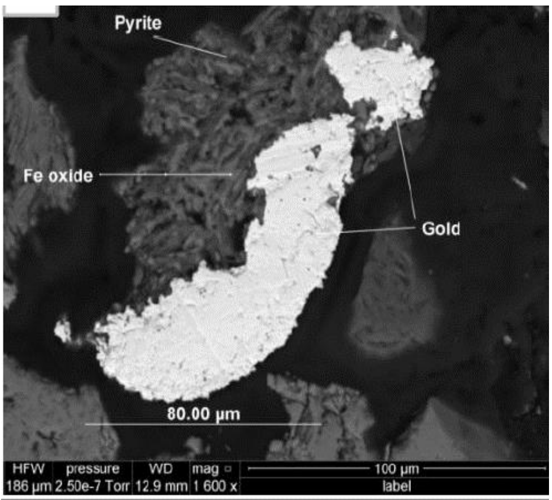

Plate III: SEM-BSE photomicrograph of gold grain associated with pyrite and Fe-oxide (E) from +25µm fraction

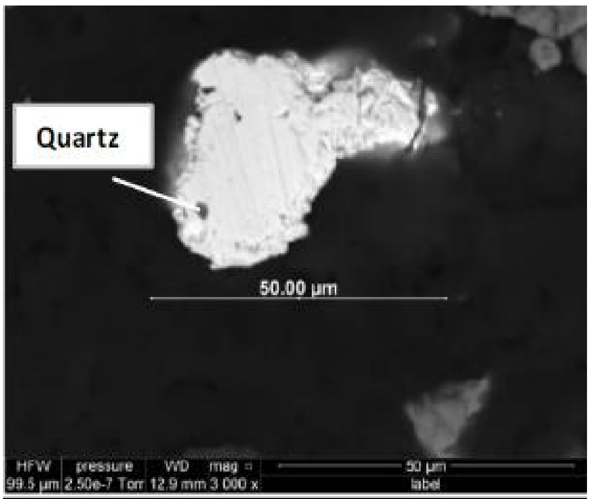

Plate IV: SEM-BSE photomicrograph of liberated gold grain with quartz inclusions from +25µm fraction (F)

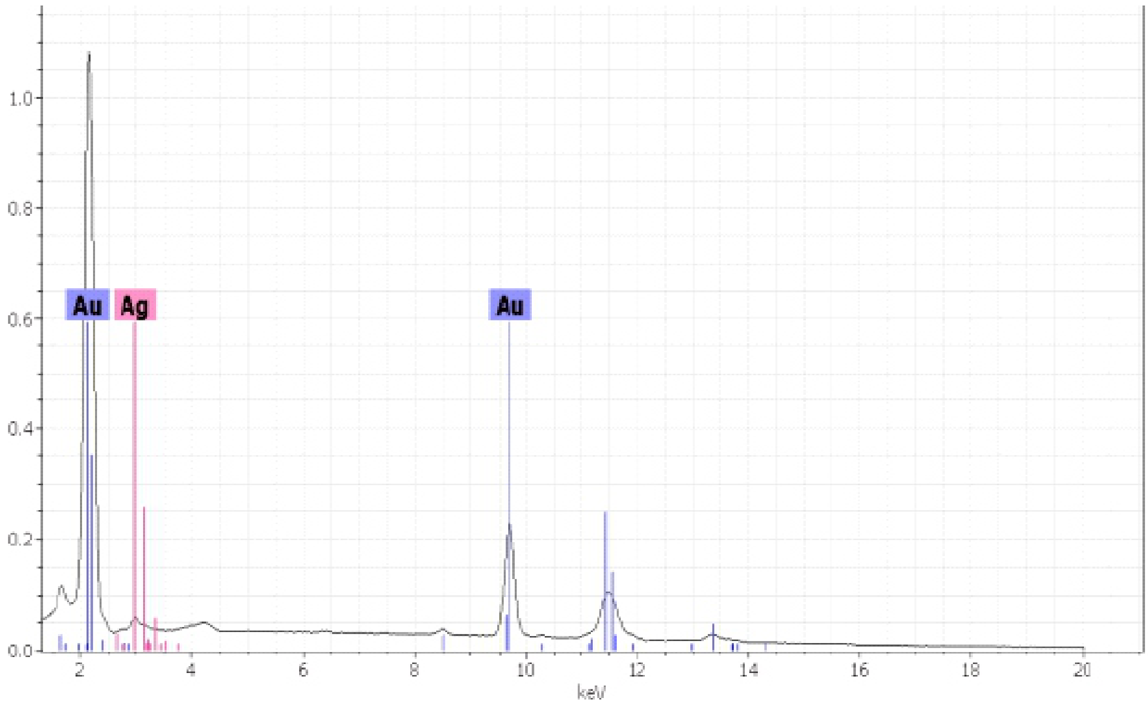

Figure 4: Results of EDS spot analysis of gold grain A in Plate V

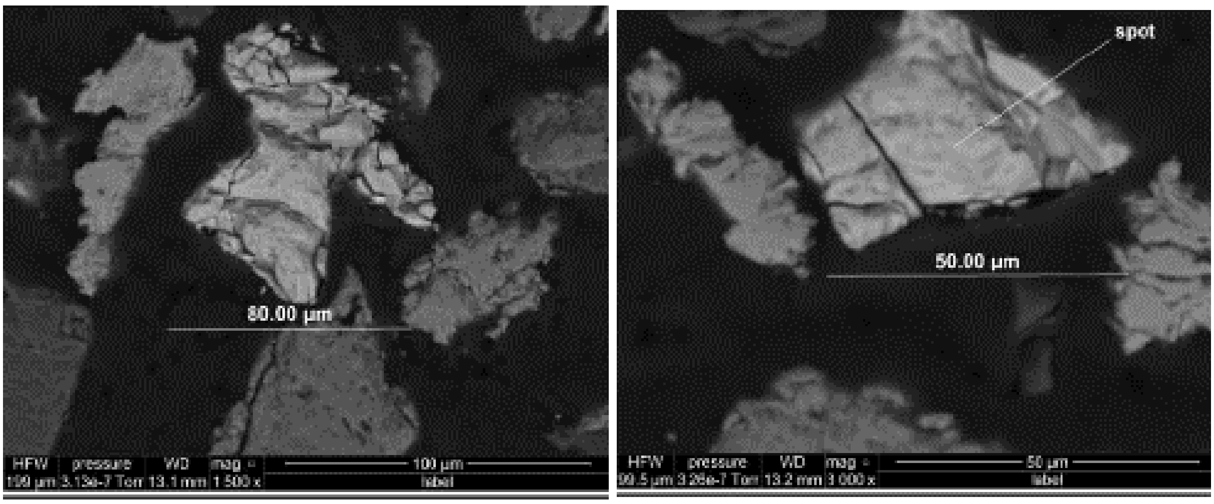

Plate VI: SEM-BSE photomicrographs of silver bearing tetrahedrite grains

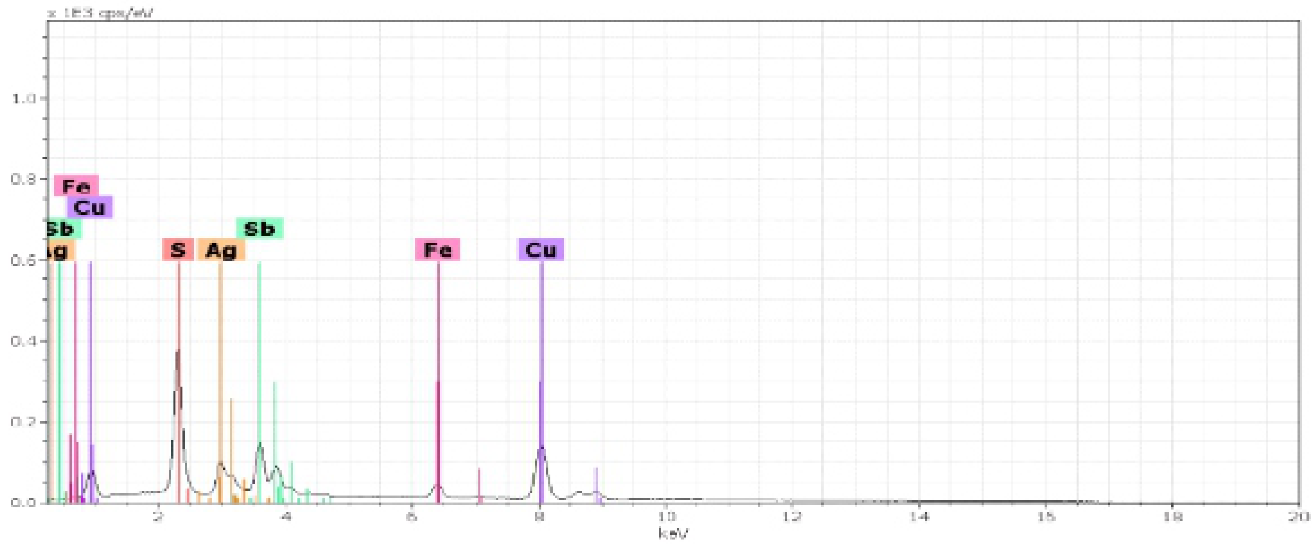

Figure 5: Result of EDS spot analysis of tetrahedrite grains in Plate VI

CONCLUSION

The characterization studies carried out on blended Ngaski gold ore crushed and milled to 100% passing 75 μm and wet-screened at 25 μm to remove slimes using heavy liquid separation (HLS) at liquid density of 2.96 resulted in +97% of the material reporting to the floats in both fractions. 0.71% of +25 μm fraction reported to the sink fraction while 2.87% of the -25 μm fraction reported to its corresponding sink fraction containing liberated gold.

The XRD results of the sink fractions showed quartz (SiO_2) to be the predominant mineral followed by pyrite (FeS_2), pyrrhotite ($Fe_{1-x}S$), hematite (Fe_2O_3) with cerussite ($PbCO_3$), chromite ($(Fe,Mg)(Cr,Al)_2O_4$), k-felspar ($KAlSi_3O_8$) and epidote ($Ca_2(Al,Fe)Al_2O(SiO_4)(Si_2O_7)(OH)$) as the minor minerals. The +25μm fraction contained quartz and pyrite as the abundant minerals while the -25 μm contained quartz as the predominant mineral with pyrite being fairly abundant.

Scanning electron microscope - Energy Dispersive Spectroscopy (SEM-EDS) conducted on aliquot of the sinks from the +25 and -25 μm showed the presence of gold in Ngaski gold ore. The micrographs showed that most of the gold grains were liberated but contained quartz inclusion while some were associated with Fe-oxide, pyrite, epidote, chromite and ilmenite. The SEM-EDS also revealed the particle sizes of Ngaski gold to range from <10 to ~ 200 μm in +25 μm fraction and <5 to ~25 μm in -25 μm fraction. The high levels of liberation and exposure of gold in the sink fractions as shown in the micrographs revealed the amenable of Ngaski gold ore to processing by gravity and leaching methods.

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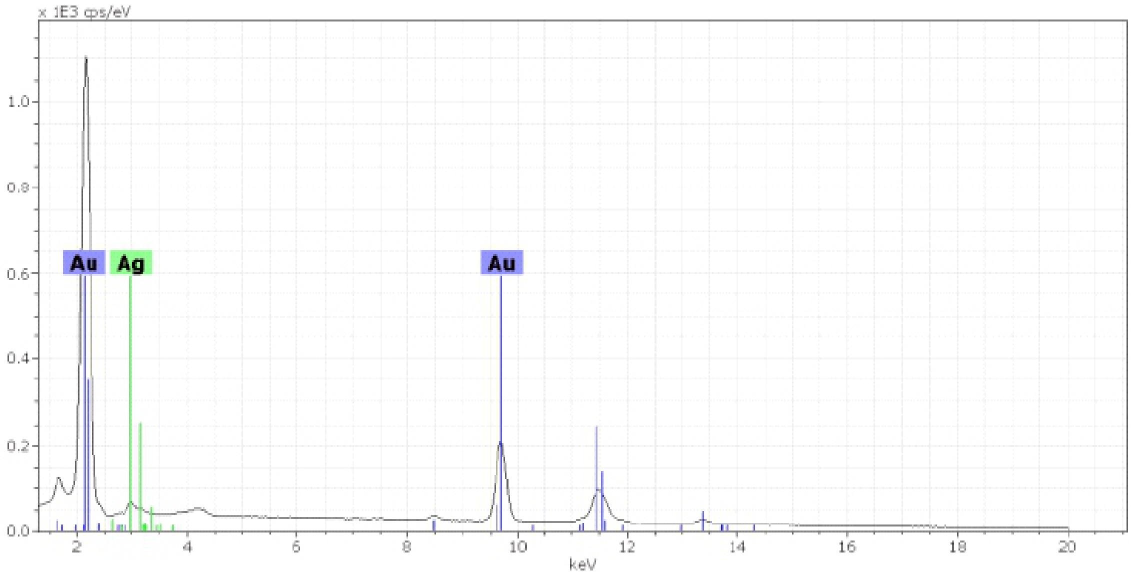

Figure 3: Result of EDS spot analysis of gold containing grains from +25µm fraction in Plate IV

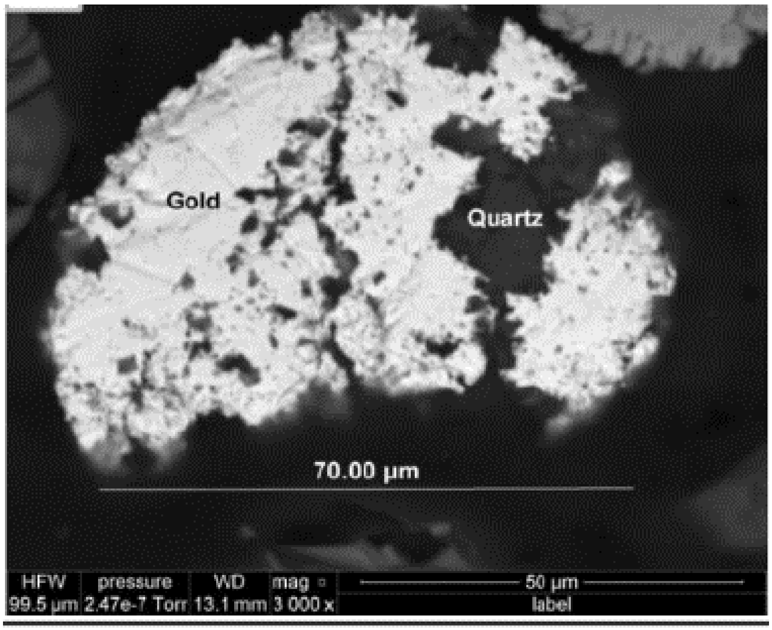

Plate V: SEM-BSE photomicrograph of gold grain with quartz inclusion

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Sirajo, Yaro, Thomas, Maina and Asuke (2020) Nigerian Mining Journal, (18) 90 - 96

Determination of Work Index of Bassa-Nge Iron Ore Deposit

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Abstract

The work index of Bassa-Nge iron ore deposit in Kogi State of Nigeria was determined using the modified Bond's method. The sample used for the test was sourced from Egeneja Village of Bassa-Nge Local Government Area, using the grab and trench sampling technique. Granite of known work index used as reference ore was obtained from Sabo village in Kaduna State. The reference and test ores of equal and known weights were ground using a ball mill under the same grinding conditions (dry) respectively. The size analyses of the feed for both reference and test ore was found to be 289.58 µm and 332.32 µm, and the ball mill discharge was 206.84 µm and 200.81 µm respectively. The granite used as reference ore with work index of 15.13 kWh/t was used to calculate the work index of the test ore and was found to be 10.39 kWh/t. The grindability of the ore was determined using Mathur's grindability scale and was found to be a type B medium soft texture iron ore that can be fragmented.

INTRODUCTION

The ability of a nation to exploit its minerals lies fully in characterization of its minerals deposits in a manner that will expose the minerals potentials and hence attracting the much-needed investors. The identification and characterization of a mineral is one of the fundamental parameters necessary in the development of the process route of that mineral ore (Hope *et al.,* 2001).

Comminution is a process in which solid materials are reduced in size and this takes place in mineral processing plant as a sequence of crushing and grinding process. Crushing reduces the particle size of run-of-mine ore to such a level that grinding can be carried out until the mineral and gangues are substantially produced as separate particles according to Wills and Napier Munn, (2006). Work index is a comminution parameter which expresses the resistance of an ore to crushing and grinding. Numerically, it is the kilowatt hour per short-ton required to reduce the ore from theoretically infinite size to 80% passing 100 microns (Oladunni *et al*; 2015). In another development, determination of work index can be achieved using the comparative method called the Berry and Bruce method. This method requires the use of a reference ore of known grindability. Determination of the required amount of energy to effect rock-breakage is of fundamental importance in process design because it helps in the selection of the appropriate comminution equipment and avoidance of energy wastage. Table 1 presents the work index of some Nigerian iron ores.

Table 1: Some Nigerian Iron Ores Work Indices

Iron ore	Work index (kWh/t)
Hematite	2.0 – 29.4
Illimenite	10.70- 16.40
Magnetite	2.4 - 19.20
Iron ore unidentified	2.3 -33.6
Toto Muro	10.00
Koton Karfe (Calcined) and Uncalcined	11.33 (calcined) and 17.00 (uncalcined)
Birni Gwari	18.55
Gujeni iron	13.96
Itakpe	9.34
Ochokochoko	9.47

(Sources: Agava *et al*, 2016)

Related Literature Review on Work Index Determination

Oladunni *et al*; (2015), determined the work index of Gyel-Buruku columbite ore in Plateau State and reported that the value obtained lies favourably within the work indexes of 3.94 - 10.81 kWh/t for columbite minerals cited in the literatures. Obassi *et al*., (2015), investigated the work index of lead ore using granite as a reference material with work index of 15.13kWh/t. Reported that the work index of the test ore was found to be 14.37kWh/t and asserted that, the value obtained is within the range indicated by previous research work on lead ores.
Allen, (2017), investigated work index of Ishiagu galena ore using Bond's work index and found out that 5.14kWh/t was required to grind 1000kg of test ore to 80% passing sieve size of 100μm.
Therefore, the need to determine the work index of Bassa-Nge iron ore deposit in Kogi State using the modified Bond's method as remedy for the development of a process route for the beneficiation of the ore deposit to metallurgical grade became necessary. The Bassa-Nge iron ore deposit is located in Egeneje hill in Bassa-Nge Local Government Area of Kogi State. It is about 40km from Ajaokuta Steel Company.

MATERIALS AND METHODS

Materials

The materials used were the homogenized raw sample of the iron ore and granite.

Sample Collection

The iron ore raw samples were collected from three different locations A, B and C of the ore deposit in Egeneja village in Bassa-Nge Local Government Area of Kogi State using trend and grab method of sampling. The A, B and C locations where the iron ore samples were sourced measured 150cm^2 with each excavation been 50cm length, 50 cm width and 500cm depth and 3 meters apart.

Method

The Modified Bond's Method called Berry and Bruce method of determining the work index of an ore was used. This is because of its simplicity and it requires the use of a reference ore known work index. The procedure used in the determination of Bassa-Nge iron ore include the following: crushing the ore sample and pulverizing it using Denver laboratory jaw crusher and pulverizing machines respectively. The pulverized samples of 150 grams each of the test and reference ores were ground in the Denver laboratory Ball mill machine for an hour and sieved for fifteen minutes using the automatic Denver laboratory sieve shaking machine. The sieves were arranged on the basis of √2 from the coarsest to the finest sieve in stack on the machine. The appropriate sieve aperture sizes used ranges from 355μm to 63μm (+355 μm, −355 + 250 μm, −250 + 180 μm, −180 + 125 μm, −125 + 90 μm, −90 + 63 μm, −63 μm).

The work index of the test ore was determined when equal work input (W) was expended for equal weights of the reference and test ore, ground under the same conditions in the same laboratory ball mill using 1 and 2.

$$E_r = E_t = W_{ir}\left[\frac{10}{\sqrt{P_r}} - \frac{10}{\sqrt{F_r}}\right]$$

$$= W_{it}\left[\frac{10}{\sqrt{P_t}} - \frac{10}{\sqrt{F_t}}\right] \text{ kWh/t} \qquad \text{Equ. 1}$$

Where, E_r is the energy input in grinding the reference ore,
E_t is the energy input in grinding the test ore,
W_{ir} is the work index of the reference ore,
W_{it} is the work index of the test ore,
F_r is the 80% passing feed of reference ore,
F_t is the 80% passing feed of test ore,
P_r is the 80% passing product of reference ore,

P_t is the 80% passing product of test ore.

Therefore, the work index of the test ore can be determined using the comparative equation 2 of Berry and Bruce's method after determining the cumulative 80% weight passing of the feed and discharge of the test and reference ore.

$$W_{it} = W_{ir} \frac{\left[\frac{10}{\sqrt{P_r}} - \frac{10}{\sqrt{F_r}}\right]}{\left[\frac{10}{\sqrt{P_t}} - \frac{10}{\sqrt{F_t}}\right]} \mathrm{kW/t} \qquad \text{Equ. 2}$$

Where Wit = Work index for test ore, (kWh/t,
Wir = Work index for reference ore, (kWh/t,
P_r = 80% passing size of circuit product for reference ore,
P_t = 80% passing size of circuit product for test ore,
F_r = 80% passing size of new feed for reference ore,
F_t = 80% passing size of new feed for test ore.

RESULTS AND DISCUSSION

Results of Bassa- iron ore and Granites as Feeds and Discharges (Products) of the Ball Mill.

The particle size distribution the feeds and products of the ball mill are presented in Tables 2 3, 4 and 5 while Figures 1, 2, 3 and 4 show the plots of cumulative percentage retained and passing against sieve size fractions of the test and reference ores respectively.

Table 2: Particle Size of the Feed of Test ore (Bassa-Nge iron ore)

Sieve size Range (µm)	Weight retained (g)	% weight retained	Nominal aperture	Cumulative %weight retained	Cumulative % weight passing
+355	17.90	17.90	355	17.90	82.10
-355+250	15.80	15.80	250	33.70	66.30
-250+180	18.80	18.80	180	52.50	47.50
-180+125	19.50	19.50	125	72.00	28.00
-125+90	11.80	11.80	90	83.80	16.20
-90+63	01.60	01.60	63	85.40	14.60
-63	14.60	14.60	-	100	0.00

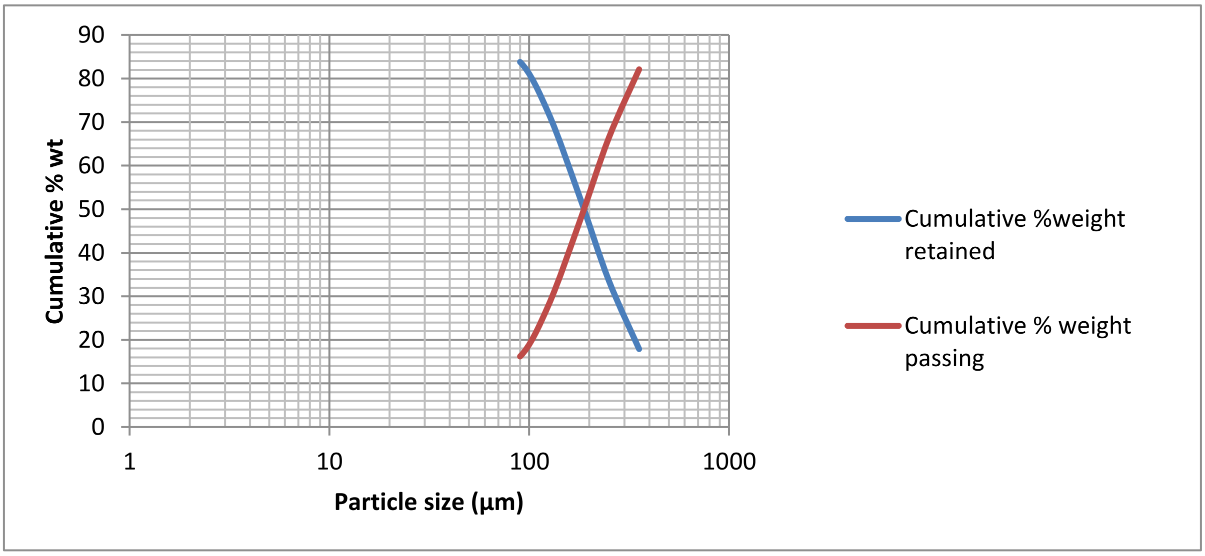

Figure. 1: Cumulative % weight retained and passing against sieve size fractions (µm) of Bassa- Nge iron ore as feed to ball mill.

Determination of the 80% passing sieve size fraction. From Table 2, using Gaudin Schumann expression given by Allen, (2017):

$$\text{Size2} = \frac{(\textit{percentage passing size } 2)^2}{(\textit{percentage passing size } 1^2} \times \text{Size1} \qquad \text{Equation 3}$$

$$\text{X } (\mu\text{m}) = \left(\frac{\frac{80}{100}}{\frac{82.10}{100}}\right)^2 \times 350 = \quad 332.32 \ \mu\text{m}$$

Table 3: Particle Size Analysis of the "Product" of Test ore (Bassa-Nge iron ore)

Sieve size Range (µm)	Weight retained (g)	% weight retained	Nominal aperture	Cumulative % Weight retained	Cumulative % Weight passing
+355	04.74	04.74	355	04.74	95.26
-355+250	06.00	06.00	250	10.74	89.26
-250+180	15.26	15.26	180	26.00	74.00
-180+125	20.03	20.03	125	46.03	53.97
-125+90	23.40	23.40	90	69.43	30.57
-90+63	11.77	11.77	63	81.20	18.80
-63	18.80	18.80	-	100	0.00

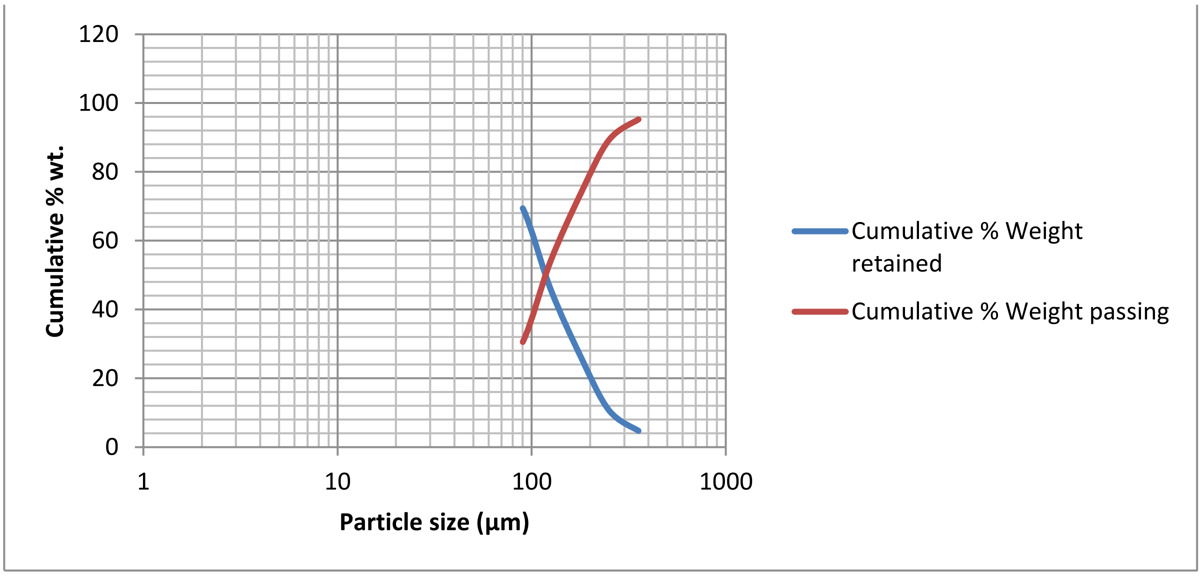

Figure 2: size against % cumulative retained and % cumulative passing of Bassa-Nge iron ore as product of the ball mill

Calculation using the result in the Table 3 and equation 3:

$$\text{X}\mu\text{m} = \left(\frac{\frac{80}{100}}{\frac{89.26}{100}}\right)^2 \times 250 = 200.81 \ \mu\text{m}$$

Table 4: Particle Size Analysis of the Feed Reference ore (Granite)

Sieve size Range (μm)	Weight retained (g)	% weight retained	Nominal aperture	Cumulative % Weight retained	Cumulative % Weight passing
+355	11.47	11.47	355	11.47	88.53
-355+250	20.37	20.37	250	31.84	68.16
-250+180	13.09	13.09	180	44.93	55.07
-180+125	14.31	14.31	125	59.24	40.76
-125+90	21.65	21.65	90	80.89	19.11
-90+63	18.62	18.62	63	99.51	0.49
-63	0.49	0.49	-	100	0.00

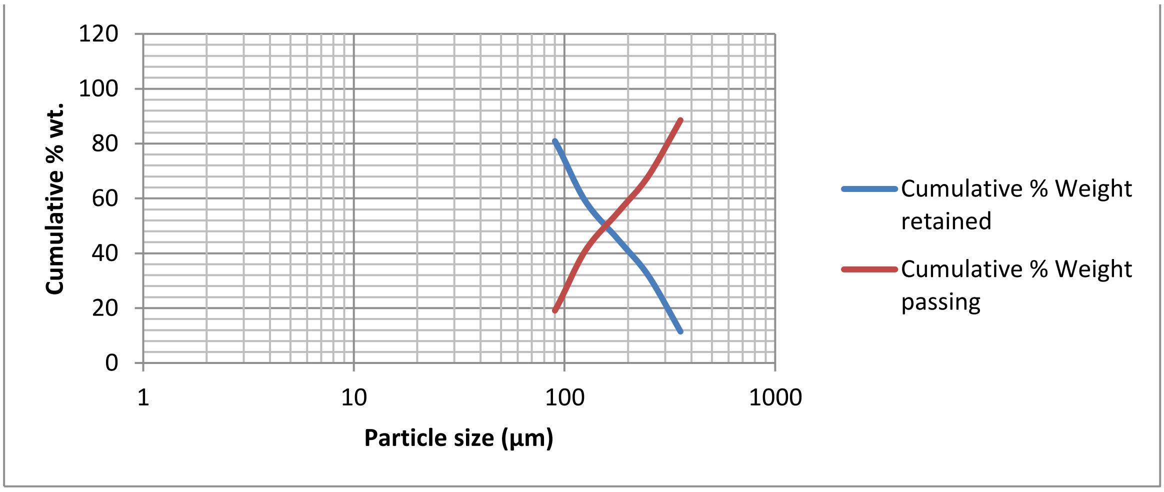

Figure 3: Particle size against cumulative % wt. retained and cumulative % wt. passing of Granite feed to ball mill

Calculation from the Table 4, using equation 3:

$$X\ (\mu m) = \left(\frac{\frac{80}{100}}{\frac{88.53}{100}}\right)^2 \times 355 = 289.88\ \mu m$$

Table 5: Sieve analysis of the Product of Reference Ore (Granite)

Sieve size Range (μm)	Weight retained (g)	% weight retained	Nominal aperture	Cumulative % Weight retained	Cumulative % Weight passing
+355	4.21	4.21	355	4.21	95.79
-355+250	7.84	7.84	250	12.05	87.95
-250+180	27.12	27.12	180	39.17	60.83
-180+125	15.56	15.56	125	54.73	45.27
-125+90	25.15	25.15	90	79.88	20.12
-90+63	16.40	16.40	63	96.28	3.72
-63	3.72	3.72	-	100	0.00

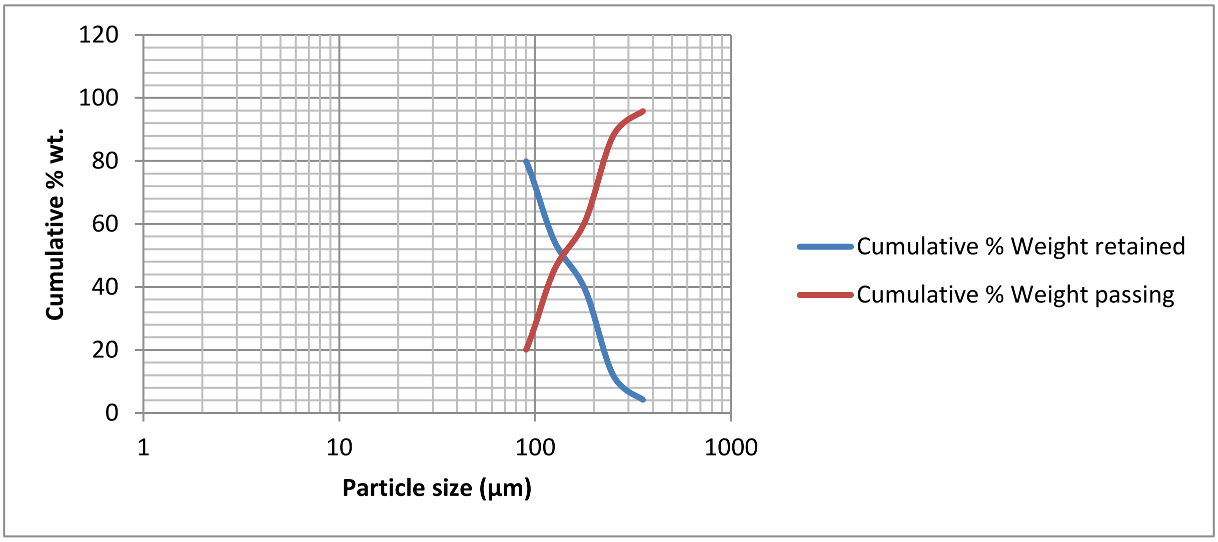

Figure 4: cumulative % wt. retained and passing of Granite product against sieve size fractions (µm) of the ball mill.

Calculation from the Table 5, using equation 3:

$$X\,(\mu m) = \left(\frac{\frac{80}{100}}{\frac{87.95}{100}}\right)^2 \times 250 = 206.84\ \mu m$$

Determination of the Work Index of Bassa-Nge Iron Ore

Granite as a reference ore and Bassa-Nge iron ore as the test ore, the work index of the test ore was calculated using the determined parameters below and equation 2:

Reference ore (Granite): P_r = 206.84 µm, Fr = 289.88 µm; Wir = 15.13 kWh/t

Test ore (Bassa-Nge): P_t = 200.81 µm; F_t = 332.32 µm,

Using Equation 2 as expressed bellow:

$$Wi = 10 \text{ x } 15.13 \times \left(\frac{\frac{1}{\sqrt{206.84}} - \frac{1}{\sqrt{289.88}}}{\frac{1}{\sqrt{200.81}} - \frac{1}{\sqrt{332.32}}}\right)$$

$$= 10.39 \text{ kWh/tonne.}$$

The energy expended in grinding the iron ore from 80% passing feed of 332.32µm to 80% passing product 200.81µm using the Bond's energy equation is found to be 10.39 kWh/tonne.

The results obtained from the test revealed that iron ore can be comminuted and the energy 10.30 kWh/t required to comminute the iron ore is less when compared to that of the reference ore granite (15.13kWh/t), This could be attributed to the nature of wrought texture of the iron ore with fragmented slip lines on the iron ore surface morphology. Also, the result revealed that 10.30kWh/t energy required to comminute the iron ore is less when compared to that of Birnin Gwari (18.55kWh/t), Koton-Karfe (11.33 kWh/t) and Gujeni (13.96kWh/t) (Salawu, 2015). But higher when compared to Itakpe (9.34kWh/t) and Ochokochoko (9.47 kWh/t). This can also be attributed the texture of the Bassa-Nge iron ore which appeared to show fault slip lines on the surface of the ore. However, the work index of the Bassa-Nge iron ore compares favourably with the work index of some iron ore presented in Table 1. Furthermore, it also signifies that approximately 10.30kWh/t of energy is required to reduce one tonne of the ore from 80% passing particle size 332.32µm to 80% passing particle size 200.81µm for one hour of grinding.

CONCLUSION

Determination of work index of Bassa-Nge iron ore, Kogi State was carried out. The work index of the iron ore was found to be 10.30kWh/t and using Mathur's grindability curves scale the ore was found to be a type B grade with soft medium texture that can be fragmented. All the results obtained compared favorably with those cited in the literatures.

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Performance of locally formulated Collector from Sesame Oil on flotation of Maru Copper Ore Deposit, Kaduna State, Nigeria

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Abstract

Maru Copper Ore, which contains Ccuprite (Cu_2O) and tenorite (CuO) copper phases was floated using locally prepared collector from sesame oil. The sesame seeds were obtained from Kachia in Kaduna State, Nigeria, its oil extracted and the oleic acid content was upgraded from 37.27% to 56.32% by distillation process. The floatation was conducted on three different particle sizes of – 125 + 90, – 90 + 63 and – 63 + 45 μm, and pH value of 9.5. Sodium silicate and pine oil were used as depressant and frother respectively. The flotation test result using the formulated collector showed that concentrates from – 90 + 63, μm particle size has the highest copper grade of 29.32% Cu and a recovery of 97.09% compared to 34.3% Cu with 97.59% recovery obtained using the standard oleic acid collector. The formulated reagents indicated favorable results though recorded lower assay compared to the standard reagent performance but has satisfied the minimum copper content requirement of 25% Cu content in concentrates for the metal extraction.

Keywords: Maru, Cuprite, Tenorite, Flotation, Formulation, Collector

Introduction

The discovery of copper deposits in Nigeria; Azara and Akiri in Nasarawa State, Anka and Maru in Zamfara State and areas along Benue trough– brought to fore the need to harness the benefits of this very valuable ore by developing process route for the Beneficiation of the copper ore. This is in tandem with the assertion by Thomas (2008) who said a newly discovered ore will be subject to investigation geared towards unbundling its nature and characteristics, thus permitting appropriate selection of steps for its Beneficiation. Hence, the need for this work which aims at identifying the specific nature and peculiarities of the Maru copper ore which, like most Nigerian deposits; lacks this information necessary for effective concentration of the ore and maximum metal recovery.

Copper has been one of the most important metals for over five thousand years Usaini (2011), and because of the art of making and shaping of copper and copper alloys, the traditional industries continue to survive and flourish. Development of renewable non-conventional sources of energy and the superiority of copper in solar heating is expected to dictate the use of the metal in large quantities in the near future (Prasad, 1998). In the coming decades, copper scarcity is likely to result in lean ore quality, which in turn will lead to a higher Gross Energy Requirement (GER) for copper production (Hansem, 2013).

The major problem faced by the Mineral sector in Nigeria is the loss in foreign earnings, this is as a result of exportation of unprocessed minerals (run-of-mine ore – R.O.M.) by artisans and small-scale miners. Loss of employment opportunities, skill acquisition through experiential learning and technology transfer, political power, technological independence and leverage; also plague the sector as a result. The need to reverse this trend necessitated this research work, which is part of the holistic attempt to provide an indigenous approach to copper value addition domesticated in adaptable local materials and technology.

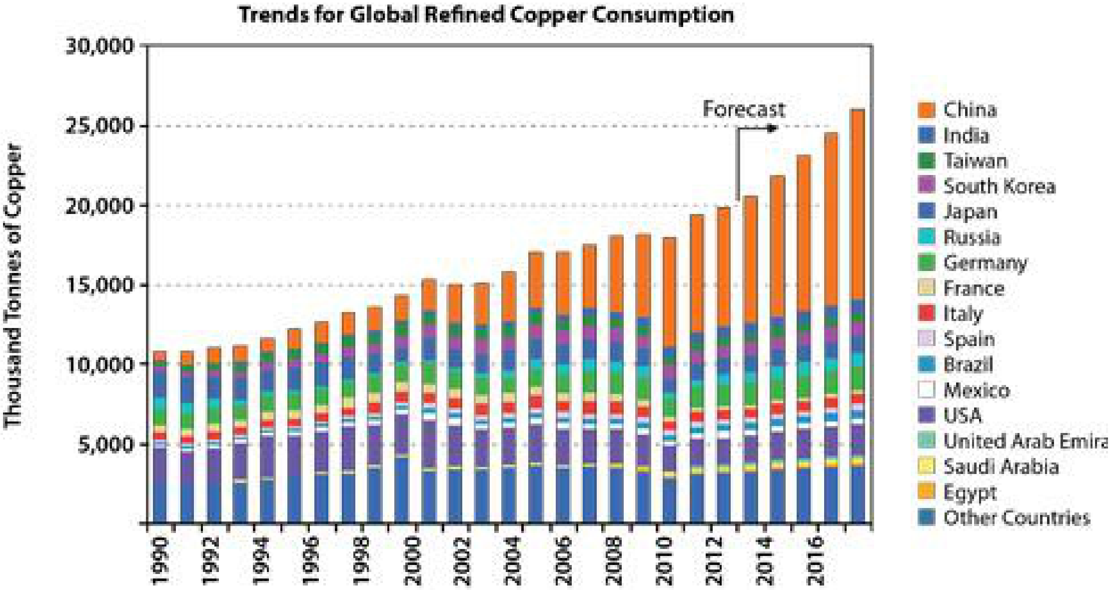
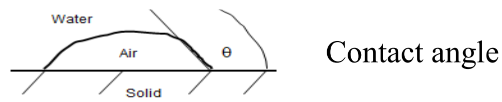


Figure 1.0: Projection of Copper global demand (Source: Usaini, 2017, Gonzalez, 2012

Flotation, specifically froth flotation, is both a physical and chemical process of concentrating finely ground ores that depends on the selective adhesion to air of specific minerals while others remain submerged in the pulp.

The process involves treatment of an ore pulp with suitable reagents to create conditions favourable for the attachment of certain (hydrophobic) mineral particles to air bubbles and render the other solids hydrophilic or water -loving.

The air bubbles carry the selected minerals to the surface of the pulp and form a stabilized froth, which is skimmed off while the other minerals remain submerged in the pulp.



Young Equation: $\gamma_{SA} = \gamma_{SW} + \gamma_{WA} \cos\theta$

where: γ_{SA}, γ_{SW}, and γ_{WA} are the surface free energies at the air-solid, water-solid, and air-water interfaces
θ is the contact angle

Figure 2.0: Contact angle between bubble and a particle (Source: Demise, 2014)

Although flotation was originally developed in the mineral industry, the process has been gradually extended to other fields.

To carry-out successful flotation, the mineral should be coarser than 48 mesh (about 295 microns or 0.295 mm in diameter) because beyond this limiting size, mineral cannot be effectively recovered; consequently, an ore that is to be floated must first be ground fine enough so that the desired mineral is all, smaller than this limiting size.

Flotation provides selectivity hence can be used to achieve specific separations in complex ores. It is best applied to very fine particles of minerals so that downward pull of gravity will be insufficient to overcome its adhesion to an air-water interface.

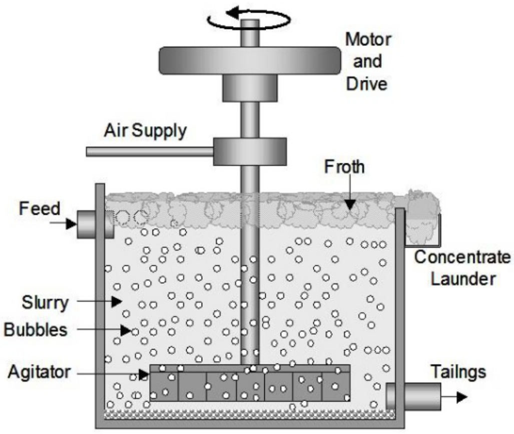


Figure 3.0: Sketch of Flotation Machine in operation

Mechanisms involved in Flotation

The essential mechanism of flotation is the attachment of mineral particles to air bubbles in such a manner that the particles are carried to the surface of the ore pulp, where they can be removed. The process encompasses the following steps:

a) Grinding the ore to a fine size to liberate the valuable minerals from one another and from the gangue minerals.
b) Making conditions favourable for the adherence of the desired minerals to air bubbles.
c) Creating a rising current of air bubbles in the ore pulp.
d) Forming a mineral-laden froth on the surface of the ore pulp.
e) Removing the mineral-laden froth.

Although grinding may not be considered part of the flotation process it is of importance, consequently affecting the separation process. Optimum flotation requires complete separation of valuable mineral from waste rock (gangue) but because it is not economically feasible it is rarely attained. Thus, leading to complications in subsequent floatation steps.

Location and Accessibility of Maru Copper Deposit

Nigeria's Copper deposits occurs in many Sstates; Abia; in the south east, Benue and Nasarawa Sstates in north central, Gombe in north east and Zamfara in north west, Nigeria.

The study area is located in an approximate area of 20km^2 consisting part of sheet 531 Maru NE. It lies within longitude 12^0 23.6^{i}N and 12^0 21.5^{i}N as well as latitude 6^0 27.2^{i}E and 6^0 24.43^{i}E, which forms part of Maru Local Government Area in Zamfara state. It has prominent villages such as Karakai, Gamagiwa and Gabbro amongst other settlements. The area is accessible via a tarred and untarred road, series of footpaths and cattle tracks as well as along river channels for traverse, (Abubakar, 2013).

Oxide of Copper Ores and Minerals

More than 120 oxide-containing minerals have been identified, mainly from the Central and South African regions, but only a few of these minerals are of economic value. Some of the most important copper oxide minerals are listed in Table 1.0.

In most cases, oxide copper ores contain more than one copper oxide minerals, and also contain mixtures of sulphide and oxide copper minerals. From a processing point of view, the oxide copper ores can be divided into the following five groups:

In oxide ores copper is predominantly Malachite with significant quantities of Cobalt oxides. According to the mineral composition, these ores can be sub-divided into two main groups:

(a) *Oxide ore that contains carbonaceous gangue minerals* (carbonate, dolomite) with little or no silica; and
(b) *Oxide ore, where silica is the predominant gangue mineral.*

The gangue composition of the ore plays a decisive role in selection of reagent scheme for beneficiation of the ore. These ores also contain cobalt minerals, mainly carrollite ($CoCuSO_4$) and Cobaltite (CoAsS).

Copper oxide mixed ore – Type 1: The main copper minerals found in these ores include Malachite, Pseudo-malachite, Chrysocolla and some Tenorite. These ores also may contain mainly siliceous gangue minerals, including Spherocobaltite as the main cobalt minerals. The carbonaceous types also contain an appreciable amount of clay slime minerals.

Copper oxide mixed ore – Type 2: In contrast to Type 1, this ore type contains Cuprite, Malachite and Azurite as the main copper oxide minerals. This ore type predominantly contains carbonaceous gangue, and usually, significant amounts of clay-like slimes.

Mixed copper sulphide oxide ores: These contain varieties of both sulphide and oxide minerals and are the most complex copper-bearing ores from a beneficiation point of view. The major copper minerals present in this ore type include Bornite, Chalcocite, Covellite, Malachite, Cuprite and Chrysocolla. In some cases, significant amounts of cobalt minerals are also present in this ore.

Copper oxide gold ores: Although this ore type is not abundant, they are of significant value because they contain gold. Only a few deposits in Brazil and Australia are known. The copper in these ores is represented by cuprite, native Copper, Antlerite and Tenorite. The gold is associated with cuprite, as an auricupride and several sulphosalts. The major problem associated with treatment of this ore type is the presence of large amounts of clay slimes in the form of Iron hydroxide and Illite.

Table 1.0: Some economically valuable copper oxide minerals

Mineral	Chemical formula	Cu content (% Cu)	Specific gravity (SG)	Colour
Cuprite	Cu_2O	88.8	5.9	Brick red
Tenorite	CuO	80.0	6.5	Black
Malachite	$Cu_2(OH)CO_3$	57.4	3.9	Green
Azurite	$CuO_3(OH)_2(CO_3)_2$	55.3	3.7	Blue
Brochantite	$CuO_4(OH)_6SO_4$	56.6	3.9	Emerald green
Atacamite	$Cu_2(OH)_2Cl$	44.6	3.8	Green, blue
Antlerite	$Cu_3(OH)_2SO_4$	54.0	3.9	Emerald green
Chrysocolla	$CuO.SiO_2$	10-36	2-2.4	Blue
Chaecantite	$CuSO_4.5H_2O$	25.5	2.2	Deep blue

(Schlesinger, 2011)

Copper Ore Concentration by Froth Flotation

The run-of- mine ore of copper from the mines will usually be comminuted first, to achieve liberation of the various grains in the ore matrix, and then copper minerals and waste rock are separated at the mill using froth flotation. The copper ore slurry from the grinding mills is mixed with milk of lime (water and crushed limestone) to give a basic pH, pine oil (a by-product of paper mills) to make bubbles, an alcohol to strengthen the bubbles, and a collector chemical called potassium amyl xanthate (or the potassium salt of an alkyl dithiocarbonate).

The xanthates are added to the slurry in relatively small quantities. Xanthate is a long hydrocarbon (5 carbons) chain molecule. One end of the chain (the ionic dithiocarbonate) is polar and sticks to sulfide minerals while the other end is nonpolar, containing the hydrocarbon chain is hydrophobic - it hates being in the water and is attracted to the non-polar hydrocarbon pine oil molecules.

Carboxilic acid flotation of malachite has been commercially used for over 70 years. This collector is prepared by heating a mixture of hydrolysed palm oil (or oleic acid) and fuel oil in a 3:1 ratio. This mixture is manly used for recovery of malachite from siliceous ores. The use of carboxylic acid for malachite flotation from carbonaceous ores resulted in both reduced concentrate grade and recovery.

Cationic flotation of malachite, using mono- and diamines in alkaline pulp, was also examined. Malachite floats readily using mono-amines under laboratory conditions.

It should be pointed out that there are several varieties of malachite, $Cu_4(PO_4)_2(OH)_4 \cdot OH$. Pseudo-malachite is difficult to float, and it is well known that pseudo-malachite can be floated with anionic collectors, but responds poorly to the sulphidization method.

In a number of oxide ores, cuprite (Cu_2O, Cu = 88.8%, SG = 5.9) is present as secondary minerals together with sulphides, malachite and tenorite. Cuprite can be floated using either sulphidization or anionic flotation methods. The flotation properties of cuprite are somewhat different from that of malachite. For example, using a sulphidization method for flotation of cuprite requires higher dosages of sulphidizer.

Some ore deposits contain cuprite as the principal mineral. Typically, these deposits contain appreciable amounts of slimes and clay minerals. The laboratory studies conducted on these types of ore indicated that improved metallurgical results can be achieved using the sulphidization method with ester-modified xanthate (Bulatovic, 1998).

Tenorite (CuO; Cu = 80%, SG = 6.5) is usually present in mixed copper oxide and sulphide ore. The flotation properties of tenorite are similar to that of cuprite.

Flotation practice in beneficiation of oxide copper minerals

Selection of a reagent scheme for beneficiation of oxide copper ores depends on many factors; some of the more important ones being:

- Type of oxide copper minerals present in the ore.
- Type of gangue minerals – some ore types contains silicate gangue free of slimes, which are the most amenable to flotation. Ores with dolomitic gangue can be beneficiated using sulphidization only. These ores usually contain an appreciable amount of clay slimes that have a detrimental effect on flotation. Some oxide ores contain talc, iron hydroxides and iron oxides. In general, each ore type requires the selection of different reagent schemes.
- Degree of liberation – the relatively fine-grained ores are more amenable to flotation than the disseminated ores, which require finer grinding.
- Chemical composition and physical structure of the copper minerals play an important role in the floatability of oxide copper minerals. Oxide copper minerals are often porous and aqueous soluble. Because of that, they tend to slime during grinding.

The fatty-acid-based collectors have been employed for the past 60 years for flotation of oxide copper/cobalt minerals from Congo, a company formerly owned by Union Minera (Belgium).

The fatty acid modification was used in operating plants at Kolwezi, Koumbore and Kakanda. The fatty acid used was hydrolysed palm oil prepared as a mixture consisting of 75% hydrolysed palm oil, 21% gas oil and 4% Unitol.

Materials and Methods used
Sampling:
Grab sampling method was used to obtain the sample which necessitated the application of Gy's formula to obtain a good representative material. The sample was comminuted by crushing and grinding using jaw crusher and ball mill. The materials from the crushing stage provide a feed with particle size of about 2mm while the ground particles were milled to less than 1000 µm. The commuiniuted ore was classified passing through three sieves of – 125 + 90, – 90 + 63, and – 63 + 45 µm. The XRD result of the ore revealed Cuprite (Cu_2O) and Tenorite (CuO) as the copper species and gangue minerals of quartz and feldspar, (Usaimi, 2017)

Formulation of Enriched Palm kernel oil
Sesame oil was extracted by grinding the seeds into powder and making this into small balls. The 'cakes' made from the seeds were placed into a container, with allowance of space for hot water, after which they were pressed at a pressure sufficient enough to extract the oil. Pressure was maintained until all the available oil was extracted. The oleic acid content of the oil was then upgraded via a distillation process. The different boiling temperatures of the various components were established, which range from 250.1°C (Myristic acid) to 381.5°C (Docosenoic acid).

The modified process heated the oils to only 320-330°C. This expectedly evaporated all other components that have boiling temperature of 330°C and below. The liquid remaining after the process was taken for analysis to establish the degree of enrichment. Ten hours of heating was used over two days period at 320-330°C.

Concentration by flotation of Maru ore
The mineral concentration to obtain the enriched portion by froth flotation was carried out using three (03) sieve sizes, ?+ 45 µm, – 90 + 63 µm and – 125 + 90 µm; of the head samples.
A 100g sample of the ore was used for each sieve size. The sample was prepared and poured in the flotation cell with sufficient water to make slurry containing about 30% solids by weight. The air inlet valve was made sure to be closed and the impeller was turned on. The slurry was stirred for one minute. Conditioning was carried out by agitating the pulp for 2 minutes (with air inlet valve closed). Sodium Carbonate was added until pH of 9.5 was attained. After 2 minutes, the required amount of the collector; Oleic acid, (the standard collector), and then the enriched Palm kernel (local collector) and frother (pine oil) in 4ml reagents dosage were added for each of the sieve sizes used. The pulp was agitated for another 1 minute. The air inlet valve was turned on and the flotation started. The froths were collected by skimming off the froth for 3 minutes in each case. The collected concentrates and tailings were dewatered, by filtration and thermal drying. The dried weight of concentrates and tailings were recorded on data sheet and their percentages recorded.

Table 2.0: Chemical and Physical Characterization of Sesame oil

S/No	Compound name	Composition (%)	Boiling Temp °C
1	Lauric acid $C_{18}H_3ClO$	4.32	298.89
2	Oleic acid $C_{18}H_{34}O_2$	37.27	360
3	Palmitic acid $C_{17}H_{34}O_2$	19.61	351
4	Stearic acid $C_{18}H_{36}O_2$	7.39	361
5	Myristic acid $C_{14}H_{28}O_2$	1.86	250.5
6	Docosenoic acid $C_{22}H_{42}O_2$	11.31	381.5
7	Octanol $C_{12}H_{26}O$	3.64	195
8	Oxalic acid $C_{24}H_{46}O_2$	2.09	166

Plate 1: Flotation using sesame oil on Maru copper ore

Results and Discussion

Table 2.0 presents chemical composition and boiling temperature of sesame oil used in the work. The results show the oil contains three very functional compounds in flotation of oxide copper according to (Usaini, 2014 & Joseph, 2014). These are oleic acid, palmitic acid and lauric acid having a combined composition of 61.12% of the raw sesame oil. This suggests a likely successful application of the oil in flotation of copper oxide ore.

The result (Table 3.0) has shown that all the particle sizes used at this stage with Oleic acid as collector has attained very reasonable assay level and satisfied the basic smelters need of having a concentrate of 25-40% copper metal. The metal recoveries were also quite high, with 84.57% and above in all the three runs at this stage. The ratios of concentration, which give the ratio of the feed to the concentrates in weight and enrichment ratios, which is an expression of ratio of the assay (metal content) in the concentrates to that in the feed, were all reasonably good. The weights of the concentrates show steady increase along increase in surface area. This can be seen from the weight of concentrates where – 125 + 90 µm sizes gave 18.25g; – 90 + 63 µm had 25.14g and – 63 + 45 µm being the finest sieve size fractions with 33.67g. The highest assay value of the concentrates was obtained at – 90 + 63 µm sieve size fractions, which supported the earlier determination made on the liberation size of Maru ore at – 90 + 63 µm. The highest recovery was also obtained at – 90 + 63 µm sieve size fractions having 86.87%. The fact that the grades and recoveries in this process were not inversely proportional made the choice or selecting of particle size simpler for optimum copper metal recovery.

Feed		Products				Recovery %	Ratio of Concentration F/C	Enrichment ratio (c/f)
Wt (g)	Cu %	Concentrates		Tailings				
		Wt (g)	Cu %	Wt (g)	Cu %			
100	2.43	15.11	26.41	74.88	0.53	79.79	6.62	10.86
100	2.43	22.62	29.32	77.37	0.45	82.75	4.42	12.07
100	2.43	26.74	27.51	73.26	0.51	80.51	3.74	11.32

Flotation result using enriched sesame oil and varied particle size

Feed		Products				Recovery %	Ratio of Concentration F/C	Enrichment ratio (c/f)
Wt (g)	Cu %	Concentrates		Tailings				
		Wt (g)	Cu %	Wt (g)	Cu %			
100	2.43	18.25	32.7	81.74	0.40	84.57	5.48	13.46
100	2.43	25.14	34.3	74.76	0.34	86.87	3.98	14.12
100	2.43	33.67	33.8	66.32	0.39	82.47	2.92	13.85

is a flotation test result carried out eve size fractions of – 125 + 90 μm, μm and – 63 + 45 μm. The flotation of reagent dosage, pH value and ity were kept constant through-out 5 and 30% solid to water by weight ly. Enriched sesame oil (in place of ard reagent for oxide copper) was collector, pine oil as frother and cate as depressant.

ched sesame oil enhanced the ce of the locally developed reagent % Cu on – 125 + 90 μm sieve size, u on – 90 + 63 μm sieve size and Cu on – 63 + 45 μm. This has y satisfied the smelters nt of 25% Cu minimum grade for etal extraction. The recoveries were minimum and enrichment ratios of 07 and 11.32 on – 125 + 90 μm, – n and – 63 +45 μm respectively. The y performance of the upgraded il can be attributed to increase in ion of the oleic acid content of the stillation.

t the two (2) collector reagents; l, and enriched sesame oil, the ve revealed that the effect of size in very significant. While using oleic ence of particle size was seen on the weights of materials that reported as concentrates with direct increase with increase in surface area of the particles. However, the quality (copper contents) of the concentrates did not simply obey proportionally and directly increase in surface area. The 63 μm particle size contained the highest assay value of 34.3% Cu, in contrast to the 32.7% Cu and 33.8% Cu on – 125 + 90 μm and – 63 + 45 μm respectively confirming that, – 90 + 63 μm particle size is the liberation size of the ore. The – 63 +45 μm, though finer and reporting higher concentrates by weight, has lower copper contents compared with the liberation size of the ore. This therefore, means that besides loss of immense man hour, technically, the copper metal will be lost along with the tailing by over grinding. This is in addition to energy expended and the inherent difficulty in increasing surface area. The behaviour of the particle size in this work is supported by (Feng, 1999) who stated that ‘particle size is an important parameter in flotation operation. Nowadays, particle size is often measured and controlled in flotation concentrators’. (Santana, 2008), asserted that mineral particle plays a significant role on the flotation process. This means that an optimum size be determined and strictly followed for higher productivity. Equally, (Trahar, 1981) in his work titled, 'A rational interpretation of the role of particle size in flotation', confirmed that the minimum degree of hydrophobicity necessary for the flotation of a particle depends upon its size and due to that recovery size curves are valuable diagnostic aid to the assessment of flotation performance.

The performance of enriched sesame oil was very encouraging and satisfied the minimum required copper metal grade. This can be attributed to the content of the oil having active element and compounds that have the same chemical characteristics of adsorption on the surface of the selected mineral grains, as the conventional collectors. The sesame oil abilities were greatly enhanced by the distillation process that removes some of the non-active components. Collectors generally are substances used in floatation for the purpose of adsorption on the surface of minerals, (Wills, 2006), rendering them hydrophobic (or aerophilic) and facilitating bubble attachment. The efficiency of the developed oil can equally be said to be aided by the fact that the gangue minerals were predominantly silica and quartz which has the highest polarity (hydrophilicity), hence are readily submerged in the pulp

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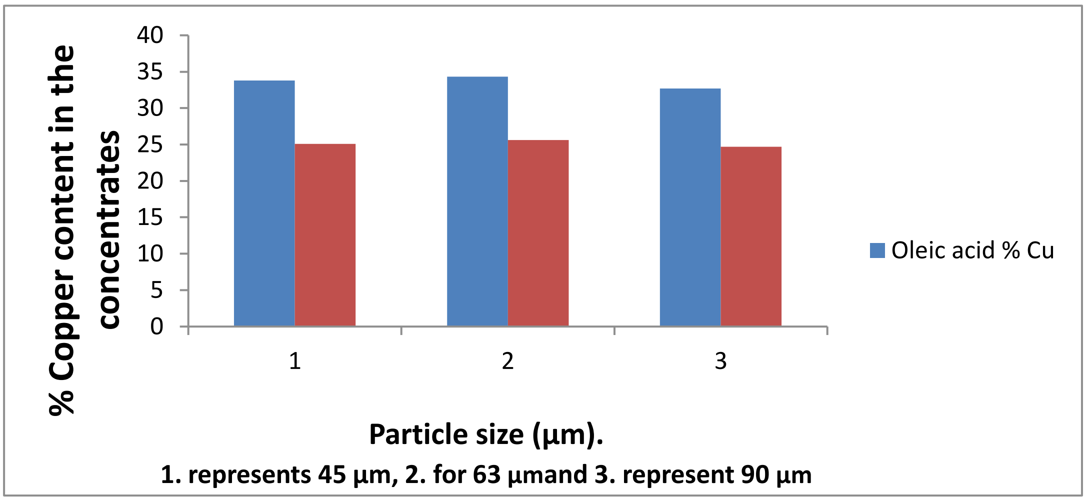

Figure 4.0: Comparison of the performance of Oleic acid against Sesame oil

Conclusions and Recommendations

The flotation test using enriched sesame oil has produced a concentrate of 24.7% Cu, 25.6% Cu and 25.1% Cu on – 125 + 90, – 90 + 63 and – 63 + 45 micro meter particle sizes (Table 3.0) compared to the results given by oleic acid as collector having 32.7% Cu, 34.3% Cu and 33.8% Cu on the corresponding particle sizes, (Table 4.0 and Figure 3.0). This has shown that the copper contents in the concentrates from all the particle size fractions using enriched sesame oil has attained the 25% minimum copper content for the extraction of copper. The result is a proof that locally derived sesame oil can be upgraded to produce an enriched substance that can effectively replace the conventional copper oxide collector with a lot of success. Further investigation is therefore suggested on other oleic acid containing oils.

Acknowledgements

The authors expressed their gratitude to the staff of Minerals & Petroleum Resources Engineering and Laboratory staff in Chemical Engineering Laboratory of Kaduna Polytechnic.

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Bida, Ajayi and Ibrahim (2020) Nigerian Mining Journal, (18) 107 - 116

Optimization of Blast Design for Drill and Blast Cost Reduction in Ashaka Cement Quarry, North Eastern Nigeria

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Abstract

Optimization of drilling and blasting operations was carried out to reduce drilling and blasting cost in Ashaka Cement Limestone Quarry. The current cost trends associated with drill and blast operation were determined. The study was conducted on two operational pits of the mine namely, North-West (N-W) and South-West (S-W) pits. The surface mine of N-W pit was faced with high cost trends in its drilling and blasting operations. The current and two proposed sets of two geometric parameters for N-W and S-W active pits were assessed during fragmentation. The estimated mean fragment size for the proposed blast parameters was within the desired fragment size (7-1000 mm) of the mine and explosive energy was effectively utilized per in-situ material blasted. The estimated total cost variation for N-W pit was N2,773,048.19 at a power factor of 0.25 kg/tonne but the total cost of variation reduces at S-W pit to N1,891,300.00 using a powder factor of 0.164 kg/tonne. The optimization of drilling and blasting costs requires characterization of rock mass strength to guide in the selection of appropriate powder factor. Thus, the right explosive type and quantity should be matched to yield optimal fragmentation to attain effective blast design. For this reason, it was recommended that the present blast design should be reviewed towards reducing explosives consumption. Key Words:

Keywords: Burden, Spacing, Explosives Consumption, Optimal Fragmentation, Powder Factor, Cost Saving

INTRODUCTION

Mining can be described as the extraction of valuable minerals as well as the removal of other of geological materials from the earth. Mining is an activity that has been carried out since prehistorical times, where stone and other metals were extracted from the earth. The process of mining has to be profitable to the miner seeking to gain from the extract, and involves prospecting for the extracts, analyzing the profitability of the mine, the process of extraction and finally the reclamation. For many years, these mining processes have led to adverse negative environmental effects either during the mining process or long after the mines have been closed. Stone as one of the valuable mineral extract has always been used since the beginning of civilization to make early tools and weapons (Hartmann, 1992).

Quarrying is an important business since it provides the means by which earth resources are extracted, processed and used for construction, agricultural and manufacturing industries. Different types of rock are quarried and crushed for different purposes. For instance, limestone is quarried and the rock processed for use in agriculture and cement production, while granite or sandstone rock is quarried for use in the construction industry. Limestone quarrying is a major economic activity in many developing countries including Nigeria. Quarrying is a very important activity in human life as it helps in the development of infrastructure such as roads, electricity, health-care center etc. in a given area. Assessment of drilling and blasting cost reduction through blast optimization has to do with the mining cost operation as a result of global and financial crises, and fluctuations in the price of technologists in the industry to search for innovative ways of reducing overall mining cost. The cost of drilling and blasting operation greatly contributes to the “high cost trends of the overall mining operation” (Anon, 2014; Anon 2012, Palangio et al, 2005 and Bozic, 1998). Drilling is one of the critical and important operations of every hard rock mine and contributes about 15% of the overall mining cost in some mining operations (Gokhal, 2010).

In view of the forgoing, it is imperative to assess the drilling and blasting operation at Ashaka Cement Limestone Quarry so as to reduce the overall cost of cement production. It should be noted that the high of cost of drilling and blasting operation of mining and quarrying have scared a lot of investors away in the mining industry and these has led to the innovative ways to reduce the overall mining cost.

Geology of the Study Area

The Ashaka Cement Quarry is located within the N-S aligned Gongola Basin of the Upper Benue Trough northeastern Nigeria. The area is geologically bounded to the south by the "Wuyo-Kaltungo High" which is the median zone of the Upper Benue Trough and also the area separating the Gongola Basin from the Yola Arm. The Wuyo-Kaltungo High is characterized by four major NE-SW and N-S trending faults, the Gombe, Bima-Teli, Kaltungo and the Burashika faults. These are basement faults reactivated during early Cretaceous times. This group of faults, together with NW-SE and N-S trending faults are believed to have played a major role in the development of early Cretaceous sub-basins in the Upper Benue Trough (Guiraud, 1990). The northern border of the Ashaka area is the Dumbulwa-Bage high which is also the northern border of the Gongola Basin, separating it from the Bornu Basin. Zaborski et al (1998) described the Dumbulwa-Bage High as shoal area during the late Cenomanian early Turonian Bima Sandstone. This is shown by the wedging out of the Middle Bima, Upper Bima and Yolde formations and the attenuation of the Kanawa Member in the Dumbulwa-Bage High. To the east and west of the Ashaka area are the Biu Plateau and the Kerri-Kerri Basin respectively [Figure 1].

Factors in Controlling Drilling and Blasting Costs in Mines and Quarries

The following are the factors in controlling drilling and blasting costs:

(a) Types of explosive to be used: Ammonium nitrate and diesel oil process have invited many operations in mine to re-examine their drilling and blasting methods to find ways of reducing costs. Under the changed scenario, drilling and blasting companies requires to optimize their blasting and costs through a blast optimization review. In the case of main explosive, charge, many operations are using more explosive, more energetic explosives than nudged with the small drilling pattern they would use for ANFO. All that results are additional throw of the cost. Some operations are also using cartridge explosives rather than bulk explosives, the additional explosive that you place into the blast hole because you are filling the annular space that you would not fill with cartridge explosives will allow you to reduce the drilling cost by expanding the drill pattern (Nanda, 2003).

Figure 1: Map showing Ashaka Cement Factory [Source: Google Map, 2021]

(b) Types of drilling pattern: The drilling pattern is also a factor controlling cost of drilling and blasting when using a 4 by 4 in drill pattern while the actual fact of the drill pattern on average is 3 by 3.5m. This small differences in average pattern dimension would already increase cost per ton by over 11%.

(c) Type of initiators: The choice of initiators is also a factor controlling drilling and blasting cost, whereby the choice could either be (i) the use of redundant path system, and (ii) the use of single path shocked tube system.

(d) The use of desk loading: Is used in many operations to reduce the quantity of explosives per delay in order to reduce vibration. Deck/ loading increase the number of initiators and primers needed, and often produces less efficient fragmentation than when using a fall column of explosive (Nanda, 2003).

Calculation of Drilling and Blasting Parameters for Quarry

In order to achieve the optimum results of blasting under all conditions, a thorough understanding of the following parameters is required in quarry: Type, weight, distribution of explosives; Nature of the rock; Bench height; Blast hole diameter; Burden; Spacing; Sub drill depth; Stemming; and Initiation sequence for distribution of explosive in powder factor (Singh, 2017).

Physico-Mechanical Properties of Rocks that affects Drilling

Below are some physical properties of rocks that affects the rate and ease of drilling and blasting.

Hardness: is the characteristic of a solid material expressing its resistance to permanent deformation. Hardness of a rock material depends on several factors including material composition and density. A typical measure is the Schmidt Rebound Hardness number test.

Abrasivity: measure the abrasiveness of a rock material against other material such as steel. It is an important measure for estimate mean of rock drilling and boring equipment. Abrasivity is highly influenced by the rock material in the rock material. The higher the quartz content the higher the abrasivity.

The permeability: is a measure of the ability of a material to transmit fluids. Most rocks including igneous, metamorphic and chemical sedimentary rocks generally have very low permeability (Audet, 2011).

The Powder Factor

Powder factor is the relationship that shows the amount of explosive that is required to blast a unit of volume of rock. It describes how much explosives are needed to blast a particular volume of rock. Powder factor should not be used to design blasts. It is more useful for accounting purpose as it determines the explosive consumption for every tonnes of ore/ rock removed. The powder factor is closely related with the efficient blasting.

Higher energy explosive, such as those containing large amount of aluminum powder, higher density can break more rock per unit weight than lower energy explosives. Most of the commonly used explosive products have similar energy value and thus, have similar rock breaking capacities. Soft and low-density rock requires less explosive energy than hard, dense rocks. Large blast holes' diameter requires less explosives per volume of rock as a larger steaming height is usually left as compared to small blast hole diameter. The powder factor is a function of the rock type, rock density and geological structure. Powder factor for surface blasting can vary from 0.1 to 1.1kg/m^3 with 0.12 to 0.45kg/m most common (Yusuf, 2015).

The powder factor for a single hole is given by this formula

$$P.F. = \frac{P \times 0.34p \times D^2}{B \times S \times (H/27)}$$

Where Pc = Powder Column (m)
P = Density of explosives (g/m^3)
D = Diameter of drilled hole (m)
B = Burden (m); S = Spacing (m);
H = Bench height (m)

$$\text{Powder Factor (PF)} = \frac{QuantityofExplosives}{VolumeofBlastedRock}$$

High explosives such as containing large amount of Aluminium, can break more rock, unit weight than lower energy explosive soft, low density rock dense. Large/ Hole pattern requires less explosive per volume of rock because a large portion of stemming is used. Poor powder factor distribution in large diameter blast holes results in coarse fragmentation (Yusuf, 2015).

RESEARCH METHODOLOGY

Site Visitation

The researchers visited the project site in order to obtain information about drilling and blasting operations, types and costs of explosives and accessories, and other operations connected to blasting.

Interviews

The interview undertaken at project site involved the Quarry personnel, Safety officer, the Driller and Blaster. Information obtained was used to deduce drilling and blasting costs after optimizing blasting parameters.

Photographs

The photographs were used to capture the quarry face and the general outlay of various pits being worked.

Simulation using Microsoft Excel

Various data obtained was used for simulation through Microsoft Excel to obtain optimal blasting techniques.

RESULT AND DISCUSSION

Data Presentation

Ashaka Quarry currently operates two pits namely N–W and S–W pits. Current blast design data for the various pits including burden, spacing, blast hole, diameter, bench height, sub-drill, stemming and drilling cost were obtained from the quarry. Due to several uncontrollable geological factors and the type of mineralized zones being mined, the quarry has adopted different parameters for drilling and blasting the limestone in the pits. The staggered drilling pattern was generally used for the production.

Drill Parameters

The drill parameters used for limestone operation in each pit are shown on Table 1. Similarly, data on explosive consumption in the quarry including the explosive type, relative explosives energy (REE), explosive density and explosive cost (including initiation system, bottom and column charges) were obtained from the quarry and these are summarized on Table 2.

Table 1: Drill Parameters obtained from the Quarry

Parameter	N – W Limestone Pit		S – W Limestone pit	
	Actual	Proposal	Actual	Proposal
Spacing (S), m	4.5	5	4.5	5
Burden (B), m	3.7	4	3.7	4
Depth m	6.5	7	6.5	7
Density of H. E	1.2	1.2	1.2	1.2
Stemming (T)	1.8	2	1.8	2
No of Holes	120	120	120	120
Bench Height (H), m	6.5	7	6.5	7
Sub – drill (U), m	0.5	1	0.5	1

Table 2: Explosive Data used for this Study

Parameter	Value
Explosive Type	Blend (70% Ammonium Nitrate and 30% Rock Gel)
Explosive Density (kg/m^3)	0.6 for ANFO /0.2 Rock Gel
Explosive Cost[₦]	23.852.00
Actual powder factor, P.F (kg/m^3)	0.23

The quality of the blast output using the current drill geometric parameter (Table 1) and the explosive data (Table 2) used by the mine lead to high fragmentation adopted at AshakaCem.

New sets of drill geometric parameters proposed for blasting the limestone in the two pits operation were also assessed and were found to be 4.5 spacing and 3.7 burden used for both N-W and S-W pits. The total costs per Bank Cubic Meter (BCM) of drilling and blasting using the mines current drill and blast parameters of the two pits drill geometric parameters for blasting limestone in the active pits were determined to be ₦2,773,048.19 for N-W pit and ₦1,891,300.00 for S-W pit.

Optimization of Drill and Blast Parameters for the Quarry

Drilling and blasting operations of the mine were closely studied to identify alternative geometric parameters for blasting based on the fragmentation model that reduces the total cost of blasting. Other technical parameters that would significantly reduce costs and improve productivity, while maintaining the desired rock fragmentation and wall control were also considered.

To assess the blast performance and further generate appropriate sets of geometric parameters for drilling and blasting in a surface mine, it is recommended to use the Kuz-Ram fragmentation model which is the best estimator (Cunmingham, 1983; 1987; 2005) of geometric parameters. It is also a tool for examining how different parameters could influence blast performance. The major factors for selecting the optimum and appropriate set of geometric drill and blast parameters of a mine include the total cost BCM blasted and the desired mean fragmentation size. The mine is committed to achieving a mean fragment size.

The effect of changes in the current drill and blast parameters on fragmentation are presented on Table 3 to 4. The two new sets of drill and blast parameters (proposal 1 and proposal 2) used for blasting both limestones in the two different pits are also shown on Table 3 to 4. For each proposal, the explosive type and density; for each proposal, the explosive type and density; REE, and loading density were maintained as it is currently being used in the mine.

Drill and Blast Cost Evaluation

The drilling and blasting performance in terms of the total cost per BCM using the current drill and blast data compared to the proposed drill and blast geometric parameters for blasting limestone in the North-West and South-West pits are shown on Tables 3 and 4 respectively.

Table 3: Evaluation of Drilling and Blasting for N-W pit

Parameter	N-W Limestone Pit	
	Actual	Proposal 1
Burden (B) m	3.7	4
Spacing (S) m	4.5	4.8
Hole Diameter mm	120	120
Depth (D) m	6.5	6.5
Sub-drill (V) m	0.5	0.5
Stemming (T) m	1.8	1.8
Number of Blast Holes	120	120
Limestone Estimated Tonnage (tons)	30,519.45	30,519.45
Explosive mass per Hole (kg)	77.44	54.21
Cost of high explosive per Hole (N)	27,918.67	19,543.79
Charge length (m)	5.7	5.7
Total mass of explosive (kg)	22,147.84	15,504.06
Total cost of high explosive (₦)	7,984,739.28	5,589,523.71
Total cost of blast hole with O.B (₦)	21,629,619.25	19,234,403.71
P.F (kg per Tons)	0.25	0.17
Total cost variation		2,773.048.19

Table 4 Evaluation of Drilling and Blasting for S-W Limestone pit

Parameter	S-W Limestone Pit	
	Actual	Proposal 2
Burden (B) m	3.7	4
Spacing (S) m	4.5	4.8
Hole Diameter mm	120	120
Depth (D) m	6.5	6.5
Sub-drill (V) m	0.5	0.5
Stemming (T) m	1.8	1.8
Number of Blast Holes	120	120
Limestone Estimated Tonnage (tons)	72,738.02	83,878.82
Explosive mass per Hole (kg)	66.16	45.87
Cost of high explosive per Hole (N)	23,852.00	16,537.05
Charge length (m)	4.7	4.7
Total mass of explosive (kg)	18,921.76	13,118.82
Total cost of high explosive (₦)	6, 821,672.92	4,729,596.99
Total cost of blast hole with O.B (N)	20,466,552.92	18,374,476.99
P.F (kg per Tons)	0.26	0.15
Total cost variation		

Performance of N-W Pit

It is observed in table 4.3 that by adopting to the proposal in the N-W pit the powder factor (PF) would reduce fairly from 0.25 to 0.174kg/mg3, the explosive mass per hole of drilling and blasting would be reduce from 77.44 to 54.21kg. The cost of high explosives used for blasting will be reduced from 27,918.67 to 19,543.79, the total mass of explosive used for drilling and blasting was reducing from 22,147.84 to 15,504.06kg, the total cost of high explosive used for drilling and blasting was reduced from ₦7,984,739.28 to ₦5,589,523.71, the total cost of blast hole with overburden used was reduced from ₦21,629,619.25 to ₦19,234,403.71, total cost variation used in N-W pit is ₦2,773,048.19

Performance of S-W pit.

As depicted on Table 4 the set drill and blast parameters for the proposal 2 with the lowest blasting is a better alternative for drilling and blasting in the S-W pit. To achieve this, the burden should be increased from 3.70 to 4m; the spacing increased from 4.70 to 4.8m; the stemming height increased from 1.80 to 2.00 (T) m and the PF reduced from 0.260 to 0.15kg/m^3. By adopting this proposed alternative for S-W pit the total cost of variation will now be reducing to ₦1,891,300.00.

Discussion

This study was carried out to determine if changes in the design of drilling and blasting can reduce the cost incurred. The result from the study has shown that the cost per unit of explosive and accessories used in blasting can be reduced by increase of burden and spacing and reduction of powder factor. It also revealed the cost per unit of explosives and some accessories used in blasting. Figs. 2 and 3 show the cost of variation of explosive estimation sheet N-W. The powder factor used for blasting of the drill hole was 0.25kg/ton to blast 120 holes and the cost used in blasting these holes was ₦2,773,048.19. In the second pit covering south-west, the powder factor used for blasting of the drill hole was 0.164kg/ton to blast 120 holes and the cost used in blasting this hole was ₦1,891,300.00 showing reduction in the cost to the tune of N881,746.19. Hence, the higher the powder factor the higher the cost of blasting and vice versa.

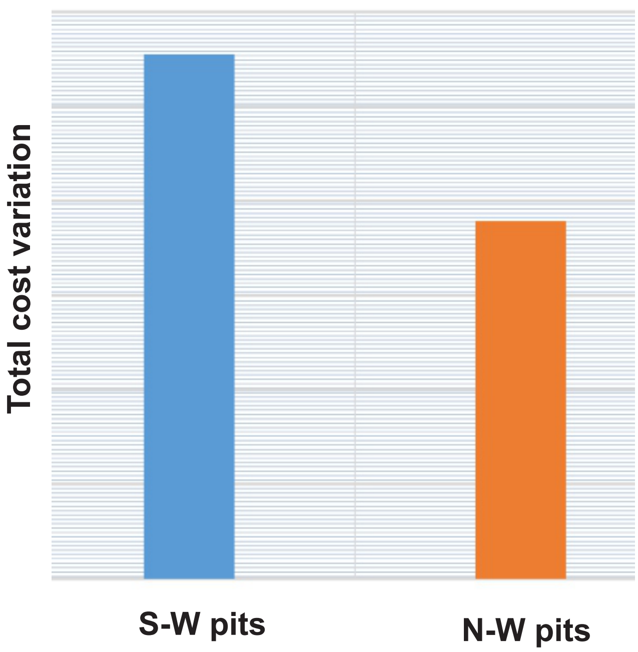

Figure 2: Comparison of total cost variation between S-W and N-W pits

EXPLOSIVES ESTIMATION SHEET			
HIGH EXPLOSIVE (H.E)	**PIT:1 N-W**	**FARM TRACK**	
ACTUAL DEPTH (M)	6.5	SPACING (M)	4.5
NUMBER OF HOLES	120	BURDEN (M)	3.7
STEMMING DEPTH (M)	1.8	VOLUME (M^3)	12987.00
DEPTH (M)	4.7	TONNAGE (TON)	30,519.45
H.E DENSITY	1.2	PF (KG/TON)	0.25
DIAMETER (MM)	120	14400	
	WEIGHT/HOLE	**QTY/HOLE**	**TO T.H.E/HOLE (KG)**
H.E WEIGHT PER PCS	63.85	22.97	63.85
	QTY RQD	**TOT. H.E (CTN)**	**TOTAL H.E (KG)**
TOTAL (CTN & KG)	2756.11	**306.23**	**7655.87**
H.E DEPTH (M)	13.57	**4.70**	

Figure 3: Cost of variation of explosive estimation sheet of pit N-W.

CONCLUTIONS AND RECOMMENDATIONS

Conclusion

The optimization of drilling and blasting cost requires characterization of the rock mass strength to guide in the selection of appropriate powder factor. For effective blast design, it is important to match the right explosive type and quantity that will give optimal fragmentation. Thus, good knowledge of drilling and blasting parameters, rock characterization, explosive strength, explosive accessories, blast design and initiation system are very essential in the optimization of drilling and blasting for the overall economy of limestone fragmentation.

Recommendations

The following measures are hereby recommended to improve the productivity of limestone at Ashala Limestone quarry:

(i) The blast design should be reviewed for possible reduction in the explosives consumption.

(ii) To ensure efficient utilisation of blasting energy in the limestone breaking process, there is need to ensure further training and re-training of staff which will in turn lead to higher efficiency and hence higher productivity.

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