KWAME NKRUMAH UNIVERSITY OF SCIENCE AND TECHNOLOGY

COLLEGE OF ENGINEERING

DEPARTMENT OF MATERIALS ENGINEERING

APPLICABILITY OF CLAY POZZOLANA IN HYDRAULIC BACKFILLING OF STOPES AT THE ANGLOGOLD ASHANTI, OBUASI MINE

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A THESIS SUBMITTED TO THE DEPARTMENT OF MATERIALS ENGINEERING IN PARTIAL FULFILMENT OF THE REQUIREMENTS FOR THE AWARD OF MASTER OF SCIENCE DEGREE IN ENVIRONMENTAL RESOURCE MANAGEMENT.

By

SAMUEL NANA ESHUN BSc (HONS)

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CERTIFICATION

It is hereby declared that this thesis is my own work towards the MSc. Degree and that, to the best of my knowledge, it contains neither material previously published by another person nor material which has been accepted for the award of any degree of the University, except where due acknowledgements have been made in the text.

Samuel Nana Eshun ……………….……. ..…………………

(5832711) Signature Date

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Certified by:

Dr Simon Yawo Gawu ……………….……. ..…………………..

Signature Date

Certified by:

Professor Samuel Kwofie ……………….……. ..…………………..

Signature Date

ABSTRACT

The quest for a cheaper alternative or supplement to ordinary Portland cement as binder in the backfilling of mined out stopes in underground mines motivated this study. Clay pozzolana has been proven to substitute about 30% of Portland cement in general construction works and is cheaper. However, very little is known about its applicability in mine backfill. This study therefore sought to investigate whether clay pozzolana could be applied in the case of hydraulic backfill in the AngloGold Ashanti, Obuasi Mine. Samples of the classified tailings from the AngloGold Ashanti, Obuasi Mine were prepared with different amounts of ordinary Portland cement and clay pozzolana combinations into 50mm diameter by 120mm high cylindrical specimens and then tested for compressive strength after curing under humid conditions for various periods. Samples of the tailings were also subjected to particle size distribution and sulphate content analyses.

The results indicated that generally, backfill with 10%, 25%, 30% and 35% of the ordinary Portland cement content replaced with clay pozzolana had strength values comparable to those obtained for samples containing ordinary Portland cement alone. Ten per cent (10%) pozzolana content replacement showed consistent maximum strength characteristics after curing beyond 7 days. Therefore, in terms of strength; it is recommended that 10% of ordinary Portland cement could be substituted with clay pozzolana in the hydraulic backfilling of voids in the AngloGold Ashanti, Obuasi Mine. The particle size distribution analysis indicated that the tailings are suitable in terms of its drainage characteristics for hydraulic backfilling of voids. The results of the sulphate content analysis indicated that the sulphate content of the tailings obtained from AGA, Obuasi Mine for the study was higher compared with the upper limit set by ASTM Standard C94 and European

Standard EN 1008.Using 2013 as the baseline, an amount of Gh¢12.00 would be saved on every tonnage of cement used. This represents 3% of the amount of money spent on every tonnage of cement used.

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LIST OF ABREVIATIONS

AGA – AngloGold Ashanti

ASTM – American Society for Testing and Materials

BRRI – Building and Road Research Institute

CRF – Cemented Rock Fill

CSIR – Council for Scientific and Industrial Research

CSH – Calcium sulphate hydrate

CU – Coefficient of uniformity

GCS – George Capendelle Shaft

GGBS – Ground granulated blast furnace slag

KMS – Kwesi Mensah Shaft

LHD –Load haulage dump

OPC – Ordinary portland cement

SCM – Supplementary cementitious material

SLC – Sub level caving

STP – South treatment plant

UCS – Uniaxial compressive strength

UG – Underground

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CHAPTER 1: INTRODUCTION

1.1 Background

Over the past hundred years, underground mining for gold has been on-going at Obuasi in the Ashanti Region of Ghana. In the years preceding the 1990s, ore recovery in the underground mine at Obuasi depended only on sublevel caving method of mining, which was self-filling. A restructuring programme to increase production and modernize the mine led to the adoption of the open stoping method of mining in most blocks of the mine. The sublevel caving method is associated with development of sinkholes. Currently, only one out of the eight active blocks of the mine is being mined with the sublevel caving method. The open stoping method of mining allows more ore to be recovered at a time, hence results in the creation of large voids which require backfilling. Mine backfill, therefore, has become an integral part of the operations in the Obuasi mine.

Backfill is the term for material used to fill voids (empty stopes) created by mining activity (De La Vergne, 2003). The roles of the backfill include:1) Provision of support to the walls and thus confines the fractured rock and preventing unravelling; 2) improving the stability of the ground for further ore extraction; 3) providing a working platform on which miners and equipment can operate; 4) acting as a method of mine waste disposal. From the perspective of waste disposal, Grice (1998) defined backfill as any waste material that is placed into voids mined underground for the purposes of either disposal or to perform some engineering function. The waste materials are often placed with very lean cement or other pozzolanic binders to improve the strength properties.

Over the period of hydraulic backfill operations in Obuasi, Portland cement has been used as the binding material in the backfill preparation. The cost of Portland cement, however, continues to increase, and hence has increased the operational cost. This has led to the search for alternative binding material that can supplement or partially substitute cement, and at the same time, achieve strengths comparable to those currently achievable with the use of ordinary Portland cement alone in the backfilling operations. Grice (1998) reports that the cost of cement can represent more than half of the operating cost of the backfill system. According to him, there has been continuous research around the world into alternative binders and the most common solution has been the use of pozzolans such as fly ash and blast furnace slags.

Grice (1998) indicates that Mount Isa Mine in Australia uses ground copper furnace slag to replace half of the cement in backfill and save over 25,000 tonnes of cement annually as a result. He also indicated that Olympic Dam also utilises fly ash from Port Augusta. In his presentation on the filling of 'The Abandoned Mine Land' in North Dakota, Dodd (2000) estimated that nearly $500,000, representing 18% of the total pressurized grout remote backfilling projects cost was saved by using fly ash over a period of five years. Currently, slag produced in northern Ontario is used primarily for mine backfill operations in that region (Shnorhokian, 2009).

Today, pozzolana, a supplementary binding material to Portland cement developed by the Building and Road Research Institute (BRRI) of the Council for Scientific and Industrial Research, Ghana, is being manufactured in the country. Pozzolana which is produced by calcining clay is found to cost less compared to Portland cement. In Ghana, clay is abundant and found in all regions (Kesse, 1985).

As at the end of 2013, the cost of 50kg bag of ordinary Portland cement in the Obuasi Municipality was Gh¢20.00 whilst for clay pozzolana it was Gh¢14.00 for the same weight. Based on the above market figures, for each tonnage of cemented hydraulic fill placed underground at a binder utilization rate of 5%, if a minimum of 10% of the cement is substituted with clay pozzolana, Gh¢0.60 is saved. Therefore, in a particular month if the total cemented hydraulic fill tonnage is 35,000 the company can save up to Gh¢ 21,000.00

1.2 Problem statement

Portland cement is the binding material used in combination with the mine tailings and water for the cemented hydraulic backfilling of the mined out voids in the AngloGold Ashanti (AGA), Obuasi Mine. However, the cost of Portland cement continues to increase, making the cost of backfill operations increasingly expensive. In Ghana, the cost of Portland cement has increased over 500 percent within the last ten years (Adu-Boateng and Bediako, 2006). The situation is compounded by progressive increase in cement quantities required for hydraulic backfill on the mine year after year. This is as a result of progressive increase in the voids that require cemented hydraulic fill.Currently, the average cost of cement per ton of hydraulic fill placed underground in the Obuasi mine ranges between Gh¢20 and Gh¢22.5.

The tailings used for the hydraulic backfill in the AngloGold Ashanti, Obuasi Mine is obtained by milling sulphide-based ore and, therefore, very rich in sulphate. Sulphate attacks products of cement hydration and weakens the resulting backfill. Pozzolans are, however, known to increase the resistance to sulphate attack and durability of concrete products (Thevarasa et al, 1979; Mertens et al, 2008).

1.3 Significance of the study

The importance of the study is pivoted around economics, technology and advancement of the knowledge about the use of supplementary binding materials in hydraulic backfill.

Over the past six years, AngloGold Ashanti, Obuasi Mine spends an average of US$ 2,089,912 annually on Portland cement for hydraulic backfill. Any reduction in the quantity of cement consumed annually that will arise from successful application of a low cost supplementary binder such as clay pozzolana, will be to the economic advantage of the Gold mining company. Utilization of the clay pozzolana by AngloGold Ashanti Mine will promote local building materials development.

Technologically, this study establishes the suitability or otherwise of clay pozzolana for hydraulic backfilling of mined out voids in underground mine. The study also adds to the stock of knowledge in the area of the use of supplementary cementitious materials for hydraulic backfill.

1.4 Objectives

The main focus of the study was to evaluate the potential of using clay pozzolana as a partial substitute for Portland cement to produce hydraulic backfill.

The specific were:

1. To evaluate whether clay pozzolana and ordinary Portland cement mixture used as binder, will produce strength of backfill comparable to those obtained from ordinary Portland cement (OPC) alone.

2. Determination of the optimum amount of clay pozzolana that can replace cement to produce backfill of adequate strength.

3. Evaluation of the amount of money that could be saved by applying clay pozzolana in backfill at the Obuasi AngloGold Ashanti Mine.

1.5 Scope of work

This study was to establish how the pozzolana made principally of clay in Ghana can be applied in combination with Portland cement in the hydraulic backfilling of the mined out voids at the AngloGold Ashanti, ObuasiMine.

The scope of activities involved the determination of compressive strength of backfill specimens prepared with only ordinary Portland cement and classified tailings from the AGA, Obuasi Mine. It also involved determining the compressive strength of backfill specimens prepared from ordinary Portland cement with 10 to 40 percent replacement with clay pozzolana. Analysis of the particle size distribution of the tailings was also undertaken. Cost benefit in the use of clay pozzolana for the hydraulic backfill operations in the AGA, Obuasi Mine was also analysed.

CHAPTER 2: LITERATURE REVIEW

2.1 Mining Methods

There are two distinct types of mining methods. They are stable stope and caving (Rankine, 2002). In between the two extreme methods is a spectrum of others. According to Hustrulid and Bullock (2001), these include Room and Pillar; Vein Mining; Long Wall Mining; Block Caving; Shrinkage Stoping; Sublevel Open Stoping; Bighole Stoping and Vertical Crater Retreat. Fernberg (2007) listed in addition to the above, cut and fill mining method.

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2.1.1 Stable Stope Mining

Hustrulid and Bullock (2001) describe stope as underground excavation made by removing ore from surrounding rock. There are three stable stope methods. Namely, open stoping, cut and fill and room and pillar mining methods (Rankine, 2002).

In an open stoping mining operation, ore body is divided into separate stopes for mining (Rankine, 2002). Between stopes, ore sections are set aside for pillar to support the hanging wall (Hustrulid and Bullock, 2001). The solid rock within each stope is blasted and the fragments removed through created accesses called crosscuts and drives, for processing, thus leaving empty stopes or voids that need to be filled.

Cut-and-fill mining is applied to mining steeply dipping ore bodies in strata with good to moderate stability, and a comparatively high-grade mineralization (Fernberg, 2007). Cut-and-fill mining excavates the ore in horizontal slices, starting from a bottom undercut and advance upward. Fernberg (2007) explained that the ore is drilled, blasted, loaded and removed from the stope, which is then backfilled with deslimed sand tailings from the dressing plant, or waste rock carried in by Load Haul Dump (LHD) from development drives.

Room and pillar method of mining is designed for flat bedded deposits of limited thickness. The method leaves pillars to support the hanging wall. The pillars are not recoverable (Hustrulid and Bullock, 2001). For general mine stability and to limit surface subsidence, backfill is introduced between pillars after the ore has been mined.Pillars are designed to fail after backfill has been placed around them (Masniyom, 2009).

2.1.2 Caving Mining

Caving mining requires a rock mass where both ore body and host rock fracture under controlled conditions. As the mining removes rock without backfilling, the hanging wall carries on caving into the voids. Continuous caving is important, to avoid creation of cavities inside rock, where a sudden collapse could induce an inrush (Fernberg, 2007).There are two types of caving mining. They are sublevel caving (SLC) and block caving.

Sublevel caving adapts to large ore bodies, with steep dip and continuity at depth. SLC extracts the ore through sublevels, which are developed in the orebody at regular vertical spacing. Each sublevel features a systematic layout with parallel drifts, along or across the ore body (Fernberg, 2007).

Block-caving is a large scale production mining method applicable to low grade, massive ore bodies with: (1) large dimensions both vertically and horizontally, (2) rock mass that can be broken into blocks of manageable sizes, (3) ground surface which is allowed to subside (Fernberg, 2007).

2.2 Mine backfill

Mine backfill refers to waste materials, such as waste development rock, deslimed and whole mill tailings, quarried and crushed aggregate and alluvial or an eolian sand, which is placed into underground mined voids for the purposes of either disposal and/or to perform some engineering function (Benzaazoua et al, 2002). Sargeant (2008) simply describes "backfill" as an engineering material that provides a level of ground support that allows for safe and economic operation of underground mines. According to Fernberg (2007), backfill affords the opportunity for more selective mining and better recovery of ore, thereby increasing both mine life and total return on investment. Backfill is also considered an essential tool to help preserve the structural integrity of the mine workings as a whole, and to help avoid stressing ground to the point where rock bursts take place (Fernberg, 2007).

In the early twentieth century, backfill consisted of hand and pneumatically placed sand, which did not provide the level of support required (Sargeant, 2008). Grice (1998) indicates that one of the earliest records of backfilling in Australia, as a discrete technique, was the placement of aggregate from lead jig wastes at Mount Isa in 1933, tipped directly from the mill by conveyor to square set timber stopes.

Attempt in the early twentieth century to improve backfill performance and its ability to support load resulted in the use of timber to provide confinement.The need to further improve performance led to the introduction of binders in backfill development (Petrolita et al, 2005, cited in Sargeant, 2008).

The first recorded use of Portland cement in backfill occurred in 1957 by Falconbridge Ltd at the Hardy Mine in Sudbury (Petrolita et al, 2005, cited in Sargeant, 2008). The Portland cement used was cheaper and resulted in the development of a backfill with improved strength properties. Backfill strength is required in stopes to maintain a free standing height of fill in the primary stope while ore in the secondary stope is extracted (Sargeant, 2008).

Sometimes backfill only acts as void filler and needs only sufficient strength to prevent any form of remobilisation. Where backfill is used as an engineering material, it requires sufficient strength to be exposed by ore pillar mining in tall vertical faces or undercuts (Grice, 1998). Backfill methods are used when ground cannot be left open for any length of time after ore has been mined out (Masniyom, 2009). Lean cement addition to the waste material is used to generate unconfined compressive strengths ranging from 0.5MPa to 4 MPa.

2.2.1 Types of backfill

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According to Grice (1998), there are three major traditional backfill methods. These are paste backfill, hydraulic backfill, and rock fills. Cooke (2007), on the other hand, states that there are two predominant backfill types. According to him, they are hydraulic fill and paste fill. He argues that backfill types that do not fall into this classification are generally applications where material is being disposed of underground for environmental reasons and that backfill is not an integral part of the mining operation.

Sivakugan (2008) indicates that mine fills can vary from boulder-size aggregates to very fine clay fractions. Sometimes a small dosage (e.g. 3%-5%) of pozzolanic binder such as cement, fly ash, gypsum or slag is added to the mine fill to improve stability.

The hydraulic fill, cemented hydraulic fill and paste fill are generally placed within the stopes in the form of slurry at 65%-80% solid content by weight. They are mixed at a plant far away and are transported through pipes and bore holes into the stopes (Sivakugan, 2008).

According to Fernberg (2007), a host of factors have to be taken into consideration when designing a backfill regime. These are the geology and dimensions of the ore body and its dip and grade; the physical and mechanical properties of both the ore and its host rock; environmental considerations; fill material resources; mining method; production capacity and operations schedules.

2.2.2 Paste backfill

Paste backfills were developed originally by Preussag in Germany and utilised at the Bad Grund Mine in the late seventies (Grice, 1998). Development in South Africa and Canada substantially refined the system (Grice, 1998). Paste backfill consists of the full size fraction of the tailings stream prepared at a high slurry density. The slurry behaves as a non-Newtonian fluid, which means that it requires an applied force to commence flowing (Grice, 1998). Paste fill originally used non-cycloned mill tailings mixed with cement. Fernberg (2007) reports that coarse tailings permit very high solids content of up to 88% to be pumped at high pressure, and high setting strengths were achieved. Grice (1998) reports that higher slurry densities improve the water to cement ratios and for a given tonnage of pastefill, lesser quantities of cement are required to achieve comparable strengths to hydraulic backfill. Reduced permeability due to the inclusion of fines also results in negligible water drainage from the paste fill. The typical range of slump values used in paste fill applications is between 150 mm to 250 mm (Cooke, 2007).

2.2.3 Hydraulic backfill

Hydraulic backfill can be defined as any kind of backfill carried by water through pipelines into an underground mine. Following the ore extraction or milling process, the tailings are classified according to particle size in devices known as hydrocyclones (Baker, 1986). This process produces separate slimes and sand fractions. Suitable tailings for hydraulic fill should not contain more than 10% by weight of the size fraction less than 10µm in size (Grice, 1998). This is done to improve the drainage characteristics of the fill. Typically, 100 mm per hour is required (Cooke, 2001). The slimes are disposed of due to their poor permeability and are generally stored in a surface storage facility.

Hydraulic backfill slurries are transported by gravity through boreholes and pipelines to the stopes (Grice, 1998). Where the hydraulic fill is to be exposed, cement addition rates of around 6% are typically required. Hydraulic fill could, therefore, be cemented or uncemented depending on the required function. Bulkheads are required to initially contain and drain the excess water from hydraulic backfill (Straskraba and Abel, 1994).

Sargeant (2008) notes that hydraulic backfill is usually prepared with a dry solids content of approximately 55 – 65 per cent. Grice (1998) and Baker (1986) on the other hand, argue that the solid concentration can be raised to over 70 per cent by weight. Over the past decade, there has been a steady increase in the solid content of the hydraulic fill slurry placed in mines in an attempt to reduce the quantity of water that must be drained and increase the proportion of solids. Currently, solid contents of 75% to 80% are common (Sivakugan et al., 2005).

Grice (1998) notes that approximately 10 per cent cement is commonly added to the slurry for capping of the back fill. This practice increases the strength of its upper surface and promotes faster development of a stope (Baker, 1986). Hydraulic fills are usually the most effective in reducing mining induced ground movements (Straskraba and Abel, 1994).

2.2.4 Dry rock fill

Rock backfill is a technology which transports backfill materials such as stones, gravels, soil or industrial solid waste through manpower, gravity or mechanical equipment in order to form compressible backfill body (Yao et al., 2012). Rock fill is used for filling secondary and tertiary stopes, and is usually a convenient and economic means of disposal of waste from development (Fernberg, 2007). Dry rock fill is most suited for the cut and fill mining method. The main advantage of this method is that it is inexpensive (Crandall, 1992). The disadvantage, however is that it may result in relatively loose, uncompacted fill. Unconsolidated waste fill can result in unsafe caving or sloughing conditions if too much fill face is exposed.

2.2.5 Cemented rock fill

Grice (1998) described cemented rock fill as a blend of cemented hydraulic fill and waste or crushed rock deposited in underground voids. It is used when storage of waste rock is required and the excess void spaces need filling.

Cemented rock fill (CRF) originally consisted of spraying cement slurry or cemented hydraulic fill on top of stopes filled with waste rock, as practiced at Geco and Mount Isa Mines (Fernberg, 2007). Nowadays, cement slurry is added to the waste rock before the stope is filled.

According to (Grice, 1998) the advantages of the cemented rock fill, are: (1) it trebles the tonnage rate of filling (doubling the volumetric rate); (2) itincreases the strength performance; (3) it reduces equivalent cement consumption to below 2% by weight.

2.3 Constituents of backfill

Backfill can be described as a mix of classified or unclassified tailings, a binding material and water. The strength development and the ability of backfill to perform according to expectation depend, to a large extent, on the characteristics of the constituent materials and the control of the mix.

2.3.1 The tailings

The extraction of metals from the earth involves the fragmentation, crushing, and grinding of rocks that host the ore in order to liberate it. The goal is to facilitate the separation of the gold from the gangue material (Shnorhokian, 2009).

In AGA, Obuasi Mine, milling is done by combination of impact and abrasion processes to produce particle sizes having 80% passing 75μm sieve size (AGA Obuasi Mine Metallurgical Services Manual, 2005). After milling, the gold bearing sulphide material is floated and sent to the bio-oxidation section for further treatment. The remaining waste after extracting the valuable minerals is called tailings. Tailings is one of the major raw materials for hydraulic and paste backfilling (AGA Obuasi Mine Backfill Operations Manual, 2012). Tailings can range from fine sand down to clay-sized particles, and the final sizing is dependent on the level of grinding carried out during processing (Potvin et al, 2005)

Depending on the type of backfill, the tailings may be classified or may not. The classification makes it possible for a range of particle sizes consistent with the desired type of backfill and required engineering properties to be obtained. According to Sargeant (2008), the hydraulic fill uses the mill tailings minus the finer fraction but the paste backfill incorporates the full range of particles including the ultra-fine. According to Archibald (2003), the material employed for hydraulic backfill preparation should comprise of 15% or less of aggregate blend size finer than 20µm. Grice (1998) reports that to ensure that acceptable permeability of the placed hydraulic fill is achieved, not more than 10% by weight of the tailings should be of particle sizes less than 10µm.

In the account of Sargeant (2008), the particle size distribution can provide some information about the engineering properties of the tailings, defined by its Coefficient of Uniformity (C_u). Well graded tailings (C_u = 4 – 6) have a wider range of particle sizes and are associated with lower void ratios. Uniformly graded tailings (C_u = 1) have a narrower range of particle sizes and are associated with higher void ratios (Sargeant, 2008). He further states that generally, hydraulic backfills are prepared from uniformly graded tailings while paste backfills are prepared from well graded tailings. Depending on the degree of crushing and grinding employed to liberate the valuable mineral, the final particle size distribution of the tailings may vary from mine to mine or between the types of ores (Potvin et al, 2005).

According to (Zhou et al., 2004) gold ores which form the source of the tailings are commonly classified by the metallurgist into two major categories. Namely, free milling, and refractory ores. Zhou et al (2004) indicates that typically, free-milling ores are defined as those ores in which over 90% of gold can be recovered by conventional cyanide leaching whilst the refractory ores are defined as those that give low gold recoveries or give acceptable gold recoveries only with the use of significantly more reagents or more complex pre-treatment processes.

According to Shnorhokian (2009), the mineralogical and geochemical properties of the tailings are site-specific and even change after deposition due to on-going reactions between the various components. The most important aspect of the mineralogy is the percentage of sulphides. The amount of sulphides has a direct impact on the density of the tailings and consequently, on the quantity of binder necessary to add per volume unit (Benzaazoua et al, 2003).The increasing presence of soluble sulphates leads to an increased slowing of hardening process due to inhibition of cement hydration in the presence of sulphates (Benzaazoua et al., 2003).When sulphur is exposed to water and oxygen, it begins to oxidise, generating sulphuric acid and heat. The sulphates liberated can attack cement bonds in the fill during curing and reduce the strength of the fill (Potvin et al, 2005).

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According to Potvin et al (2005), some minerals produce elongated or platy particle shapes which can influence such properties as permeability, density and, in some cases, strength. An example of how mineralogy can affect strength is the case of mica (Potvin et al 2005). Mica minerals are characterised by their platy geometry and smooth surfaces, both of which reduce the strength of fill.

The ore being treated at the South Treatment Plant (STP) at AngloGold Ashanti, Obuasi Mine, is the refractory type, with the gold particles encapsulated by sulphide minerals, predominantly, Arsenopyrite (FeAsS), Pyrrhotite (Fe_7S_8) and pyrite (FeS_2) (AGA Metallurgical Service Department Manual, 2005).

2.3.2 The binder

The second material in the backfill is the binder or the cementitious material. The various materials that contribute to the strength of concrete either by chemical or physical action are collectively referred to as the cementitious materials (Neville and Brooks, 2010). The main types of binders are cements and pozzolans. Cements can be used solely as a backfill binder, while most pozzolans require the addition of lime (Masniyom, 2009). Pozzolans are often used in conjunction with cements, since lime is a natural by-product of the cement reactions. However, since some slags and all pozzolans must wait for lime to be generated first by the cement reaction, they do not contribute to overall backfill strength until after a period of weeks or even months (Masniyom, 2009).

Grice (1998) recommends typical cement addition rates of up to 6% by weight to hydraulic fill and 4.5% to 5% by weight to paste fill and rock fills. The cemented rock fills which utilise hydraulic fill as the source of fines are the most efficient consumers of cement with total addition rates as low as 2% (Grice, 1998). The relationship between binder content and fill strength is not linear (Potvin et al, 2005). They further indicate that considering the number of variables that impact on strength, it is necessary to conduct site specific laboratory testing to establish their relationships. Henderson and Lilley (2001 cited in Potvin et al, 2005) found the relationship between cement content and strength of aggregate fills as follows:

$$UCS = 63\ (c/n)^{1.23} \qquad 2.1$$

Where: c = cement content by weight

n = porosity

UCS = uniaxial compressive strength

Portland cement

Portland cement is the name given to a cement obtained by intimately mixing together calcareous and argillaceous, or other silica, alumina, and iron oxide bearing materials, burning them at a clinkering temperature, and grinding the resulting clinker (Neville and Brooks, 2010). Ackim (2011) also described Portland cement as a crystalline material that, upon hydration, forms cementitious compounds.

Cement can broadly be classified as either hydraulic or non-hydraulic. The non-hydraulic cement such as lime and gypsum must be kept dry in order to harden whilst the hydraulic cement such as Portland cement reacts once it comes into contact with water and hardens irrespective of the amount of water present. The water cement reaction produces hydrate which is not water soluble and, therefore, can harden even under water.

2.3.3 Chemistry of cement

The raw materials used in the manufacture of Portland cement consist mainly of lime, silica, alumina and iron oxide. These compounds interact with one another in the kiln to form a series of more complex products (Gogue, 1955, cited in Neville and Brooks, 2010).

Ordinary Portland Cementis composed of four main reactive phases. These are: tricalcium silicate ($3CaO.SiO_2$ or C_3S), dicalcium silicate ($2CaO.SiO_2$or C_2S), tricalcium aluminate ($3CaO.Al_2O_3$ or C3A), tetracalcium aluminoferrit ($4CaO.AI_2O_3Fe_2O_3$ or C_4AF) (Shnorhokian, 2009). The variations in percentage compositions of these compounds influence the properties of the cement. The silicates, C_3S and C_2S, are the most important compounds which are responsible for the strength of hydrated cement paste (Neville and Brooks, 2010). Neville and Brooks (2010) also revealed that in reality, the silicates in cement are not pure compounds, but contain minor oxides in solid solution. According to them, these oxides have significant effects on the atomic arrangements, crystal form and hydraulic properties of the silicates. C_3A contributes little to the strength of cement except at the early stage. However, it is beneficial in the manufacture of cement in that it facilitates the combination of lime and silica. C_4AF is also present in cement in small quantities. Compared with the other three compounds, it does not affect the behaviour significantly. It, however, reacts with gypsum to form calcium sulphoferrite and its presence may accelerate the hydration of the silicates. The chemical composition and function of cement constituents are presented in Table 2.1

Table 2.1: Oxide composition of Portland cement and their contribution to strength gain.

Name of compound	Oxide composition (Abbreviation)	Approximate percentage	Function
Tricalcium Silicate	$3CaO.SiO_3$ (C_3S)	45-55%	Mainly responsible for Early strength (1 to 7days)
Dicalcium Silicate	$2CaO.SiO_3$ (C_2S)	20-30%	Mainly responsible for later strength (7 days and beyond)
Tricalcium Aluminate	$3CaO.Al_2O_3$ (C_3A)	6-10%	C3A increases rate of hydration of C3S. C3A gives flash set in absence of gypsum
Tetracalcium Aluminoferrite	$4CaO.Al_2O_3Fe_2O_3$ (C_4AF)	15-20%	It hydrates rapidly but its contribution to strength is uncertain and generally very low.

Source:http://www.rdso.indianrailways.gov.in/uploads/files/1296810736168-ch06.pdf

In addition to the main compounds, there exist minor compounds, such as MgO, TiO_2, Mn_2O_3, K_2O, and Na_2O which usually amount to not more than a few per cent of the mass of cement (Neville and Brooks, 2010). Two of the minor compounds are of interest: the oxides of sodium and potassium, Na_2O and K_2O, known as the alkalis. They have been found to react with some aggregates. The product of alkali-aggregate reaction causes disintegration of the concrete. It has also been found to affect the rate of strength gain of concrete (Neville and Brooks, 2010).

The chemical composition of a typical ordinary portland cement expressed in terms of oxides is presented in Table 2.2

Table 2.2: Chemical composition of OrdinaryPortland Cement

OXIDE	WEIGHT (%)
SiO_2	21.45
TiO_2	0.22
Al_2O_3	4.45
Fe_2O_3	3.07
MgO	2.42
CaO	63.81
Na_2O	0.20
K_2O	0.83
P_2O_5	0.11
SO_3	2.46

Source: http://iti.northwestern.edu/cement/monograph/Monograph3_6.html

Pozzolana

The American Society for Testing and Materials (ASTM Standard C618), describes pozzolans or pozzolana as siliceous and aluminous mineral substance which, though having no cementitious qualities themselves, in their finely divided form, react with lime in the presence of water at room temperature to form compounds possessing cementitious properties.

Pozzolans may be naturally occurring or artificially produced. Naturally occurring pozzolans include clays, shales, opaline materials, volcanic tuffs and pumicites whereas artificial pozzolans are mainly obtained from industrial wastes and include fly ash, silica fume and some slags (Thevarasa et al, 1979). Both the natural and artificial pozzolans are generally described as supplementary cementitious materials (SCM). According to Bensted and Barnes (2002), cited in Ostnor (2007), all pozzolans have to be rich in reactive silica or alumina plus silica.

According to Thevarasa et al (1979) some of the advantages of pozzolana cement over ordinary portland cement, are: 1) they are cheap; 2) they improve plasticity and 3) they have higher resistance to sulphate attack. Mertens et al (2008) also emphasize that pozzolans are known to increase durability, lower the heat of hydration, increase the resistance to sulphate attack and reduce the energy cost per cement unit.

According to Donovan (1999), extensive testing has indicated that there are differences in strength gain between cement-only and cement plus pozzolan fills. He indicated that the strength gain is dependent on the type of pozzolanic material used, the amount of the pozzolanic material (by weight) added, and the curing conditions.

Replacement of pozzolana for portland cement causes decrease in early strength (Ostnor, 2007). According to Yueming et al. (1999) the low activity of fly ash and other pozzolans can be attributed to two factors: a) the glassy surface layer of pozzolanas is dense and chemically stable. This layer prevents the interior constituents, which are porous and amorphous and therefore more reactive, from taking part in the pozzolanic reaction. b) The silica-alumina chain of pozzolanas is firm and must be broken if activity is to be enhanced. Yueming et al. (1999)

established that the ultimate strength of pozzolan containing cements can be higher than that of the parent Portland cement. To achieve this ultimate strength, (Pu, 1999), explained that with the addition of pozzolanas, the active silica and alumina present in the pozzolanas will have secondary reactions with the free lime produced during hydration of portland cement, removing it from solution and producing stable calcium hydrosilicates and calcium hydroaluminates.

Most of the pozzolanic and cementitious materials investigated for use in mine backfills are also investigated by civil engineers for admixture into cement and concretes (Masniyom, 2009). These include: blast furnace slags, fly ash, silica fume, and calcined clay. The two most popular pozzolanic materials used in backfill operations are fly ash and granulated blast-furnace slags (Donovan, 1999).

a. Blast furnace slag

Ground granular blast furnace slag (GGBS) is a by-product of iron and steel production and has been used routinely in the manufacture of cements since the latter half of the nineteenth century (Smart and Van Heerden, 2008). According to Tarr and Pratt (2010), ground granulated blast furnace slag has excellent cementitious properties. Smart and Van Heerden (2008) state that similar 28 days strength of backfill is achieved economically with the ground granulated blast furnace slag as with ordinary Portland cement. Masniyom (2009) observed that, as a result of selective cooling, four distinct blast furnace slag products exist. These are: air-cooled, foamed, granulated, and pelletized. The typical chemical composition of smelting slag is Fe (as FeO, Fe_2O_3) at 30 – 40%, SiO_2 at 35 – 40%, Al_2O_3 at up to 10%, and CaO at up to 10%, (Shi et. al., 1999, cited in Sergeant, 2008).

Benzaazoua et. al. (2003) observed that slag based binders hydrate at a slower rate than portland cement and fly-ash based binders.

b. *Fly ash*

Masniyom (2009) describes coal ash as a waste product derived from the combustion of coal in thermal power generation plants. He explains that coal ash consists of fine particles known as fly ash that flow with flue gases and coarser particles, called bottom ash, which collect at the bottom of the boiler. According to him, combusted coal results in approximately 10 % (by mass) coal ash, of which 85% is fly ash and the remainder is bottom ash. Bottom ash is considerably coarser and less reactive than fly ash.

ASTM Standard C618 classifies fly ash into two types. These are Class C, which has higher lime content and is self-cementitious and Class F, which must be activated through the addition of lime.

Fly ash consists mostly of silicon dioxide (SiO_2), aluminium oxide (Al_2O_3) and iron oxide (Fe_2O_3). They are also pozzolanic in nature and react with calcium hydroxide and alkali to form cementitious compounds.

Fly ash is used in concrete for road construction, masonry, and in controlled density fills for residential sub-footings. It can also be used as filler in asphalt roofing products and in composites such as ceramics and plastics. Fly ash is sometimes used as a soil amendment and in production of potting soil. Finally, fly ash is used in mine reclamation projects to fill surface and underground mines and to treat acid mine drainage and soils (Dodd, 2000).

c. Silica fume

Silica fume is a by-product of electric arc furnace production of metallic silicon and ferrosilicon alloys. It is an extremely fine material which is chemically nearly pure silica and has an average particle size of two orders of magnitude smaller than Portland cement (Crandall, 1992).

According to Masniyom (2009), there is no known application or study of silica fume as an alternative binder for use in mining backfill. However, it has had considerable use in civil engineering practice as an alternative binder for concrete.

d. Calcined clay

Calcined clay pozzolans are produced by burning suitable clays at temperatures between 600-900 ºC (Sarfo-Ansah, 2010). The product is milled usually to cement fineness before it fully develops pozzolanic activity (Sarfo-Ansah, 2010). In the account of Bensted and Barnes (2002), cited in Ostnor (2007), the thermal treatment destroys the crystal structure of the clay minerals which is transformed to an amorphous and very reactive structure. Calcination, according to them has two positive effects. These are: (1) it reduces the high water demand associated with the presence of clay minerals and (2) it increases the active phase content.

Studies by Atiemo (2005) on the clay pozzolana produced in Ghana showed that the 28-day compressive strengths of the pozzolana cement mortars with up to 30% pozzolana content, satisfied the class 32.5 cement as recommended by EN 197-1 (2000) for concrete works and general construction. However, there are very limited studies on the application of calcined clay in mine backfill.

Cement hydration and pozzolanic reaction

When cement comes into contact with water, the silicates and the aluminates present in Portland cement form products of hydration or hydrates which with time produce a firm and hard mass (Neville and Brooks, 2010). The two calcium silicates (C_3S and C_2S) are the main cementitious compounds in cement.

In contact with water, C_3A and C_4AF react almost instantaneously leading to the setting of the cement. Both C_3A and the C_4AF react with calcium sulphate to produce ettringite ($3CaO{\cdot}Al_2O_3{\cdot}3CaSO_4{\cdot}31H_2O$), which hydrates further to form a solid solution of the low-sulphate sulphoaluminate, $3CaO{\cdot}Al_2O_3{\cdot}CaSO_4{\cdot}12H_2O$, usually existing in a solid solution with C_4AH_{13}

The overall non-stoichiometric reactions and their products can be represented as:

$$C_3A + 3CaSO_4 \cdot 2H_2O \rightarrow C_3A \cdot 3CaSO_4 \cdot 31H_2O$$
$$\rightarrow C_3A \cdot CaSO_4 \cdot 12H_2O + C_4AH_{13}$$

and

$$C_4AF + 3CaSO_4 \cdot 2H_2O \rightarrow C_3A \cdot 3CaSO_4 \cdot 31H_2O$$
$$\rightarrow C_3A \cdot CaSO_4 \cdot 12H_2O + C_4AH_{13}$$

(Lea, 1970)

Concurrently, C_3S and C_2S react with water, albeit more slowly, producing afwillite ($3CaO{\cdot}2SiO_2{\cdot}3H_2O$) and lime $Ca(OH)_2$.

The overall non-stoichiometric reactions that lead tohardening of the cement products can be represented as:

$$2\beta - C_2S + 4H_2O \rightarrow 3CaO \cdot 2SiO_2 \cdot 3H_2O + Ca(OH)_2$$

$$2C_3S + 4H_2O \rightarrow 3CaO \cdot 2SiO_2 \cdot 3H_2O + Ca(OH)_2$$

Whereas the hydration of cement produces lime and raises the pH, the pozzolans consume lime and result in calcium silicate hydrates (Shnorhokian, 2009).

If a concrete contains a pozzolan, less cement is required to obtain a specified strength (Dunstan, 2011). The amount of cement will vary depending on the reactivity of the pozzolan. A highly reactive pozzolan has more cementitious strength value than a low reactive pozzolan (Dunstan, 2011).

2.3.4 Water

Water is described as a universal solvent and a medium for many chemical reactions. It is essential for hydration, the early and late strength development of both hydraulic and non-hydraulic cements.

In many of the concrete specifications, quality of water for concrete mixing is covered by a clause which states “that the water must be fit for drinking”. Such water rarely contains more than 2000 parts per million of dissolved solids and as a rule, not less than 1000ppm of dissolved solids (Neville and Brooks2010).They however, state that some water which may not be suitable for drinking may still be safe for mixing concrete. According to them, any water with a pH of 6 to 8 which does not taste saline or brackish is suitable for use, and that water for mixing is also suitable for curing. However, they warn, that it is essential that curing water be free from substances that attack hardened concrete.

In the case of mine backfill preparation, Corina-Maria and Cornelius (2010) state that the main concerns are the pH of the water and the amount of dissolved salts, particularly chloride and sulphate salts.They emphasize that in particular, acidic water with a pH below 6.5 and sulphate salts can react with the hydration products from the binders and lead to long-term loss of strength and durability of the paste.

Acid waters with pH values less than 3.0 may create handling problems and should be avoided if possible. Organic acids, such as tannic acid, can have a significant effect on strength at higher concentrations (Masniyom, 2009). Benzaazoua et al. (2002) showed that the mixing water is an important parameter that affects the quality of the paste backfill for two reasons: (i) the water-to-binder ratio affects the backfill strength acquisition process, and (ii) the water chemistry interferes with the cement chemistry and alters the hydration processes.

Depending on availability, the water can either be recycled process water from the mine or fresh water (Benzaazoua et al., 2004).

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2.4 Strength of backfill

The strength of mine backfill is commonly and conveniently referred to as its compressive strength, and in an unconfined state, defined as the limit of the backfill material's compression resistance to load (Sargeant, 2008).

Compared to the rock mass that surrounds it, backfill is relatively soft and does not provide much direct support, since its main function is to impart lateral confinement pressure against the rock walls or pillars that support the rock (Sargeant, 2008). Even though backfill is not as strong as rock, depending on its expected function, its mode of operation allows it to provide a level of local and regional stability necessary for the safe and efficient operation of mines.

The backfill cohesion is directly dependent on the binder quality and its potential to resist possible harmful chemical reactions such as hydration inhibition and sulphate attack that can occur within a sulphide- and/or sulphate-rich backfill (Benzaazoua et al., 2002). Grice (1998) reports that cement addition rates up to 6% by weight is used to generate cohesion values of around 250kPa to 500kPa and unconfined compressive strengths from 0.75MPa to 4MPa in hydraulic fill. He further held that,these strengths are sufficient to enable the backfill to be exposed in vertical walls up to 40 metres wide and virtually unlimited in height.

Cemented backfill fails when there is a sudden loss of cohesion, due to the propagation of micro cracks throughout the backfill mass, followed by shear failure along a plane (Sargeant, 2008). In most underground mining environments the predominant ground stress condition is one in which compressive stresses exist and in which failure will be predominantly caused by such stresses through the processes of compressional shear failure (Masniyom, 2009). The shear strength of a cohesionless fill can be represented by the following equation (Bell, 1992, cited in Donovan, 1999):

$$S = \sigma tan\emptyset \qquad 2.2$$

Where:

S = the shear strength of the fill (kPa)

σ = the normal stress on the failure plane (kPa)

Ø = the angle of internal friction of the fill

In cemented fills, Donovan (1999) indicates that cement bonds that form between fill particles provide a cohesive component to the fill's shear strength which is absent in an uncemented fill. The shear strength of a cemented fill can be expressed using the following equation (Terzaghi et al., 1996, cited in Donovan, 1999):

$$S = \sigma tan\emptyset + C \qquad 2.3$$

Where,

S = the shear strength of the fill (kPa)

σ = the normal stress on the failure plane (kPa)

Ø = the angle of internal friction of the fill

c = the cohesion of the cemented fill (kPa)

2.5 Cost of mine backfill

According to Grice (1998), one of the essential requirements of backfill is that it must be low cost. According to him, typical costs of backfill range from $2 to $20 per cubic metre, depending on the duty required of the backfill. These costs can be a significant contribution to the operating costs of the mine. Where cemented backfill is used, these costs tend to be between 10% and 20% of the total operating cost of the mine with cement representing up to 75% of that cost. Cannors (2001), (cited in Tarr and Pratt, 2010), corroborates that cement is the largest material cost component in the backfill process. To help offset the cost associated with the use of Portland cement, many mining operations in Canada and around the world use custom blends of Portland cement (Tarr and Pratt, 2010). This involves blending pozzolanic materials with the ordinary portland cement. The additives are used to increase the durability and the strength of the mixture, and appreciably reduce the binder costs (Benzaazoua et al., 2002). Masniyom (2009) indicate that capital costs vary widely for different backfill types and from site to site, depending on the availability and quality of the potential backfill materials and their proximity to the mine.

CHAPTER 3: MATERIALS AND METHODS

3.1 Binding materials

Ordinary Portland cement of class 35.5R which meets EN 197-1 standards, being manufactured by Ghana Cement Company Limited (GHACEM) was obtained in 50kg bag from cement retailing shop in Obuasi for the study. The clay pozzolana manufactured by Pozzolana Ghana Limited was also obtained in 50kg bag from a retail outlet in Obuasi.

3.2 Backfill specimen preparation

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Samples of classified tailings taken from the backfill plants at the Kwesi Mensah Shaft (KMS) and the George Capendelle Shaft (GCS) at the AngloGold Ashanti, Obuasi Mine, were air dried and thoroughly mixed together to obtain a homogenous mix before blending with binder.

A known quantity of the classified tailings was weighed. Five percent ordinary Portland cement by weight of the dried tailings was added and thoroughly mixed. Water was added to obtain a homogenous mixture called backfill. The density of the slurry was 1780 Kg/m^3. The water used in the study was the same as that used for backfill preparation in the mine. The mixing was done manually, using metal scoop. The 5 percent binder is consistent with the design mix proportions in the backfill operations at the AngloGold Ashanti, Obuasi Mine.

In the subsequent backfills prepared, various portions of the 5 percent Portland cement were substituted with clay pozzolana. The proportions of the Portland cement substituted were: 10%, 25%, 30%, 35% and 40%.

The backfill test specimens were made by placing the prepared backfill in 50mm diameter by 120mm high cylindrical moulds, made from PVC material. The moulds were perforated at one side and covered with brattice cloth to drain the excess water, similar to what is done underground. The moulds were filled in three layers of approximately equal volumes. The specimens were stored covered in a humid room until they were due for testing. The humidity of the room was approximately 70%, similar to the underground conditions. Four specimens were made for each curing period for each level of ordinary Portland cement (OPC) substitution. The curing periods were 7, 14, 21, 28 and 56 days.

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For each of the pozzolana content used, thus 0%, 10%, 25%, 30%, 35% and 40%; twenty (20) specimens were produced. Out of the 20, four (4) specimens were tested for compressive strength after curing for 7, 14, 21, 28 and 56 days. The overall number of specimens was one hundred and twenty (120). Figure 3.1 is the picture of the prepared samples.

Figure 3.1: Prepared backfill specimens ready for curing

3.3 Unconfined compressive strength test

The uniaxial compressive strengths (UCS) of the cylindrical backfill specimens were tested by placing the samples one at a time in the 50kN ELE compressive strength testing machine and gradually applying load to the specimen till it failure. The maximum load at which the fracture occurred was recorded. The compressive strength, σ, was then calculated using the relations: $\sigma = \frac{F}{A}$ 3.1

Where:

F = Failure load

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A =Cross sectional area of the specimen

The test was conducted on all the four samples representing a particular replacement level and the average determined. This step was repeated for the rest of the replacement levels. A picture of the test set up is found in figure 3.1.

Figure 3.2: Unconfined compressive strength test set up

3.4 Particle size distribution analysis

The solid component of the slurry was separated by filtration. The wet solids moulded into a circular tablet, was divided into approximately four equal parts and the opposite sides taken for the wet sieve analysis. In the wet sieve analysis, the material was thoroughly washed over sieve with aperture size 53μm (0.053mm) and the underflow collected in another container. The material retained on the sieve was oven dried and graded through a stack of ASTM sieves arranged in decreasing order of size of aperture. With the aid of a mechanical vibrator, the material was graded through the sieves. The amount of solids retained on each sieve was weighed and recorded in table 4.1 in appendix I. The underflow of sieve 53μm was subjected to further size analysis, using the cyclosizer (see Fig 3.3). The cyclosizer is a laboratory precision apparatus for the rapid and accurate determination of the particle size distribution within the sub-sieve range sizes falling within 44μm and11μm in diameter. The percentage weight passing each sieve was determined and plotted against the sieve size.

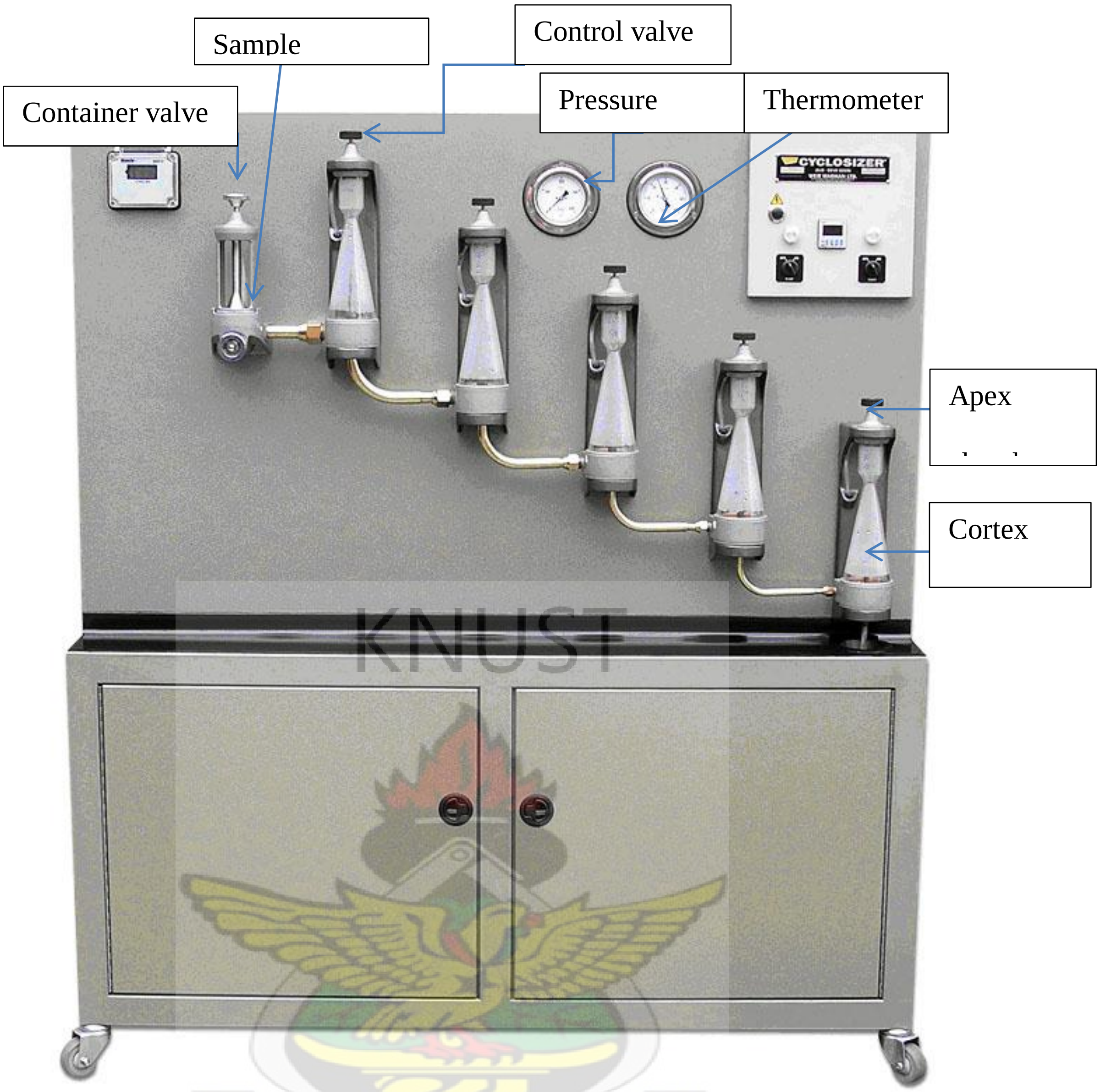

Figure 3.3: Cyclosizer for sub-sieve particle size analysis

3.5 Sulphate content analysis

20g of the sample of dried tailings was measured into 500ml HDPE bottle. 400ml distilled water was added. The water/tailings mixture was agitated vigorously for 1 minute and left covered for 12 hours after which the mixture was agitated again and filtered. The filtrate was collected in a 600ml beaker. 10ml of the filtrate was measured and transferred into a sample vial which was placed into the sample chamber of the colorimeter unit to calibrate it to the zero mark. The sample vial was removed from the sample chamber. One sulphate test tablet was added to the sample vial and crushed immediately with a plastic rod and tamped carefully to dissolve and disperse completely throughout the sample vial. The vial was capped and the outside wiped clean before placing in the sample chamber of the colorimeter unit. The *test* key was pressed and the results in parts per million (ppm) appeared on the display within few 3 seconds.

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CHAPTER 4: RESULTS AND DISCUSSION

4.1 Particle size distribution of the tailings

The particle size distribution curve of the tailings is presented in figure 4.1

The raw data for the plot is given in Table A1 in appendix A

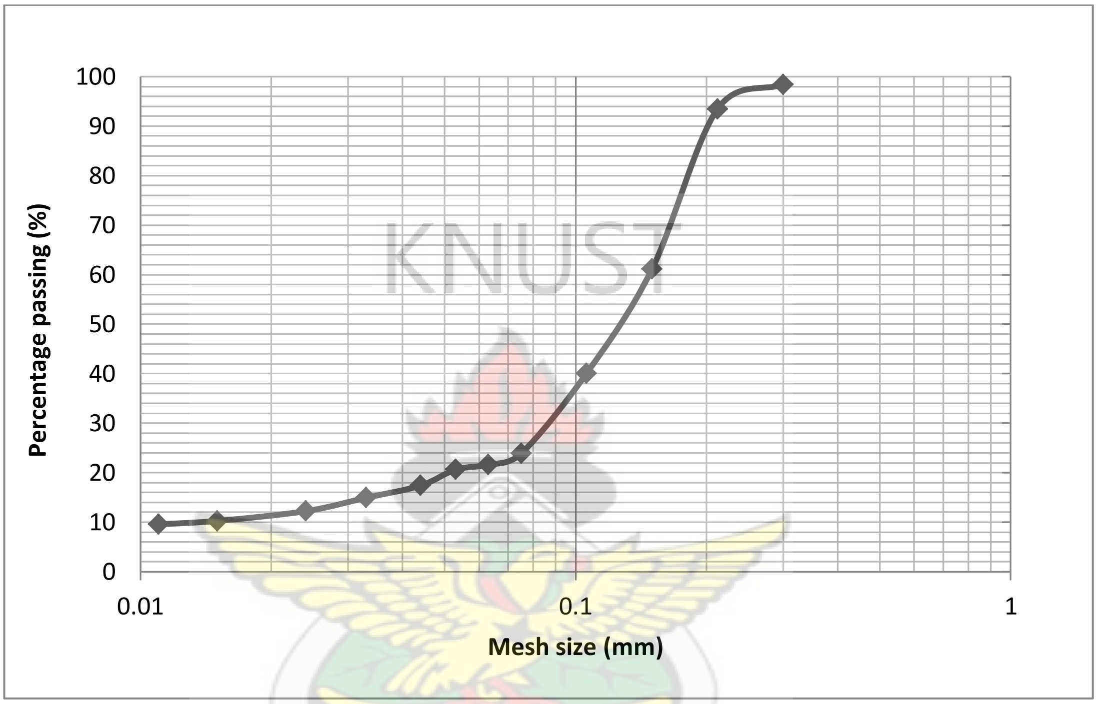

Fig. 4.1: Particle size distribution curve of the sample of tailings from AGA, ObuasiMine.

From the figure above, the particle sizes of the tailings used for the study range from 0.011mm to 0.3mm.

4.2 Unconfined compressive strength

The results obtained from the unconfined compressive strength test on specimens of backfill with various binder contents and different curing ages are presented in figures 4.2.The controlled sample is the one containing Portland cement as the only binder.

4.2.1 Effects of pozzolana content on compressive strength

The variations of compressive strength values for all specimens tested after 7 days, 14 days, 21 days, 28 days and 56 days of curing are presented in figure 4.2. Tables A2, A3, A4, A5 and A6 in appendix A presents the strength values (average of 4 specimens) obtained.

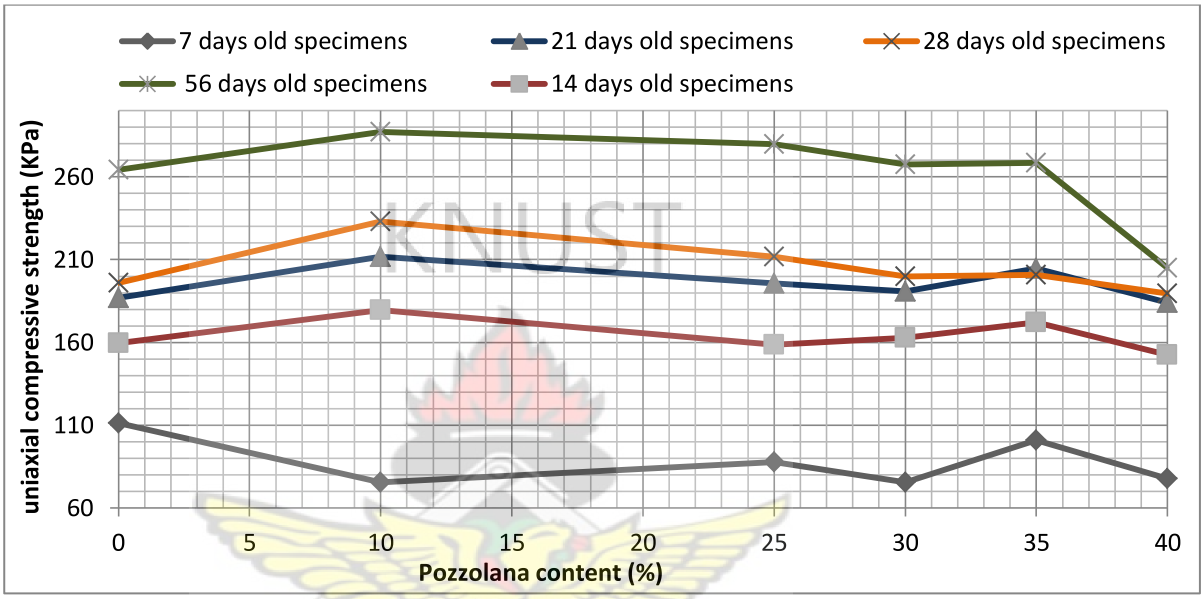

Figure 4.2 Variations of unconfined compressive strength values (average of 4 specimens) for all specimens tested after 7 days, 14 days, 21days, 28days and 56days of curing

The strength obtained for the 7 day old specimens without pozzolana (OPC alone) was 111kPa. The strength dropped with the replacement of 10% of the OPC with clay pozzolana. The 25%, 30%, 35% and 40% cement replacement levels also had strengths lower than the controlled samples. However, the strength properties of the 25%, 35% and 40% were higher than the 10% and 30% replacement levels. The results clearly showed that for the 7 days cured period, the specimens of backfill with portland cement alone performed better in terms of strength, compared to those specimens with some proportions of the cement replaced with clay pozzolana.

The results of the 14 days cured specimens showed a different pattern from the 7 days. The 10%, 30% and the 35% cement replacement levels recorded higher strength values than the controlled specimens. The 40% replacement level recorded the lowest strength value whilst the 10% replacement level obtained the highest strength.

The results show that, the strength performances by the pozzolana contained backfills are increasing faster with increase in the number of curing days, compared to the controlled speciemens. For instance, the respective increases in strengths of the 10%, 25%, 30%, 35% and 40% replacement specimens from 7 days to 14 days are 137%, 81%, 116%, 71% and 96% compared to 44% of the controlled samples.

After curing for 21 days, the test results showed that the 10%, 25%, 30% and 35% replacement levels produced strengths values higher than the controlled samples with the 40% obtaining strength values lower than the controlled case. The highest strength was obtained by the 10% replacement level whilst the lowest strength was obtained by the 40% replacement level.

The individual replacement levels gaining strengths higher than the controlled case is increasing. After 14 days of curing, three replacement levels, thus 10%, 30% and 35%, showed better strengths. However, after 21 days of curing, four replacement levels, thus, 10%, 25%, 30% and 35%, showed better strengths than the controlled.

The 28th day after filling is critical to the mining cycle. Usually, mining close to hydraulic fill commences after 28 days of curing the hydraulic fill.

The results of strength performances by all specimens tested on the 28th and the 56th day showed that the specimens with 10%, 25% 30% and 35% replacement levels continue to perform better than that of portland cement alone whilst the 40% obtained strength values lower than the base case. The 10% replacement level obtained the highest strength whilst the 40% replacement level obtained the lowest strength value.

4.2.2 Effect of curing on compressive strength of the backfill

The summary of results of compressive strength obtained for the various pozzolana content used and the curing periods are presented in Table 4.1

Table 4.1: Compressive strength for various pozzolana content and curing ages

Curing time (days)	Unconfined compressive strengths of the various replacement levelsPozzolana content (%)					
Cure time(days)	0	10	25	30	35	40
7	111.09	75.64	87.74	75.38	100.86	77.86
14	159.45	179.48	158.66	162.88	172.11	152.34
21	186.86	211.64	195.56	190.82	204.52	183.97
28	196.09	232.99	211.90	199.78	200.96	189.50
56	264.35	287.28	279.72	267.62	268.41	204.97

Variation of compressive strength with curing time is shown in figure. 4.3

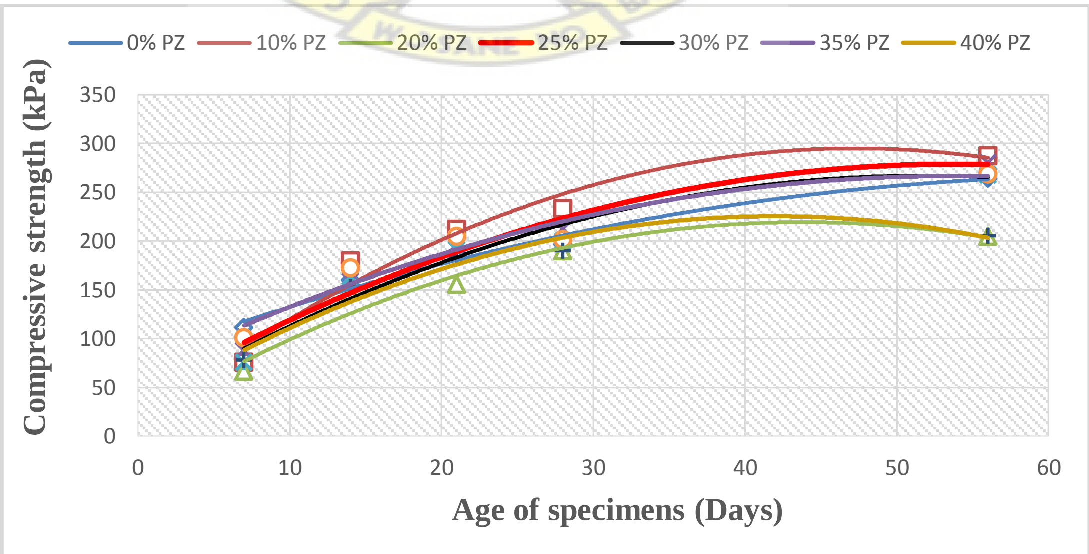

Figure 4.3: Variation of compressive strength with curing period

It is observed that the compressive strength values obtained for pozzolana substituted backfills, cured for seven days, were lower than those obtained for ordinary portland cement backfills for the same period. There were however, rapid strength gains for the pozzolana specimens after the 7 days. This explained by Yueming et al. (1999) as follows:

a) The glassy surface layer of pozzolana which is dense and chemically stable and prevents the interior constituents which are porous and amorphous and therefore more reactive, from taking part in the pozzolanic reaction. b) The silica-alumina chain of pozzolanas is firm and must be broken if activity is to be enhanced. Masniyom (2009) also attributes the slow reaction of pozzolans at the early stages to the fact that pozzolans have to wait for Portland cements to hydrate and release lime into solution before they begin reaction. Generally strength increases with curing period as expected.

Ostnor (2007) also observed the slow hardening rate of pozzolans at start but indicates the long term strength can be substantial. Pu (1999) explained why pozzolana containing cement can produce better strength in the long term. According to him, the free lime produced during hydration of cement has low strength and stability. The lime also provides avenue for attack by sulphate solutions, leading to lower strength and less durability of cement paste and concrete. With the addition of pozzolans, the active silica and alumina present in the pozzolans will have secondary reactions with the free lime, removing it from solution and producing stable calcium hydrosilicates and calcium hydroaluminates.

The tailings used for the study had higher sulphate concentration compared with the upper limits set by the Americans Society for Testing and Materials (ASTM C94) and the European Standard (EN 1008). This may have led to the reduced strength of the backfill with only portland cement. Generally, clay pozzolana is rich in silica and alumina. The silica and alumina react with the lime produced by the portland cement during hydration, therefore producing more mechanically stable compounds, hence better strength gain.

4.3 Sulphate content analysis of the tailings

The result of the sulphate content analysis of the tailings is presented in Table 4.2.

Table 4.2: Sulphate content of tailings compared with standards

Obuasi tailings	ASTM C94	EN 1008
3500 ppm	3000ppm	2000ppm

The result obtained from the sulphate content analysis of the tailings sampled from the AngloGold Ashanti, Obuasi Mine for the study compared with the upper limits set by the American Society for Testing and Materials (ASTM) Standard C94 and the European Standard 1008, show clearly that the Obuasi tailings has higher sulphate content.

4.4 Savings with the use of clay pozzolana in backfill

The results obtained from the study indicates that the compressive strength obtained for some of the backfill specimens which were prepared by replacing some portions of the Portland cement with clay pozzolana, produced comparable strengths to those obtained for Portland cement alone. Therefore, the economic viability of substituting part of the Portland cement required in the backfill with clay pozzolana is analysed in this section.

The annual consumption of ordinary Portland cement by AngloGold Ashanti, Obuasi Mine and the associated costs are shown in Table 4.2.

Table 4.3: Cement consumption for hydraulic fill at AGA, Obuasi Mine

Year	Cement consumed for backfill (tons)	Cost per ton of cement (Gh¢)	Total cost of cement (Gh¢)
2008	16,622	160.00	2,659,468.80
2009	9,913	210.00	2,081,812.32
2010	12,533	300.00	3,759,894.00
2011	13,593	360.00	4,893,616.80
2012	13,772	380.00	5,233,462.60
2013	18,547	400.00	7,418,724.00

Source: AngloGold Ashanti, Obuasi Mine stores department

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Cost of a tonnage of pozzolana in Obuasi Municipality in 2013 is Gh¢280.00

To determine the amount of money that could be saved in any year by substituting portions of the Portland cement with clay pozzolana, Let:

s= savings per unit weight of Portland cement

x= cost per unit weight of Portland cement

z = cost per unit weight of pozzolana

a = percentage of the Portland cement to be replaced

Then,

$$s = \frac{a}{100}(x - z) \qquad 4.1$$

For t tons of cement, total savings (S_{Total}) is given as $s \text{ x } t = \frac{a}{100}(x - z) \text{x } t$ 4.2

Using equation 4.1with the optimum pozzolana content of 10% and cost per ton of pozzolana of Gh¢280.00, savings, $s = \frac{10}{100}(400 - 280) = Gh¢12.00$ would have been saved on every ton of cement used.

Again, using equation 4.2 and the total weight of Portland cement of 18,547 tons; cost per ton of Portland cement of Gh¢400.00 and cost per ton of pozzolana of Gh¢280 in 2013 with optimum pozzolana content of 10%, the savings for that year would have been S_{Total} = $\frac{10}{100}(400 - 280)$x $18{,}547 = \text{Gh}¢222{,}564$

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CHAPTER 5: CONCLUSIONS AND RECOMMENDATIONS

5.1 Conclusions

Based on the study, the following conclusions were made:

- Clay pozzolana has the potential to replace Portions of the portland cement to produce backfill of strength comparable to those obtained from Portland cement alone.
- Replacing10%, 25%, 30% and 35% of the ordinary Portland cement content with clay pozzolana produced backfill with strengths comparable to using ordinary Portland cement alone.
- Replacing 10% of the ordinary Portland cement content with clay pozzolana produced backfill with the highest compressive strength relative to the other replacement levels and the ordinary Portland cement alone after curing beyond seven days.
- Assuming that the 10% pozzolana content is used for backfill in 2013, an amount of Ghȼ222,564 would be saved by the mine on cement. This amount representing a 3% saving on the cost of cement.

5.2 Recommendation

It is recommended from the study that

- Ten per cent (10%) of ordinary Portland cement could be substituted with clay pozzolana in hydraulic backfill.
- Further research into the mineralogical and chemical characteristics of the tailings is recommended, since it could affect the reaction of the tailings with cement and the ultimate strength of the backfill.
- Long term strength of backfill containing pozzolana should be investigated.

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APPENDIX

Appendix A: particle size distribution of the sample of tailings from AGA, Obuasi mine.

Table A.1: Results of particle size distribution analysis of sample of tailings from AGA, Obuasi Mine

Mesh size (mm)	Weight retained (g)	% Weight retained	% weight passing
0.011	9.00	0.68	9.58
0.015	26.20	1.97	10.25
0.024	35.80	2.69	12.23
0.033	32.6	2.45	14.93
0.044	43.10	3.24	17.38
0.053	13.00	0.98	20.62
0.063	30.40	2.29	21.60
0.075	215.00	16.17	23.88
0.106	280.70	21.12	40.06
0.150	429.20	32.26	61.18
0.212	66.10	4.97	93.46
0.300	20.28	1.56	98.44

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Table A.2: Compressive strength values for all specimens tested after 7 days of curing, (average of 4 specimens).

Clay pozzolana content (%)	Compressive strength (KPa)
0	111.09
10	75.64
25	87.74
30	75.38
35	100.86
40	77.86

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Table A.3: Compressive strength values for all specimens tested after 14 days of curing, (average of 4 specimens).

Clay pozzolana content (%)	Compressive strength (kPa)
0	159.45
10	179.48
25	158.66
30	162.88
35	172.11
40	152.34

Table A.4: Compressive strength values for all specimens tested after 21 days of curing, (average of 4 specimens).

Clay pozzolana content(%)	Compressive strength (kPa)
0	186.86
10	211.64
25	195.56
30	190.82
35	204.52
40	183.97

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Table A.5: Compressive strength values for all specimens tested after 28 days of curing,(average of 4 specimens).

Clay pozzolana content(%)	Compressive strength (KPa)
0	196.09
10	232.99
25	211.90
30	199.78
35	200.96
40	189.50

Table A.6: Compressive strength values for all specimens tested after 56 days of curing, (average of 4 specimens).

Clay pozzolana content (%)	Compressive strength (kPa)
0	264.35
10	287.28
25	279.72
30	267.62
35	268.41
40	204.97

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