CHARACTERIZATION AND BENEFICIATION OF ZURAK LEAD-ZINC ORE DEPOSIT, PLATEAU STATE

BY

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DECLARATION

I declare that the work in the thesis entitled ‘Characterization and Beneficiation of Zurak Lead-zinc Ore Deposit, Plateau State’has been performed by me in the Department of Metallurgical and Materials Engineering. The information derived from the literature has been duly acknowledged in the text and a list of references provided. No part of this thesiswas previously presented for another degree or diploma at any university.

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CERTIFICATION

This thesis entitled 'Characterization and Beneficiation of Zurak Lead-zinc Ore Deposit, Plateau State' by Yunana Solomon MASOK meets the regulations governing the award of the degree of Master of Science in Metallurgical and Materials Engineering of Ahmadu Bello University, Zaria, and is approved for its contribution to knowledge and literary presentation.

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DEDICATION

This thesis is dedicated to God and my wonderful parents, Late Mr. Solomon Masok and Mrs. Esther S. Masok.

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ABSTRACT

In this work, the characterization and beneficiation of Zurak lead-zinc ore deposit was carried out. The sample of the ore was sourced from Zurak village in Plateau state. The sourced sample was crushed and pulverized using laboratory jaw crusher and pulverizing machines. The pulverized sample was characterized to determine its chemical and mineralogical compositions, liberation size, work index and beneficiated to metallurgical grade using the most suitable concentration method adopted for the ore. The result of the chemical analysis of the ore using X-ray fluorescence revealed that the run-of-mine ore contained 6.33%Pb and 4.65%Zn on the average, which meets the typical standard feed grades of 1 – 5%Pb and 1 – 10%Zn usually required for mining lead-zinc ore. The mineralogical analysis using X-ray diffraction technique revealed that the ore contained 7.11% galena (PbS) and 7.51% sphalerite (ZnS) as the major valuable minerals; 3.04% magnetite (Fe_3O_4), 1.5% rutile (TiO_2) as minor minerals; and 65.65% quartz (SiO_2) as the main gangue. The liberation size of the ore was found to be -90 +63μm and work index of the ore was determined as 11.41kWh/tone using the Bond modified method. Froth flotation method was adopted for the beneficiation of the ore and two batches of tests were conducted on the ore. For the first batch of the test, -90 +63μm was used, lead and zinc concentrates assaying 63.02%Pb and 57.92%Zn with recoveries of 93.41% and 76.08% were obtained, while for the second batch -180 +125μm was used and lead and zinc concentrates assaying 55.79%Pb and 30.25%Zn with recoveries of 70.29% and 52.38% were obtained respectively. Based on the results obtained from the tests conducted, Sullivan concentrator of Cominco Ltd, British Columbia flowsheet was adopted and modified as the conceptual flow-sheet for the beneficiation of the Zurak lead-zinc ore. It was therefore concluded that froth flotation is the most suitable method of beneficiating the Zurak lead-zinc ore deposit at a liberation size of -90 +63μm.

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ABBREVIATION AND NOMENCLATURE

Abbreviations

AAS	Atomic Absorption Spectrophotometer
FGN	Federal Government of Nigeria
GDP	Gross Domestic Product
MMSD	Ministry of Mines and Steel Development
MSMD	Ministry of Solid Minerals Development
NEITI	Nigeria Extractive Industries Transparency Initiative
RMRDC	Raw Materials Research and Development Council
SEM	Scanning Electron Microscopy
USA	United States of America
USEPA	United States Environmental Protection Agency
XRD	X-ray Diffraction
XRF	X-ray Fluorescence

Nomenclature

c	Velocity of light (m/s)
D_h	Specific gravity of the heavy mineral
D_l	Specific gravity of the light mineral
D_f	Specific gravity of fluid (water)
E_1	Former energy level of electron before filling vacancy
E_2	New energy level of electron after filling vacancy
E_r	Energy input in grinding reference ore (Kwh/tonne)
E_t	Energy input in grinding test ore (Kwh/tonne)
E_u	Energy input in grinding unknown ore (Kwh/tonne)
F_r	80% passing feed of reference ore (μm)

F_t	80% passing feed of unknown ore (μm)
F_u	80% passing feed of unknown ore (μm)
h	Plank's constant (J/s)
P_r	80% passing product of reference ore (μm)
P_t	80% passing product of unknown ore (μm)
P_u	80% passing product of unknown ore (μm)
S	Specific gravity
w	Weight (g)
v	Volume (cm^3)
W_i	Work index (Kwh/tonne)
W_{ir}	Work index of reference ore (Kwh/tonne)
W_{iu}	Work index of unknown ore (Kwh/tonne)
W_{it}	Work index of test ore (Kwh/tonne)

CHAPTER ONE

1.0 INTRODUCTION

Solid minerals development plays very significant role in the economies of both the mineral-rich and consuming countries. While the few developed mineral-producing countries such as Australia, Canada, and United States of America (USA) have benefited greatly from mining and mineral processing and utilization, only a handful of their counterparts in the developing regions have succeeded in transforming the revenues accruing from mining into concrete and demonstrable sustainable development (NEITI, 2010).

1.1 Background of the Study

The new world economic order and the demand for mineral commodities has risen over the last few years largely due to the growth of the emerging economies of Asia, thus raising the global mineral commodity prices, with worldwide increases in exploration activities. This has necessitated the Federal Government to develop a policy framework that will ensure an efficient and effective utilization of the nation's mineral resources in order to accelerate economic growth of Nigeria through the provision of foreign exchange earnings. In addition, solid minerals will provide local raw materials for industries and bring vital infrastructure and wealth to rural areas.

Nigeria is endowed with several mineral resources that are widely distributed. There are over thirty-four mineral commodities spread across the entire country, which include gold, coal, bitumen, iron ore, tantalite/columbite, lead-zinc sulphides, barytes, cassiterite, gemstones, limestone, talc, feldspar and marble (MMSD, 2009).

Lead is widely used in the electrical industry (as storage batteries and in sheathing electric cables); mechanical industry (as radiator shielding and paints); metallurgical industry (bearing and fusible alloys); chemical industry (as paints and pigments, lining pipes); pharmaceutical industry, medical industry (as x-ray apparatus); oil industry (as anti-knock component)(Microsoft Ecarta, 2009). It is also used in the military industry (as ammunitions); nuclear industry (as a protective shielding around nuclear reactors, particle accelerators, x-ray equipment, and containers used for transporting and storing radioactive materials);and other sectors (Encyclopædia Britannica, 2010).

Similarly, zinc products are used in several major industries, including construction (for galvanizing and steel manufacturing); transportation; machinery manufacturing; electrical, chemical and other industries (USEPA, 1994). Zinc oxide is well suited for use in rubber, paints, inks, dyes, oil additives, wood preservatives, fungicides, varnishes, and linoleums(USEPA, 1994). Zinc oxide is used in these products because of its opacity to ultra-violet light and high refractory index, durability, and electrostatic properties(USEPA, 1994).

1.2 Statement of the Problem

The beneficiation route of any ore deposit must be designed to suit its peculiarities. This is because the geological characteristics of every mineral deposit in the earth crust are unique. This is applicable to the Zurak lead-zinc ore. Presently, Zurak ore deposit has no reliable data that describes its mineralogical characteristics and its beneficiation route. Hence, this research work sought to address the following concerns:

1. develop the appropriate technology for the beneficiation of Zurak ore deposit from low grade to metallurgical grade,
2. ensure acquisition of appropriate skills by local technologists and engineers.

1.3 Justification of the Research

As a result of government policies on privatization of the solid mineral industry and the reforms in the solid mineral sub-sector in order to diversify the Nigerian economy, the nation will require the beneficiation of lead-zinc ore deposits for use as alloying elements in the local iron and steel industry.

The exploitation of the mineral endowments spread across the nation for the creation of wealth for all Nigerians, and generating employment, reducing poverty, promoting rural economy and significant contribution to our Gross Domestic Product (GDP) partly necessitated the consideration of this research work.

1.4 Aim and Objectives

The aim of this research work was to characterize and beneficiate Zuraklead-zinc ore (longitude $10°28^{1}$E to $10°52^{1}$E and latitude $9°15^{1}$N to $9°16^{1}$N) in Plateau State, and this was achieved through the following objectives:

1. determining the chemical and mineralogical characteristics of the ore,
2. determining the liberation size and work index of the ore,
3. determining the suitable concentration method for the ore,
4. the use of the data generated from 1 – 3above, to develop a suitable flow sheet for the beneficiation of the ore to metallurgical grade.

1.5 Scope of the Research

The research work involved field work to collect representative samples of the Zurak ore deposit; laboratory analysis to determine the chemical and mineralogical composition of ore; the concentration of the ore using froth flotation method; and the development of a conceptual flowsheet for the ore deposit.

The ore's work index and size/assay analysis were also determined. Available geological data as well as appropriate literature on lead-zinc ore processing were collated and reviewed.

CHAPTER TWO

2.0 LITERATURE REVIEW

2.1 Previous Works on Lead-Zinc Ores

Damisa (2008) worked on the process design for the beneficiation of Nahuta lead deposit in Bauchi State. He concluded that the major economic minerals present in Nahuta deposit are Lead and copper in the forms of galena (PbS) and covellite (CuS); that bulk floatation followed by differential floatation is best to achieve high recovery in the beneficiation of the ore. Ajayi (2005) used froth floatation to concentrate Abakaliki lead-zinc sulphide ore deposit. Greg (1993) worked on the oxidation and chloride leaching of sulphide concentrates. He concluded that all chloride systems studied can be used to leach base metals from sulphide concentrates with metal extractions of greater than 80% being achieved at 90^0C with a retention time of 24 hours, but complete metal dissolution cannot be obtained and chlorine cannot be used to extract nickel, copper or zinc preferentially from pentlandite, chalcopyrite or sphelarite.

2.2 Morphology of Lead-Zinc Minerals

Although more than 2,800 mineral species were identified, only about 100 are considered ore minerals. There are more than 60 known lead-containing minerals but the most important primary ore of the metal is the lead sulphide, galena (Encyclopædia Britannica, 2010). Galena (PbS) often contains silver, zinc, copper, gold, cadmium, bismuth, arsenic, and antimony. Other commercially significant lead-containing minerals are cerussite ($PbCO_3$) and anglesite ($PbSO_4$). These are known as secondary minerals because they are derived from galena through natural actions, such as weathering (Encyclopædia Britannica, 2010).

Galena is very widely distributed, and has up to 86.6%Pb, with a specific gravity of 7.4 – 7.6. It occurs in veins, often with sphalerite, pyrite, chalcopyrite, tetrahedrite, and gangue

minerals such as quartz, calcite, dolomite, barite, and fluoride. It is also found in pegmatites, and as replacement bodies in limestone and dolomite rocks, with garnets, feldspar, diopside, rhodonite, and biotide. Galena often contains up to 0.5% Silver, and is important source of the metal (Wills, 2006).

2.3 Nigerian Lead-Zinc Mineral Deposits

Lead-zinc ore deposits abound in Nigeria. It is estimated that there are at least 30 lodes with an aggregate length of about 6000m (MSMD, 2006).The lodes cover over 560Km in a narrow belt from Adun in Cross River State through Isiagu in Imo State and Abakaliki in Ebonyi State, to Arufu in Taraba State to Wase and Zurak in Plateau State and Gwona in Bauchi State (RMRDC, 2004). Other deposits are located in Abia, Adamawa, Bauchi, Benue, Enugu and Nassarawa. The total estimated reserve is 20,000 million tons (RMRDC, 2003).

Table 2.1: Lead-zinc ore deposits in Nigeria

S/No	State	Location	LGA	Total (tons)
1	Ebonyi	Ishiagu Abakaliki Ameri Ameka	Ivo Abakaliki	1,040,000
2	Enugu	Enyigba Nyeba	Umulopa	450,000
3	Plateau	Wase Zurak	Wase	Na
4	Nasarawa	Akwanga Arufu zone	Akwanga	Na
5	Cross River	Adua Oban	Akamkpa	150,000
6	Bauchi	Gwona	Alkaleri	Na
7	Adamawa	Numan	Numan	Na

Source: RMRDC, 2004 inTechnical Brief on Minerals in Nigeria-Lead/Zinc. No 16(NEITI, 2010).

2.4 Mineral Processing Conceptual Parameters

Ores are usually subjected to some concentration processes that can separate the various minerals into two or more products. This is done for the purpose of beneficiating the ore for its desired metallic value (Sirajo, 2008). Beneficiation of minerals is a chemical process of improving the inherent properties and quality of an ore in order to maximize its utility (Olokesusi, 2010). Separation is achieved by utilizing some specific differences in physical or chemical properties between the valuable and gangue minerals in the ore (Sirajo, 2008).

After geological survey has been carried out, the standard procedure usually followed in the development of conceptual flowsheet for the beneficiation of a newly discovered ore deposit are as follows (Yaro, 1997 in Thomas, 2007):

1. Chemical analysis of a representative sample of the ore to establish the chemical composition of the ore and also to confirm the result of the geological investigation and probably mining engineers who had already work on the ore deposit.
2. Size/assay analysis to establish the distribution of both the valuable and gangue minerals in the various size fractions.
3. Mineralogical analysis of the ore to establish the major, minor and trace minerals in the ore, allocation of each element to each mineral, the grain size of each mineral in the ore (particularly the mineral of economic importance), and the degree of association of the valuable and gangue minerals.
4. Liberation studies to establish the liberation size of the individual mineral ore; here emphasis is placed on the grain size of the valuable minerals and probably the gangue which may likely affect the subsequent separation process.

5. Determination of the work index of the ore which will be used in calculating the energy requirements for the comminution of the ore from a certain size to any size and for the selection of the appropriate comminution equipment for the ore.
6. Concentration test: the method of concentration to be used in the beneficiation of the ore will depend on the information obtained in 1 – 4 above.

From the information above, the size fraction to which the ore will be ground in order to liberate all the minerals is obtained, and also how the different minerals in the ore are going to affect the concentration process.Meaning whether there will be interaction between the valuable mineral and other associated minerals, i.e. whether it is finely disseminated, or it is an inclusion in the rock, or intimately associated. From this also, the feasible method of concentration of the valuable minerals will be conceived depending on the size fraction in which the mineral of interest lies. This will also show whether waste rejection technique is feasible and how much of the material will be lost with the waste. If it is found out that the minerals are liberated at coarse sizes

, the gravity method of separation can be used in order to separate the valuable minerals from the gangue. To do this at laboratory scale, heavy liquid analysis will be carried out on a representative sample ground to reasonable size fractions. Assay of the different fraction obtained at different densities will be carried out, and then a reasonable density of separation is selected. Alternatively, if the valuable mineral ore can only be liberated at fine particle size, then the possibility of using froth floatation technique will be considered and laboratory scale test will be carried out (Yaro, 1997).

Beneficiation processes may be classified into:

1. Physical separation: Gravity, magnetic, froth floatation (fine particle separation method), electrostatic or high tension and sorting, and optical.

2. Chemical separation: Leaching, precipitation, solvent extraction, carbon adsorption, electro-winning, autoclaving, chlorination.
3. Biological separation: Bacterial leaching
4. Pyro-metallurgical separation: Thermal treatments, which may include roasting, calcination, pelletizing and sintering.

2.5 Comminution

This is one of the two fundamental operations in mineral processing carried out in order to liberate valuable minerals which are finely disseminated and intimately associated with the gangue before separation can be effected (Thomas, 2007). Comminution encompasses all the terms, crushing, grinding and other words or phrases associated with the size reduction or severance of ores. It is mainly concerned with the relationship between energy input and the product particle size produced from a given feed size. Throughout the process industries, solids are reduced by different techniques for different purposes. Huge chunks of crude ores are crushed to manageable size that they can be further worked on. Reducing the particle size also enhances the reactivity of solids; it allows separation of undesirable constituents by mechanical means; and it reduces the bulk of fibrous substances for easy working (Gupta, 2003).

Major purposes of comminution are:

1. Liberation of one or more economically important minerals from the gangue components in an ore matrix.
2. Exposure of a large surface area per unit mass of material to facilitate some specific chemical reaction, such as leaching.
3. Reduction of the raw material to the desired size for subsequent processing or handling.
4. Satisfaction of market requirements concerning particle size specifications.

2.5.1 Particle size analysis

This is the most important method of size analysis and is accomplished by passing a known weight of sample material successfully through finer sieves and weighing the amount collected on each sieve to determine the percentage weight in each size fraction. Sieving is carried out wet for fine particles, and dry for coarse particles. The sieves are usually agitated to expose all the particles to the openings (Wills, 2006).

The process of sieving may be divided into two stages; first the elimination of particles considerably smaller than the screen apertures, which should occur fairly rapidly, and secondly, the separation of the so-called "near size" particles, which is a gradual process rarely reaching final completion. Both stages require the sieve to be manipulated in such a way that all particles have opportunities of passing the apertures, so that any particle which may blind the apertures may be removed. The effectiveness of a sieving process depends on the amount of material put on the sieve (the "charge"), the sieving time, and the type of movement imparted on the sieve (Finch, 1982 in Sirajo, 2008).

2.5.2 Size/assay analyses

The size/assay analyses are usually carried out to determine the distribution of various minerals specifically valuable and major valuable metal bearing minerals of the ore in the various size fractions. The analysis is used to obtain information on the possibility of using waste rejection techniques to reject the gangue minerals at the coarse size as much as possible. It is also used to obtain clues on the liberation size of the various minerals in the ore and their degree of association (Yaro, 1997 in Thomas, 2007).

2.6 Gravity Concentration

Gravity concentration is the processes for the separation of minerals of different specific gravities by inducing variable movement in response to gravitational force and one or

more natural or applied force with the aid of a flowing film. It is widely used, sometimes along with other processes in particle floatation, magnetic separation and chemical treatment. The method is adapted for fine-grained ores of size range between 1.5 and 0.1mm. Usually, the ore is ground to such a degree that the metal bearing constituents are mechanically liberated from the gangue. The most important unit which is often used on an industrial scale is the spiral separation, i.e. Humphrey's Spiral, due to its simplicity and environmental friendliness (Wills, 2006). The major demerit of gravity separation process is the recovery of fines and it is inherent in the process itself, even with advanced (or more sophisticated) slime gravity concentrators, the practical lower limit of particle size which can be handled is still about 10μm (Yaro, 1997).

Gravity separation processes are effective, practical and efficient in the treatment of many ores. But their applicability to the treatment of any specific ore or in combination with other processes must be determined through the mineralogical composition and characteristics of representative samples of the ore and from careful and comprehensive testing (Yaro, 1997).

2.7 Flotation

Flotation process has proved to be very efficient and effective for the concentration of galena and many other sulfide ores. The separation is achieved based on differences in physicochemical surface properties of the individual mineral constituents, particularly, the wettability by water (Yaro, 1997).

Floatation method is adapted to a much greater extend for the separation of residual gangue from concentrates and of metal sulfides or phosphorous minerals from concentrates already produced (Sirajo, 2008).

The effectiveness of the technique, however, is dependent on four factors related to the nature of the ore body: the degree of oxidation of the ore, the abundance of iron sulfides, the presence of copper minerals, and the basic or acidic nature of the gangue(Weiss, 1985).

In addition to the factors related to the mineralogy of the ore, other factors are also important to the success of flotation activities. These factors include uniformity of particle size, use of reagents compatible with the mineral, and water conditions that will not interfere with the attachment of the reagents to the mineral or air bubbles(USEPA, 1982).

2.7.1 Froth flotation

This is both a physical and chemical process of concentrating finely ground ores that depends on the selective adhesion to air of specific minerals while others remain submerged in the pulp. This process involves treatment with suitable reagents of an ore pulp to create conditions favorable for the attachment of certain hydrophobic or aerophilic mineral particles to air bubbles and render the other solids hydrophilic or water loving, submerged in the solution (Thomas, 2007). The bubbles carry the selected minerals to the surface of the pulp and form a stabilized froth which is skimmed off, while the other minerals remain submerged in the pulp (Wills, 2006).

The importance of flotation in modern mineral processing practice results from a number of factors which include some of the following reasons. It is necessary for mining and treating ores in which the minerals are finely disseminated and require further grinding to liberate the valuable minerals. In many complex ores it is desirable to separate the concentrate to facilitate subsequent treatment, e.g. galena (PbS), sphalerite (ZnS), pyrite

(FeS)and other mineral separation are desired, but the spread of density between these minerals is so small that gravity method is unsatisfactory.

The success of froth flotation operation lies in the correct selection of the reagents to be added to the aqueous medium. Some of these reagents affect processes in the aqueous phase, while others affect them in the froth phase (Ajayi, 2005).

2.7.2 Chemistry of flotation

Flotation agents or reagents are substances added to the mineral pulp prior to or during floatation in order to make possible or facilitate the process. The flotation agents are known as collectors, depressants, frothers and modifiers (Damisa, 2007).

Below is a description of the function ofeach group(U.S. EPA, Office of Water, 1982; Weiss, 1985; U.S. DOI, Bureau of Mines, 1985a):

1. Collectors (promoters):- Cause adherence between solid particles and air bubbles in a flotationcell. Examples are:
    i. Xanthates: potassium amyl xanthate (PAX), potassium isobutyl xanthate (PIBX), sodium isoppropyl xanthate (SIPX), sodium ethyl xanthate (SEX), potasium ethyl xanthate (PEX).
    ii. Dithiophosphates: thionocarbamates, xanthogen formates, thiocarbanilide.
2. Frothers: - Stabilize air bubbles by reducing surface tension, thus allowing collection of valuable material by skimming from the top of the cell. Examples are pine oil, alcohols like methyl isobutyl cabinol (MIBC), polyoxyparafins, cresylic acid (xylenol).
3. Modifiers (Activators): - Promote flotation in the presence of a collecting agent when added to mineral pulp.pH modifiers such as lime (CaO), soda ash (Na_2CO_3), caustic soda (NaOH); acids like sulfuric acid (H_2SO_4) or hydrochloric acid (HCl).

i. Cathionic modifiers: Ba^{2+}, Ca^{2+}, Cu^{+}, Pb^{2+}, Zn^{2+}, Ag^{+}.

ii. Anionic modifiers: SiO_3^{2-}, PO_4^{3-}, CN^{-}, CO_3^{2-}, S^{2-}.

iii. Organic modifiers: Dextrin, starch, glue.

4. Depressants - React with particle surfaces in the flotation cell to keep materials from remaining in the froth. Instead, materials fall to the bottom as tails. Example is copper sulphate ($CuSO_4$).

For example, flotation of galena in the presence of sphalerite is possible by adding sodium cyanide (NaCN) and xanthate. Sodium cyanide depresses the sphalerite but not galena. After the galena has been collected from the froth, sphalerite can be floated by adding copper sulfate ($CuSO_4$) and xanthate (Considine, 1975).

Several factors are important when conditioning the ore for flotation with chemical reagents. These include thorough mixing and dispersal of reagents through the pulp; repeated contact between the reagents and all the relevant ore particles; and time for the development of contacts with the reagents and ore particles to produce the desired reactions. Reagents may be added in a number of forms, including solid, immiscible liquid, emulsion, and solution in water. The concentration of reagents must be closely controlled during conditioning; adding more reagent than is required may retard the reaction and reduce efficiency(Fuerstenau, 1976).

2.7.3 Floatation concepts

The flotation operation comprises the following successive steps.

1. The raw ore material is ground usually to fine mesh sizes.
2. The pulp density is adjusted to a consistency of 15 to 35% solids and the resulting feed is fed to the flotation cell.
3. Addition to the pulp of various reagents which have a number of functions is effected.

4. Aeration, either by agitation or by air injection, that impregnates the pulp thoroughly with bubbles, which are essentially gas-filled holes of macroscopic dimensions in the pulp, is carried out.
5. The attachment of minerals to the bubbles, which is the heart of the flotation process, occurs. Once this occurs, gravity completes the separation readily. In the present example, galena attaches itself to the bubbles and thereby its effective density is drastically reduced. This causes galena to buoy up in spite of the fact that it is, in reality, much heavier than the siliceous gangue.
6. The bubbles are collected as froth at the surface.
7. The galena-bearing froth is removed from the top and the impoverished tailing is drawn away from the bottom of the cell. A device in which all these processes take place is called a flotation cell.

2.7.4 Flotation circuit for complex sulfide minerals

For complex or polycrystalline sulfide ores containing copper, lead, zinc and iron, floatation is the only method that can be used to beneficiate the ores and wide variety of flow sheets are in use, some involving sequential or differential floatation; others bulk floatation of copper and lead minerals followed by single mineral separation (Wills, 2006).

The major processing problems encountered are related specifically to the mineralogy of the assemblies. Due to the extremely fine dissemination and interlocking of the minerals, extensive fine grinding is often needed, usually to well below 75μm (Wills, 2006). Complex ores usually behave differently from single minerals. This is due to amongst other things, to the interactions between different minerals from the same ore, between minerals and the grinding media, and the electrochemical conditions present in the system (Seke, 2005) in (Damisa, 2007). A number of studies have reported that galvanic

interactions between the grinding media and different minerals as well as between minerals can affect the flotation selectivity of sulfide minerals depending on the electrochemical conditions in the mill (Mielezarski, 2003).

The standard Cu-Pb-Zn-Fe ore flow sheet involves the flotation of a Cu-Pb concentrate, followed by Zn and the pyrite flotation (Dow Chemical Co., 2005). In some mills, it is not economical to separate the copper and lead, but in others separation is made by treating the bulk concentrate. This may be done either by depressing the copper with cyanide, or the lead with SO_2 at low pH (5-7) or dichromate. The SO_2 method is the most common (Dow Chemical Co., 2005), (Wills, 2006).

The choice of the method for separating copper minerals from the lead minerals depend on the response of the minerals and the relative abundance of the copper and lead minerals. It is preferable to float the mineral present in the least abundance and galena depression is usually performed when the ratio of lead to copper in the bulk concentrate is greater than unity (Wills, 2006).

Lead depression is also undertaken if economic amounts of chalcocite or covellite are present as these minerals do not respond to the depression by cyanide, or if the galena is oxidized or tarnished and does not float readily. It may also be necessary to depress the lead minerals if the concentration of copper ions in solution is high, due to the presence of secondary copper minerals in the bulk concentrate(Wills, 2006). The standard copper depressant, sodium cyanide, combines with these ions to form complex cuprocyanides:

$$3CN^- + Cu^{2+} \rightarrow [Cu(CN)_2]^- + \frac{1}{2}C_2N_2$$

Increase in cyanide addition serves to accelerate the dissolution of secondary copper minerals (Wills, 2006). Depression of the galena is achieved using sodium dichromate,

sulfur dioxide and starch in various combinations, whereas, copper minerals are depressed using cyanides, or cyanide-zinc complexes (Wills, 2006).

Cyanide depression is not used if economic quantities of chalcocite or covellite are present in the bulk concentrate, since it has little depression action on these minerals. As cyanide is a very effective sphalerite depressant, most zinc reporting to the bulk concentrate is depressed into the copper concentrate which may incur the smelter penalties. Cyanide, however, has little action on galena allowing effective floatation of galena from chalcopyrite, and hence, a low lead-copper concentrate. Lead is never paid for in a copper concentrate, and is often penalized (Wills, 2006).

Sphalerite (ZnS) must be activated by copper salt before it becomes floatable with xanthane or similar collectors. The copper coated zinc mineral thus behaves in floatation as a copper mineral and its separation from copper is extremely difficult (Dow Chemical Co., 2005).

2.8 Determination of Liberation Size of the Valuable Minerals in the Ore

Liberation is the release of the valuable minerals from the associated gangue minerals at the coarsest possible size by comminution. If such an aim is achieved, then not only is energy saved by the reduction of the amount of fines produced but any subsequent separation stage becomes easier and cheaper. If high-grade solid products are required, then good liberation is essential. In practice, complete liberation is not achieved, even if the ore is ground down to the grain size of the desired mineral particles (Wills, 2006).

The degree of liberation refers to the percentage of the mineral occurring as free particles in the ore after grinding in relation to the total mineral content in the ore. This can be high if there are weak boundaries between the mineral and gangue particles, which is mainly the case with ores composed mainly of rock forming minerals, particularly

sedimentary materials. Usually, if the adhesion between the valuable mineral and the gangue is strong during comminution, the various constituents are cleft across. This produces much middling and low liberation. New approaches to increasing the degree of liberation involves directing the breaking stresses at the mineral grain boundaries, so that the rock can be broken without breaking the mineral grains. High degree of liberation may only be possible through intensive grinding, which may reduce the particle to such a fine size that separation becomes very inefficient. In practice, ores are ground to an optimum mesh to be determined by laboratory and pilot scale tests to produce an economic degree of liberation (Wills, 2006).

2.9 Grindability – Methods of Determining Work Index

Grinding characteristics of an ore refers to the ease with which the ore can be comminuted. It is an important factor in estimating the power requirement of industrial grinding operation. The most widely used parameter to measure grindability of an ore is the Bond's Work Index (W_i), which is defined as the power in Kwh per ton required to grind an ore from an infinite size to 80% passing 100 μm. Several empirical methods have been developed for evaluation of grindability of ores (Levin, 1989) in (Wills, 2006). Most of them have not been widely accepted because the results obtained do not correlate with each other or with the corresponding industrial data. The most widely used method is the Standard Bond's Method.

2.9.1 Standard Bond method

The Bond grindability test and the resulting work index obtained have been successfully used for predicting ball mill and rod mill energy requirements and for selection of plant scale comminution equipment. The work index has also been widely used to evaluate the grinding efficiency in operating plants. Although the Bond method test provides an accurate measurement of ore grindability, the test procedures are time consuming. This

method involves the use of a standard laboratory ball mill specifically made for that purpose with specified charge ball with varying diameters. The ore is ground in this ball mill at various revolutions of the mill. However, when making plant evaluations, a simplified procedure requiring less time which does not affect the accuracy is employed. Several methods to simulate the Bond work index parameters are used and some have been reported in literature. For example, (Smith, 1968) in (Sirajo, 2008) used batch type grindabilty test to arrive at the work index. The batch type result was found to compare favorably with the standard grindability test data. The advantage is that less time is required to determine the work index.

2.9.2 Modified Bond's method

Another method of determining the work index is method of Barry and Bruce, otherwise known as the Modified Bond's Method. The method requires the use of a reference ore whose work index is known using the Bond's Energy Equation (Wills, 2006).

$$\mathrm{W} = \frac{10\mathrm{W}_i}{\sqrt{\mathrm{P}}} - \frac{10\mathrm{W}_i}{\sqrt{\mathrm{F}}}$$

Where, W = Work input, W_i = Work index, P = product, F = feed

The work index of the unknown ore is determined when the work input (W) is the same for the identical weights of the reference and unknown ore, ground under the same conditions in the same laboratory ball mill. The energy input of the unknown ore therefore equals that of the reference ore. For the above conditions where r and u denote the reference and unknown ores respectively, the following holds according to Wills (2006):

$$E_r = E_u = W_{ir}\left[\frac{10}{\sqrt{P_r}} - \frac{10}{\sqrt{F_r}}\right] = W_{iu}\left[\frac{10}{\sqrt{P_u}} - \frac{10}{\sqrt{F_u}}\right]$$

Where, $E_r$is the energy input in grinding the reference ore,

$E_u$is the energy input in grinding the unknown ore,

$W_{ir}$is the work index of the reference ore,

$W_{iu}$is the work index of the unknown ore,

$F_r$is the 80% passing feed of reference ore,

$F_u$is the 80% passing feed of unknown ore,

$P_r$is the 80% passing product of reference ore,

$P_u$is the 80% passing product of unknown ore.

Therefore, the work index of the unknown ore is:

$$W_{iu} = W_{ir}\frac{\left[\frac{10}{\sqrt{P_r}} - \frac{10}{\sqrt{F_r}}\right]}{\left[\frac{10}{\sqrt{P_u}} - \frac{10}{\sqrt{F_u}}\right]}$$

This method is relatively simple and is called the comparative method of determining the grindability parameter i.e. work index (Wills, 2006).

2.10 Analytical Methods of Metallurgical Analysis

Metallurgical analysis involves the qualitative as well as the quantitative analysis of various metallurgical raw materials like ores and products like metals, alloys and slags (Yaro, 1997). The analytical methods employed in metallurgical analysis can be classified into classical and instrumental methods.

The classical methods are slow and for practical purposes, the instrumental methods which are faster and quite accurate are used (Gilchrist, 1989).

In the instrumental methods of analysis, measurement is made upon the system as a whole. A particular physical or chemical property of the system is measured in order to get the desired analytical results. Comparison of the results of the samples of the results

to be analyzed with standard samples of known composition is necessary in this method (Yaro, 1997).

The common instrumental analytical techniques used are atomic absorption spectrophotometry, x-ray fluorescence, x-ray diffraction, scanning electron microscopy and electron probe micro analysis.

2.10.1 X-ray fluorescence (XRF)

An x-ray source is used to irradiate the specimen and to cause the elements in the specimen to emit (or fluoresce) their characteristic x-rays, by electrons ejected from the atom making up the material.A detector system is used to measure the positions of the fluorescent x-ray peaks for qualitative identification of the elements present, and to measure the intensities of the peaks for quantitative determination of the composition. All elements but low atomic number elements: H, He, and Li, can be routinely analyzed by XRF (Huang, 1992). The x-rays will have a precisely defined energy and the wavelength (velocity of electron) which are related by the Plank's energy equation:

$$E_1 - E_2 = hc$$

Where, $E_1$is the former energy level of the electron before filling the vacancy

$E_2$is the new energy level of the electron after filling the vacancy

$h$is Plank's constant, $c$is the velocity of light

As the energies or wavelengths are characteristics of the atoms of the elements emitting the x-rays, they are used for qualitative analysis while the intensity of these characteristics x-rays is related to the concentration of the particular element in the material and is used for qualitative elemental analysis (Funtua, 1999 in Thomas, 2007).

XRF can be used extensively for the analysis of solids, powders, and liquids. The technique is nondestructive, rapid, precise, and potentially very accurate.

2.10.2 Scanning electron microscopy (SEM)

SEM is often the first analytical instrument used when a "quick look" at a material characterization is required and the light microscope no longer provides adequate resolution. This method is used to study the properties of materials at microscopic level. The most important applications are the investigation of the micro-structure and analysis of in-homogeneity such as defects, dislocations, submicroscopic precipitates, inclusions, domains, etc. It is also used for determining the composition of individual grains of particles and surface properties like topography.

In SEM, a small area of the specimen of a material is irradiated with direct electron beam and characteristics x-rays are produced which are used for the qualitative and quantitative analysis. As the beam interacts with the sample it creates various signals (secondary electrons, internal currents, photon emission, etc.), all of which can be appropriately detected (Bindell, 1992). The most important of the produced electrons (x-rays), called backscattered or primary reflected electrons are detected and amplified. The output from this is a function of the composition of the area irradiated by electron beam (Thomas, 2007). The intensity of the backscattered electron signal is dependent on the atomic number of the specimen and specimen surface of different chemical composition will produce different signal intensities and appear different on the SEM screen. The signal intensities are used for quantitative analysis.

No elaborate specimen-preparation techniques are required for examination in the SEM, provided that the specimen is vacuum compatible, and large and bulky specimens may be accommodated. It is desirable that the specimen be rendered electrically conducting; otherwise a sharp picture will not be obtained (Encyclopædia Britannica, 2010).

2.10.3 X-ray diffraction (XRD)

XRD analysis is a non-destructive method of analysis. It is used for both qualitative and quantitative analysis of materials. This method can be used to determine the actual compound(s) present in a material and can also distinguish the different crystal forms of the compounds. XRD can also be used to measure the structural properties (strain state, grain size, epitaxy, phase composition, preferred orientation, and defect structure) of these phases and to determine the thickness of thin films and multilayers, and atomic arrangements in amorphous materials (including polymers and at interfaces (Toney, 1992).

XRD is based on the diffraction of x-rays by polycrystalline solids and the reflections are called powder diffractometry. Crystals deflect X-rays when the condition satisfying the Bragg's equation is fulfilled. The Bragg's equation is given as Bragg's law ($n\lambda = 2d \sin\theta$, where d is the spacing between atomicplanes in the crystalline phase and n is an integer called order of reflection). The intensity of the diffracted x-rays is measured asa function of the diffraction angle 2θ and the specimen's orientation (Brundle, 1992).

These characteristics x-rays which are deflected by the crystal are detected by a detector and registered and displayed on paper chart recorder as series of peaks (x-rays patterns). Each compound has a unique x-ray powder pattern (fingerprints) because the line position depends on the unit cell size and the type of atoms present and their intensities of powder lines are used for qualitative phase analysis, while quantitative phase analysis depends on the accurate determination of the line intensity (Markowiez, 2002, in Thomas, 2007).XRD offers unparalleled accuracy in the measurement of atomic spacing and is the technique of choice for determining strain states in thin films. XRD is noncontact and nondestructive, which makes it ideal for in situ studies.

CHAPTER THREE

3.0 RESEARCH METHODOLOGY

3.1 Sample Collection

50Kg of the weathered ore sample was collected from the deposit site at Zurak using the grab and trench sampling techniques which involvedcollection at random, small quantities of the ore from different artisanalspots in the deposit area and then combining them.

3.2 Equipment and Reagents Used

The following equipment and reagentswere used to carry the research work:

1. Manual sieves, collection dishes and measuring cylinder.
2. AR9400 XP + sequential XRF and uniquant software.
3. PANalyticalX'Pert Pro diffractometer.
4. Phenom Pro scanning electron microscope.
5. A Bico jaw crusher, Bico rod mill and Bico pulverizer.
6. Jones riffle.
7. Endecotts sieve shaking machine and various sieves.
8. Wemco flotation cell.
9. Flotation reagents (water, sodium silicate, zinc sulphate, lime, potassium exthy xanthate, methyl isobutyl ketone and copper sulphate).
10. Digital pH metre, analogue and digital weighing beams.
11. Gallenkamp oven.
12. Buck scientificatomic absorption spectrophotometer.

3.3 Physical Properties of the Zurak Lead Ore

The run-of-mine ore was visually inspected. The colour of the run-of-mine ore was dark grey, which strongly suggested that the ore may contain lead mineral (Klein, 2002). The streak of the ore which waslead grey to blue black confirmed the ore as that of lead (Klein, 2002).

Plate 3-I: Picture of a crude sample of Zurak ore containing galena and sphalerite

To determine specific gravity of the ore, 100gof the dried Zurak ore was placed into 500cm^3 of water in a graduated measuring cylinder. It was ensured that the ore particles were completely wetted and entrained air was allowed to escape. The volume of water which represents the volume of the ore was observed to increase by 29cm^3.

$$S = \frac{w}{v}$$

Where, *S*is specific gravity, *w*is weight (g), and *v*is volume (cm^3) of the ore.

3.4 Chemical Analysis

The chemical analysis of the ore was carried out usingXRFat the Geology Department, University of Pretoria, South Africa.The run-of-mine ore was sieved to -90μm particle size and 100g was sent for analysis.

The samples were prepared as pressed powder briquettes. The AR9400 XP + Sequential XRF and Uniquant software were used for the analyses. The software analyzed for all elements in the periodic table between Na and U, but only elements found above the detection limits were reported. The values were normalized, as no LOI was done to determine the crystal water and oxidation state changes. All elements were expressed as oxides.

The result of the chemical analysis are presented in Tables 4.2 and 4.3.

3.5 Mineralogical Analysis

The mineralogical analysis of the ore was also carried out usingXRDat the Geology Department, University of Pretoria, South Africa.

The samples were prepared according to the standardized PANalytical back loading system, which provides nearly random distribution of the particles.They were analyzed using a PANalyticalX'Pert Pro powder diffractometer in θ–2θ configuration with aX'Celerator detectorand variable divergence- and receiving slits with Fe filtered Co-Kα radiation(λ=1.789Å). The phases were identified using X'PertHighscore plus software.

The relative phase amounts (weight%) were estimated using the Rietveld method (Autoquan Program). Errors are on the 3 sigma level in the column to the right of the amount. Amorphous phases, if present were not taken into consideration in the quantification.

The XRD patterns of the Zurak lead ore is shown in Fig. 4.1 and the mineralogical analysis is presented in Table 4.4.

The SEMwas performed at the Chemical Engineering Department, Ahmadu Bello University, Zaria, Nigeria. The micrographs are shown in Plates 4-I to 4-II.

3.6 Work Index

The Bond modified method (i.e. Barry and Bruce comparative method) was used to determine the grindability of the ore (i.e. the work index of the ore). The method required the use of a reference ore of known grindability.

25. 3kg of the run-of- mines ore of the Zurak lead ore sample was taken as the feedfrom which 1.7kg was obtained through coning and quartering. Equal quantity of the reference ore was also taken. Both the reference and the test ore were crushed using aBico Jaw Crusher.The samples were further splitusing a Jones Riffle. 500g of the samples were sieved for 30miutes on an Endecott sieve shaking machine, to determine the particle size at 80% passing. The appropriate sieve aperture sizes ranging from 1700μm to 63μm were used based on the square root of 2 ($\sqrt{2} = 1.414$) method.Iron ore from Itakpe was used as the reference ore. The analysis were conducted at the Mineral processing laboratory of the Nigerian Institute of Mining and Geosciences, Jos, Nigeria. The feed to the roller mill are shown in Tables 4.1 and 4.3 respectively.

1.7kg of the reference and test ores wereeach ground dry, in a Bico RodMill at the Mining Community Resource Centre, Jos, for 30minutes and their power consumptions recorded. The samples were further splitusing a Jones Riffle. 500g of the product samples were sieved for 30minutes to determine the particle size at 80% passing. Sieve aperture sizes ranging from 500μm to 63μm were used.The rod mill discharge are shown in Tables 4.2and 4.4 respectively.

3.7 Sieve/Particle Size Analysis

Sieve analysis was carried out to determine the particle size distribution of the ore in each sieve size fraction. 637.83g of Zurak leadores was ground for 10minutes in a Bico Pulverizer and sieved on an Endecott sieve shaking machine for 30minutes.Sieve aperture sizes ranging from 500μm to 63μm were used.Each sieve size fraction obtained was weighed and recorded in Table 4.5.

3.8 Size-Assay Analysis

The assay of the various size fractions obtained from the sieve/particle size analysis were determined using a Buck Scientific Atomic Absorption Spectrophotometer (AAS) Model 210 VGP version 3.74, at the Chemistry laboratory, Plateau State Polytechnic, Barkin Ladi, Nigeria.

The samples were prepared by digesting 0.1g of each size fractionusing aquaregia. Each solution was aspirated by means of pneumatic nebulizer into the spray chamber from which only about 10% passes into the atomizer (flame) and was analyzed.

The chemical analysis are presented in Table 4.6.

3.9 Liberation Size Analysis

The liberation size of the Zurak lead ore was determined. The aim of the liberation test was to determine the particle size at which the valuable mineral can be freed from the gangue.

3.10 Ore Concentration

The specific gravities of the two main minerals of interest in the ore (galena and sphalerite) were used to determine the Concentration Criterion (CC) for determining the susceptibility of the ore to gravity separation process(La-Plante & Spiller, 2002).

$$CC = \frac{(D_h - D_f)}{(D_l - D_f)}$$

Where, $D_h$isthe specific gravity of the heavy mineral,

$D_l$is the specific gravity of the light mineral,

$D_f$is the specific gravity of fluid.

Thus, for the Zurak lead-zinc ore,

D_g = the specific gravity of galena = 7.6,

D_s = the specific gravity of shalerit) = 4.0,

D_w = the specific gravity of water = 1.0

Hence,

$$\text{CC of Zurak ore} = \frac{(\mathrm{D_g} - \mathrm{D_w})}{(\mathrm{D_s} - \mathrm{D_w})}$$

3.11 Froth Flotation Process

From the result of the size-assay analysis, more valuable were retained in the -90+63µm and -180+125 size fractions. Hence, the two stage selective flotation of the ore was carried out in two batches at the Minerals processing laboratory of the Nigerian Institute of Mining and Geoscience, Jos.

In the first batch, the Zurak lead ore sample was ground using a Bico pulverizering machine, sieved to -90+63µm particle size and 350g fed into the cell ofa Wemco flotation machineand mixed with1500cm^{3}of water of pH 6.8. The cell was positioned under the impeller at a clearance of 1cm from the bottom of the cell. The mixture was agitatedfor 5 minutes and it blended into a pulp.0.12g of Sodium silicate ($NaSiO_2$) and

0.08g of Zinc sulphate ($ZnSO_4$) were addedto the pulp after 1 minute to depress the silica and sphalerite respectively.The pulp pH was measured using a digital pH meter and maintained at 9.0using Lime (CaO) water. 0.02g ofcollector;Potassium ethyl xanthate ($C_2H_5OCS_2K$ or simply PEX) was added and the pulp conditioned for another 5 minutes.3 drops of frother; Methyl Isobutyl Ketone (MIBK) was added and after an additional 5 minutes of conditioning, the air tap was opened and air bubbles introduced into the pulp.The galena froth formed was scooped into a pan using a spoon,every 10 seconds for about 4 minutes, as the bubbles emerge.The air flow was shut as froth activity in the cell waned.

After the lead flotation, the PH of the pulp was adjusted to 11 using lime. 0.35gof activator; Copper sulphate ($CuSO_4$) was added in order to activate the sphalerite. It was noticed thatthe pulp turned leafy green from previous black colour.0.02g of PEX was added, after conditioning for 2 minutes, 3 drops of MIBK was added and airflow was opened after 2 more minutes of conditioning.The sphalerite froth was again scooped into a pan using a spoon,every 10 seconds for about 2 minutes, as the bubbles emerge.The tailings left in the cell was also collected.

After the cell had been emptied, it was washed and rinsed using distilled water for the second batch of flotation test using a similar procedure as outlined above.

The ore sample was again ground and sieved to -180+125µm particle size. 500g of the ore was fed into the flotation celland mixed with2000$cm^3$of water of pH 6.8. The mixture was agitatedfor 5minutes and it blended into a pulp.0.143g of $NaSiO_2$ and 0.1g of $ZnSO_4$ were addedto the pulp after 1 minute.The pulp PH was maintained at 9.0using lime water. 0.03g ofPEX was added and the pulp conditioned for another 5 minutes.3 drops of MIBK was added and after an additional 5 minutes of conditioning, the air tap was

opened and air bubbles introduced into the pulp.The galena froth formed was scooped into a pan using a spoon,every 10 seconds for about 6 minutes, as the bubbles emerge.The air flow was shut as froth activity in the cell waned.

The pH of the pulp was adjusted to 11 using lime. 0.6gof $CuSO_4$ was added. It was noticed thatthe pulp turned leafy green from previous black colour.0.03g of collector; PEX was added, after conditioning for 2 minutes, 3 drops of frother was added and airflow was opened after 2 more minutes of conditioning.The sphalerite froth was againscooped into a pan using a spoon, every 10 seconds for about 4 minutes, as the bubbles emerge.The tailings left in the cell was also collected.

Data from selective flotation galena:

Weight of sample:	350g and 500g;	Impeller speed:	1500rpm;
Conditioning time:	5 minutes;	Air flow rate:	1.2L/m;
Pulp pH:	9;	Solid/liquid ratios:	23.33% and 25%
Collector:	Potassium ethyl xanthate;	Collector dosage:	0.12kg/tonne;
Frother:	MIBK;	Froth dosage:	3 drops;
Activator/PH regulator: Lime;		Activator dosage:	1kg/tonne;
Depressant:	Zinc sulphate, supplemented by sodium silicate;		

Depressant dosage: 0.2kg/tonne zinc sulphate and 0.15kg/tonne sodium silicate.

Data from sequential flotation of sphalerite:

Conditioning time:	5 minutes;	Pulp pH:	11;
Collector:	potassium ethyl xanthate;	Collector dosage:	0.12kg/tonne;
Frother:	MIBK;	Froth dosage:	3 drops;
Activator/PH regulator: Lime;		Activator dosage:	1.2kg/tonne;
Depressant:	Copper sulphate;	Depressant dosage:	0.7kg/tonne.

After the flotation tests the concentrate and tailings from both batches were decanted and driedin a Gallenkamp Oven, weighed and analyzed using AAS at the Plateau State Polytechnic, Barkin Ladi.

CHAPTER FOUR

4.0 RESULTS AND DISCUSSIONS

4.1 The Zurak Lead-Zinc Ore Deposit

Wase Local Government Area is located about 216Km from Jos, the capital of Plateau State, Nigeria. It lies between longitude $9°50^{1}$E to $10°56^{1}$E and latitude $8°30^{1}$N to $9°25^{1}$N in the South-Eastern part of the Plateau State. Zurak, where the lead-zinc ore deposit is located lies between longitude $10°28^{1}$E to $10°52^{1}$E and latitude $9°15^{1}$N to $9°16^{1}$N.

As at the time of this research, the estimated reserve of the deposit was not ascertained(RMRDC, 2004).

4.2 Physical Properties of the Zurak Lead-Zinc Ore

Table 4.1: Physical properties of Zurak ore

Property	Diagnostic Feature
Colour	The run-of-mine ore is dark grey
Specific Gravity	3.45
Streak	Lead grey to blue-black

Other Characteristics	Bright metallic lustre.

Table 4.1 presents the result of the physical test. The result reveals that the major mineralsin the sample are lead – galena (PbS) and zinc – sphalerite (ZnS). The colour of the run-of-mine ore is dark grey, and the streak of the ore islead grey to blue black. These features indicate that the ore is predominantly lead-zinc and the stated features of the ore agreed with those sighted in Klein(2002).

4.3 Chemical Analysis

The result of the chemical analysis (XRF) of the Zurak ore and the estimated metal contents of minerals of interest are presented in Tables 4.2 and 4.3 of this work.

Table 4.2 : Chemical composition of the Zurak run-of-mine ore (%)

Oxide	SiO_2	TiO_2	Al_2O_3	Fe_2O_3	MnO	MgO	CaO	Na_2O	K_2O	P_2O5
NimN cert.	52.64	0.2	16.5	8.91	0.18	7.5	11.5	2.46	0.25	0.03
NimN	48.80	0.16	22.96	8.08	0.17	5.23	11.28	2.88	0.27	0.02
P2	51.77	0.86	17.86	3.91	0.11	0.33	6.55	<0.01	2.84	0.26

Cr_2O_3	NiO	V_2O_5	ZrO_2	SO_3	WO_3	Cl	CuO	ZnO	Co_3O_4	PbO
0.0044	0.0153	0.0393	0.0031				0.0018			
<0.01	0.02	0.03	<0.01	<0.01	<0.01	0.02	<0.01	0.01	0.01	<0.01
<0.01	<0.01	0.02	0.04	2.57	0.01	0.09	0.02	5.79	0.01	6.82

BaO	Rb_2O	HfO_2	Y_2O_3	CeO_2	Nb_2O_5	U_3O_8	SrO	MoO_3	Yb_2O_3	SnO_2	PtO_2
<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.03	0.01	<0.01	<0.01	<0.01
0.03	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.02	<0.01	<0.01	<0.01	<0.01

ThO_2	Ta_2O_5	La_2O_3	Nd_2O_3	Dy_2O_3	Er_2O_3	Pr_6O_{11}	Sb_2O_3	CdO	Ag_2O	Total

										100.23
<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	99.99
<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.02	0.02	0.01	99.95

Table 4.3:Elemental composition of the Zurak run-of-mine ore (%)

Oxide	Percentage (%)	Element	Percentage (%)
PbO	6.82	Lead	6.33
ZnO	5.79	Zinc	4.65
Fe_2O_3	3.91	Iron	2.74

The result of the chemical analysis (XRF) of the ore is presented in Table 4.2, while estimated metal contents of minerals of interest of the ore is presented in Table 4.3. The resultreveals that the run-of-mine ore (head sample) contains 6.33% Pb,4.65% Zn and 2.74% Fe on the average. Based on this result, it is evident that the ore meets the typical standard feed grades of 1 – 5% Pb and 1 – 10% Zn required to mine lead-zinc ore deposit (Wills, 2006).

4.4 Mineralogical Analysis

The XRD patterns and mineralogical analysis of the Zurak lead ore are presented in Figure 4.1 and Table 4.4 below respectively.

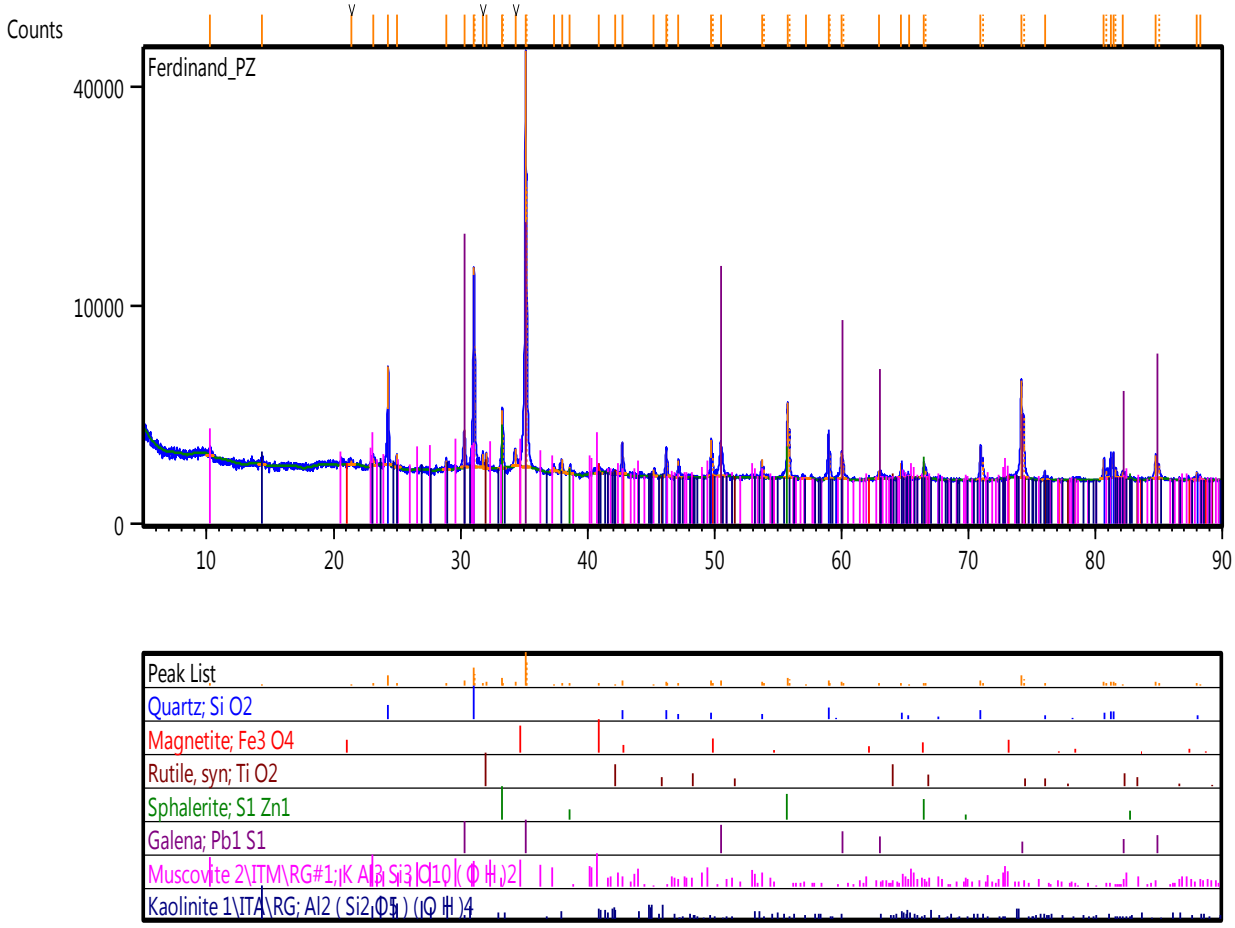

Figure 4.1: XRD patterns of the Zurak ore

Table 4.4: XRD analysis of Zurak ore (%)

Mineral	Chemical formula	Weight %	3 σ error
Galena	PbS	7.11	0.25
Kaolinite	$Al_2Si_2O_5(OH)_4$	5.38	1.83
Magnetite	Fe_3O_4	3.04	0.54
Muscovite	$KAl_3Si_3O_{10}(OH)_2$	9.8	1.47
Quartz	SiO_2	65.65	1.71
Rutile	TiO_2	1.5	0.51
Sphalerite	ZnS	7.51	0.33

Table 4.5: Mineralogical composition of Zurak ore (qualitative)

Major minerals	Minorminerals	Gangues
Galena (PbS)	Magnetite (Fe_3O_4)	Quartz (SiO_2)
Sphalerite (ZnS)	Rutile (TiO_2)	Kaolinite [$Al_2Si_2O_5(OH)_4$]
		Muscovite [$KAl_3Si_3O_{10}(OH)_2$]

Table 4.4 and Figure 4.1 are the mineral compositions and x-ray diffractogram pattern of the ore sample respectively. The result reveals that the major valuable minerals present are 7.11wt.% galena (PbS) and 7.51wt.% sphalerite (ZnS). Other associate minerals are 65.65wt.% quart (SiO_2), 1.5wt.%rutile(TiO_2), 5.38w.t% kaolinite [$Al_2Si_2O_5(OH)_4$], 3.04wt.% magnetite(Fe_3O_4) and 9.8wt.% muscovite ($KAl_3Si_3O_{10}(OH)_2$) as minor and gangues. From the result the percent weight distributions of the valuable minerals are closely related, though zinc sulphide seems to have the highest percent weight when compared to lead sulphide. This phenomenon could be attributed to the mineralization process of the ore deposit (Klein, 2002).

TheSEM micrographs in Plates 4-I to IIIfurther reveals the interlocking nature of the minerals.

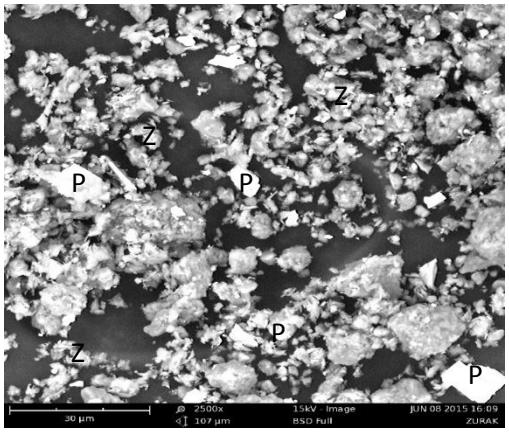

Plate 4-I: Micrograph showing the morphology of Zurak ore (-90µm) at 2000x
P = Galena; Z = Sphalerite

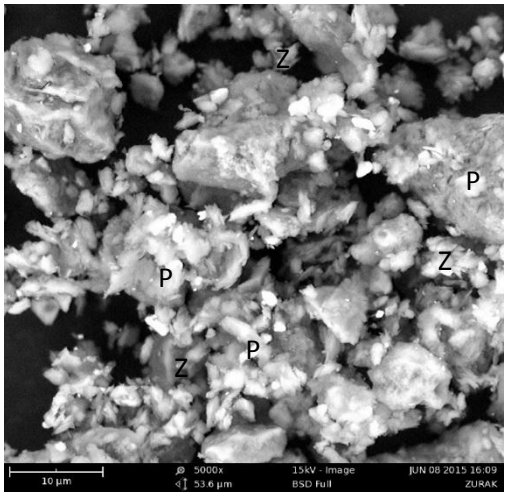

Plate 4-II: Micrograph showing the morphology of Zurak ore (-90µm) at 5000x

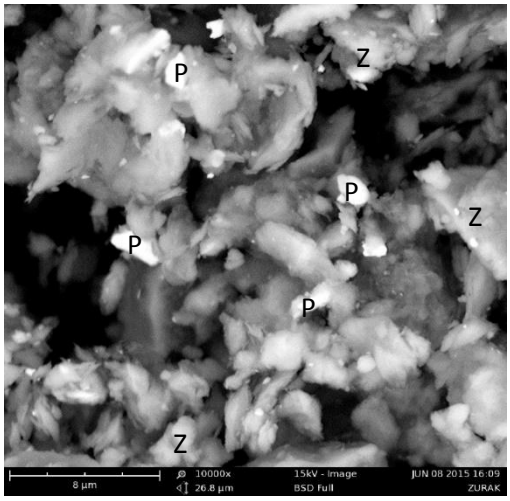

Plate 4-III: Micrograph showing the morphology of Zurak ore (-90µm) at 10000x

4.6 Work Index

The Bond modified method (i.e. Barry and Bruce comparative method) was used to determine the grindability of the ore (i.e. the work index of the ore). The method required the use of a reference ore of known grindability.The reference ore used was Itakpe iron ore whose work index is 8.71Kwh/tonne (Smith & Lee, 1968 in Sirajo, 2008). The grinding was conducted dry.

Table 4.6: Feed to the rod mill of reference ore (Itakpe iron ore)

Sieve size	Weight retained	Cumulative weight (%)

(μm)	(g)	(%)	Retained	Passing
+1700	83.12	17.27	17.27	82.73
+1168	72.03	14.96	32.23	67.77
+1000	90.13	18.72	50.96	49.04
+699*	50.12	10.41	61.37	38.63
+500	33.41	6.94	68.31	31.69
+350*	12.27	2.55	70.86	29.14
+250	43.31	9.00	79.86	20.14
+180	29.11	6.05	85.91	14.09
+125	21.10	4.38	90.29	9.71
+90	26.41	5.49	95.78	4.22
+63	12.22	2.54	98.31	1.69
-63	8.12	1.69	100.00	-
	481.35	100.00		

*699 and 350 mesh sieves were used in the absence of 710 and 355 mesh sieves.

Table 4.7: Rod mill discharge of reference ore (Itakpe iron ore)

Sieve size	Weight retained		Cumulative weight (%)	
(μm)	(g)	(%)	Retained	Passing
+699	-	-	-	-
+500	20.43	4.79	4.79	95.21
+350	40.17	9.42	14.21	85.79
+250	79.44	18.64	32.85	67.15
+180	43.11	10.11	42.96	57.04
+125	109.44	25.68	68.64	31.36
+90	60.41	14.17	82.81	17.19

+63	49.34	11.58	94.39	5.61
-63	23.91	5.61	100.00	-
	426.25	100.00		

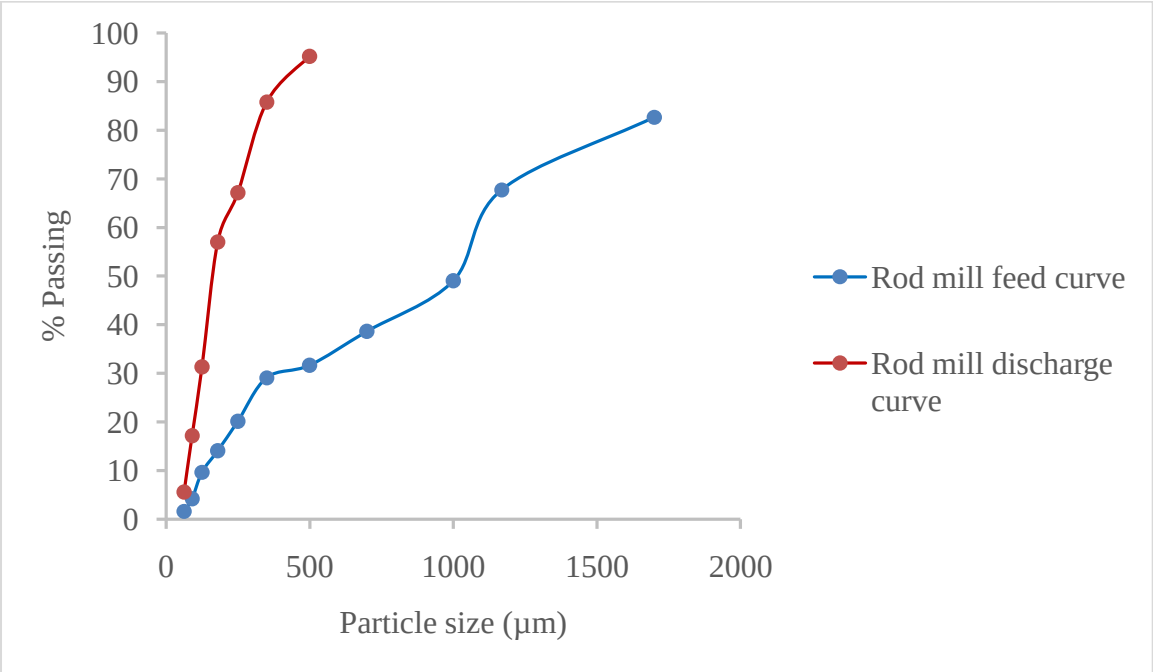

Figure 4.2: Plot of cumulative sieve analysis for the reference ore (Itakpe iron ore)

Table 4.8: Rod mill feed of test ore (Zurak galena)

Sieve size	Weight retained		Cumulative weight (%)	
(µm)	(g)	(%)	Retained	Passing
+1700	57.06	11.54	11.54	88.46
+1168	35.91	7.26	18.80	81.20
+1000	32.63	6.60	25.40	74.60
+699	81.32	16.45	41.85	58.15
+500	21.61	4.37	46.22	53.78
+350	74.13	14.99	61.22	38.78

+250	41.75	8.45	69.66	30.34
+180	15.68	3.17	72.84	27.16
+125	47.63	9.63	82.47	17.53
+90	14.28	2.89	85.36	14.64
+63	41.96	8.49	93.85	6.15
-63	30.41	6.15	100.00	-
	494.37	100.00		

Table 4.9: Rod mill discharge of test ore (Zurak galena)

Sieve size	Weight retained		Cumulative weight (%)	
(μm)	(g)	(%)	Retained	Passing
+699	-	-	-	-
+500	-	-	-	-
+350	24.93	5.23	5.23	94.77
+250	56.40	11.84	17.07	82.93
+180	35.71	7.50	24.57	75.43
+125	145.04	30.45	55.02	44.98
+90	56.51	11.86	66.89	33.11
+63	122.73	25.77	92.66	7.34
-63	34.96	7.34	100.00	-
	476.28	100.00		

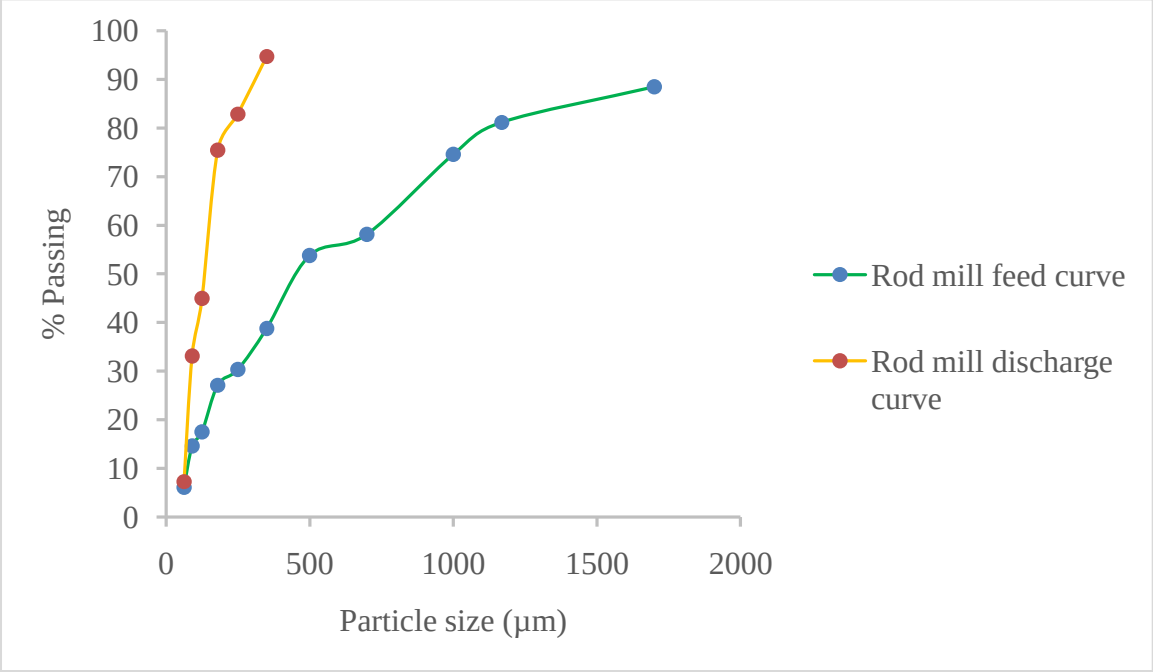

Figure 4.3: Plot of cumulative sieve analysis for the test ore (Zurak galena)

From Figure 4.2, the cumulative weight percent passing against particle size for the reference ore (Itakpe iron ore) were obtained:

80% passing of feed size, F_r = 1650µm

80% passing of product size, P_r = 325µm

Work index of the reference ore W_i = 8.71Kwh/t (Smith & Lee, 1968 in Sirajo, 2008)

The energy expended in grinding the iron ore from 80% passing feed (1650µm) to 80% passing product (325µm) using the Bond's energy equation is given by:

$$E_r = 10W_i\left(\frac{1}{\sqrt{P}} - \frac{1}{\sqrt{F}}\right) \text{Kwh/tonne} \qquad \text{Equation 4.1}$$

$$= 10 \text{ x } 8.71\left(\frac{1}{\sqrt{325}} - \frac{1}{\sqrt{1650}}\right)$$

$$= 87.1(0.05546 - 0.02462) = \ 2.6863\text{Kwh/tonne}$$

Since the grinding was conducted dry, then the energy expended in grinding the iron ore from 80% passing feed (1650μm) to 80% passing product (325μm) using the Bond's energy equation is multiplied by 1.30. Thus,

$$E_r = 1.30 \text{ x } 2.6863 = 3.4922\text{Kwh/tonne}$$

Also, from Figure 4.3, the cumulative weight percent passing against particle size for the test ore (Zurak galena) were obtained:

80% passing of feed size, F_t = 1100μm

80% passing of product size, P_t = 215μm

The energy consumed in grinding the test ore is given by:

$$E_t = 10W_i\left(\frac{1}{\sqrt{215}} - \frac{1}{\sqrt{1100}}\right)$$

$$= 0.306W_i$$

Since the same amount of energy was consumed in grinding both the iron ore (reference ore) and the galena (test ore), then:

$$0.306W_i = 3.4922\text{Kwh/tonne}$$

Solving for W_i,

$$W_i = 11.41\text{Kwh/tonne}$$

The work index of the test ore (Zurak galena) is 11.41Kwh/tonne

The value of the work index of the Zurak lead ore obtained in this work is 11.41Kwh/tonne on the average. This value means that approximately 11.41Kwh/tonne of energy is required to reduce 1tonne of the Zurak ore from 80% passing particle size 1100μm to 80% passing particle size 215μm. This indicate that the ore can be classified as a type B ore with medium soft texture using grand-abilities test curves (Thomas, 2007). This is very significant because energy is very expensive and grinding is the most energy intensive operation in mineral beneficiation (Wills, 2006).

The value of the work index obtained for ore may be said to be valid over the size range of the particle size considered (-63μm to 1700μm). This is because breakage characteristic of most ores varies with particle size. The result obtained compares favourably with the ones sighted in the literature (Wills, 2006).

4.7 Sieve/Particle Size Analysis

The sieve and particle size analyses results are presented in Table 4.10 below.

Table 4.10: Particle size analysis of Zurak ore

Sieve size range	Sieve fractions weight		Nominal aperture	Cumulative %	
(μm)	(g)	(%)	size (μm)	Undersize	Oversize
+500	2.63	0.44	500	99.56	0.44
-500 to +350*	31.46	5.21	350	94.35	5.65
-350 to +250	45.06	7.46	250	86.88	13.12
-250 to +180	71.18	11.79	180	75.09	24.91
-180 to +125	71.31	11.81	125	63.28	36.72
-125 to +90	183.04	30.32	90	32.96	67.04
-90 to +63	154.89	25.66	63	7.31	92.69
-63	44.12	7.31			
	603.69	100.00			

*350 mesh sieves was used in the absence of 355 mesh sieve.

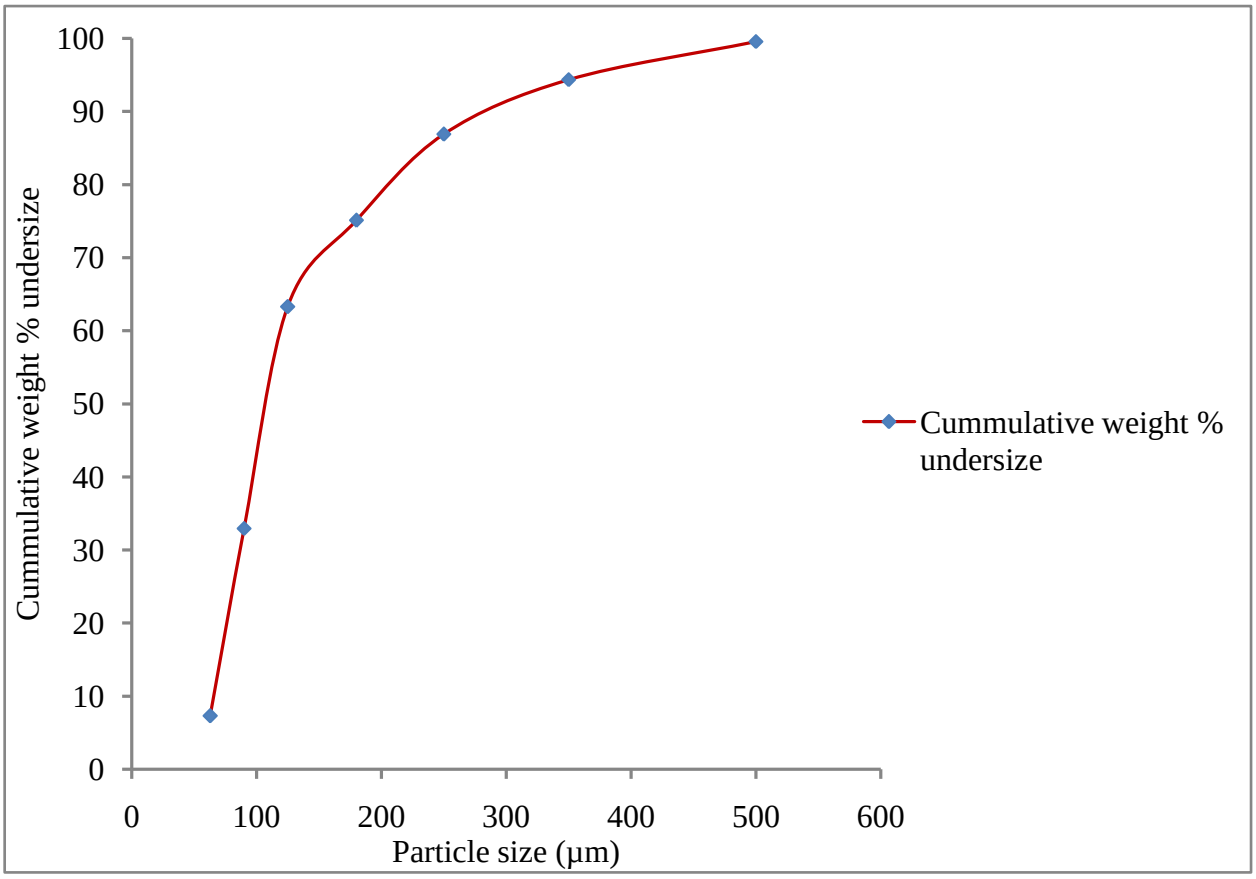

Figure 4.4: Particle size analysis of Zurak ore

Table 4.10 and Figure 4.4 present the result of the sieve/particle size analysis. The result reveals the significance of size analysis in assessing the performance of grinding circuits. The product size is usually quoted in terms of one point on the cumulative undersize curve, this often being the 80% passing size. Although this does not show on the overall size distribution of the material, it does facilitate routine control of the grinding circuit. For instance, if the target size is 80%, -90µm, then, for routine control, the operator needs only screen a fraction of the mill product at one size. If it is found that, say, 50% of the sample is -90µm, then the product is too coarse, and control steps to remedy this can swiftly be made(Wills, 2006).

4.7 Size-Assay Analysis

Table 4.11 shows the results of thesize-assay analysis performed on the ore to determine the most suitable size fraction for the concentration of the ore by froth flotation method.

Table 4.11:Size-assay analysis for Zurak ore

Size range (μm)	Sieve fraction Weight		Assay (%) A		Weight of metal (g) M x A		Distribution (% Metal) MA/∑(MA)		Cumulative (%)		
	(g) M	(%)							Over size	Distribution	
			Pb	Zn	Pb	Zn	Pb	Zn		Pb	Zn
+500	2.63	0.44	4.32	1.18	0.11	0.03	0.08	0.03	0.44	0.08	0.03
-500 +350	31.46	5.21	12.35	3.28	3.89	1.03	2.75	1.04	5.65	2.83	1.07
-350 +250	45.06	7.46	17.07	5.57	7.69	2.51	5.43	2.53	13.12	8.26	3.61
-250 +180	71.18	11.79	19.54	9.12	13.91	6.49	9.83	6.56	24.91	18.09	10.16
-180 +125	71.31	11.81	22.76	18.16	16.23	12.95	11.47	13.08	36.72	29.55	23.24
-125 +90	183.04	30.32	20.03	12.91	36.66	23.63	25.90	23.87	67.04	55.46	47.11
-90 +63	154.89	25.66	28.62	15.46	44.33	23.95	31.32	24.18	92.79	86.78	71.29
-63	44.12	7.31	42.43	20.08	18.72	8.86	13.22	8.95			
	603.69	100	23.45	13.16	141.54	79.45					

$$\text{Overall Assay } = \frac{\sum_n^i MA}{\sum_n^i M}\text{x}100 = \frac{\sum_{-63}^{+500} MA}{\sum_{-63}^{+500} M}\text{x}100$$

$$\therefore \text{Overall Assay of Lead} = \frac{141.54}{603.69}\text{x}100 = 23.45\%\ \text{Pb}$$

$$\text{Overall Assay of Zinc} = \frac{79.45}{603.69}\text{x}100 = 13.16\%\ \text{Zn}$$

The result of the size assay analysis is presented in Table 4.11. The result reveals the most suitable size fraction to be used for the concentration of the ore by froth flotation method.

The particle size-assay analysis showed that the calculated overall assay is23.45% Pb and 13.16% Zn and much of the minerals are present within the -90 +63μm size fraction

(31.32% Pb, 24.18% Zn). It was obviousthat flotation at the -63µm size fraction will not be economical for beneficiating the ore even though it has the highest assay (42.43% Pb, 20.08% Zn) because of high energy requirements to reduce the ore to that particle size.

The result also demonstrate that, for instance, if the material was initially screened at 350µm and the coarse fraction discarded, the bulk required for further processing would be reduced by 5.65%, with loss of only 2.83%Pb and 1.07% Zn. This confirmed the earlier statement made by Wills (2006) that comminution is an energy intensive process.Though needed to liberate the mineral of interest, it must be conducted taking into consideration the economy of the process.

4.8 Liberation Size

Table 4.11 above also presents the result of the liberation study of the ore. From the result, the highest degree of liberation of the minerals of interestlies within ranges of -90+ 63µm and -63µm. This means that the lead ore may be ground to its optimum mesh for selective flotation of the sulphides. The bulk of the ore within these size fractions will therefore contain almost all the free mineral particles and middling fractions that may require further grinding to promote optimum release of the valuable minerals, in which selectiveand sequential flotation will separate the galena from the sphalerite, and the sphalerite from the gangue respectively. This is important because achieving good liberation at coarse a size as possible, saves comminution energy and facilitates mineral separation at lower costs. Complete liberation is seldom achieved in practice.

4.9 Ore Concentration

The Concentration Criterion (CC) of the two main minerals of interest in the ore (galena and sphalerite) was found to be 1.87 which is less than 2.5 ideal for separation of minerals using gravity concentration process (La-Plante & Spiller, 2002). However, it is

widely reported in literatures (Weiss, 1985; Wills, 2006) that froth flotation is most suitable for the beneficiation of the lead-zinc sulphide ores due to the ability of sulphide to react with conditioning reagents to create effective separation of lead from zinc. Hence, the adoption of froth flotation method for the beneficiation of the Zurak ore sample.

4.10 The Froth Flotation Process

The results obtained during the course of the concentration tests are presented in the Tables 4.12 to 4.15 below.

Table 4.12: Pulp parameters of Zurak ore

Parameter	Value
Weight of ore	350g and 500g
Volume of water	1500cm^3 and 2000cm^3
Dilution ratio	4.29 and 4
Pulp density	23.33% and 25%

350g and 500g of the pre-concentrated ore wererespectively mixed with 1500cm^3 and 2000cm^3 to give 23.33% and25% of the pulp. Mineralogical studies and assaying showed that the lead ore – galena (PbS) contains significant amount of zinc in the form of sphalerite (ZnS) only. This makes the ore a simple one. Therefore in the beneficiation to recover galena (the lead mineral), efforts was made to also be made to recover the zinc mineral because of its significance.

4.10.1 Selectiveflotation of galena

Selective flotation was carried to recover the galena from the ore using potassium ethyl xanthate as collector and methyl isobutyl ketone as frother. Zinc sulphate and sodium silicate were used to depress the sphalerite and silica respectively. Lime was used to

maintain the pH at 9 and also to depress any pyrite present (USEPA 1994 & Wills, 2006).

4.10.2 Sequential flotation of sphalerite

Sequential flotation of sphalerite was carried out after the recovery of the galena. Lime was added to the pulp to raise the PH to 11 and also further depress any pyrite present. Copper sulphate was added to activate the sphalerite. Potassium ethyl xanthate and methyl isobutyl ketone as collector and frother respectively (USEPA 1994 & Wills, 2006).

Table 4.13: Metallurgical statement of distribution of metals of Zurak ore

Size range (μm)	Stream				Weight of metal (g) M x A		Distribution (%) MA/∑(MA)	
	Product	Weight(g) M	Assay (%) A					
			Pb	Zn	Pb	Zn	Pb	Zn
-90 +63	Feed	350.00	23.45	13.16	82.08	46.06	100.00	100
	Lead conc.	108.21	63.02	5.91	68.19	6.40	83.08	13.88
	Zinc conc.	52.15	5.78	57.92	3.01	30.21	3.67	65.58
	Tailings	163.09	1.14	2.28	1.86	3.72	2.27	8.07
-180 +125	Feed	500.00	23.45	13.16	117.25	65.80	100.00	100
	Lead conc.	127.08	55.79	15.84	70.89	20.13	60.47	30.59
	Zinc conc.	116.02	22.17	30.25	25.72	35.10	21.94	53.34
	Tailings	224.03	4.02	3.33	9.01	7.64	7.68	11.34

For the first batch of the flotation test at -90 +63μm, lead concentrate assaying 63.02%Pb, 5.78%Zn, andzincconcentrate assaying57.92%Zn, 5.91%Pb were obtained. While the second batch of flotation test at -180 +125μm, lead concentrate assaying

55.79%Pb, 22.17%Zn andzincconcentrate assaying30.25%Zn, 15.84%Pb were also obtained.

The high value of percent lead recovered in the lead concentrate could be attributed to the reagents used in the selective flotation of the lead mineral and to some extend the degree of freedom of the lead mineral in the ore sample. This phenomenon enables the migration and recovering of the lead mineral into the lead concentrate phase of the flotation process when compared to the zinc mineral recovery in the same phase. Similar reason can be advanced for the case of recovering of zinc and lead in the zinc concentrate produced. However, comparing the percent assays and recoveries of lead and zinc concentrates of the two different sieve sizes. Sieve size of -93+63µm produces the highest percent assays of 63.02%Pb and 57.92%Zn and recoveries of 83.08% and 65.58% against the percent assays of 55.79%Pb and 30.25%Zn and recoveries of 60.47% and 21.94% of sieve size of-180+125µm of lead and zinc concentrates respectively. This could be attributed to the reason that -93+63µm is the liberation size of the ore and hence the liberty for the high proportion of the minerals of interest to be retained in that sieve size. This phenomenon enhances the high percent recoveries of lead and zinc in the lead and zinc concentrates when compared to sieve size -180+125µm. The grades obtained from the beneficiation of the ore compared favorably with the metallurgical standard required for smelting of lean-zinc concentrate (Weiss, 1985; Wills, 2006; Ajayi, 2005).

4.11 Metallurgical Accounting Permutations

For the control of the concentrating process using the froth flotation method, metallurgical balance was used to determine the distribution of the various products and

the values contained in them and the gangue. Data obtained in the flotation process is shown in Table 4.14 and the metallurgical analysis carried out.

Table 4.14: Metallurgical Accounting for Zurak ore

Size range (μm)	Product	Weight (g)	Assay (%)	
			Pb	Zn
-90 +63	Feed	350	23.45	13.16
	Pb Conc.	P	63.02	5.91
	Zn Conc.	Z	5.78	57.92
	Tails	T	1.14	2.28
-180 +125	Feed	500	23.45	13.16
	Pb Conc.	P	55.79	15.84
	Zn Conc.	Z	22.17	30.25
	Tails	T	4.02	3.33

For the -90 +63μm particle size range, let,

F = 350g

$p_1 = 23.45\%, p_2 = 63.02\%, p_3 = 5.78\%, p_4 = 1.14\%$

$z_1 = 13.16\%, z_2 = 5.91\%, z_3 = 57.92\%, z_4 = 2.28\%$

Also, for the -180 +125μm particle size range, let,

F = 500g

$$p_1 = 23.45\%, p_2 = 55.79\%, p_3 = 22.17\%, p_4 = 4.02\%$$

$$z_1 = 13.16\%, z_2 = 15.84\%, z_3 = 30.25\%, z_4 = 3.33\%$$

Using the Three product (bi-metallic formula) for the -90 +63μm particle size range,

$$P = Fx\frac{(p_1 - p_4)(z_3 - z_4) - (z_1 - z_4)(p_3 - p_4)}{(p_2 - p_4)(z_3 - z_4) - (z_2 - z_4)(p_3 - p_4)}$$

$$= 350x\frac{(23.45 - 1.14)(57.92 - 2.28) - (13.16 - 2.28)(5.78 - 1.14)}{(63.02 - 1.14)(57.92 - 2.28) - (5.91 - 2.28)(5.78 - 1.14)}$$

$$= 350x\frac{22.31x55.64 - 10.88x4.64}{61.88x55.64 - 3.63x4.64} = 350\ x\frac{1241.33 - 50.48}{3443.00 - 16.84}$$

$$= 350x\frac{11190.85}{3426.16} = 121.65$$

∴ P = 121.65g of Lead Concentrate

Similarly,

$$Z = Fx\frac{(z_1 - z_4)(p_2 - p_4) - (p_1 - p_4)(z_2 - z_4)}{(z_3 - z_4)(p_2 - p_4) - (p_3 - p_4)(z_2 - z_4)}$$

$$= 350x\frac{(13.16 - 2.28)(63.02 - 1.14) - (23.45 - 1.14)(5.91 - 2.28)}{(57.92 - 2.28)(63.02 - 1.14) - (5.78 - 1.14)(5.91 - 2.28)}$$

$$= 350x\frac{10.88x61.88 - 22.31x3.63}{55.64x61.88 - 4.64x3.63} = 350\ x\frac{673.25 - 80.99}{3443.00 - 16.84}$$

$$= 350x\frac{592.26}{3426.16} = 60.50$$

∴ Z= 60.50g of Zinc Concentrate

Also, for the -180 +125μm particle size range,

$$P = 500x\frac{(23.45 - 4.02)(30.25 - 3.33) - (13.16 - 3.33)(22.17 - 4.02)}{(55.79 - 4.04)(30.25 - 3.33) - (15.84 - 3.33)(22.17 - 4.02)}$$

$$= 500x\frac{19.43x26.92 - 9.83x18.15}{51.77x26.92 - 12.51x18.15} = 500\ x\frac{523.06 - 178.41}{1393.65 - 227.06}$$

$$= 500x\frac{344.65}{1166.59} = 147.72$$

∴ P =147.72g of Lead Concentrate

And,

$$Z = 500x\frac{(13.16-3.33)(55.79-4.04)-(23.45-4.04)(15.84-3.33)}{(30.25-3.33)(55.79-4.04)-(22.17-4.02)(15.84-3.33)}$$

$$= 500x\frac{9.83x51.77-19.43x12.51}{26.92x51.77-18.15x12.51} = 500\,x\frac{508.90-243.07}{1393.65-227.06}$$

$$= 500x\frac{265.83}{1166.59} = 113.93$$

∴ Z =113.93g of Zinc Concentrate

4.11.1 Recoverypermutations

The recovery in a mineral concentrating operation is the percentage of the total metal content in the ore that is recovered in the concentrate. The purpose of calculating recovery is to determine the distribution of the metal in the feed or heads among the various products of the mineral concentrating operation. Below is the permutation for the recovery of the minerals of interest (lead and zinc) of the flotation process.

$$\text{Recovery, } R = \frac{Cc}{Ff}x100$$

Thus, for the -90 +63µm particle size range,

$$\text{Recovery of Lead, } R_P = \frac{Pp_2}{Fp_1}x100 = \frac{121.65x63.02}{350x23.45}x100 = 93.41\%$$

$$\text{Recovery of Zinc, } R_Z = \frac{Zz_3}{Fz_1}x100 = \frac{60.50x57.92}{350x13.16}x100 = 76.08\%$$

Also, for the -180 +125µm particle size range,

$$\text{Recovery of Lead, } R_P = \frac{147.72x55.79}{500x23.45}x100 = 70.29\%$$

$$\text{Recovery of Zinc, } R_Z = \frac{113.93x30.25}{500x13.16}x100 = 52.38\%$$

The recovery of lead and zinc to the galena and sphaleriteconcentrates, at -90 +63µm particle size rangeare93.41% and 76.08% respectively.This means that 93.41% of the lead mineral and 76.08% of zinc mineral in the run-of-mine ore were recovered into the concentrates. Similarly, at -180 +125µm size range, the recovery of lead and zinc to the galena and sphalerite concentratesare 70.29% and 52.38% respectively.

It can therefore be concluded that the finer the ore material, the higher the recovery.

4.11.2 Ratio of concentration permutations

The ratios of concentration is the weight of the feed (heads) to the weight of the concentrate.

$$\text{Ratio of Concentration, } K = \frac{F}{C}$$

Hence, for the -90 +63µm particle size range,

$$\text{Ratio of Concentration of Lead, } K_L = \frac{F}{P} = \frac{350}{121.65} = 2.88$$

$$\text{Ratio of Concentration of Zinc, } K_Z = \frac{F}{Z} = \frac{350}{60.5} = 5.79$$

Similarly, for the -180 +125µm particle size range,

$$\text{Ratio of Concentration of Lead, } K_L = \frac{500}{147.72} = 3.38$$

$$\text{Ratio of Concentration of Zinc, } K_Z = \frac{500}{113.93} = 4.39$$

The ratios of concentration for lead and zinc are 2.88 and 5.79 respectively at -90 +63µm particle size range. Similarly, the values are 3.38 and 4.39 respectively for -180 +125µm particle size. This means that the ratio of the weight of the feed (heads) to the weight of the concentrateare 2.88 and 5.79 respectivelyfor lead and zinc at -90 +63µm particle size range; 3.38 and 4.39 respectively for -180 +125µm particle size.

The ratio of concentration is sometimes referred to as the grade of the concentrate and itis a measures of the efficiency of the froth flotation using conventional collectors.

4.12 Cost -Profit Analysis of Mining and Processing of the ZurakOre

Value of the mineral (Overall recovery)

Value = Assay x Recovery x price (Hartman, 1987)

Profit = Value – cost (Hartman, 1987)

4.12.1 Lead

Value of Lead = 0.23 x 0.93 x ₦340,000/tonne

= ₦72,726/tonne

Profit = ₦72,726 – 45,000 = ₦27,726/tonne

= 61.61%

4.12.2 Zinc

Value of zinc = 0.13 x 0.76 x ₦270,000/tonne

= ₦26,676/tonne

Profit = ₦26,676 – 16,000 = ₦10,676/tonne

= 67.73%

4.13 The Developed Flowsheet of Zurak Lead-Zinc Ore

A flowsheet is a pictorial representation showing all the steps in mineral concentration process (Kumar, 2001) which facilitates visual comprehensionusing symbols to denote the machines and other features. Analysis of a flowsheets shows the bottlenecks,

potential sources of troubles, loopholes in process/system and identifies the critical units in the process.

In designing the suitable flow sheet for the concentration of an ore, the primary grind size was the major consideration.This is mainly because the flotation response depends on the level of liberation of the minerals in the ore. The purpose of the primary grind is to promote economic recovery of the valuable minerals (Wills, 2006). The target grind size was estimated based on previous works and literature on lead-zinc ore concentration (Wills, 2006); and from the mineralogical evaluation of the Zurak lead ore deposit.

The principal economic minerals of the Zurak ore are galena (PbS) and sphalerite (ZnS), iron being slightly present as magnetite.

The process of beneficiation of Zurak lead ore deposit consists of two stages. The first stage is the comminution process where the run-of-mine ore(6.33% Pb,4.65% Zn) is crushed using jaw crusher. The discharge from the crusher is screened, the oversize (+2000μm) is recycled back to the crusher while the undersize (-2000μm) passed on to rod mill for further grinding to pass -90+63μm (23.45%Pb, 13.16%Zn). The discharged from the rod mill is conveyed to the flotation units where the second stage of the process begins. The second stage is the flotation process, where ground 80% passing,-90+63μm is used form a pulp 25% solid and fed to the flotation cell. The pulp is conditioned by adding reagents (collector:Potassium-ethyl-xanthate at 0.12kg/tonne; depressant: Zinc-sulphate, at 0.2kg/tonne, activator: lime at 1kg/tonne and frother:MIBK at 3drops) and the pH was maintained at 9. Air was introduced into the flotation cell by opening the valve and lead concentrate floated and cleaned. The tailing from the lead flotation is reconditioned with reagentsat a pH of 11 with the introduction of air, zinc concentrate

floated and cleaned. The gangue consisting essentially of quartz and kaolinite are discarded.

The finalstage of lead flotation is the de-zincing of thelead cleaner concentrate. The zinc mineral is activated with copper sulphate and the galena is depressed by raising the pH to 11 by the addition of lime. A rougher de-zincer concentrate is cleaned and the dezincer tailing is the final lead concentrate(63.02%Pb, 5.91%Zn) with lead recovery of 83.08%. The lead scavenger tailing is conditioned with copper sulphate, prior to feeding to zinc rougher flotation where reagents specified in Table 4.14 are added to the cell. A rougher concentrate is floated at pH 10.6 and fed to acleaner where is collected as zinc concentrate(57.925%Zn, 5.78%Pb) with zinc recovery of 65.58%. The developed flowsheet for the beneficiation of the Zurak lead ore deposit is shown in Figure 4.5.

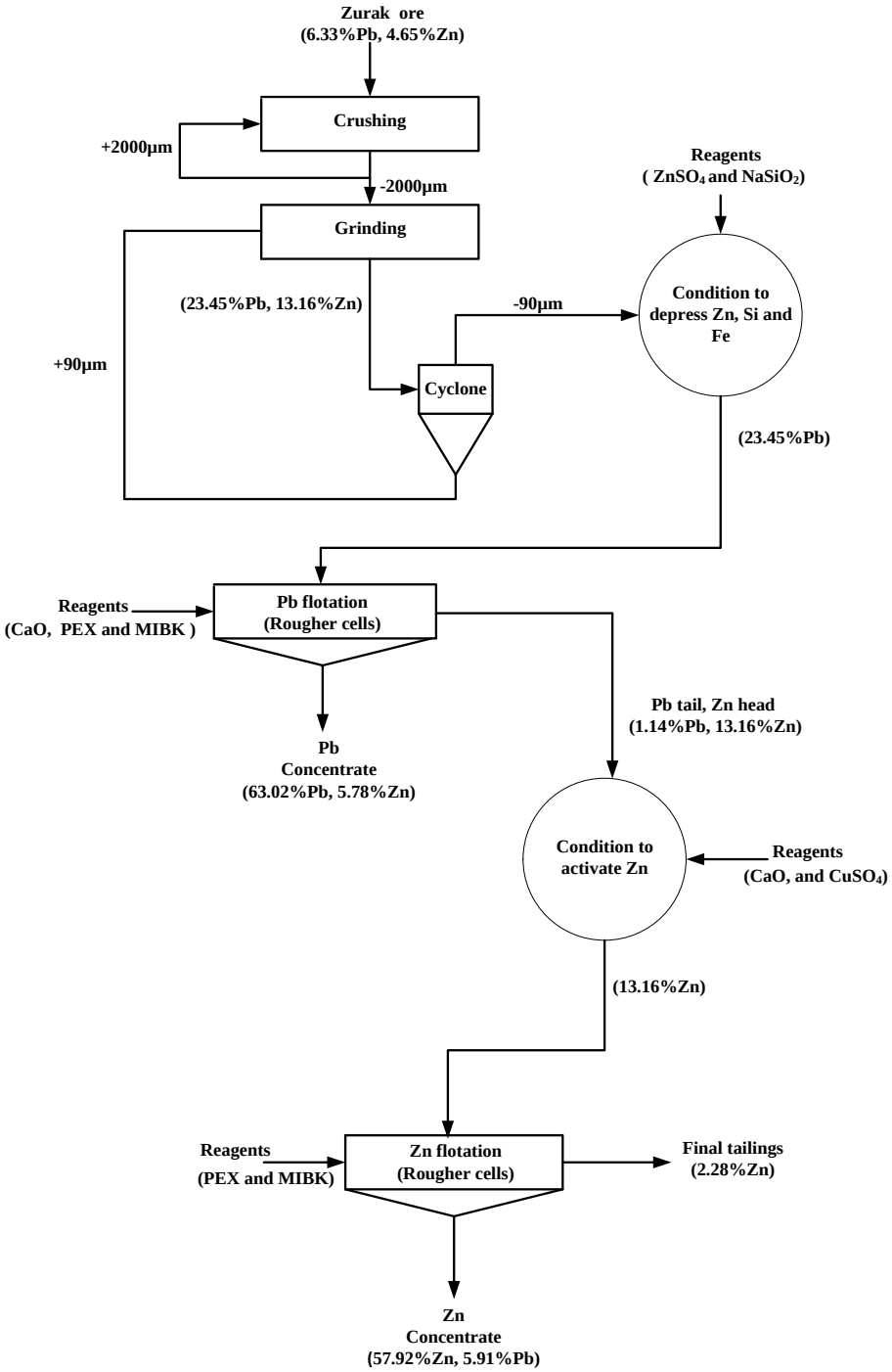

Figure 4.5: The developed flowsheet for the beneficiation of Zurak lead-zinc ore.

CHAPTER FIVE

5.0 CONCLUSION AND RECOMMENDATIONS

5.1 Conclusion

The occurrence of the lead deposit and its associated minerals is of great importance to the socio-economic development of Zurak Community, Wase Local Government Area, Plateau State and Nigeria in general. The Zurak ore deposit is predominantly made up of lead and zinc in the forms of galena (PbS) and sphalerite (ZnS) assaying 6.33% lead and 4.65%Zn as the major minerals constituents. Other minerals associated with the ore are magnetite (Fe_3O_4), rutile (TiO_2), quartz and kaolinite ($KAl_3Si_3O_{10}(OH)_2$) as minor minerals constituents. The liberation size of the ore was found to be -90+63µm and the work index of the ore was also found to be 11.41Kwh/tonne making the ore a type B grade(medium soft texture ore) using the grindability curves graph. The concentration criterion (CC) of ore was found to be 1.87 an indication that, the ore cannot be concentrated using the gravitymethod. However, literature has shown that froth flotation is the most suitable method of concentrating lead-zinc minerals. The first batch of the flotation test at -90 +63µm produced lead and zinc concentrates assaying 63.02%Pb and 57.92%Zn with recoveries of 83.08% and 65.58%, while the second batch at -180 +125µm, produced lead and zinc concentrates assaying 55.79%Pb and 30.25%Zn with recoveries of 60.47% and 21.94% respectively. All the grade of lead and zinc concentrates obtained met the metallurgical requirement required for smelting lead and zinc concentrates.

The flowsheet of Sullivan concentrator of Cominco Ltd, British Columbia, was adopted and modified as the conceptual flow sheet for the beneficiation of Zurak lead ore deposit.

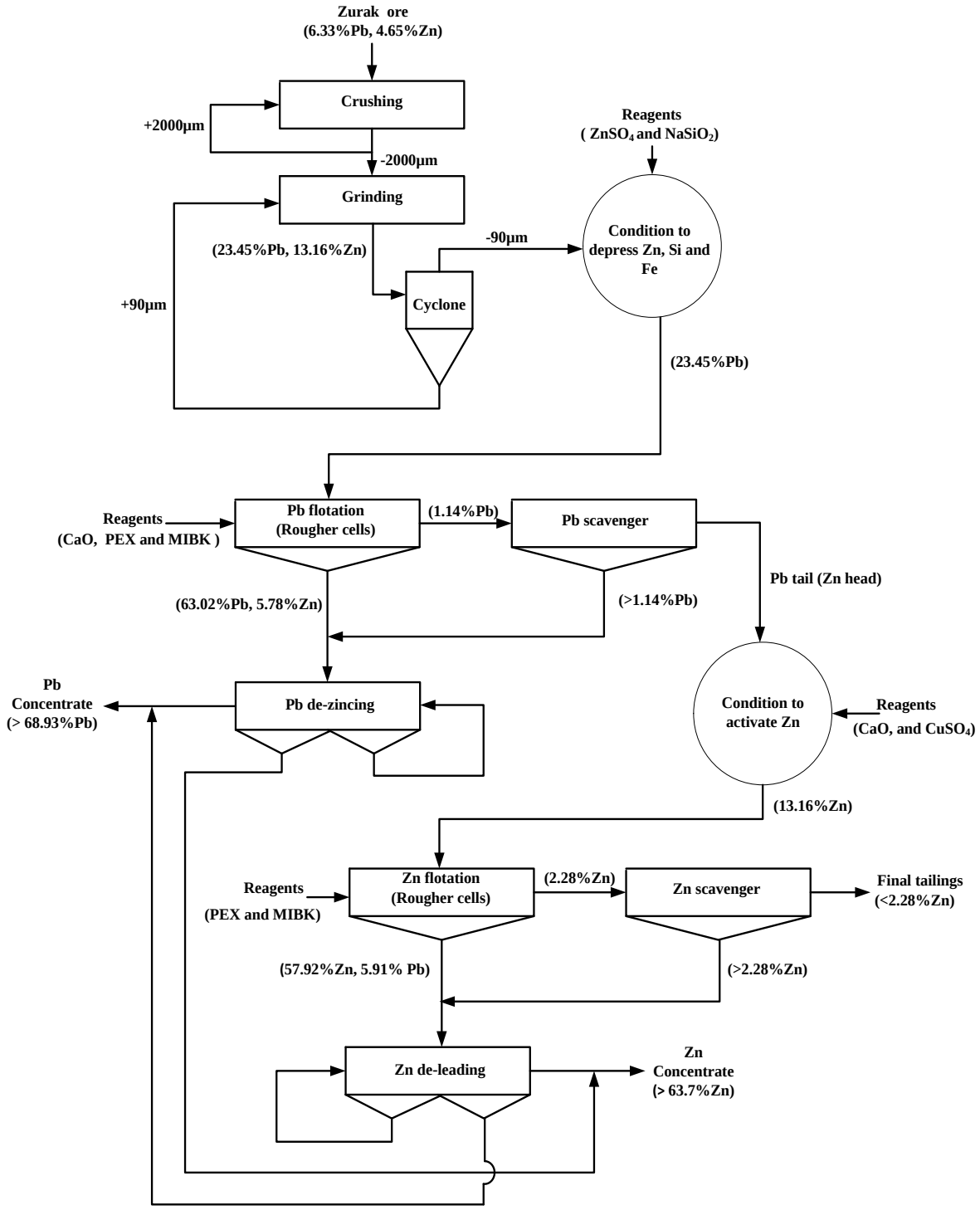

Figure 5.1: Proposed conceptual flowsheet for Zurak lead-zinc ore

5.2 Contribution to Knowledge

Presently, there is no previous work on the characterization and beneficiation of Zurak lead-zinc ore. This research work specifically contribute to knowledge in the following ways:

1. provides information on the chemical and mineralogical characteristics of the Zurak lead-zinc ore,
2. the determined liberation size, work index and suitable concentration technique for beneficiation of Zurak lead-zinc ore will be known,
3. the process route for the beneficiation of the zurak lead/zinc ore will be established.

5.3 Recommendations

1. The beneficiation of the ore using locally made reagents should be investigated to reduce the dependence on standard imported reagents.
2. the kinetic and the chemistry of interaction of the standard and local reagents on the froth flotation of the Zurak should be investigated.
3. The Federal Government should as a matter of urgency, enact a deliberate policy that will compel all investors in the mining industry to add value to minerals in the country before they are exported.

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APPENDIX A

List of Photographs

Plate A-I: Bico jaw crusher

Plate A-II: Jones riffle

Plate A-III: Bico rod mill

Plate A-IV: Endecotts sieve shaker

Plate A-V: Bico pulverizer

Plate A-VI: Wemco flotation cell

Plate A-VII: Measuring water pH using adigital pH metre

Plate A-VIII: Mineral laden froth

PlateA-IX: Weighing reagent using a digital weighing beam

PlateA-X: Flotation reagents used in the flotation of Pb-Zn ore

PlateA-XI: Scoopingof mineral laden froth

PlateA-XII: Gallenkamp oven used for drying concentrates and tailings