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NIGERIAN MINING JOURNAL

INFORMATION FOR AUTHORS

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Papers should be typewritten with double line spacing and wide margins on one side only. Each page should be numbered. The first page should include a concise title of the paperand the author(s) name(s), affiliation(s) and addressees). In order to maintain consistency, titles such as Engr, Dr, Prof, should be avoided as they frequently change. The authors have to secure the right of reproduction of any material that has already been published elsewhere.

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Journals: Ojo, O and Brook, N: (1990) The effect of moisture on some mechanical properties of rock. Mining' Science and Technology, 8(2), 14-26

Unpublished work: Umoru, E.E. (1996) Re-appraisal of mining methods of Ameka Lead/zinc deposit. Unpublished HND project, Kaduna Polytechnic Kaduna, Nigeria.

Proceedings: Bello,S.B. (1996) Optimisation of mning parameters of sub-basalt underground mine development of solid mineral for national self-reliance and economic growth. Edited by E.O.A. Damisa, Engineering Conference Series, pp 202-215

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Analysis of Drill Bit Dull Grading in the Niger Delta

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Abstract
This paper is simply an overview of the various analysis carried out on drill bit dull grading in the Niger Delta (Offshore) during the drilling of three different wells; well 13, well 14, and well 15. When a section of the hole has been drilled and the bit is pulled from the wellbore, the nature and degree of damage to the bit is determined by using the qualitative-visual inspection and/or the quantitative-gauge ring method before it is carefully recorded on the bit record. A system known as the Dull Bit Grading System, a system devised by the International Association of Drilling Contractors-IADC is used to facilitate the grading process before it is then recorded on the bit record. This bit record would help in proper selection of the type of bit to be used for the Niger Delta formation.
Keywords: Drilling, Niger Delta, Bit, Analysis, Grading, Well, Offshore

Introduction
Dull grading is the determination of the degree of material wear as it relates to the tooth/insert cutting structure bearing wear and gauge wear. Dull grading was established in 1961 by AAODC as the first dull grading standard and was adopted by the oil industries in 1963. It was later improved by the International Association of Drilling Contractors (IADC) who introduced a three-part system of dull grading in the early 1970s. The cutting structure and bearings were graded on basis of wear in eighths [1/8] (Baker, 1995). This reflected the amount of wear experienced or the proportion of total 'life' used. A cutting structure that was one quarter [1/4] worn would be graded 2. A bearing with half its life remaining would be graded 4. The final element was a measure of the amount the bit was under gauge. A grade would look like this:**T3B61/8"**. This TBG system (tooth bearing gauge) is still sometime used, but it lacks flexibility and contains scant information. In 1987 the IADC introduced a new eight point system of dull grading, which was updated in 1992 (Reedhycalog, 2001). When a section of hole has been drilled and the bit is pulled from the wellbore, the nature and degree of damage to the bit must be carefully recorded. A system, known as the **Dull Bit Grading System**, has been devised by the IADC to facilitate this grading process. This is the standard system for rig site dull grading of drill bits. The objectives of this paper are to improve bit type selection, identify the effects of WOB, RPM, and evaluate bit performance.

The IADC Dull Grading System
The IADC Roller Bit Dull Grading System and First Revision of The IADC Fixed Cutter Dull Grading System are used for dull grading. Although there are some small differences between the fixed cutter and the roller cone systems, the eight categories are used for both fixed cutter and roller cone as presented in Table1.
The first four points relates to cutting structure only. Of these the first two points are used to grade cutting structure wear: firstly the wear of the inner teeth (or inserts or cutters, or cutting elements), then the outer teeth (or inserts or cutters, or cutting elements).
The major dull is indicated by a two-letter code, followed by its location. For roller cone bits, the condition of the seals or bearings is noted. Whether the bit is in gauge, or if not, by how much it is under gauge is shown in the next category. Then any other dull characteristics are described, using the two letter code system, finally, the reason the bit is pulled is declared

Table 1: Eight Column Systems used for Dull Grading Bits (After Reedhycalog, 2001)

1	2	3	4	5	5	7	8
Inner Teeth	Outer Teeth	Major Dull Character	Location	Seals/ Bearings	Gauge 1/16ths	Other Dull	Reason Pulled
I	**O**	**D**	**L**	**B**	**G**	**O**	**R**

Materials and Methods

The IADC system has been widely incorporated into computer data bases through the use of prescribed fields and alpha-numeric characters. These are used to describe the cutting structure conditions, and "other" dull conditions as well as the reason the bit was pulled or the run terminated. This system provides valuable information to drilling personnel. The information generated assist rock bit manufacturers in designing new products for drilling the Niger Delta formation.

Understanding how to analyze a dull bit will enable the drilling engineer to improve bit selection and operating parameter for future drilling in the Niger delta.

Methods Used in the Analysis of Drill Bit Dull Grading

There are two methods used which are qualitative grading and quantitative grading

Qualitative Grading

Qualitative grading can be made by visual inspection. Visual inspection is the most widely used method for the analysis of dull grading as shown in Table 2.

Table 2: Bit dull Grading Descriptive Notations.

Condition	Abbreviation
Bent Legs	BL
Damage Bit	DB
Eroded Nozzle	EN
Lost Nozzle	LN
Plugged Nozzle	PN
Shirttail Damaged	SD

Quantitative Grading

Quantitative measurement of bit dull grading is by the use are gauge ring and quarter wear Gauge

Gauge Ring

The gauge ring is used to measure the outside diameter of the gauge hole of the bit each time the bit is pulled out of the hole. Maintaining a full gauge hole is very important in drilling operation. The selected gauge ring should be placed flush with the back of two cones. The distance measured in inches between the gauge and the back of the third cone if a three-cone bit is used. The wear is reported in 1/32inches. Actual under-gauge will be two-thirds of measured under gauge.

Quarter Wear Gauge

The quarter wear gauge is used to measure the quarter that drill round. This is only applicable to PDC bits and roller cone components are presented in Figure 1.

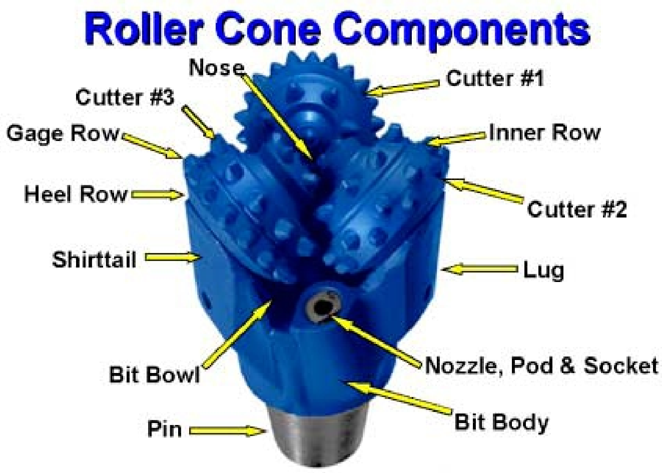

Figure 1: Roller Cone Component

Dull Grading a Bit (Qualitative)

Examine the cutting structure looking down on the bit by Starting with cone 1, the teeth and cone condition. Look for worn, broken, lost or chipped teeth, junk damage, cone run-together, and rotate the cone to determine seal and bearing condition and moving clockwise repeat this process for cones with two and three and typical dull bit is shown in Figure 2

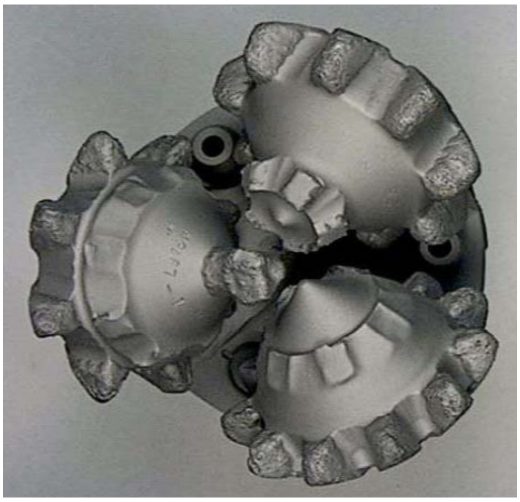

Figure 2 : Dull Bit (Qualitative)

Dull Grade a Bit Using Quantitative-Gauge Ring

Gauge ring methods in the analysis of dull bit using the "TWO THIRDS RULE" are as follows:

Place the ring gauge touching two gauges, measure the gap at the closest point on third cone. This distance multiplied by 2/3rds is the amount out of gauge (given in 1/16ths) as shown Figure 3

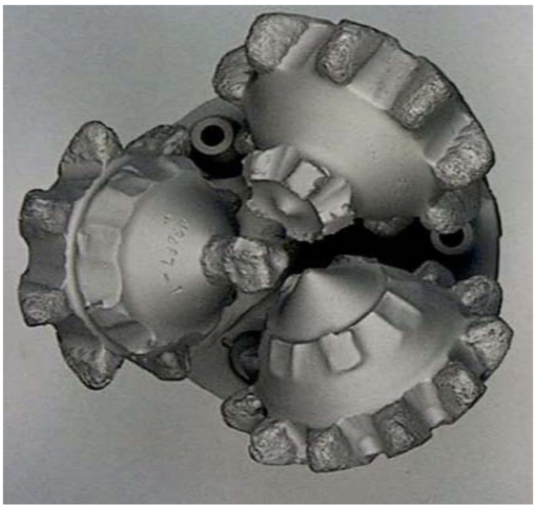

Figure 3: Dull Bit (Quantitative)

Determination of a Dull Bit

A dull bit could be was determined as follows: Ball plug weld and pressure plug for leakage, Pin for cross-threading, shoulder for improper make-up torque, cone gauges and arm condition. Look for or lost inserts, erosion and ST damage. Finally, identify reservoir cap and nozzle area for damage or erosion.

The above examinations can now be used to analyze the condition of the bit, if it has worn out or not and if it has the type of wear associated with it.

Sources of Errors in Dull Grading a Bit

Consideration is given to three categories of error that may affect dull grades; inaccurate grading, improper grading and inaccurate focused grading.

a. Inaccurate Grading

More importantly, through inexperience, grading can be in error. This can result from common mistakes which include; Incorrect distinction between inner and outer cutting elements, taking the worst cutting structure wear rather than the average, grading a bit as under-gauge because some of the gauge elements are worn and grading a seal as failed because one assembly is lower than the other two.

b. Improper Grading

Some grading made at the rig site error may be due to deliberate action, reflecting a poorer bit condition than actually applies. Curiously, such grading often coincides with cases where a bit has been pulled for 'failure', but on reaching the surface is found to be in excellent condition.

c. Inaccurate Focus

Occasionally the focus of the estimates may be faulty. On the basis of previous offset data, including dull grades that were available to aid selection, the bit manufacturer recommended a bit known to offer superior life in such application to the competititor bits that had been used.

The focus of the early dull gradings, in failing to identify the major dull characteristics, delayed the introduction of the optimum bit type.

Results and Discussion

A bit record (Tables 1, 2 and 3) will always be kept by the operating company, drilling contractor and/or bit vendor. This bit record is used to store the following information about the bit after it has completed its run: the bit size, type and classification, the operating parameters, the condition of the bit when pulled (IADC bit grading) and the performance of the bit.

The result of bit records obtained from offshore drilling (OML 111) from different wells, the IADC drill bit dull grading for the Niger Delta formation is simply analyzed. Here, data are obtained from 3 different wells during drilling Well 13, Well 14 and, and Well 15.

The bit record for well 13, 14 and 15 are given below containing the IADC Dull Grading. Table 2 shows the bit record for well 13, Table 3 shows the bit record for well 14 and Table 4 shows the bit record for well 15. The analysis of drill bit dull grading were carried out and inserted into its cell so as to further analyze the drill bit along with the formation. In addition, from the bit records for the 3 wells, one can easily determine the degree of material wear as it relates to the tooth/insert cutting structure, bearing wear and gauge wear by the use of the Table 1. This table gives straight forward analysis of the bit as it relates to wear, therefore provide information as regards wear of bit and the Niger Delta formation

Conclusion

This paper examined analysis of drill bit dull grading in the Niger delta. It can then be concluded that:

1. Wear characteristics are informative.
2. Dull bit analysis is a powerful diagnostic tool.
3. Wear can be related to operating practices.
4. Wear can be related to bit design features.
5. Accelerated wear can often be corrected.
6. Analysis of dulls can yield better bit choices for Niger Delta formation.
7. Analysis of dulls can improve drilling operation for Niger Delta formation.

References

Reedhycalog (2001). Drill Bit Design and Selection, Houston, Texas, USA. pp 57 – 66

Baker Hughes (1995): Drilling Engineering Workbook; a Distributed learning course. Houston, Texas, USA. pp 117 – 144

SITP (2001), Well Engineering Drill Bits, Shell Petroleum publication, pp8.1 – 8.37

ExxonMobil (1998): Drilling Engineering, Analysis of Dull Bits, Exxon Mobill Publication, Unit XI, pp 1 -14.

Table 2 Bit Record for Well 13

BIT RECORD FOR WELL 13																	
SIZE	MAKE	MODEL	TYPE	TFA	SERIAL NO.	DRILLED FOOT	HOURS RUN	ROP	IADC DULL GRADING								IADC CODE
									I	O	Dull	Loc	B-S	Gage	Other dull	Reason	
8.50	HGH	GTX	MTB	1.5217	C59-DL	1,912	27	70.8	5	5	CT	A	E	3/16	NO	TD	145
8.50	HGH	MX-C3	Insert	2.49	R78DP	4,172	27	154.5	2	2	WT	A		IN	NO	TD	
8.50	HYC	DS56	PDC	1.2149	23446	11,546	76.5	150.9	2	3	CT	N	X	1/16	LT	TD	
8.50	HYC	DS56	PDC	1.45	200418	11,395	82	139.0	3	3	CT	A	X	IN	NO	TD	
8.50	HYC	RS-162	PDC	1.18	201986	26,067	203.5	128.1	2	3	CT	A	X	1/16	WT	TD	
8.50	HYC	DS71	PDC	2.0686	H48887	7,628	109	70.0	3	3	CT	NT	X	1/16	WT	BHA	M422
8.50	SMT	M36	PDC	2.3869	1382	4,863	20	243.2	3	3	6	NT	X	1/16	PN	TD	M433
8.50	SMT	M36	PDC	2.3869	551389	4,554	23.5	193.8	1	3	CT	T	X	I	PN	TD	M433
8.50	SMT	M36	PDC	2.3869	JR 5513	7,512	31.5	238.5	1	1	CT	S	X	I	WT	TD	M433
8.50	SMT	M-36	PDC	1.48	JR5762	14,534	101	143.9	2	2	WT	N	X	1/16	CT	TD	M433
8.50	SMT	M16	PDC	2.2273	JS1191	4,112	60	68.5	2	2	WT	A	X	I	NO	TD	M442
8.50	SMT	M36	PDC	2.2703	JS1389	20,830	130.5	159.6	2	2		N	X	1/16	CT	TD	M433
8.50	SMT	M36	PDC	2.1476	JS4991	2,209	48.25	45.8	3	4	wt	c	a	XI	ct	td	M441
8.50	SMT	M36	PDC	2.2488	JS5806	5,183	39	132.9	5	5	BT	A	X	1/16	CT&JD	TD	M442
8.50	SMT WS	M36	PDC	2.66	JS6742	8,351	107.5	77.7	8	8		A	X	2-Jan	RO	PR	
8.50	SMT	M36	PDC	1.89	JS7559	9,417	63.5	148.3									
8.50	SMT	M36	PDC	1.96	JS7887	6,262	41.5	150.9									
8.50	SMT	M36	PDC	2.25	SC 0331	4,464	33.5	133.3	3	3	BT	S	X	1/16	PN	TD	
8.50	SMT	M36	PDC	2.0525	SC0284	1,500	17.5	85.7	1	1	WT	N	X	I	ET	TD	M442
8.50	SMT	M36	PDC	2.2488	SC0328	25,266	136	185.8	2	2	CT	S	X	1/16	ER	TD	M422
8.50	SMT	M36	PDC	2.2665	SC0329	11,528	57	202.2	3	3	BT	A	X	1/16	PN	TD	M446
8.50	SMT	M36	PDC	2.6630	SC0330	3,169	24	132.0	1	1	CT	T	X	I	NO	TD	M433
8.50	SMT	M36	PDC	2.2488	SC0331	6,986	57	122.6	2	2	BT	S	X	1/16	PN	TD	M422
8.50	VRL	MKF58	PDC	2.0126	cp5800c	4,734	26.5	178.6	5	5	BT	A	X	1/4	ER	ROP	M442

Table 3 Bit Record for Well 14

BIT RECORD FOR WELL 14																	
SIZE	MAKE	MODEL	TYPE	TFA	SERIAL NO.	DRILLED FOOT	HOURS RUN	ROP	IADC DULL GRADING								IADC CODE
									I	O	Dull	Loc	B-S	Gage	Oth	Reason	
12.25	SMT	M40	PDC	2.5986	JS4967	6,064	61.5	98.6	1	1	PN	A	X	I	NO	TD	
12.25	HGH	MX C-1	MTB	2.41	LOODK	1,516	15.5	97.8	4	4	wt	A	E	1/8	NO	TD	
12.25	HGH	MX-C1	MTB	2.4053	M33DF	3,916	27	145.0	2	2	CT	M	E	1/8	WT	TD	1/1/2007
12.25	HGH	R/B		2.41	M83DX	1,479	16	92.4									
12.25	HYC	DS71	PDC	2.2764	100816	4,316	48	89.9	2	2	WT	A	X	IN	PN	TD	M442
12.25	HYC	DS71	PDC	2.5410	10816	1,930	28	68.9	1	1	WT	A	X	1/8	PN	TD	M442
12.25	HYC	DS-113	PDC	2.4850	23080	4,050	45	90.0	1	2	WT	G	X	2	CT	TD	M333
12.25	HYC	DS113	PDC	1.9880	23081	5,678	81.5	69.7	2	2	WT	A	X	I	NO	TD	
12.25	HYC	RS130	RSB	1.2088	24294	7,131	61.5	116.0	2	2	WT	A	X	1/8	CT	BHA	
12.25	HYC LIH	RS190	RSB	1.1904	201009	5,844	46.5	125.7									
12.25	HYC	RS163	PDC	2.0985	201347	1,715	25	68.6	1	2	WT	G	X	I	NO	TD	
12.25	HYC	RS130	PDC	1.50	202352	26,223	253	103.6									
12.25	HYC	RS130	PDC	1.21	202763	7,220	63	114.6	1	1		A	X	1/16	NO	TD	
12.25	HGH	MX-C1	MTB	2.41	S92JW	1,428	25.5	56.0	2	2	WT	A	E	I	NO	TD	M442
12.25	SMT	M40	PDC	2.3408	JR7965	3,768	25	150.7	2	2	CT	G	X	I	PN	TD	M433
12.25	SMT	M40	PDC	2.60	JS 4967	3,080	20.5	150.2	1	1	PN	A	X	I	NO	TD	
12.25	SMT	M40	PDC	2.3408	JS3391	3,197	25	127.9	3	6	WT	A	X	3/16	CT	TD	
12.25	SMT	M40	PDC	2.9698	JS3945	6,347	67.5	94.0	1	1	WT	A	X	I	CTX3	BHA	M442
12.25	SMT	M40	PDC	2.5986	JS3946	8,291	113	73.4	2	2	WT	A	X	1/16	CT	TD	M442
12.25	SMT	M40	PDC	2.5986	JS5247	1,606	27	59.5	3	3	CT	N/S	X	1/16	JD	MOTOR FAILURE	M442
12.25	SMT	M40	PDC	2.5986	JS6472	2,484	27.5	90.3	2	2	CT	A	X	1/16	PN	DMF	
12.25	SMT	FDSS	MTB	1.8040	LE0088	1,396	21	66.5	3	3	ER	N/S	4	1/8	CT	TD	135

Table 4 Bit Record for Well 15

BIT RECORD FOR WELL 15																	
SIZE	MAKE	MODEL	TYPE	TFA	SERIAL NO.	DRILLED FOOT	HOURS RUN	**ROP**	IADC DULL GRADING								IADC CODE
									I	O	Dull	Loc	B-S	Gage	Oth	Reason	
17.50	HGH	MX-1	MTB	2.41	578-DA	1,108	11	100.7									
17.50	HGH	MAXGT1	MTB	2.4053	A75DG	2,884	17.5	164.8									1/1/2005
17.50	HGH	MX1	MTB	2.4053	M16DL	3,613	23	157.1	1	1	SS	A	E	1	BU	HT TD	127
17.50	HGH	MX1	MTB	2.4053	p57da	4,628	30	154.3	2	3	WT	A	2	1/8	ER	TD	115
17.50	HGH	MX-1	MTB	2.4053	X741A3	2,914	23.5	124.0	1	2	WT	A	E	1/16	NO	TD	1-1-7
17.50	HGH	MaxGT1	MTB	2.4053	X-74-GT	2,518	25	100.7	6	7	WT	A	E	1/4	BU	PR/BHA	135
17.50	HGH	MAXGT1	MTB	2.4421	X74GT157	2,955	15	197.0	1	1	NO	A	2	I	NO	TD	115
17.50	HYC	**T11C**	MTB	2.18	Y54723	8,691	72.5	119.9	3	3	WT	A	E	1/8	BT	TD	115
17.50	HYC	**T11C**	MTB	2.18	Y54724	7,201	76	94.8	1	4	CT	C3	E	I	NO	TD	115
17.50	Reed	EMS13	MTB	2.41	A09222	2,802	17	164.8									
17.50	Reed	EMS13	MTB	2.2457	LU8088	2,876	42	68.5	3	3	ER	A	E	3/16	WC	TD	135
17.50	SMT	MSDGHC	MTB	2.4053	LF 4943	3,482	33	105.5	1	1	WT	A	E	I	N	TD	136
17.50	SMT	MSDGHC	MTB	2.2434	LF4943	3,817	35.5	107.5	1	1	WT	A	E	1	N	TD	136
17.50	VRL	ETR1	MTB	2.11	164401	2,941	35	84.0	2	2	CT	A	E	1/16	WT	TD	115m

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Processing and Recovery of Salt from Keana Salt Sand Deposit, Nasarawa State

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Abstract
The processing and recovery of salt as-received filtrate brine solution of Keana salt-sand deposit, Keana raw pond water, Laboratory filtrate raw pond water brine solution and Lab.Tap water filtrate brine solution of Kaena salt-sand deposit were investigated. The samples were obtained from Keana Local Government Area of Nasarawa State while the Tap water was obtained from Mineral Resources Engineering, Kadpoly Assay Lab. The water samples were analyzed using atomic absorption spectrometer (AAS) technique, the results of the analyses revealed that the as-received filtrate brine solution of the Keana salt-sand deposit contains on the average 91.74%Ca^{2+}, 0.0024% Fe^{2+}, 7.77% Mg^{2+}, 0.0192% of F^-, 0.359%Cl^- and 0.0680% NO_3- while the as-received Keana raw pond water contains 64.34% Ca^{2+}, 0.713%Fe^{2+}, 1.314% Mg^{2+}, 0.099% F^-, 31.58% Cl^- and 1.878% of NO_3^-. 10kg of the salt sand sample was sieved and then dissolved in 1 liter of the Lab. Tap and as-received Keana raw pond water respectively. The filtrate brine solutions were evaporated to dryness using electric hot plate at a constant temperature of 100°C and the evaporation process timed. From the result it is observed that the percentage of the salt recovered increase as the volume of filtrates brine solutions is increased with increase in the time required for the evaporation to take place. The quantity of the salt samples that are recovered from the as-received filtrate brine solution are higher than the quantity of the salt samples that are recovered from the as-received raw pond water, Lab. Tap water filtrate brine solution but lower than the quantity of salt recovered from the laboratory filtrate raw pond water brine solution for 10 to 510ml of the brine solution samples respectively. The time required for recovering salt from 10 to 510ml of the different samples of the filtrate brine solutions varies, with the Lab. filtrate raw pond water brine solution having the least total time of 758minutes compared to others while the total amount of salt sample recovered is 153.09g more than the as-received raw pond water brine solution (110.04g, 770 minutes), Lab. Tap water filtrate brine solution (22.0g, 819minutes) but less than the as-received filtrate brine solution (166.8g, 845 minutes). This reason could be attributed to the low pH value of the brine solution of pH 3.93 which makes the brine solution more acidic and thermodynamically unstable compared to the Lab. Tap water filtrate brine solution (pH 4.10), as-received filtrate brine solution (pH9.00) and that of the as-received raw pond water (pH8.00), thus enhancing fast evaporation of the water and quick seeding of the salt grains from the water sample than others.

Significance of the research: to determine the time taken to recover salt from the as-received filtrate brine solution of Keana salt-sand deposit, as-received Keana raw pond water, Laboratory filtrate raw pond water brine solution and Lab. Tap water filtrate brine solution of Kaena salt-sand deposit. Also to confirm if the superstition-al belief of the locals that only the water from the pond can be used as solvent alone for the recovery of the salt from Keana salt sand deposit.

Key words: Processing, Recovery, salt, Keana salt-sand deposit

Introduction

Nigeria's potential in solid minerals is enormous; there is hardly any state in the country where solid minerals do not occur in commercial quantities. Although, Nigeria can boast of up to 33 different solid minerals in its 36 states, some of these have been fully explored and their quantum ascertained while further investigation is required to determine the quantum of the occurrence of others not yet listed. Those known to exist in commercial quantities include feldspar and quartz and these industrial minerals cut across the entire states (Famuboni, 1990).

The relevance of these industrial minerals to the technological development of the nation can not be over emphasized. The high cost burden of importing industrial raw materials, example salt create effect on the foreign exchange and thereby affecting the economic of the country, on this basis for sourcing and upgrading the Keana through the kinetic recovery of the salt. Therefore the availability of other Nigerian industry that utilize salt as their raw material will depend on the continuous processing and supply of salt which is the Kaena salt sand deposit. Also because the materials are bulky, there is need to source them locally not only to reduce dependence on the imported source, but because scarce foreign exchange will also be saved. (Famuboni, 1990).

The growth of the modern chemical process industries have been characterized by an increase in the range of products offered for sale with a consequent increase in the amounts and variety of material processed and a demand for an increase in the purity of the products. This reflects the growth in sophistication of the chemical processes themselves, both in terms of the chemical reactions used to create the products and also in the separation procedures used to provide raw materials and to purify the final products. Some of the wide range of separation techniques used in modern chemical industry with examples of their application is listed in Table 1.

Separation processes are fundamental part of almost every chemical process. Indeed, there are usually far more separation stages in the overall process than there are chemical reactions. Separation starts with the extraction of raw materials and continues to the purification and isolation of the final product without an efficient separation technology, raw materials would be in short supply, more expensive and of lower quality. Efficient separation processes which allow useful intermediates and by products to be recovered and make it possible for un-reacted starting materials to be re–cycled needs to be developed. Without efficient purification, final products might well be un-sale-able since they would be hopelessly contaminated with materials from the reaction mixture. It is the number and complexity of the separation steps which are necessary to give an acceptable product which accounts for the major part of the manufacturing costs (Corning, 1989).

Solid – liquid extraction is amongst the most commonly employed methods of separation and it's particularly common in the isolation of starting materials. It is also a technique which can reasonably be used on a large scale operation. Although the detailed design of equipment and its mode of operation will differ from industry to industry or even between competitive variants of the same process, the principles underlying solid-liquid extraction remain the same, whatever the overall process. Consequently, a detailed study of such a basic step or unit operation can shed light on the operation of whole range of processes; the concept of the operation of whole range of the process and the concept of the unit operation can be seen as a way of unifying a whole range of processes, thereby simplifying both the design and operation of chemical process plant (Corning, 1989).

The choice of Separation Technique

The choice of separation technique is wide, separative method to be employed is conditioned by both technical and commercial factors, some of which are briefly outlined below:

The first stage is selecting the most suitable method is to decide the type of separation that is capable of giving the required result under reasonable and realistic operation conditions i.e. whether each method is feasible. This stage reduced the choice and many proven methods which can be satisfactorily applied can be directly excluded.

The next stage is to decide between the remaining possibilities on the grounds of the technical efficiency, but in general terms it is necessary to evaluate whether the general method (or more particularly the specific operating mode) under consideration can ensure product of the appropriate quality of the specified rate. That is to say whether it has the required degree of selectivity and can offer a reasonable recovery and operate at a suitable through put. In most cases selectivity is not a fixed parameter but as will be shown later, depends intimately on the mode of operation. Both selectivity and recovery may be increased by combining several stages within the overall separation operation, either repeatedly using the same method or by combining methods. Such processes tend to have reduced through put compared with single stage operations and are generally more complex and costly, so that the minimum number of stages should be employed consistent with the required grade of product. Here economic decisions are often as important if not more, than to technological ones (Corning, 1989, Aye et-al, 2005).

Table 1.0: Showing Some Applications of Solid-Liquid Extraction

S/No	Solid feed	Solvent	Product
1	Sugar-cane	Water	Cane sugar (sucrose)
2.	Palm kernels (oil-seed)	Hexane (naptha fraction)	Palm kernel oil (oil)
3.	Roasted copper (sulphide) ore	Water	Copper sulphate
4.	Gold ore	Dilute potassium cyanide solution	Gold
5.	Rock phosphate	Sulphuric acid	Phosphoric acid
6.	Fish (halibut or cod) livers	Ether	Fish liver oil
7.	Sand (sea-shore)	Water	Salt-free sand
8.	Titanium dioxide $TiCl_2 + 4NaOH = TiO_4 + 4NaCl + 2H_2O$	Water	Purified (pigment grade) titanium dioxide)

Source: (Corning, 1989)

Types of Salt

The three well-known types of salt in the world are:

I. Unrefined salt (sea salt): Different natural salt have different minerals, giving it a unique flavor. Natural sea salt harvested by hand has a unique flavor varying from region to region. Some advocates for sea-salt assert that unrefined sea salt is healthier than refined salt because it contains the entire four (4) cationic electrolytes in the body i.e. sodium, potassium, magnesium and calcium and other vital minerals needed for optimal body function, (http://www.saltmatters.org 2009).

Sea-salt which is unrefined salt is bitter because of its magnesium and Calcium compound, it is rarely eaten. The refined salt industry and cities scientific studies say that raw sea and rock salt do not contain enough iodine salt to prevent iodine deficiency diseases. Unrefined sea salt are commonly used as ingredients in bathing additives and cosmetic products, example is bath salt which uses sea salt as its main ingredients combined with other ingredients used for its healing and therapeutic effects. (http://www.salt.gov.uk/tvads.html, 2009).

ii. **Refined salt (table salt):** This type of salt is mainly sodium chloride and is widely used today. Refined salt 99% sodium chloride contains substances that make it free-flowing (anti-caking agent) such as sodium silico-aluminate or magnesium carbonate. It is commonly practice to put a desiccant such as few grains of uncooked rice in the salt shaker to absorb extra moisture and help to break chumps when anti-caking agents are not enough. (http://www.salt.sense.co.uk.htm, 2009).

In some European countries where drinking water fluoridation is practiced, fluorinated table salt is available. In France 35% of sold table salt contains either sodium fluoride or potassium fluoride, another additive that is widely important for pregnant women which is folic acid (vitamin B9). It gives the table salt a yellow colour. (http://www.saltsense.co.uk.htm, 2009).

iii. **Iodized salt:** this is a table salt mixed with a minute amount of potassium iodide, sodium, magnesium and calcium. Iodized salt in United State contains 46 – 77ppm while in United Kingdom, the iodine content of iodized salt is recommended to be 10 – 22ppm which makes it to be very common in United State, Australia, Newzeland, Nigeria than in United Kingdom. The amount of iodine and specific volume of iodine compound added to salt varies from country to country. (http://www.salt.org.u/arch.html, 2009).

Theoretical consideration

Salt is produced by the evaporation of sea water or brine from other source such as brine well as salt lakes under vacuum. This involves the use of heat energy with implications for CO_2 emission, the vacuum process maximizes energy efficiency which is closely monitored for both commercial and environmental reasons. The steam used for the evaporation process is generated in accordance IPPC regulations and wherever possible is reused within the manufacturing process http://www.eu-salt.com/index3.htm, 2009). Salt is found in association with other soluble minerals such as gypsum, sylvite and anhydrite. Salt also occurs in liquid form known as brine. Examples of the world renowned brines are the Great Salt Lake in Utah (U.S.A), the Dead Sea, and the Caspian Sea (U.S.S.R). Sometimes salt occurs as salt domes which are vertical pipe like bodies of salt which have apparently punched their way upward from an underlying bed of salt. These salt domes are searched for because of the petroleum frequently associated with them. These occurrences exist in Texas and Louisiana in the United States of America. The best known commercially viable natural brines have a salt content of 22 to 26% of which NaCl (Sodium chloride) represent 95 to 98%. Under favourable conditions those with 10% total salts with NaCl (Sodium Chloride) content is not less than 90% may be economically exploited also. Normally, where salt content is less than 16%, the brines are brought to saturation point by the cheap solar and wind evaporation by adding NaCl (sodium chloride) to the solution. The temperature of evaporation varies with the type of salt required (Carter, et al 1963).

Rock salt mining does not require energy input, this is considerably lower than for evaporated salt. All manufacturers monitor and seek to maximize their energy efficiency, users are encouraged to weigh the overall energy impact including the lower distribution energy usage of indigenous supplies. Unrefined sea salt and rock salt made by evaporating sea water is often sold for use as condiments because it contains trace elements of other minerals which are removed during the refining process; it may have health advantage over normal table salt. Rock and sea salt are usually referred and sold at NastrumMutriaticum in homeopathy and reported by followers to be a deep acting and powerful curative when taken over long period of time. (http://www.saltsense.co.uk/aloutsalt-envol.htm, 2009)

Uses of Salt

From the days of the cave men, humans have discovered ingenious means to use salt to enhance the quality of our lives. So valuable is this common mineral that wars have been waged and revolution fraught for access to salt. Its largest use is largely invisible to the public; salt is a mineral that almost everyone in the world consumes. Although world wide food uses account for 17.5% of salt production, the majority is sold for industrial use. (http://ww.saltsense.co.uk/aboutsalt-facts01.htm, 2009).

Salt keeps our industries alive today, the properties of chlorine and sodium, and the principal compounds from which make it one of the most important basic raw materials for industries, the others are coal, limestone and sulphur (Robert, 1960).

Salts are basic for the production of seventy five (75) important industrial chemicals, and brines are necessary for twenty four (24). In industry, salt is used for the manufactures of soap and dyes, the processing of textiles and leather, dust and ice control, for water treatment and also in metallurgy. Salt is used to remove traces of water from aviation fuel after it is purified, also used in the manufacture of chemical like chlorine (Cl), chlorates (ClO_3), hydrochloric acid (HCl) and also related compounds. Salt is used in the production of glass, caustic soda for paper industry, it is also used in combination with lime to produce soda ash (sodium carbonate Na_2CO_3) http://www.saltsense.co.uk/aboutsalt-facts.of.htm, 2009).

As the knowledge and ambition developed, salt came to be used due to its transparency to infrared; salt crystals are used for making prisms and lenses of instruments used in the study of infrared radiation, chlorine which is used for the manufacture, is used for the manufacture of polyvinyl chloride (PVC) which contains a vast number of products including blood bags and tiny catheters used to keep premature babies alive. Salt is employed in refrigeration, meat packaging, and fish curing and in the processing of dairy products in food industry. Salt provides a vital diet supplements and as a means of food preservation, it brings to food more than one of the four (4) taste sensation i.e. sweet, salty, sour and bitter. It enhances other tastes; it is also used to regenerate our water softeners to protect the pipes and appliances in our homes (http://www.saltsense.co.uk, 2009).

In 2002, the total world production of salt was estimated at 210 million tones, the top five (5) producers being the United State 40.3 million tones, China 32.9 million tones, Germany 17.7 million tones, India 14.5 million tones and Canada 12.3 million tones. The figures are for sodium chloride in general and not just for table salt.

Properties of Salt

Salt with its IUPAC name sodium chloride (NaCl) and other names which is common salt, halite and table salt is 60.633% elemental chloride (Cl) and 29.337% sodium (Na) with the atomic weight of 35.4527 for chlorine (Cl) and 22.989768 for sodium (Na) has the following properties:

i. It is a white, colourless crystal or powder with a melting point of 801^0c.
ii. It is odourless with a molar mass of 58.44277g/mol and a density of $2.16g/cm^3$.
iii. It has a face center cubic crystal structure
iv. It is soluble in water at 35.9g/100ml (25^0c)
v. It has a molecular weight of 58.4428 with a specific gravity of 2.1 to 2.6 (sodium chloride).
vi. It has a eutectic composition of 23.31% with a freezing point of eutectic mixture of 21.12^0c (-6.016^0F)
vii. Salt (sodium chloride) is neutral to PH of aqueous solution.

Salt has a related compounds i.e. sodium fluoride (NaF), sodium bromide (NaBr), and sodium iodide (NaI) as other related anions while lithium chloride (LiCl), potassium chloride (KCl), rubidium chloride (RbCl), Cesium Chloride (CsCl) Magnesium chloride ($MgCl_2$) and calcium chlorides ($CaCl_2$) as other related cations. Sodium acetate is the related salt. http://ww.saltsense.co.uk/aboutsalt-fact01.htm, 2009).

Table 2.0: showing the solubility of salt at various temperatures (salt per 100 pounds of brines).

Temperature ^{0}F	**Temperature ^{0}C**	**Percentage of salt (%)**
-6	-21.11	23.31 eutectic point
0	-17.78	23.83
10	-12.22	24.7
20	-6.67	25.53
30	-1.1	26.16
32	0	26.29
32.2	0.1	26.31 transitional point
40	4.44	26.33
50	10	26.36
60	15.56	26.395
70	21.11	26.45
80	26.67	26.52
100	37.78	26.68
125	51.67	26.92
150	65.56	27.21
175	79.44	27.62
200	93.33	27.91
212	100	28.12
220	104.44	28.29
227.5	108.7	28.46 (boiling point at 1 atmospheric pressure)

Source: http://www.saltsense.co.uk/aboutsalt-fact01.htm, (2009)

Geology and the location of Keana Salt Deposit

Salt is the most familiar of all minerals and has played an important role from the beginning of man. It is used for a great variety of purposes in the chemical, metallurgical and ceramic industries, and in agriculture, medicine and the household. Salt is so widespread in its uses that the materials made from it or requiring its use in their manufacture are continually met in everyday life. Some of its uses are the manufacture of industrial chemicals like soda, sodium bicarbonate, caustic soda, chlorine and certain acids; the smelting and refining are ores and metals, the making of soap and dyes, the tanning of leather, the preservation of foods, the making of explosives, and the bleaching of cotton and paper. The few indigenous sources of salt are by no means sufficient for the country's need, most of which have to be met by imports. The folded cretaceous rocks of the Benue valley gives rise to many dilute brine springs that are the centers of village salt industries. Such springs are known to occur in many parts of the country, but mainly in Nasarawa, Imo, Cross river, Benue, Plateau, Sokoto, Adamawa and Bauchi states (Minerals and industry in Nigeria, 1987).

The Keana brines field is notable for its mineral endowment such as the occurrences of salt and brines, lead zinc (Pb-Zn sulphides), limestone and barite ($BaSO_4$) mineralization among others (Minerals and industry in Nigeria, 1987).

In fact the Keana and Awe areas became popular for their local salt production and trading from time immemorial. Salt has therefore played a great role as a domestic and industrial requirement in the area. Brine which is mainly composed of mainly salts of Na^+ and Cl^-. Traces of metal radicals like K, Ca, Mg, Ba, Sr, Cu and Mn have been observed to be associated with the brines. Chloride concentration of between 38 and 135ppm has been reported from the study area (Offodile, 1976).

The origin of the brine is still debatable while pioneer workers (Tattan 1974, Cartchley and Jones 1965) are of the view that the brines were due to salt impregnations in the pore spaces in the sands. Phoenix (1966) linked them to evaporitic origin with a possibility of the existence of a salt dome beneath the cretaceous sediments. Recent studies, (Offodile 1976b) suggested a magmatic origin for the brines.

Occurrence of Salt in Nigeria

No deposits of rock salt have been founded in Nigeria yet. The only known sources of salt in the country at present are natural brines and sea water. The average sea water contains salt in the following of percentages by weight.

NaCl	2.72%
$MgCl_2$	0.30%
$MgSO_4$	0.25%
$CaSO_4$	0.13%
KCl	0.08%
$MgBr_2$	0.01%
$CaCO_3$	0.01%
Total	3.50%

This would give the composition of total salts as follows

NaCl 77.7%, MgCl2 9.43%, MgSO4 6.29%-CaSO4 3.71%-

KCl 2.29%, MgBr2 0.29%, CaCO3 0.29%, Total 100%

Nigerian Local Natural Brines

A number of saline springs and ponds occur within a relatively narrow belt which extends in a N. N.E – S.S.W. direction from Gombe to Afikpo, along the generally low- lying and gently undulating plains of the Benue and Cross river drainage systems. The brines issue from clays, shales, sandstones and conglomerates of the upper cretaceous age, and are almost invariably situated in valleys often flooded during the rainy season and sometimes for one or two months after. Gravity surveys indicate the belt of occurrence of brines, anomalous features elongated in a N.E – S.W direction – Benue trough, and corresponding to a series of folds and fractures related to several tertiary intrusions. These intrusions are in turn responsible for galena, sphalerite, baryte, fluorite and subordinate pyrite and silver mineralization. The brine springs are frequently found near the axis of anticlines in fractured and mineralized areas. The Nigerian brines are located in four (4) main areas namely: Gombe, Shendam, Abakeliki and Ikom – Calabar, where they have given rise to small salt industries.

1. **Gombe area:** This area has over fifteen localities mostly south of Gombe, all of which are salt production centers. The brine springs of Ayabe, Bumanda, Gyakam. Jebero, Jebieb, Jende, Langa, MutumDaya and Todi emerge from the Bima sandstone, while the Gombe sandstone is probably the source of brines at Gubja, Pindiga, Takulma, Tumu and Zanga. The concentration of the brines is not known but is reported to be rather weak. At Gyakam the spring is hot and sulphurous. The chemical analysis of the final product of evaporation of the Bumanda brine gives the following result (Beltare, 1971): NaCl 93.3%, SiO_2 2.6%, CaO 1.3%, SO_4^{2-} 0.4%, MgO trace, Fe trace.
2. **Shendam area:** Over 11 locations are known, south of Shendam Arufu and Akwana location issue from shales of the Eze-Aku shale group. All the other brine springs emerge from marine sediments of the Lafia – Wukari faces. Awe and Akwana have been salt – producing centers for many generations as observed in their more advanced local manufacturing techniques and organization. Analysis of source of the springs and bore holes in this area shows average values of about 1% total dissolved solids with over 90% NaCl and trace amounts of magnesium, Borou, Iodine, and bicarbonates. (Beltaro, 1971).
3. **Abakiliki area:** This area comprises three (3) zones of salt occurrence, namely, Ogoja, Ameri, and Afikpo zones which are underlain by sediments of the Asu River group and of the Eze – Aku shale group.

i. Ogoja zone: There are over ten (10) known salt producing centers in this zone, the most important districts is the Yacha – Gabu and western Yala. Chemical analysis of the final product after evaporation shows 74–76% NaCl, the remaining percentage being composed principally of calcium, magnesium, and potassium salts.

ii. Ameri zone: In this zone, over 11 localities, including Abakiliki and Ameri towns have brine deposits which were located in the course of mining galena and sphalerite. Chemical analysis of the evaporate shows total solids of 1.2% predominantly NaCl (sodium chloride). Studies show that the salt content increases with increasing depth

iii. Afikpo zone: Achara, Ezeukwu, Okayi, as well as the famous Okposi and Uburu salt lakes, characterize the salt zone. Samples from the Okposi and Uburu brines yielded less than 1% total solids, predominantly NaCl. Local salt industries have existed in Uburu and Okposi long before the advents of Europeans in Nigeria.

iv. Ikon – Calabar area: Little is known about the brines of this area. There are, however, indications of salt industries existing or at least having existed at the following localities: Ikom, Okumi, Odukpani, Abia and Dauara. Besides the four main areas considered, other scattered occurrences of brine are reported at Gumel, Bage, Nefada, in Kano state and also Keana in Nassarawa state. The salt content is low, of the order 0.08% on the average. (Tattan 1942).

Research Methodology

Field Sampling

The samples used for this research work were obtained from the stockpile, Keana salt-sand deposit at Keana, Nassarawa state. The Keana salt-sand deposit was collected at different stockpile, based on the years of storage after being dried. The raw water from the river was collected with 0.75litres of plastic bottle container (3 in number) and another 0.75litres of plastic bottle container (3 in numbers) were also used to collect the filtrate containing salt after being processed locally (filtration).
1kg of the end product (salt) was also collected after being processed locally

using evaporation method by boiling off the filtrate using pot and firewood. The salt and deposit was also collected (5kg), 1kg from each stockpile, the salt-sand deposit were stockpiled based on their years of storage. The local method of processing Keana salt-sand deposit involved the use of a log of wood which was carved u-shaped, three (3) perforated clay pots which contained the mixture of the salt-sand deposit and the raw water from the river, a leather carpet which enhances the flow of the filtrate (salt water), Apashi clay is used with a coconut shell to block the perforated hole to prevent the salt-sand from coming out and also three (3) pots used for the collection of the filtrate (salt water). A log of wood which was curved u-shaped and the three (3) perforated clay pots were mounted on an inclined stone, the leather carpet was placed on the surface of the log of wood to enhance the flow of the filtrate (salt water). The coconut shell and the Apashi clay were used to cover the perforated hole to prevent the salt-sand from coming out along with the filtrate (salt water) which was then evaporated to dryness locally using firewood and a pot by boiling to get the end product (salt).

Materials and Method

Materials

I. Conical flask (10 in number)
ii. Raw water sample and the filtrate water sample.
iii. Hot plate.
iv. Measuring cylinder.
v. Weighing balance
vi. Two (2) perforated plastic containers and two (2) imperforated Plastic container used as filtrate collectors.

Procedure

i. The conical flask (10 in number) were taken and weighed using a weighing balance, five (5) of the conical flask were used for the raw water (river) while the other five (5) for the filtrate (salt water).
ii. The pH of the two water samples were determined using the pH meter, the pH of the raw water was 7.00 while the filtrate was 9.00 respectively.
iii. 10ml, 50ml, 100ml, 150ml and 510ml of the two water samples were measured using the measuring cylinder.
iv. The two water samples were taken to the hot plate for evaporation to dryness. The hot plate was adjusted to $90^{o}C$ which was used for the evaporation of samples
v. 10ml of the filtrate (salt water) and 10ml of the raw water evaporated to dryness at the same time, also 50ml of the raw water and the filtrate (salt water) evaporated to dryness at the same time but 100ml, 150ml and 510ml of the raw water and the filtrate (salt water) evaporated to dryness at different time intervals.
vi. The salt recovered was allowed to cool and their weights were taken using the weighing balance. The time of recovery of each sample and the weight of the salt recovered were recorded.
vii. 1kg of the salt-sand deposit was weighed using a scale and another 1kg was weighed using the same procedure.
viii. 1litre (1000ml) was measured using a measuring cylinder.
ix. The 1kg of salt-sand was mixed with the 1litre (1000ml) of tap water in a perforated rubber with a collector, the same procedure was used for the other 1kg of salt sand and 1litre (1000ml) of the raw water (river water).
x. The two (2) samples were allowed to stay for 24hours in order to get a filtrate from the two (2) mixed samples.
xi. The two filtrates were obtained, out of the 1000ml (1litres) of the tap water; 700ml was recovered as filtrate while 750ml was recovered out of 1000ml of the raw water.
xii. The two salt-sand samples mixed with tap water and the one mixed with raw water (1kg of each salt sand deposit) were allowed to dry under the sun and reweighed again. The weight of the salt-sand mixed with tap water out of the 1kg was 0.9kg. And that of the salt-sand mixed with raw water out of the 1kg was 0.86kg.

xiii. The pH of the two filtrates were obtained using the pH meter, tap water mixed with salt-sand was 4.10 wile the pH of raw water mixed with salt sand was 3.93.

xiv. 10ml, 50ml, 100ml, 150ml, and 510ml, of the two filtrates were measured separately using a measuring cylinder. The weight of the conical flask (10), five for the tap water filtrate and the other five for the raw water filtrate, were taken using the weighing balance.

xv. The two filtrates were taken to the hot plate for evaporation to dryness. The salts were recovered at different time interval.

xvi. The salt recovered was allowed to cool and their weights were taken using the weighing balance. The time of recovery of each salt sample was recorded.

Discussion of results

Chemical Analysis of the water samples

Table 4.0 and 4.1 shows the results of the chemical composition of the as-received filtrate brine and the as-received raw water from water salt pond. From the results of the chemical analysis of the water samples, the water sample of as-received filtrate salt water has 91.74% calcium, 0.0024% iron, 7.77% magnesium, 0.0192% fluoride, 0.359% chloride and 0.0680% nitrates. The as-received Keana raw pond water from the pond has 64.34%calcium, 0.713% iron, 1.314% of magnesium, 0.099%fluoride, 31.58% chloride, and 1.878% nitrate. Elements like potassium, sodium and iodine were unable to be determined. From the analysis the as – received filtrate of the sand-salt has high percentage of calcium and magnesium and low percentages of chlorine, nitrates and iron ions than the as-received raw water from the salt water pond. This could be attributed to the reasoning that some of the elements present in the salt-sand may have been leached into the filtrate water while chlorine, nitrates and iron that are highly present in the salt-sand may have undergone reaction with other elements in the as-received raw water their by bringing their low concentration in the filtrate water.

Recovery of Salt from the Brine Solutions

Table 4.2 shows the results of the times, weights and the percentages of salt recovered from the as-received filtrate brine solution of Keana salt-sand deposit, as-received Keana raw pond water, Laboratory filtrate raw pond water brine solution and Lab. Tap water filtrate brine solution of Kaena salt-sand deposit. From the result in the table 4.2a it is observed that the percentage of the salt recovered increases as the quantity of filtrate increase with increase in the time required for the evaporation to take place. The weights of salt samples that are recovered from the as-received filtrate brine solution (10-510ml) is higher than the weights of the salt samples that are recovered from the as-received Keana raw pond water, Laboratory filtrate raw pond water brine solution and Lab. Tap water filtrate brine solution for 10 to 510ml samples respectively. The time required for recovering of the salt from 10 to 200ml of the different samples of filtrates brine solutions varies with the Laboratory filtrate raw pond water brine solution having the least total time of 758minutes compared to others while the total amount of salt sample recovered is 153.09g more than the as-received Keana raw pond water (110.04g, 770 minutes), Lab. Tap water filtrate brine solution (22.0g, 819minutes) but less than the as-received filtrate brine solution (166.8g, 845 minutes) see figure 1.0. This reason could be attributed to the low pH value of the brine solution (pH 3.93) which make the water sample more acidic compared to the Lab. Tap water filtrate brine solution (pH 4.10) and that of the as-received Keana raw pond water (pH8.00) but more less basic than the as-received filtrate brine solution (pH9.00), thus enhancing fast evaporation of the brine and quick seeding of the salt grains from the Laboratory filtrate raw pond water brine solution sample than others.

Conclusion

In conclusion the Keana salt is high calcium based salt that can be recovered using both the Keana salt water pond and any clean tap water. The recovering of salt samples from filtrate of the Keana salt-sand using tap water disbelieved the superstition statement that said that only the filtrate of the salt water pond from the Keana village where the deposits is located can be uses only to produce salt. Therefore, the Keana salt-sand deposit can be listed as another potential source of raw material for the production of salt for both domestic and industrial used.

Recommendation

The chemical composition of the salt produced using various brines should be determined to ascertain their qualities.

References

Adetoye, B.O., (1998): Preliminary return on barites occurrence in Nasarawa State, Geological Survey Department, Lafia.

Benkhehlil, J., (1989); the origin and evolution of the cretaceous Benue trough, Nigeria Journal and Africa Earth Science Vol. 8

Beltare, F., (1971): The possibility of salt industry in Nigeria. G.S.N Report 1992.

Carter J.D., Barber, W.M, Tait, E.A., (1963): The geology of part of Adamawa, Bauchi and Borno provinces in North-eastern Nigeria, Bulletin of Geological Survey Nigeria, No. 30, PP 108.

Carter, J.D., Janes, G.P., (1965): An interpretation of the Geology and gravity anomalies of the Benue valley, Nigeria Overseas Geological Survey Geophysics paper, pp1 – 26.

Farrington, J.L., (1952): A preliminary description of the Nigeria lead-zinc field, Economic geology, pp 47, 583 – 608.

Ford, S.O., (1980): The economic mineral resources, Journal and Earth Evolution Science, 2/1981 Pub. Fredr.Berlin Vieweg and John.

Offodile, M.E., (1976a): The geology of the middle Benue, Nigeria volume 4, Paleo Institution Publication University of Upsallah.

Offodile, M.E., (1976 b): The geology and tectonics of the Awe brine field, Journal Minerals and Industries, vol. 30 (1), pp189 – 201.

Offodile, M.E. and Reyment, R.A., (1976): The Stratigraphy of the Keana-awe area of the middle Benue region of Nigeria. Volume 7.Bulletin of Geol. Inst. Uni. of Upsallah. N.S.

Ogbonnaya, O., (1997): The mineral wealth of Nigeria, Federal University of technology Owerri. First published 1997

Raeburn, C and Jones, B. (1934): The Chad basin geology and water supply. GSN bulletin No. 15, 61p.

Robert L. Bates (1960): Geology of industrial rocks and minerals Dover publications New York, First edition 1960.

Tattan, C.M. (1942): Preliminary report on salt industry. Eastern Nigerian GSN report NO. 778.

Thomas, D.G. Aye, A.E, Salawu, O.M. (2005): Journal of Engineering Technology and Industrial applications Vol. 2, Number 1, Page 49.

Minerals and Industry of Nigeria (1987) with notes on the history of geological survey of Nigeria, second edition published by Authority of the government of the Federal republic of Nigeria printed by Zewods, Printing Services Kaduna.

Welz, A.B., (1998): Atomic absorption spectrometer third edition, Wiley VCH Withinhein, Germany.

www.nassarawastate.org,(2009)

Nassarawa state development agency (1979)

http://www.saltmatters.org, (2009)

http://www.saltgov.uk/tv.ads.html,(2009)

http://www.saltsense.co.uk.htm, (2009)

http://www.salt.org.u/arch.html, (2009)

http://www.eu-salt.com/index3htm,2009

http://www.saltsense.co.uk/aboutsalt-envo1.htm, (2009)

http://www.saltsense.co.uk/aboutsalt-fact01.htm,(2009)

Table 4.0: Elemental composition of as-received filtrate brine solution **from Keana (sample A)**

Element	Formula	Percentage (%)
Calcium	Ca^{2+}	91.74
Ferrous	Fe^{2+}	0.0024
Magnesium	Mg^{2+}	7.77
Sodium	Na^{2+}	Not determined
Potassium	K^{+}	Not determined
Fluoride	F^{-}	0.0192
Chloride	Cl^{-}	0.359
Nitrate	NO_3^{-}	0.0680
Iodine	I^{-}	Not determined

Table 4.1: Elemental composition of as-received Keana raw pond water **sample from Keana (sample B)**

Element	Formula (ionic state)	Percentage (%)
Calcium	Ca^{2+}	64.34
Ferrous	Fe^{2+}	0.713
Magnesium	Mg^{2+}	1.314
Sodium	Na^{2+}	Not determined
Potassium	K^{+}	Not determined
Fluoride	F^{-}	0.099
Chloride	Cl^{-}	31.58
Nitrate	NO_3^{-}	1.878
Iodine	I^{-}	Not determined

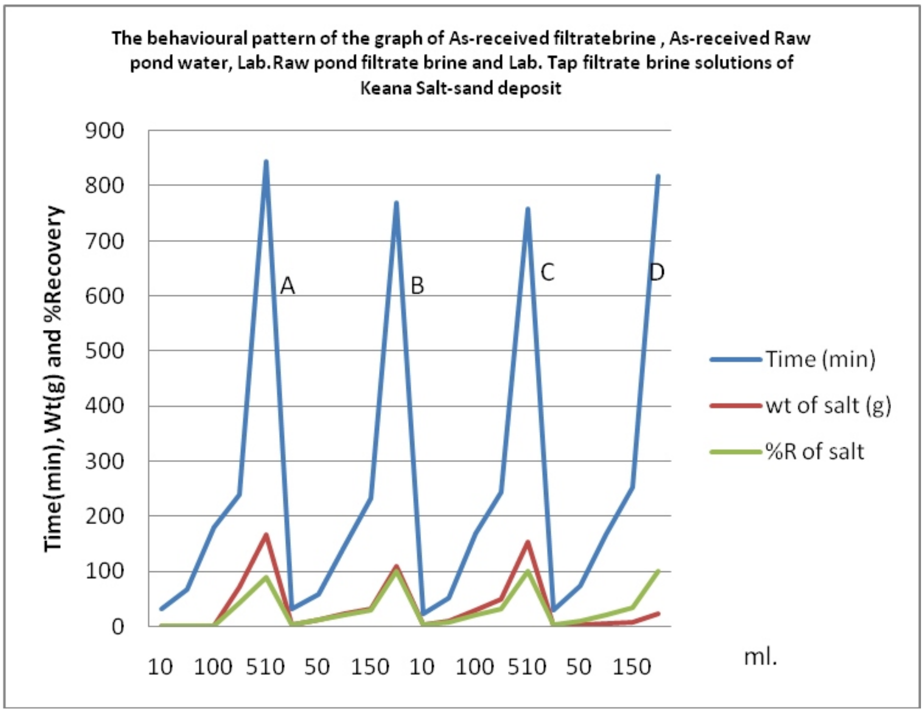

Fig.1.0: The behavioural pattern of the graph of as-received filtrate brine(A), as-received raw pond water(B), Lab.raw filtrate brine (C) and Lab. Tap filtrate brine (D) solutions of Keana salt-sand deposit

Table 4.2. (a) Result of the salt samples recovered from the as-received filtrate brine solution of Keana salt-sand deposit, as-received Keana raw pond water, Laboratory filtrate raw pond water brine solution and Lab. Tap water filtrate brine solution of Kaena salt-sand deposit

Content of filtrate	As-received Filtrate brine solution (pH:9.00)			As- received Raw pond water (pH: 8.00)			Lab. Filtrate Raw pond water brine solution (pH:3.93) (1 litre for 1kg of salt sand)			Lab. Tap water filtrate brine solution (pH: 4.10) (1litre for 1kg of salt sand.)		
	Time of recovery (minutes)	Weight of recovered salt (g)	% recovered	Time of recovery (minutes)	Weight of recovered salt (g)	% recovered	Time of recovery (minutes)	Weight of recovered salt (g)	% recovered	Time of recovery (minutes)	Weight of recovered salt (g)	% recovered
10ml	33	0.02	0.0119	33	2.06	1.8720	24	2.10	1.3717	30	0.57	2.5815
50ml	69	0.35	0.2098	59	11.63	10.5688	53	9.39	6.1336	74	2.02	9.1485
100ml	181	0.67	0.4016	145	21.41	19.4565	169	29.13	19.0280	168	4.46	20.1992
150ml	241	71.04	42.5899	233	32.12	29.1893	244	48.20	31.4847	254	7.34	33.2427
510ml	845	166.8	88.0093	770	110.04	99.9997	758	153.09	99.9998	819	22.08	99.9997

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Correlation of Grindability of Granite Rock, Penetration Rate and Bit Wear, Piccolo Granite Quarry, Ore, Nigeria

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Abstract
This work correlates the parameters of grindability of granite rock, penetration rate and bit wear in granite quarry. In the investigation, the grindability test of the granite rock at 150µm and the 80% passing of sieve size was determined using Bond developed closed circuit grindability test. The penetration rate and bit wear under the influence of the rock mechanical properties were also examined. The data obtained were analyzed using SPSS computer software package. The result shows that a medium correlation exists between the grindability, penetration rate and bit wear of the granite rock sample. It was also found out that an increase in hole depth results in a decrease in penetration rate and increase wear rate while increase in number of revolution will lead to increase in grindability to a certain limit. The results of the investigation will be useful database for mine/quarry operators and prospective mine investors in the mining industries.

Keywords: Granite rock, grindability, penetration rate, bit wear, correlation

Introduction
Grindability and work index are important properties of rock that refer to the relative ease with which the rock material can be comminuted. It is important to measure the parameters for newly discovered rock as well as rock from the existing mine periodically in order to access the efficiency of the selected processing equipment. The rate of grindability of particles has been related to their material properties. Clarke (1982) measured both the tensile strength and the unaxial compressive strength of limestone particles and also determined the grindability of the particles in the laboratory ball mill. He found out that the particle in a size reduction process has some probability of being grinded and in any size fraction, a proportion of the particles selected for grinding and the remainder passes through the process unbroken. Using this concept, a selection function was defined, that is, the probability of breakage each size reduction. As with the rate of breakage, the selection function can be used in combination with the breakage function to describe the size reduction process.

Grindability index is a numerical indication of the capacity of a material to be ground. It has been shown that the data from grindability test are used to evaluate crushing and grinding efficiency. One of the most widely used parameters to measure ore grindability is the Bond work index, W_1. If the breakage characteristics of a material remain constant overall size ranges, then the calculated work index would be expected to remain constant since it expresses the resistance of material to breakage. However for most naturally occurring raw materials, difference exist in the breakage characteristics depending on particle size, which can result in variation in the work index. For instance, when a material breaks easily at the boundaries but individual grains are tough, the grindability increases with fitness of grind, consequently, work index values are generally obtained for some specified grind size which typifies the comminuting operation being evaluated, (Ojo and

Olaleye, 1999). Grindability is based upon performance in a carefully defined piece of equipment according to a strict procedure. Bond has devised several methods for predicting ball mill and rod-mill energy requirements that provide an accurate measure of ore grindability. The rate at which particles grind in a size reduction process can be represented by a first-order equation.

Thus, absolute grindability of size

$$i = K_1 M_1 \qquad ...1$$

where K_1 is the grindability of size fraction i (the first order rate constant) and M_1 is the mass fraction in size i. This equation has been found to be generally applicable to grindability process, (Pijush, 2005).

Properties such as compressive strength, grain size, shear strength, cleavage, fracture, abrasiveness and hardness affect grindability. A low grindability index value corresponds to a high cuttability that is, high resistance to cutting action. The higher the resistance of a material to grinding, the lower its grindability. A fractured material can be easily grinded than a material that is intact. Also a rock that has a high tendency to split along certain directions also has high grindability. A material possessing high compressive and tensile strength also possess a low grindability value, (Olaleye and Ojo, 2004). It has also been found out that grain size of rock material affects rock grindability because rock with fine grain size has a high grindability than a rock with coarse grain size. The bonding property of a rock is also of paramount important because a friable rock with a loose bonding property like kaolin can be easily comminuted than a coherent rock like quartzite that has strong bonds and therefore has low grindability.

The basic factors affecting the penetration rate are bit selection, bit weight, rotation speed and formation properties. According to Gatlin (1988), drilling rate is not linearly proportion to the rotary speed in hard formations because some finite concurrent application of torque is required for a bit tooth to fracture the rock. The proper bit selection is a factor in reducing the number of bit changes, increasing penetration rates and reducing drilling where bit costs are minimal in relation to the huge cost of even an hour of delay. The success of a bit choice and the suitability of new bit design in various rock types can be evaluated from drilling records, (Clark 1982). According to Garner and Allen (1997), a small bit diameter may appear attractive, as it favours the rate of penetration and is as well economically advantageous considering the purchase price and regrinding costs, considerations of blasting technology and the drilling conditions such as flushing effect, wear on the drill and number of possible regrinding, may however direct the choice of a larger diameter. Also, the drilling rate is greater in a porous rock than in a dense rock and porous zones of the same formation usually have lower compressive strengths than the less porous sections.

Correlation analysis is used to investigate the variations between variables and also to measure the magnitude of such variations. The rule of thumb for interpreting coefficient of correlation, r, is presented in Table 1, (Okoko, 2000).

Table 1: Rule of thumb for interpreting coefficient of correlation, r, (Okoko, 2000)

Value of coefficient of correlation, r (positive and negative)	Interpretation
0.90 – 1.00 (-0.90 to –1.00)	Very high correlation
0.70 – 0.90 (-0.70 to –0.90)	High correlation
0.50 – 0.70 (-0.50 to –0.70)	Moderate correlation
0.30 – 0.50 (-0.30 to –0.50)	Low correlation
0.00 – 0.30 (0.00 to –0.30)	Slight correlation

Location of the Study Area

The determination of the penetration rate and bit wear was carried out in Piccolo granite quarry, Ore, Nigeria (Fig. 1) while the grindability test was carried out in the Rock Mechanics Laboratory of The Federal University of Technology, Akure, Nigeria. Ondo State is located in the western part of Nigeria on longitude 4^0 - 6^0 E and latitude 6^0 - 8^0 N. The area is located within the tropical west forest and savannah. Most of the rainfall is concentrated between April and October, and the wet and dry seasons are well marked. Temperature is high throughout the year with a mean of 27^0C and the relative humidity ranges between 60 and 80%.

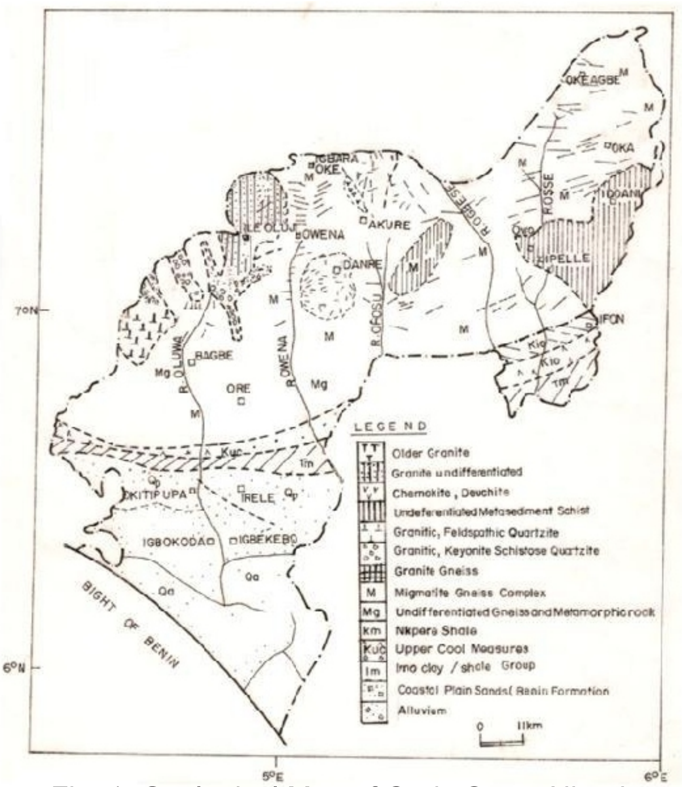

Fig. 1: Geological Map of Ondo State, Nigeria Showing the Study Area.

Materials and Method

Granite rock samples were collected from Piccolo granite quarry, Ore, Nigeria. The samples were worked upon in the laboratory to determine the grain size distribution, grindability while the wear and penetration rates of the drill bits were determined in the quarry during operation. Button bit was selected for the investigation due to its characteristic design and ease of measurement.

Determination of Grindability: Crushing and grinding operations were carried out on the samples in the laboratory using jaw crusher and pulverizer respectively. The sieve analysis was carried out on a representative sample (approximately 500g) of the crushed feed in a set of sieve under the influence of a sieve shaker. The result of the sieve analysis is presented in Table 2. A graph of the result of sieve analysis was plotted in which the feed size F, that is, the size in microns through which 80% of the feed passed was determined from the plot. The grindability test was carried out at 150μm sieve size, (Ojo and Olaleye, 1999). The mill was charged with 20 balls weighing 1575g and 200g of feed. The mill was run for 100 revolutions, N_1. At the end of the cycle, the mill was discharged into a receiving pan and the ball charges screened out. The grounded material was sieved through a 150μm sieve taking small portion at a time. The weight of +150μm (oversize) fraction and -150μm (undersize) fraction were taken and recorded. The grindability of the first cycle was determined from the following relation:

$$Grindability, Gr = \frac{Weight\,(W_{+150})}{No.\ \ of\ \ \mathrm{Re}\,v(N_2)} \qquad ...2$$

The same process was repeated for the second trial and the result recorded. A new feed of the same weight as -150 μm was added to the retained fraction, +150 μm to make up 200g and charged into the mill for 150 revolutions, N_2 and the grindability was also determined. The whole process was repeated for 200, 250 and 300 revolutions for both first cycle and second cycle trial. The result of the grindability test is given in Table 3

Determination of Penetration rate: In the quarry, a new working face was created for the purpose of the investigation. of depth 17m, Hole depth of 17.0m was desired but due to the undulating nature of the terrain, some of the hole were drilled to the depth 17.5m, 18.0m, and 18.5m. The time taken to drill each hole from the point of installation of the new bit till the bit was changed was recorded. Eighteen holes were drilled to various specified depth by the bit before it worn out. The result of the drilling operation is presented in Table 4.

Determination of Wear Rate: At the new quarry face, the initial length of the new button bit employed in the drilling operation was measured to be 7mm using a vernier caliper. At the end of drilling a 17.0m hole depth, the drill pipe was tripped out and the button bit was measured again. This was carried out for all the nineteen holes that were drilled by the button bit. The initial and final measurements were recorded, (Table 5).

Results and Discussion

Results: The result of the sieve analysis and the grindability test on 150μm particle size are presented in Tables 2 and 3 respectively. Table 4 shows the result of the drilling operation indicating the degree of penetration of the bit while Table 5 shows the result of wear rate of the drilling bit. Figure 2 depicts the graph of log of sieve analysis against log of cumulative % passing and Figure 3 shows the plot of drill hole depth against penetration and wear rates. The graph of grindability against mill revolution is shown in Figure 4.

Table 2: Result of Sieve Analysis of Granite Rock Sample

Nominal Aperture (μm)	Wt. Retained (g)	Wt. % Retained	Cumulative Wt. % Passing	Log of Sieve size	Log of Cum. Wt % Passing
4750	150	30.00	70.00	3.676	1.845
2000	185	37.00	33.00	3.301	1.518
1700	10	2.00	31.00	3.231	1.491
850	50	10.00	21.00	2.920	1.322
600	15	3.00	18.00	2.778	1.255
425	16	3.20	14.80	2.628	1.17
212	25	5.00	9.80	2.326	0.991
150	12	2.40	7.40	2.176	0.869
-150	37	7.40	0.00	0.000	0.000

Table 3: Result of Grindability Test on 150μm Particle Size

Cycle	No. of Rev.	1st Run		2nd Run		Average
	(N)	W_{150} (g)	$A = W_{150}/N$	W_{150} (g)	$B = W_{150}/N$	(A+B)/2 (g/rev)
1	100	3	0.030	7	0.070	0.050
2	150	4	0.027	13	0.087	0.057
3	200	12	0.060	15	0.075	0.068
4	250	19	0.076	18	0.072	0.074
5	300	18	0.060	19	0.063	0.062

Table 4: Result of Penetration Rate

Hole	Time Taken (min)	Depth (m)	Penetration Rate (m/min)	Hole	Time Taken (min)	Depth (m)	Penetration Rate (m/min)
1	26.03	17.00	0.65	10	26.10	17.00	0.65
2	25.03	17.50	0.69	11	25.58	18.00	0.70
3	26.57	18.50	0.69	12	25.47	17.00	0.66
4	27.58	18.50	0.67	13	29.00	17.00	0.58
5	26.00	17.50	0.67	14	30.00	17.00	0.56
6	28.00	17.00	0.60	15	26.00	17.00	0.65
7	28.59	18.50	0.64	16	25.00	17.00	0.68
8	27.54	18.50	0.67	17	26.41	17.00	0.64
9	26.00	17.00	0.65	18	26.31	17.00	0.64

Table 5: Result of Wear Rate

Hole	Wear Length of Bit (mm)	Depth (m)	Wear Rate (mm/m)	Hole	Wear Length of Bit (mm)	Depth (m)	Wear Rate (mm/m)
1	6.90	17.00	0.41	10	3.90	17.00	0.23
2	6.40	17.50	0.37	11	3.75	18.00	0.21
3	6.00	18.50	0.32	12	3.60	17.00	0.21
4	5.20	18.50	0.28	13	3.50	17.00	0.21
5	5.00	17.50	0.29	14	3.40	17.00	0.20
6	4.89	17.00	0.29	15	3.35	17.00	0.20
7	4.68	18.50	0.25	16	3.30	17.00	0.19
8	4.31	18.50	0.23	17	3.21	17.00	0.19
9	4.00	17.00	0.24	18	3.12	17.00	0.18

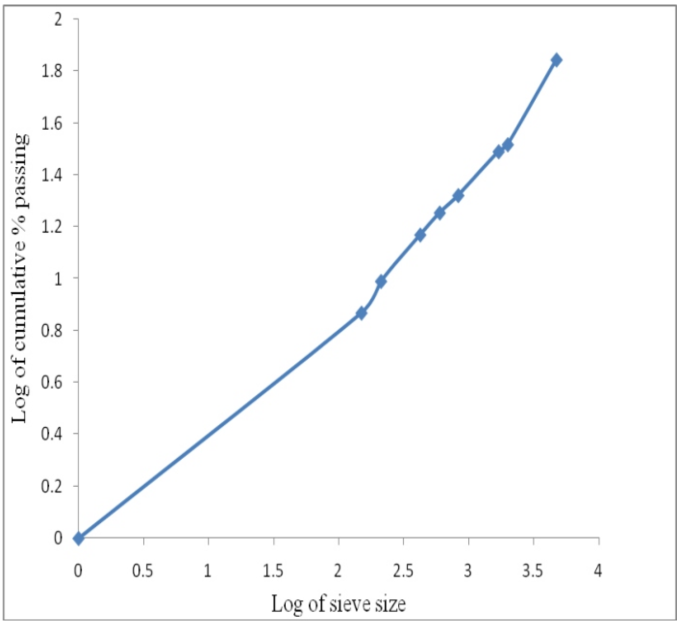

Fig. 2: Graph of Log of sieve analysis against Log of cumulative % passing

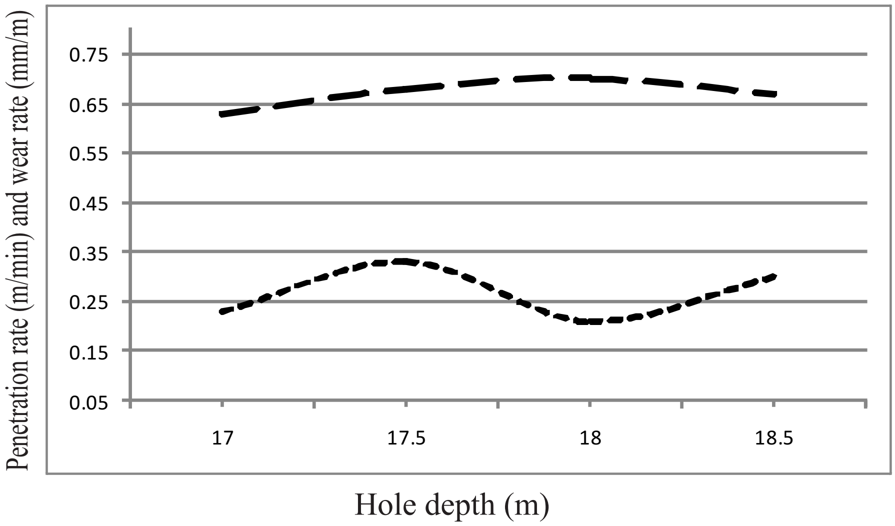

Fig. 3: Plot of drill hole depth against penetration and wear rates

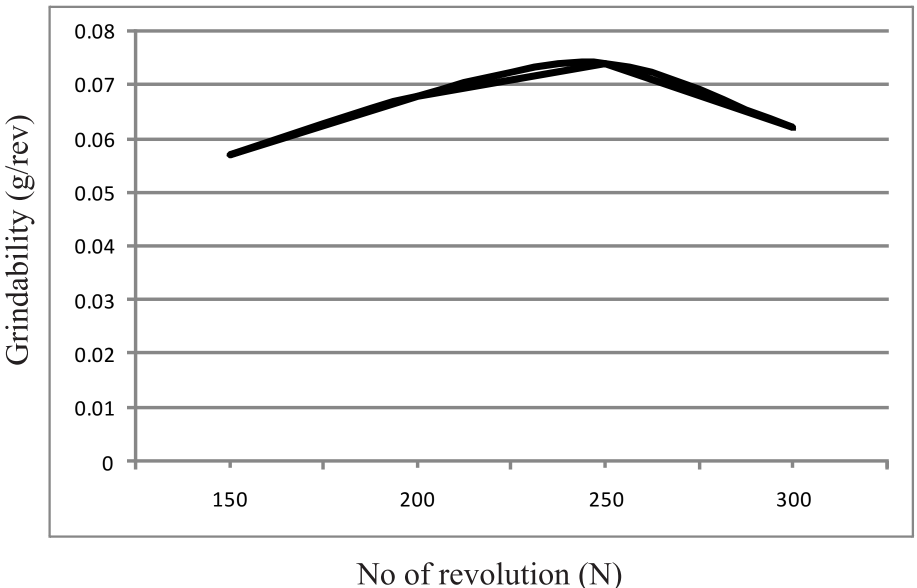

Fig. 4: Graph of grindability test on 150µm particle size against mill revolution

Discussion

From Table 2, it can be observed that the sieve size of 2000 μm has the highest percent weight retained of 37% which may be due to low reduction ratio of the pulverizer. Also, the 80% passing size is 2248μm. The graph of grindability test shown in Figure 4 depicts that the net 150μm product increases with increase in the number of mill revolution and a little increase in grindability to a certain limit. That is, the highest grindability of 0.074g/rev was attained at mill revolution of 250. At this point any increase in mill size reduction will lead to more energy input in the mill and consequently waste of energy. This implies that grindability increases with fitness of grind and individual mineral grains or particles are well liberated at the value of the equivalent grindability of the rock which is 0.062 g/rev. It also shows that work index values are generally obtained for some specified grind size which typifies the comminuting operation being evaluated. From Figure 3, it is found out that an increase in the hole depth may lead to an increase or decrease in penetration rate depending on the degree of the overburden pressure and compaction of the rock formations. That is, as hole depth increases, more time may be consumed in drilling the hole thereby reducing the rate of penetration. The graph also shows that an increase in depth may result in an increase in wear rate that is, the rate at which the bit wears will increase as depth increases thereby reducing the life span of the bit. Grindability, penetration rate and wear rate were correlated using SPSS software to share the strength of relationship between the variables. The result shows that a positive correlation value of 0.51 (moderate correlation) exists between grindability, penetration rate and wear rates.

Conclusion

Investigation was carried out to determine the correlation of grindability of granite rock, penetration rate and wear rate. It was found that increase in depth will result in a decrease in penetration rate and increase wear rate while increase in the number of revolution will also lead to increase in grindability to a certain limit. This work will be useful to mine/quarry operators and prospective mine investors in the mining industries in estimating the actual time required for a drilling bit to be changed due to wear and the quantity of bits required for a drilling cycle.

Reference

Clark, G.B. (1982): "Principles of Rock Drilling and Bit Wear, Part 1, Quart, Colorado School of Mines, Vol 77, No.1, p 108

Gatlin, C. (1988): Petroleum Engineering, Drilling and Well Completion. Prentice-Hall Inc., Englewood Cliffs, N.J., p 341

Garner, C.R. and Allen J.H. (1997): Economical Bit Runs Achievement. Drilling DCW, 37(9), pp. 42-43.

Ojo O. and Olaleye B.M. (1999): Determination of Grindability and Work Index of Itakpe Iron Ore, Global Journal of Pure and Applied Sciences, Vol.6, No.3, pp. 503-508.

Okoko E. (2000): Quantitative Techniques in Urban Analysis. Krast Books Limited. Ibadan, Nigeria. pp. 32-70.

Olaleye B.M. and Ojo O. (2004): Classification of Ondo State Intact Rock for Engineering Purposes. Journal of Sci. Eng. and Tech., Vol. 11, No 3, pp. 5753-5759.

Pijush Pal Roy (2005): Rock Blasting Effect and Operations. A.A Balkema Publishers, p. 364

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The Socio-Economic Impact of Artisanal Mining in Kuru (Naraguta Sheet 168) Plateau State, North Central Nigeria

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Abstract

A study of the sub-surface (Loto) mining activities around Kuru Village- a suburb of Jos City revealed the activities of Artisanal Miners where mining is presently being carried out on an old mining lease belonging to Bisichi Jenta Ltd. This former British mining company carried out mining activities in the 1950s with intensity in area of coverage which suggests that the company stood out amongst its peers especially with regard to the effective deployment of mechanized mining using hydraulic mining equipment including the hydraulic jigs, mine dredges and large haulage machinery. The mining of cassiterite along two relatively adjacent steams in the Kuru area witnessed the exploitation of tin and columbite initially in alluvial deposits in the downstream stream and graduating into elluvial and subsequently upstream to the primary source of decomposed granite. The hydraulic mining operation then concentrated in the mining of deposits with cut-off grades of more than 0.6g/ton. The present artisanal mining however, is engaged in mining of abandoned low grades (0.1g/ton) of tin and columbite, which at the present economic value, still provides income to the artisanal miners. The mining activity involves the random sinking of mining pits for the exploitation of the tin mineral, with devastating effect on the environment as opened pits are not covered after mining. The activity which is rudimentary and non-regulated contributes to secondary devastation of the arable land with resultant effects on the land, ecosystem, and health threads to inhabitants. Although the sub-surface mining of tin and Columbite contributes to an average daily earnings of about N250, 000.00 accruing to a total population of about 1200 local miners, the overall after effect lives much to be desired in terms of environmental degradation, potential health hazards and ground water pollution amongst others. The individual average daily earning of about N 20,500 makes economic sense for these miners especially when viewed happening in a national economy with a per capita earning of less than $1 dollar(N165) per day. The artisanal miners by all means deserve a better social contract from government to ensure a better working condition through appropriate mining regulation in line with best practices.

Key Words: Artisanal Mining; Sub-surface Mining; Tin and Columbite; Mine Devastation

Introduction

Kuru is a village in Jos South Local Government Area of Plateau State, North Central Nigeria, located on Naraguta SE, Sheet 168. The study area falls within the following coordinates latitude $9^0 40'$N – N$9^0 44'$N and longitude E$8^0 51'$ – E$8^0 53'$ (Fig 1). General Geology of the study area Kuru is generally characterized by the abundant occurrence of the Younger Granite rocks which were emplaced during the Jurassic era. The formation of the Younger Granite is associated with the hot spot magmatism. The rock bodies are massif occurring ring complexes. The Younger Granites are known for hosting tin and columbite within Jos Bukuru and environs.

The study area (Kuru) constitutes of settlements like; Science School Kuru, Kuru Karama and Kuru Babba. Kuru is an ancient mining town boarded by some other mining town such as Bisichi in the Northeast and Bukuru- Rayfield to the north. It is essentially known for the mining of tin and columbite with other associate minerals such as tantalum and kaolin.

The brief history of mining for cassiterite and columbite in the Study area as narrated by

Pwajok shows that mining started in the area in 1920. Due to fluctuation of prices of minerals in the global market, stock market; International Tin and Columbite Control (ITCC) was established to counter the fall in price whenever there is a fall in demand which is synonymous to OPEC. Sub-Surface mining (Lotto) mining started way back from 1955, and then the minerals were mined at two pounds per-cubic feet (cut-off grade) and anything less than that was considered unprofitable. Presently the artisanal miners mine grades of ore as low as 0.2 per cubic feet. Mining activities were carried out by three (3) major companies on the plateau namely; the Amalgamated Tin Mining Company of Nigeria (ATMN) and Bisichi-Jenta (established by Jentu and Tabus from U.K who owned the mined then). The Sub-surface and surface mining activities in the study consists of deposits made up of primary (decomposed granite- found in place within the parent rock) and alluvial deposits (found long streams and old river channels). Other places with primary mineralization of cassiterite and columbite include Udegi – Nassarawa State, Kuru Jenta – Within the study area Rayfield – Jos South local government.

Within the study area are lots of mine dump this was due to the reason that in the 1960s, there was no restoration clause in the law so miners dumped their waste without restoring the mines which constitute lots of unclaimed mined ponds on the Jos Plateau today. Later on, the laws were amended stating that 80% of the mines must be restore after any mining can be re-issued again.

Jig was used for the mining of both tin and columbites, using their difference in specific gravity, the processes are; first blast the rock; Use the gravity pump to wash the minerals at the paddock phase to the jig where the minerals are sorted.

The study area is vegetated with shrubs and tony plants; there are occasionally tall trees, even though there are Community Forest seen at locations. There lost of grasses found within the study area but the irony there is within the mine zone there is virtually not plant growth, just bare brown soil indicating that the soil doesn't promote plant growth.

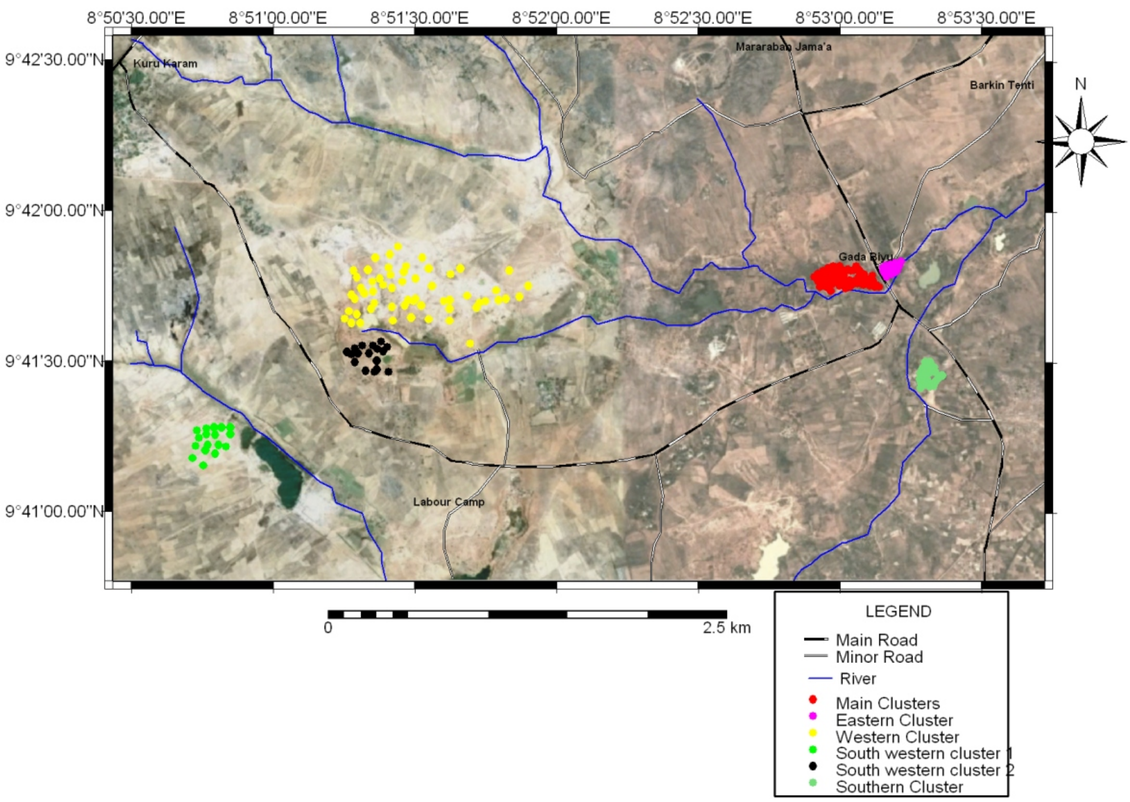

Fig.1. Satellite image showing mining clusters within Kuru and environs

Materials and Methods

The methodology of research involved on the spot assessment and field measurements of the study area using various tools which include Global Positioning System (GPS), Compass, Tape, Camera, Field notebook and writing materials. The use of Google-Earth was employed in order to extract the map of the study area. The locations of the Mining Clusters were obtained from alluvial and primary deposits with the aid of GPS which was used to pick up the coordinates and then inserted on a geo-referenced map, using the software ILWIS 3.1 Academic. The digitized map was exported out of ILWIS 3.1 Academic into Microsoft word for further editing.

Results and Discussion

The field work involved the study of six (6) Mining Clusters mainly, The Main Cluster, Eastern Cluster, Western Cluster(Primary deposit), South-Western Cluster 1, The South Western Cluster 2 and The Southern Cluster respectively all covering a total land area of 1.34km^2 (Table.1).

S/No.	Clusters	GPS Coordinates	Area of Devastation
1	Main Cluster	$9^0$41’46.91”N $8^0$53’00.55”E	0.334 km^2
2	Eastern Cluster	$9^0$41’49.37”N $8^0$53’11.62”E	0.075 km^2
3	Western (Primary Deposit) Cluster	$9^0$41’43.69”N $8^0$51’30.40”E	0.600 km^2
4.	Southern Cluster	$9^0$41’26.37”N $8^0$53’18.76”E	0.194 km^2
5	South-western cluster 1	$9^0$41’13.93”N $8^0$53’18.76”E	0.064 km^2
6	South-western cluster 2	$9^0$41’31.11”N $8^0$51’20.03”E	0.074 km^2

Location 1: The Main Mining Cluster (Gada Biyu) (N09^{0}41'50.1”; E 08^0 52'01.4”and Elevation: 1236m) (Fig.1). Artisanal mining started in Gada Biyu 2006, it was located on the opposite side of the present Gada Biyu (across the road), the mine site was later abandoned because cut-off grade of the ore was not favourable then, the miners therefore relocated to Gero, Bisichi and Barkin Ladi mining areas of Plateau State.

Location 2: Primary Columbite Paddock (PCP) otherwise known as the primary deposit (Latitude N 09^{0}41'30.2” Longitude E08051'20.0” & Elevation: 1260m). This large mine site is being reworked in search of Tin and Columbite. The deposit is from a primary source (decomposed granite) which was mined around 1950s and left un-reclaimed. The mine site cannot be restored to more than 2% as no amount of dump can be used to fill the largely excavated area (Fig.3). Abandoned mining pits litter the area. Despite its devastated nature and threat to human lives and livestock, farmers were seen using the ponds for irrigation purposes (vegetable like pepper, tomato and beans) were being plant on the higher plains. Reworking of the old mine and opening up of new mine-faces from the existing one was seen currently going on by artisanal miners.

Locations 3, 4, 5 &6. These comprise of the Eastern; Southern, South-Western 1; and the South-Western2 Clusters. The Eastern and Southern Clusters are located in alluvial deposits while the South-Western 1 and 2 Clusters are located on decomposed granite deposits. At these locations, several active and non-active mining pits were sighted with several artisanal miners on duty.

Mine Development/ Extraction: Mine Development is accomplished through a series of vertical pits (shafts) (Figs. 2 & 3) while the extraction of mineral is done using diggers and shovels. In the study area and specifically on the Main Cluster, twenty five (25) active mining pits and twelve (12) active sluice boxes were sighted (Fig.2 in-set). The non-active mine pits which by far outnumber the active ones, were found to be 67 in numbers. The tables provide the exact locations and elevations of active mine pits and ground sluice boxes which can serve as reference data for future geo-technical study of the sub-surface soil/rock structures. The development and extraction of minerals is accomplished through the mine syndicate system consisting of about ten team members. The mine syndicate is further divided into two, such that while about three (3) people are developing new shafts, seven (7) members would be involved in extraction and processing. Women are primarily involved in the haulage of ore to the ground sluice boxes. The major mining functions are as follows:

- Digging – digging extends to depth of between 30ft – 50ft, the shallowest pits being about 30ft with a diameter of 2.5 – 3ft., tools used include digger, shovel, torch light and wheel and bucket. When the wash (ore) is hit, the thickness is determined and the mining is accomplished along the strike.
- Extraction-Excavation of horizontal openings is carried out along the trend of mineralization. Horizontal opening can extend to about 60ft, depending on the extent of mineralization or availability of oxygen within the sub-surface.
- De-watering – A water pump machine is used to extract water out of the mine pit into neighbouring pit otherwise known as the 'cotonia'. Two types of water pumps are used which are; the 2-inch inlet and outlet pump and 3-inch water inlet and outlet pump. The 2-inch water pump is considered to be more efficient because of its higher water pumping height.
- Haulages – the use of wheel and bucket to extract the wash from the well to the surface is employed by the miners while women further haul the mineral ore in head pans to the ground sluice boxes locations..
- Processing – They use rudimentary method in the absence of mechanized method. Ground sluicing boxes are designed in a manner that the wash passes through several stages pushed by flowing water delivered by water pumps. The lighter materials float pass while the denser materials (those with higher specific gravity) tin, columbite and iron trailing behind.
- Mine illumination/Ventilation- Candles and torchlight are used for illumination within the shaft, though torches are more preferable because it reduces competition of air consumption between the individual and candles as well as reduces heat generation within the hole. There was no evidence of any mine ventilation with miners sweating profusely.
- Mine reclamation: The evidence of any mine reclamation in the area is only through sporadic dumping of mine spoils and dumping of domestic waste from the village(Fig. 4)

Fig.2: Rudimentary Haulage and tin Processing Technique

Fig. 3: Land devastation at Primary Deposit

Fig.4: Reclamation of an abandoned mine site using domestic waste

Marketing: In a lotto where there is a sufficient quantity of ore, one pit can produce one wheel yield of 19kg of tin, columbite and other accessory minerals. One (1) bucket wheel of hauled ore represents one (1) bucket wheel yield consisting of 19kg of tin with specific gravity 5 and 28.8kg of Columbite with a usual specific gravity 7. Accessory minerals associated with the tin ore include zircon, iron and monazite with less than 1% total. Presently, a bucket wheel will yield at least 3cups of the tin concentrate after processing. On the average in a day, the artisanal miners can deliver to the surface about 15 bucket wheels, which translates to 45cups of tin daily.

On extraction of the material, it is buretted, the purity (grade) is scaled at 18.0. This is the highest grade and is sold at about N4200 per kg. If the burette value falls between 18.6 and 19.2, then it is sold between N3500 – N4000 per kg, any burette value greater than 19.2 is considered to be low grade.

In 2006, the market situation was more favourable as one (1) bucket wheel yielded one (1) bag of llthe same production rate (15 bucket wheel per/day) and at the cost of 1cup standing at N1000 – N2300 depending on the grade of the wash, the daily revenue accrual per production pit (N2000 x 45 cups) is N90,000. The tin concentrates is sold to middle men who in turn either resell or process same for export. The tin processing sheds include: Kingsley – At Anguldi Jos; Spectrum – at Industrial Estate old Airport Road (Mainly Exported); Gidan Kumar – At Dadin Kowa Jos (Mainly export), Gidan Emma – At Dadin Kowa, Jos; Chike Mills – Bukuru (Exporter), Gidan Ayara – Anguldi, and Gidan D. B. Zang. The dressed minerals are not smelted but exported raw (Mallo, Aluwong, 2012).

The mining activity is of great economic and social significance. As it currently exists, artisanal mining in Kuru is high risk activity. The activity is practised on a small-scale by people who are often poor, though educated, they lack other employment opportunities. It is a highly unregulated sector and subject to harsh working conditions. That the artisanal miners are not trained, they often do not realize that the un-supported and/or un-reclaimed pits of sub-surface mining methods they use to mine minerals are devastating their environment and their health with potential fatal consequences. In regulated mining, the mined out area is expected to be reclamation, this involves filling the excavated area with mine spoils. This is followed by restoration which involves putting the area into use once again after exploitation for activities such as fish farming, dry season farming of assorted vegetables such as pepper, beans and potatoes. Around the study area, Community Forest Area (CFA) was spotted within the mining area these are relics of reclamation activities carried out at the advent of the mining law of 1946 by the mine operators (Mallo, 2000). The forest was subsequently handed over and managed by the community (traditional rulers). The forest is harvested within a period of ten years interval and proceeds (revenue) from the forest are converted into community use and development. The reclamation activities by the natives using assorted waste and domestic materials portend future danger to the sub-surface water quality and human health. Mining operations normally upset the equilibrium in the geological environment, which may trigger off certain geological hazards such as landslide, subsidence, flooding and erosion together with their secondary effects. Since land is a non-renewable resource at a human time scale, some adverse effects of these degradation processes on the land quality of Kuru are irreversible. The productivity of some lands of these areas have declined by 50% due to soil erosion and poor crop yield. Land degradation is a decline in land quality caused by human activities which in Kuru's case, mining.

This un-scientific method of mining leaves behind devastated land area with abandoned pits and mine dumps littering the environment. This artisanal mining activity although constitutes a menace to the environment as the mined out pits are not reclaimed through adequate re-filling to forestall roof collapse leading to land subsidence, provide a formidable source of income to the miners.

Conclusion

The rudimentary sub-surface methods of mining being deployed which is un-supported, un-illuminated and un-ventilated portends serious dangers to the miners in terms of accidental roof collapses, suffocation and other forms of health hazards. This calls for some urgent need for regulation and effective monitoring by the appropriate agencies of the Federal Government in line with best practices. For the fact that the mining activity is a formidable source of earnings supporting a large population of miners in the locality, requires that the miners should be entitled to a better social contract by the government to ensure a more scientific approach that guarantees better earnings and healthier and safer working conditions and sustainable environment.

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References

Mallo S.J. (2000). Scientific Approach to Sub-Surface (Loto) Mining Technique. Journal of EngineeringTechnology and Industrial Applications. Vol.1 number 2 (pp17-22).

Mallo, SJ: Wazoh, HN, Aluwong KC and Elam EA (2011). Artisanal Mining of Cassiterite: the Sub-Surface (loto) Approach, Sheet 390, Rayfield Jos, Nigeria. Continental J. Environmental Sciences 5 (2): pp38 - 50.

Mallo, S.J. and Aluwong C.(2012). An Investigation into the Artisanal Mining of Tin and Columbite on Abandoned Mines of Bisichi Jenta Ltd, Kuru (Naraguta Sheet 168) Plateau State, North Central Nigeria. CJEarthSci/2012/080.

Ali M. (2012) Nigerian Mining Journal, 10(2) 95 - 99

Characterisation of Manganese Oxide Secondary Tailings of Ghana Manganese Company (GMC) Limited

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Abstract

The work employed mineralogical and particle size determinations using x-ray fluorescence, sieve analysis and atomic absorption spectrometry. The results, indicated that the head sample consists of 27.43 % MnO_2 (14.39% Mn) and 23.64% SiO_2; as the major associated material, while minor associations include Al_2O_3 (9.01%) and Fe_2O_3 (9.86%). The secondary tailings material grade has fallen below 83.28% MnO_2 (52.61% Mn) of the original average grade and 53.83%-69.21% MnO_2 (35-45% Mn) of the primary tailings respectively. The material also showed +850 µm as the discretely major constituent having 25.54% retained and six fractions from +2000 µm to +500 µm, mostly coarser to medium size fractions has 74.1% cumulative weight retained. These pointed to a promising outcome if further re-enrichment is desired, especially by the source company to supplement its supply to ferroalloy industries.

Keyword: Nsutite, manganese oxide

Introduction

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

By the latter part of 1980s depletion of manganese oxide ore at the Nsuta mine was declared; thereby turning attention fully on the manganese carbonate(Anon., 2010); having an average of 34.16% Mn (Kesse, 1985). According to Christie (2010) such metallurgical grade is close to ferroalloy industry's requirement of 38 – 55% Mn. Hence, tailings enrichment is imminent as converting the manganese carbonate to oxide will require extra treatment(s) such as calcination (Amankwah et al, 1999; 2005) or leaching (Sharma, 1992). Furthermore, the technological world is having difficulty of recovery of manganese oxide through recycling (Gandhi, 2010).

The processing of the manganese oxides was done using jigs (gravity method of separation) (Kesse, 1985 and Bekoe, 1994). The primary tailings generated earlier was enriched from a range of 35%-45% Mn to ≥ 50% Mn using sorting spirals (Anon, 1999).

It is against the foregoing background that work was conducted in order to characterise the secondary tailings generated from the latter enrichment in order to establish its mineralogical and particle size compositions. These are essentially required to serve as base for ascertaining its suitability as raw material for efficient re-enrichment.

Location of Nsuta Manganese Mine

Nsuta town is about 6.5 km south-east of Tarkwa, the capital of Tarkwa-Nsuaem Municipality. The Nsuta manganese deposits are located about 60 km north of Takoradi (Fig 1) and about 4 km close to the Sekondi-Kumasi rail line near Mile 34 Post from Tarkwa on its western end (Kesse, 1985). The mine is precisely 5° 17′ North and 1° 58′ West (Anon, 2011).

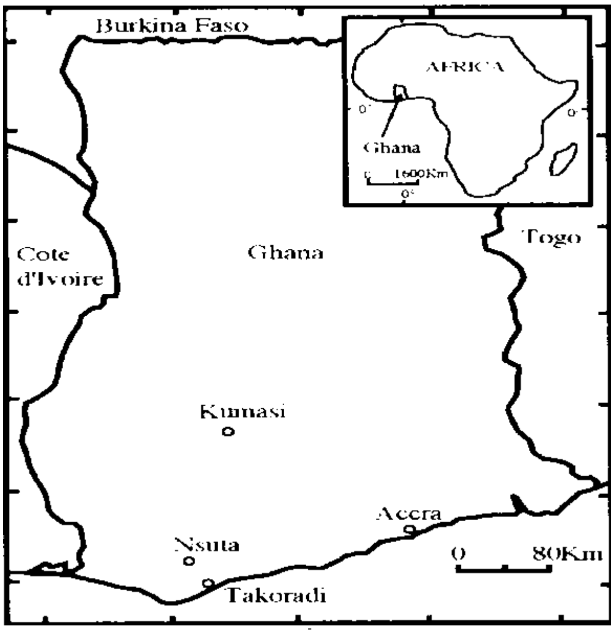

Fig1 Map of Ghana showing the location of Nsuta (Source: Nyame et al, 1995)

Methods

Samples of about 400 kg of the secondary tailings (referred to as as-received material) were obtained from the Ghana Manganese Company Limited, Nsuta, tailings dump at random points. On discovering the presence of moisture in the as-received sample materials, it was air dried for four days, at the rate of eight hours/day. This was followed by the laboratory preparation of the materials via simultaneous mixing, conning, quartering and splitting repeatedly until a manageable representative sample about 50 kg was obtained. This was further divided into two equal portions A and B using a Riffler. Portion A was taken for the characterisations, while B was kept as a backup. An X-Ray Fluorescence spectrophotometer Spectro XLAB 2000 was used in determining the major oxides in the as received tailings material head sample at the Geological Survey Department of Ghana.

1 kg of portion A was used as feed to a set of ten (ASTM E11) sieves selected based on relationship in order to make possible much closer sizing (Anon., 1977; Wills and Napier-Munn, 2006). The topmost sieve was 10 mesh (2000 μm) and the finest was 80 mesh (180 μm). The sieves were loaded with the materials and shaken with Retch sieve shaker for ten (10) minutes after which the oversize material on each sieve was weighed.

Samples from individual sieve fractions of the particle size analysis were pulverised and sieved again to < 90μm (140 mesh). A 20 ml HCl and 3 drops of HNO_3 were added to 0.5 g of the sample in a test tube and heated at 105°C on a hot plate for 10 minutes. After cooling for 15-20 minutes, each digested material was mixed with distilled water and shaken for 1 minute to attain homogenisation. The pulp was then filtered into a container using filter paper and topped up to 100 ml with distilled water. The determination of manganese content for each of the digested samples was done with Varian AA 240FS Fast Sequential Atomic Absorption Spectrometer. The machine was calibrated to the appropriate manganese wavelength of 279.5 nm and slit width of 0.2 nm before analysis.

Results and Discussions

From Fig. 2, the quantitative analysis by XRF of the head sample showed 9.01% Al_2O_3, 27.43 % SiO_2, 23.64% MnO_2 and 9.86% Fe_2O_3, while the other oxides ranged between 0.04 and 1.55%.

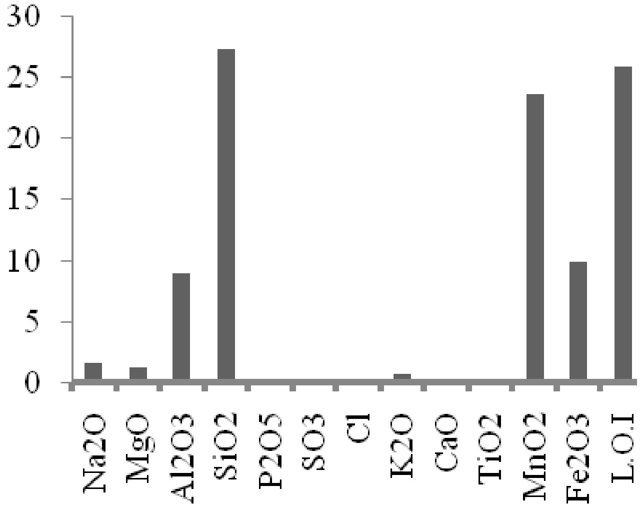

Fig.2 Percentage Compositions of the Major Oxides in the Representative Sample by XRF Analysis

The MnO_2 value of 23.64% (14.93% Mn) was lower than the average value of 83.28% (52.61% Mn) of the manganese oxide ore in the original chemical analyses (Kesse, 1985). The presence of SiO_2 up to 27.43% indicated it as the major gangue in the secondary tailings material. The other oxides such as Fe_2O_3, Al_2O_3, TiO_2, Na_2O, and MgO also exceeded their original average values of 3.88%, 2.37%., 0.07%, 0.04% and 0.09% respectively, while CaO and P_2O_5 are less than their original average values of 0.28% and 0.25% respectively (Kesse, 1985). K_2O was not detected in the original oxide samples analysis, but 0.33% of it was found in the manganese carbonate samples (Kesse, 1985). Additional work conducted by Nyame et al (1995) detected that CaO, MgO, Na_2O and K_2O all have values less than1%. The Loss on Ignition (LOI) value of 25.97% significantly acknowledged the presence of volatile matter(s).

These values signified divergence from the original ore content due to drastic fall in the % Mn content, while the other oxides' concentration is due to saturation of moisture from the tailing dumping operation. Figure 3 shows the weight percentage composition by size of the as-received material in order of magnitude; -1180 +850 μm (25.54%), -2000 +1400 μm (14.85 %), -850 +650 μm (13.97 %) and -650 +500 μm (10.59 %), while the remaining ranges had less than 10%.

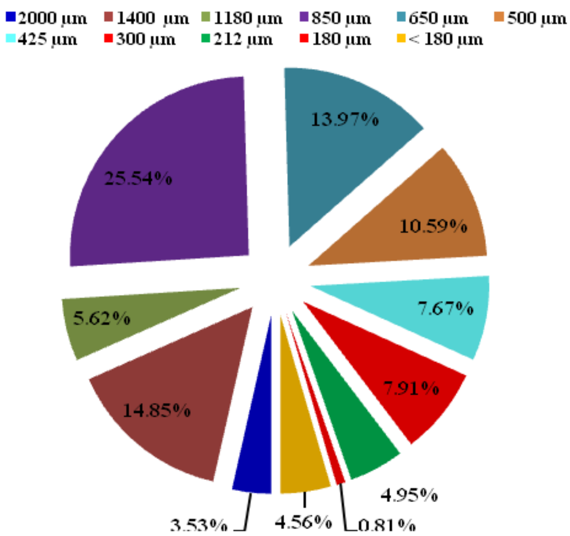

Fig. 3 Tailings Fractions Characterisation by Weight of the Representative Sample

The results translated to the fact that the composition was mostly of coarse to medium particle sizes as cumulatively 74.1% of the bulk material is made up of 2000 μm to 500 μm fractions. The remaining 25.9% covers the 425 μm to -180 μm fractions. Therefore, if the assays of the fractions' better grades fall within 425 μm to -180 μm fractions there would be the need for more energy expenditure and dust control in terms of extra comminution. It would also mean that the as-received materials were at their final scavenging stage with higher manganese concentration in the finer fractions.

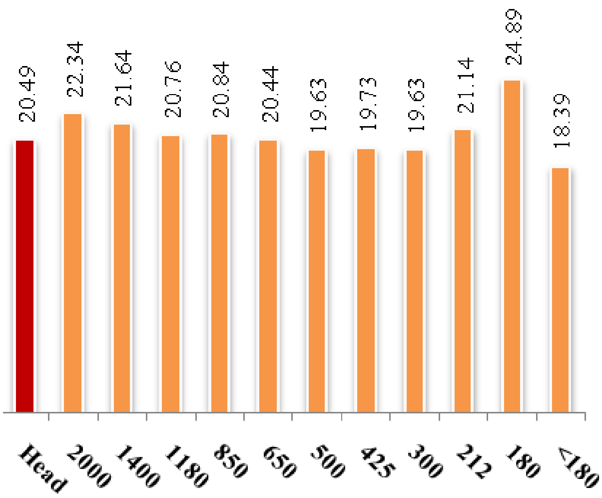

Fig. 4 Assay of the Sieved Fractions of the Representative Sample

Figure 4 depicts the AAS assay values of respective fractions ranging from a maximum of 24.89% Mn at 180 μm fraction to the lowest of 18.39 % Mn at <180 μm, whereas the fraction with the highest quantity of material (850μm) has 20.84 % Mn.

The head sample value of 20.49% Mn was deduced from calculation and found to differ from the XRF value of 14.93%. Meanwhile, it can also be seen that the fractions' maximum and minimum assays were all greater than the XRF head sample value. This could be attributed to the degree of liberation of the particle sizes at discrete fractions. A close review would also show that from 2000 μm to 650 μm fractions, the values are greater than 20% Mn and a drop to < 20% Mn from 500 μm to 300 μm, then similar rise in values with 212 μm and 180 μm and finally a drop at the -180 μm.

At this stage, if the objective was to produce aggregates for downstream consumption, then one could reduce particles to -212 +180 μm size range due to its assay content of 24.89% Mn. The latter option was because the as-received material was mostly made up of the referred fraction range. Otherwise, the entire assay values all pointed to the need for further enrichment of individual fractions.

Conclusions and Recommendations

The work conducted with the aim of characterising the manganese oxide mineral secondary tailings material deduced that there was manganese oxide in the as-received material and silica is the major associated material, while minor associations include corundum, haematite and rutile. The particle size analysis of the material gave +850 μm as the discretely major constituent having 25.54% retained and six fractions from +2000 μm to +500 μm, mostly coarser to medium size fractions has 74.1% cumulative weight retained.

Therefore, it is recommended that the GMC can proceed to recover more manganese from the secondary tailings material to supplement its supply to ferroalloy industries.

References

Amankwah, R.K., Wiafe, J. and Bekoe, A. (1999), "Production and Quality Assessment of Manganese Nodules from Nsuta Low Grade Carbonate Ore", *Ghana Mining Journal*, Vol. 5, pp. 70-74.

Amankwah, R.K. and Pickles, C.A. (2005), "Microwave Calcination and Sintering of Manganese Carbonate Ore", *Canadian Metallurgical Quarterly*, Vol. 44, No. 2, pp. 239-248.

Anon. (1977), *Test Sieving Manual*, Endecotts Ltd., London, 32 pp.

Anon. (1999), http://www.miningweekly.com/article/manganesetailings-treatment-plant. Accessed in November, 2010.

Anon. (2010), "History of Nsuta Manganese Mine", www.ghamang.net. Accessed: September, 28, 2010.

Anon. (2011), "Location of Nsuta Manganese Mine", http//www.mindat.org/maps.phy?id=17129. Accessed in March, 2011.

Bekoe, K. A. (1994), "Production and Quality Assessment of Nsuta Nodules", *Unpublished BSc Project Report*, University of Science and Technology, Kumasi, pp. 56.

Christie, T. (2010), "Mineral Commodity Report 7 - Manganese", www.crownminerals.govt.nz/cms/pdf.../report07_manganese.pdf. Accessed: September, 24, 2010.

Nyame, F.K, Kase, K. and Yamamoto, M. (1995), "Occurrence and Composition of Manganese Oxide Minerals from the Nsuta Manganese Deposit, Western Ghana", *Okayama University Earth Science Report*, Vol.2, No. 1, pp. 71-80.

Gandhi, A. (2010), *Manganese Report*, Ideas 1st Information Services Pvt Ltd, Mumbai, 72 pp.

Kesse, G.O. (1985), *The Minerals and Rock Resources of Ghana*, A.A. Balkema Publishers, Rotterdam, pp.298-339.

Sharma, T. (1992), "Physico-Chemical Processing of Low Grade Manganese Ore", *International Journal of Mineral Processing,* Vol. 35, pp. 191-203.

Wills, B.A. and Napier-Munn, T.J. (2006), *Wills' Mineral Processing Technology*, 7th edition, Butterworth-Heinemann,

Abdulraman S. O. & Olaleye, B.M. (2012) Nigerian Mining Journal, 10(2) 101 - 109
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Determination of an Appropriate Exploitation Method for Sobi and Oloje Clay Deposits for Production of Bricks and Ceramic Work; Ilorin, Nigeria

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Abstract
Current exploitation practice of Ilorin clay deposits in Nigeria has been investigated to be largely dominated by female artisans (old and young) who depend solely on crude implements such as sticks, spades and baskets with resultant premature abandonment of mined out pits. Exploratory pits were dug to probe the profile of the deposits in the two locations (Sobi and Oloje) and the borehole data of the deposits were used to produce isopach map of the deposits. Several equipment combinations were considered for the exploitation of the deposits. Stripping by bulldozer was favoured to be most appropriate.

Keywords: Exploitation, clay, deposit, production, bricks, ceramic

Introduction
Clay is one of the oldest building materials known to man and its extraction has been tailored to suit the prevailing demands (Patterson and Murray, 1983). According to Van-Amsterdam, (2008), the development of new manufacturing techniques maintains clay brick and tile as competitive building materials that have good quality, long life and minimal maintenance requirements.

The extraction of clay is in most cases directly connected to the production process. Little waste occurs because extraction is only economical where the stripping ratio is low, (Jones and Berard, 1995). Extraction of the clays may not be a continuous process, so the immediate impact and rate of change in some localities may not be seriously felt. According to Virta, (1992), when clay is compared to other industrial minerals, volumes and rates of extraction are very low.

Extraction of clays and sands has environmental impact; however, they also have potential benefits, such as the creation of nature reserves, amenity lakes and formation of repositories for various forms of waste. This is particularly useful as the impervious nature of exhausted clay pits provides an acceptable means of waste disposal. Restored land over exhausted clay pits can provide useful social amenities or be converted to agriculture or forestry use. The extraction of clay for construction products is a small percentage of the total mineral extraction, typically about 5 % (Patterson and Murray, 1983). If deposits lie deep, land usages rates are modest, whilst in delta areas extraction is managed within minimal disruption.

Extraction of clay deposits is by surface mining method (usually strip mining), in which soil and rock overlying the mineral deposit are removed. Strip mining is a type of surface mining that involves excavating earth, rock and other material to uncover a tabular, lens-shaped, or layered mineral reserve. The mineral extracted is usually rocks of sedimentary origin (Schissler, 2002). The mineral reserve is extracted after the overlaying material, called overburden is removed. The excavation of the overburden is completed in rectangular blocks in plain view called pits or strips. The pits are parallel and adjacent to each other with each strip of overburden and the mineral beneath extracted sequentially (Popov, 2001).

The uncovered mineral is excavated and hauled out of the pit to down - stream processing operations. Filling the adjacent empty pits with the overburden is systemic to the process and therefore insures the genesis of mined - land reclamation, an advantage of this method of surface mining (Matthew and Thomas, 1992).

Planning strip mining utilizes a cross - section or range diagrams of the earth to be removed and the prospect of making a profit. The variables that are involved in the economic analysis may include consideration of various mining methods/equipment combinations, mine size/equipment combination, pit layout combinations, etc. Problems could arise in areas such as geology, engineering, environmental sciences, and economics. The correct mine plan will optimize the economic return of the project (Horsley, 2002).

In this work, the stratigraphic units of Sobi and Oloje clay deposits Ilorin, Nigeria were investigated by boreholes information. The borehole data of the deposits were used to produce isopac maps to delineate beds of equal thickness in the formation. The pit sequence geometry for the extraction of clay and weathered bedrock of the deposits was determined.

Geology

General Geology of Ilorin clay deposits show that rock exposures are rare in the entire Ilorin clay deposits. Some exposures occur of coarse white granites free of ferromagnesians and quartz pegmatites close to River Fomo, in the northern fringes of Sobi area. This area is directly south of the granitic to gneissic Sobi Feldspar pebbles in excess of 30mm have been found in some borehole sections of the Sobi area, which confirms that more occurrences exist of granitic substrates either around or under the investigated area (Adeleye and Ottan, 2007).

In the southern and south-western corner of the Oloje area, minor occurrences exist of granite gneisses and quartz pegmatite dykes. Weathered gneiss outcrops also occur just to the south of the deposit. There are also localized occurrences of narrow quartz pegmatite dykes in a few cases within abandoned clay pits in the southern half of the middle Okelele deposit. Localized concentrations of quartz floats occur in a few areas within the entire clay deposits and are indications of nearness of quartz dykes and/or sills.

The ideal and complete section of the superficial deposits comprises a basal unit of weathered bed rocks (commonly above 1m thickness) overlain by a thin layer (0.1-0.2m) of mixed or inter-bedded clay and gravelly materials. This is overlain by clay beds (0.2-1.4m). The topmost unit is commonly grouped as overburden (0 - 1.5m) and comprises sandy, silty to clayed soils, cemented to uncemented rubble of weathered and unweathered rock fragments of pebble to sand - size. This last unit is unconformable above the clay, as an erosional surface separates the two. In most cases, there are no thin clayed - gravelly beds between the weathered bedrock, and the overlaying clays, hence, only three principal stratigraphic units are identifiable (Adeleye and Ottan, 2007). These are, from bottom to top as:

(a) Overburden unit
(b) Clay unit
(c) Weathered bedrock unit

However, three potential clay deposits were discovered from an age-long geological exploration of Ilorin and its environs, only two of them (Sobi and Oloje) were proved viable as a result of detailed grid drilling (Adeleye, 1992).

The dominant clay colours are various shades of grey and brown with mottled appearance very common. The colours include yellowish grey, dark yellowish-brown, pale yellowish brown, moderate brown, tending to Khaki colour is of local significance, along with dark grey and black. There is often colour 'Layering' in the field: a top greyish clay with various percentages of reddish brown stains followed by a more homogenous gray, yellowish grey. This sub-unit is often the most plastic, but it is sometimes absent in some sections. It is more prominent in a large part of the Sobi area than the Oloje area. The basal part, and the dominant, is darker coloured and displays various shades of brown and yellow including dark brown, moderate brown, dark yellowish brown, pale yellowish brown. It is only in a few localized sections that light grey clays occur below the darker clays, in the Oloje area (Adeleye and Ottan, 2007).

The clays are characteristically massive to the naked eyes. There are isolated cases of 'Pseudo-bedding' which are products of localized concentrations of dead rootlets. The false beddings often have up to 30° dip and never close to 0 - 10°. Coarse clastic elements are a common feature of the entire clays. The elements include various sizes of fresh, and ferruginised weathered rock fragments of sand to pebble sizes rarely up to 12mm size.

The Weathered Bedrocks under Ilorin clays occur directly below the clays and their associated coarse detrital grains. Drilling through the former is less noisy than through the latter. The latter contains only a few coarse particles in the top areas. They are often gradational into the overlying clays in terms of clay contents (materials less than 2 microns in size). Texturally, however, the top clayed parts of the weathered bedrock often contain ghost appearances of medium to porphyritic granites from which they were derived. This feature is accentuated by various subtle shades of grey, yellow and brown colours of this sub-unit. Rarely dark grey and black colours occur, as around Odo-Ogba. This top gradational unit shows various thicknesses ranging from 0.1m to well over 1m (proven thickness) in some localized areas. The average thickness is about 0.2m.

Below the gradational top, buff, whitish and yellowish white, very silty to sandy sub-units occur. They are of greater thickness than the gradational top. They are in various degrees or percentage of admixtures with grey and greyish clayey pockets and lenses. The general trend is that the clay contents diminish with depth towards the un-weathered bed rock. Scattered black nodules (products of diagenesis) locally occur within the gradational top; sericite often increases with depth and the greatest sericite areas match the greatest sericite areas within the overlying clays. Pseudo beddings have locally been found in the gradational top. Generally, most of the basal parts of the weathered bedrock contain up to about 15% of white Kaolinitic clay fraction. The sands and silts left are very white and appear free of labile (less chemically stable) materials.

The overburden of Ilorin clay is the most diverse unit of the three. It comprises varying proportions of loose brownish to greyish sandy-silty soil, lateritised and patchy coloured reddish brown and greyish clayey soil, loosely to strongly cemented rubble, loose coarse sands and gravels of quartz and feldspar. The rapid vertical and lateral fades changes often occur from hole to hole at the close intervals of 50m.

The coarser fractions are mostly restricted to the basal parts of the sections while the finer sands to clays generally dominate the upper parts of the overburden sections. The loose quartz gravels are also good construction materials and are more common in Oloje than Sobi area. The overburden displays wide thickness variations (Adeleye and Ottan, 2007). In Oloje area, the thickness varies from 0.2m - 1.5m. Over the economic clay block, the thickness is commonly 0.4 - 0.7m; the thickness is higher in Sobi area, being 0 - 2m and above. It is commonly 0.3 - 1m over the economic block.

Surface Mining

Surface mining is a type of mining method in which soil and rocks overlying the mineral deposit are removed. It is the opposite of underground mining, in which the overlying rock is left in place, and the mineral removed through shafts or tunnels. Surface mining is employed when deposits of commercially useful minerals or rock are found near the surface; that is, where the overburden (surface material covering the valuable deposit) is relatively thin or the material of interest is structurally unsuitable for tunnelling(as would usually be the case for sand, cinder, and gravel). Where minerals occur deep below the surface-where the overburden is thick or the mineral occurs as veins in hard rock - underground mining methods are used to extract the valued material.

Materials and Methods

The locations of the deposits were established by using Global Positioning System (GPS) and the characteristics of the deposits were confirmed from exploratory pits sparsely distributed within two different areas (Sobi and Oloje), (Figure 1). The exploratory pits of dimension 1.2m × 1.2m × 3.5m were dug and the stratigraphic units of the deposits confirmed. The borehole data of the deposits were used in preparing isopach maps of the deposits, (Figures 2-7).

Work Bench Design

In designing the bench for extraction of the clay material, the most important factor considered is the stability of the working bench, which ensures the safety of workers and equipment (Atkinson 1983). The followings parameters were also considered:

(a) The characteristics of the deposit under moisture condition (Olaleye et al., 2010).
(b) The depth and strength of the clay deposit
(c) The mining system and type of equipment and
(d) The planned volume of mining work.

Height of the Bench

The bench height was determined by the size of the pay-loader employed and mining system. The bench height was taken to be 3.5m and this ensures the required safety of mining work, high productivity of the pay-loader and planned volume of mining work.

Working Slope

The working slope of a bench was selected after careful consideration of all activities that would take place in the bench. According to Horace (1989), working slopes between 18^0 - 25^0 are normally allowable for the stability of benches for clay deposits. This range is suitable after due consideration of the geological conditions and the characteristics of the clay deposit. A working slope of 22^0 is chosen for the bench and with this the stability of the bench is assumed under dry and wet conditions.

Loader Capacity

Several equipment combinations were considered for the exploitation of the deposits. The certified capacity of a loader is based on the coefficient of extraction and utilization, bucket capacity and distance of haulage. The capacity was calculated using equation (1):

$$Q_L = \frac{60EKeKu}{T_L}, \text{ t/h} \quad . \quad . \quad . \quad 1$$

where $_LQ$is the certified capacity ,t/h; E is the bucket capacity (4t); K_e is the coefficient of excavation (0.85); K_u is the coefficient of utilization (0.83); and T_L is the total cycle time of the loader (min).

The capacity of the loader was determined as:

$$Q_L = \frac{60\times 4\ (0.83\times 0.85)}{4.64} = 36.5\text{t/h} \quad . \quad . \quad . \quad 2$$

It should be noted that the output capacity of the loader when used for extraction, haulage and dumping operations is 36.5t/h. This capacity is about one-third of the expected output capacity of 110t/h when a combined action of one loader and two dump trucks are used for the same cycle operation. If however, only the pay loader is employed for the mining operation, the pit will be underutilized, that is, only one-third of the expected production capacity would be met, unless three loaders of the same bucket capacity are employed for the operation.

Comparatively, mining the deposits with three loaders or one loader and two dump trucks to meet the production requirement of 110t/h, may be assumed to be the same economically in terms of capital costs, labour costs, operational and maintenance costs. But in haulage operation, the advantages of trucks cannot be ruled out and therefore is preferred to that of loaders. Consequently, exploitation of the clay and weathered bedrock deposits is more feasible with the use of a loader and two dump trucks in order to meet the production requirement.

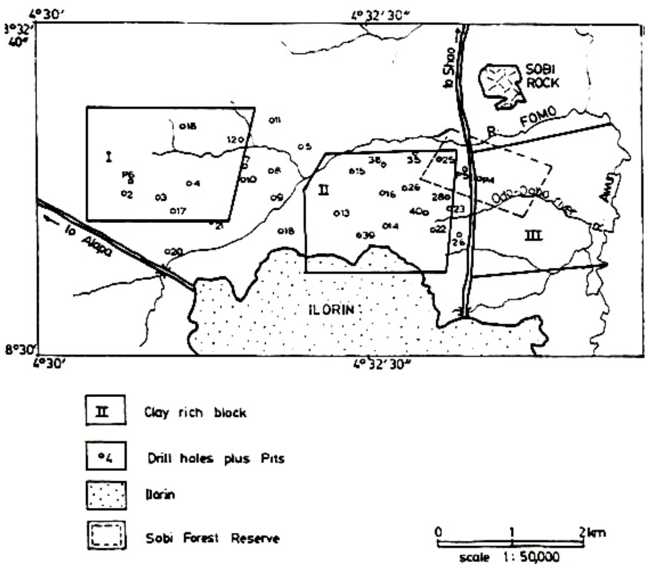

Figure 1: Map of the main clay deposit areas of Ilorin, Nigeria.

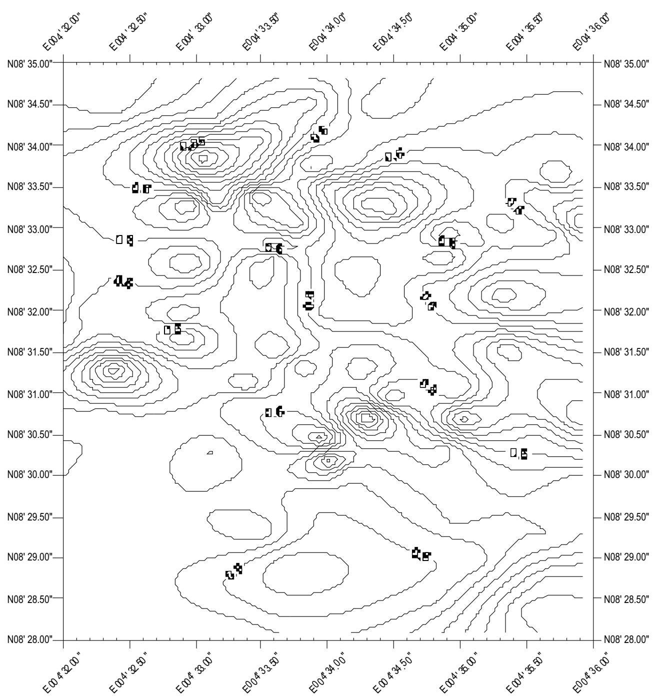

Figure 2: Isopach showing clay distribution in Sobi area

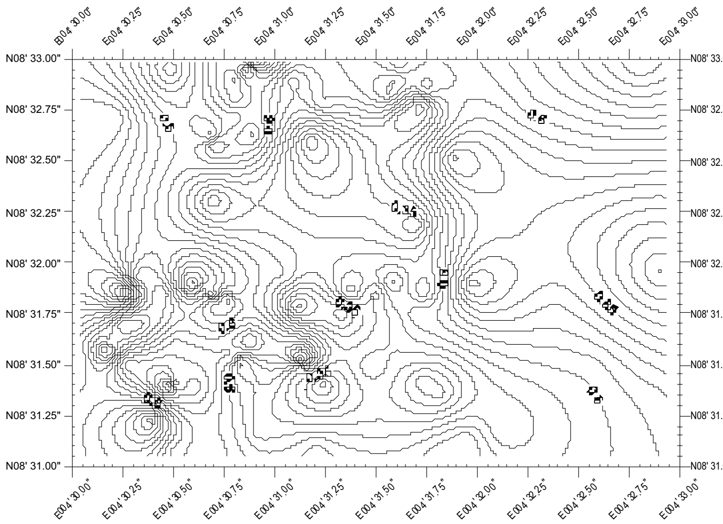

Figure 3: Isopach showing clay distribution in Oloje area.

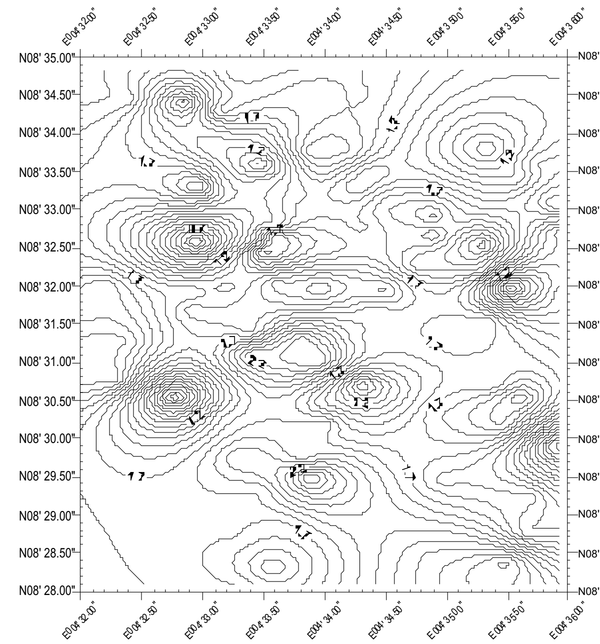

Figure 4: Isopach showing weathered bedrock distribution in Sobi area

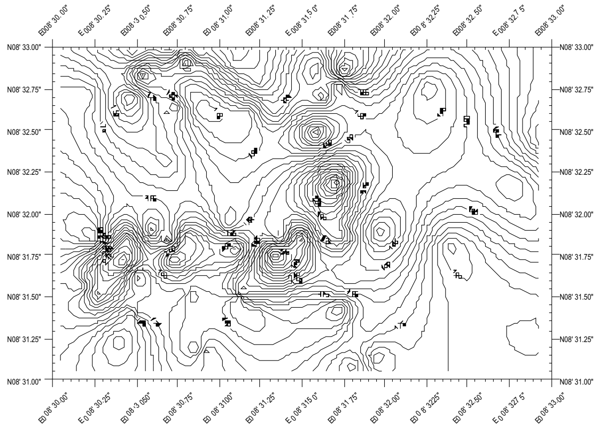

Figure 5: Isopach showing weathered bedrock distribution in Oloje area

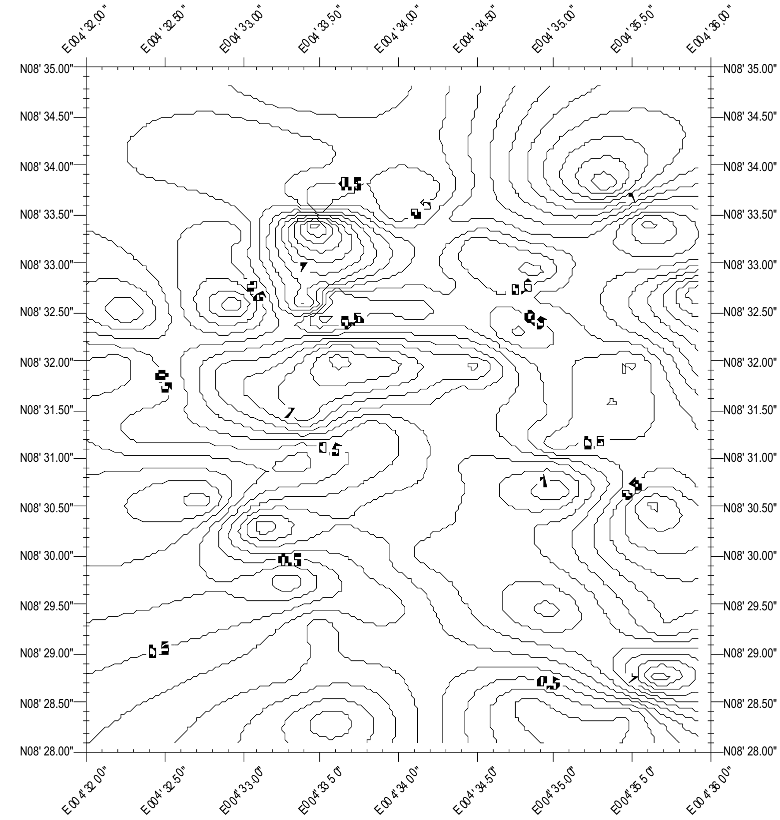

Figure 6: Isopach showing overburden distribution in Sobi area

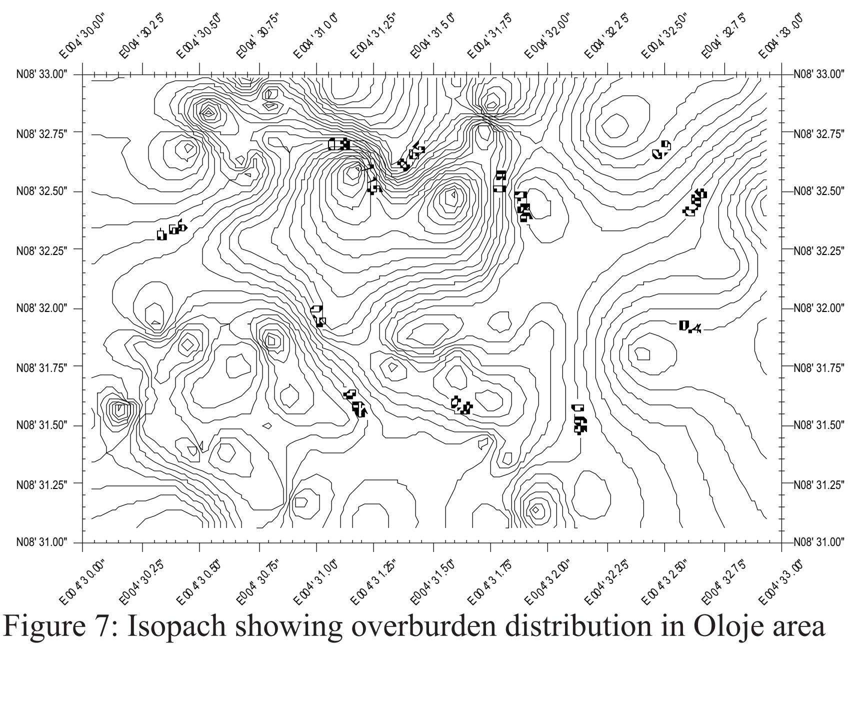

Figure 7: Isopach showing overburden distribution in Oloje area

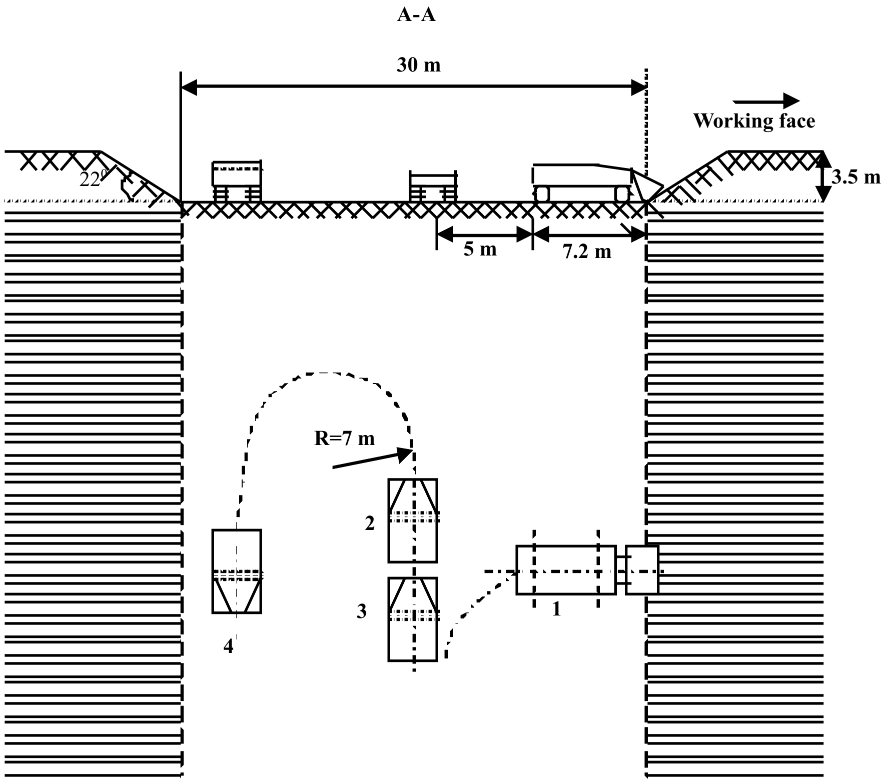

Figure 8: Cross-section of the exploitation method

Results and Discussion

Two economic blocks were delineated and the reserves are the minimum possible for the two economic blocks (Sobi and Oloje clay blocks), based on the detailed drilling. They are proven reserves based on the occurrences of almost continuous sheets of clay with a minimum thickness of 0.5m. The reserve estimate of the two economic clay blocks under study is indicated in Tables 1 and 2 while Table 3 shows the overburden reserves of the areas.

The available volume of weathered bedrock, in total is 565,970 m^3. The gross total reserve of clay and weathered bedrock under Oloje and Sobi economic blocks amount to 799,940 m^3. The overburden reserves imply that a total of 202,512 m^3 of soil, rubbly materials, sands and silts have to be stripped off in order to quarry all the clays and required weathered bedrocks under them in the two economic blocks (Sobi and Oloje clay and weathered bedrock deposits).

Table 1: Summary of proven clay reserves in Sobi and Oloje areas

Location	Area (Ha)	Average thickness (m)	Volume (m^3)
Oloje 'A' Area	15.80	0.70	110,600
Sobi 'F' Area	18.98	0.65	123,370

Table 2: Summary of proven weathered bedrock reserves in Sobi and Oloje areas

Location	Area (Ha)	Average thickness (m)	Volume (m^3)
Oloj'e'A' Area	15.80	1.60	252,800
Sobi 'F' Area	18.98	1.65	313,170

Table 3: Summary of proven overburden reserves in Sobi and Oloje areas

Location	Area (Ha)	Average thickness (m)	Volume (m^3)
Oloje 'A' Area	15.8	0.597	94,326
Sobi 'F ' Area	18.98	0.57	108,186

Conclusion

Based on the characteristics and geometry of Sobi and Oloje clay deposits; the most appropriate exploitation method for the reserve is stripping by bulldozer. Strips are parallel and adjacent to each other with each strip of overburden extracted sequentially. Filling of adjacent empty pits with the overburden is systemic to the process and insures the genesis of mined-land reclamation. The disparity in the sizes of the Oloje 'clay' and weathered bedrock samples as compared with Sobi, suggests that more care has to be taken in the striping of Oloje materials to ensure consistency in the mix for ceramic works

References

Adeleye R. (1992): Clay Occurrences within 250km of Ilorin. (unpublished report), pp. 75

Adeleye R. and Ottan A.S. (2007): Clay Investigation for Special Structural Clay Blocks in Ilorin, (unpublished report), pp. 66

Atkinson T. (1983): Surface Mining and Quarrying. Institute of Materials and Metallurgy. pp.147-152

Horace, K. C. (1989): Excavation Handbook. McGraw-Hill, New York. pp. 410.

Horsley T. P. (2002): Dollar Driven Mine Planning: The Corporate Perspective to Operational Mine Planning, AMC References Material, AMC Consultants, Australia, www.amcconsultants.com.au, 9p.

Jones J.T and Bernard M.F. (1995): Ceramics, Industrial Processing and Testing, Iowa State University Press, Amsterdam. pp. 21

Matthew H. and Thomas A. (1992): SME Mining Engineering Handbook, 2nd Edition, Vol. 2. pp. 1298-1303.

Olaleye B.M., Abdulrahaman S. O. and Saliu M.A. (2011): Investigation of Some Physical and Strength Properties of Ilorin Clay and Weathered Bedrock for Production of Hydraform Bricks, Journal of Eng. and Eng. Technology (FUTAJEET), Vol. 6, No 2, pp. 1-10.

Patterson S. H. and Murray H.H. (1983): "Clays", Industrial Minerals and Rocks, Society of Mining Engineers, New York, Vol. 4, pp. 722

Popov G. (1971): The Working of Mineral Deposits (Shiffer, V. Translator), Mir Publishers, Moscow. pp. 597

Schissler A. P. (2002): “Design and Methods of Coal Mining”, In The Encyclopedia of Energy, Elsevier Inc., Kidlington, Oxford, Volume 1, pp. 485-494.

Van-Amsterdam E. (2008): Construction Materials for Civil Engineering, JUTA & Co. Ltd, Kenwyn, South Africa. pp 258

Virta R.L. (1992): Clays, Annual Report (Draft), Bureau of Mines, U. S. Department of the Interior, Washington. pp. 13.

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