DEVELOPMENT OF A SUITABLE MINE BACKFILL MATERIAL USING MINE WASTE FOR SAFE AND ECONOMIC ORE PRODUCTION AT KONKOLA MINE (ZAMBIA)

By

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A dissertation submitted in fulfillment of the requirements for the award of Master of Mineral Sciences (M.Min.Sc) Degree

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Declaration

I, Mutawa Ackim, do hereby declare with academic honest that with the exception of quotes and work for other people, which I have duly referenced to and acknowledged herein, this dissertation is the result of my own original research work. No part of it has been presented in pursuit of another degree in this university or anywhere else.

I declare that this dissertation was written according to the rules and regulations governing the award of Master of Mineral Sciences (M.Min.Sc) of the University of Zambia.

Signature of Research Scholar……….………………………………………………………

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Certificate of Approval

This dissertation by Mutawa Ackim has been approved as fulfilling the requirements for the award of Degree of Master of Mineral Sciences by the University of Zambia.

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Dedication

To my family and especially my mum, ‘Ba Mbuya’

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Abstract

The use of backfill in underground mines has increasingly become an integral part of overall mining operations all over the world. This has largely been driven by the need to increase ore recovery and productivity. The underground mining activities have given rise to creation of large volumes of voids which, if not addressed, would lead to long term global instability of the mine. The ever-increasing depths reached by underground mining will, in the future, place even more demands on sound backfill and mine design systems if safe and efficient operating conditions are to be maintained. Environmental protection and increasing need for economic use of surface land have demanded use of underground waste as backfill material.

Geotechnical investigations have been carried out on the available mine waste materials at Konkola Copper Mine (KCM) in Chililabombwe, Zambia, to develop a suitable backfill material for safe and economic ore production at Konkola mine (Zambia). The material would need to have the strength of 1 MPa, as spelt out by Konkola mine management and suitable drainage characteristics to resist failure due to self weight in a backfilled stope and therefore facilitate ore pillar recovery.

Geotechnical tests have shown that the suitable backfill material for safe and economic ore production at Konkola Mine is the blend between the hydraulic fill material produced at 65% solid fraction at the old backfill plant and waste rock from No.3 Shaft Dump crushed to 2.83mm size. The blend ratio must be 3:7 by weight, respectively, with the addition of 4% Zambezi Portland cement. This material, mixed on surface, must be pumped to the various stopes underground at Konkola Mine and allowed to cure for not less than 28 days to attain the recommended UCS of 1 MPa needed to facilitate total ore pillar recovery.

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List of Acronyms

KCM	-	Konkola Copper Mines
UCS	-	Uniaxial Compressive Strength
KDMP	-	Konkola Deep Mining Project
LOM	-	Life of Mine
HF	-	Hydraulic Fill
MOCB	-	Modified Overcut and Bench
CHF	-	Cemented Hydraulic Fill
PC	-	Porous Conglomerate
LPC	-	Lower Porous Conglomerate
GFW	-	Geological Footwall
GHW	-	Geological Hanging wall
URD	-	Upper Roan Dolomite
SLOS	-	Sublevel Open Stopping
CRUD	-	Continuous Retreat Up Dip
PPCF	-	Post Pillar Cut and Fill
OPC	-	Ordinary Portland cement
Mtpa	-	Million tons per annum
tph	-	Tons per hour
tpd	-	Tons per day
tpm	-	Tons per month
MPa	-	Mega Pascal
kPa	-	Kilo Pascal
FOS	-	Factor of Safety
PSD	-	Particle Size Distribution
CR	-	Cemented Rockfill
CS	-	Cemented Sandfill
CT	-	Cemented Tailings
BSI	-	British Standard International
BQT	-	Backfill Quality Tester
BS	-	British Standard
RD	-	Relative Density
SG	-	Specific Gravity
LL	-	Liquid Limit

PI	-	Plasticity Index
SL	-	Shrinkage Limit
CAF	-	Cemented Aggregate Fill
CHF	-	Cemented Hydraulic Fill
LHD	-	Laud Haul Dump
PPCF	-	Post Pillar Cut and Fill
COP	-	Cost of Production
TSS	-	Total Suspended Solids
TDS	-	Total Dissolved Solids
RMR	-	Rock Mass Rating

List of Symbols

γ	-	Unit weight
k	-	Permeability
φ	-	Internal angle of friction
c	-	Cohesion
E	-	Young's Modulus
e	-	Void ratio
n	-	Porosity, %
G_s	-	Specific Gravity
ρ_w	-	Density of water
σ_1	-	major principal stress
σ_2	-	intermediate principal stress
σ_3	-	minor principal stress
σ_n	-	Normal Stress
σ_v	-	Vertical stress
σ_h	-	Horizontal fill pressure, kPa.
τ	-	Total shear stress developed along a failure plane
ψ	-	Measured failure angle of backfill in compression, ^{o}
C_u	-	Coefficient of Uniformity
C_c	-	Coefficient of Curvature
D_{10}	-	Particle diameter at which10% of the material passes,
D_{30}	-	Particle diameter at which 30% of the material passes,
D_{50}	-	Average particle size
D_{60}	-	Particle diameter at which 60% of the material passes,
η	-	Absolute viscosity of fluid flowing, Ns/m,
γ	-	Unit weight of fluid flowing
κ	-	Stiffness, N/m
k	-	Friction factor (dimensionless)
Cw	-	Mass solids concentration
N_c	-	cohesion bearing capacity factor, dimensionless
N_γ	-	Unit weight bearing capacity factor, dimensionless
N_q	-	Surcharge bearing capacity factor, dimensionless

Chapter 1

INTRODUCTION

1.1 What is Mine Backfill?

Mine backfill is material or combination of materials used to fill underground voids created by mining operations. The use of backfill in underground mines has increasingly become an integral part of overall mining operations all over the world. This has largely been driven by the need to increase ore recovery and productivity. Increased underground mining activities have given rise to creation of large volumes of underground voids which, if not addressed, would lead to long term global instability of the mine. The use of mining waste helps to stabilize the surrounding rock mass thus preventing, in severe cases, premature collapse of stope pillars into the mining voids resulting in disruption of production activities, and subsequently, loss of ore. In addition, these voids provide an opportunity for mining operations to dispose of waste materials underground rather than hoisting to surface thus helping to reduce the cost of mining. The ever-increasing depths reached by underground mining will, in the future, require sophisticated backfill and mine planning designs to ensure safe and efficient operating conditions. Environmental protection and increasing need for economic use of surface land have demanded use of underground waste as backfill material.

Konkola Copper Mines Plc, situated in Chililabombwe on the Copperbelt Province of Zambia, has embarked on the Konkola Deep Mining Project (KDMP) at its Konkola Mine to increase copper ore production from the current 2.4 million tonnes to 7.5 million tonnes per annum. The company has recognized that attainment of the above mentioned milestones will rely greatly on the successful implementation of a *Backfill System* at Konkola Mine. The project will access the orebody containing about 250 million tonnes copper ore at 4% contained copper that lies below the current production levels of 950m at number No.1 Shaft. The KDMP project is expected to extend the Life of Mine by over 30 years from the time of completion in 2012.

The future mining operations commencing after completion of the KDMP project at Konkola Mine will utilize a *Primary/Secondary* mining system. This system will comprise of two phases of extraction namely, primary and secondary. During the *primary phase*, pillars of ore will be left underground to give structural support. In the *secondary phase*, the voids created during the primary phase will be systematically filled with the engineered backfill material which would need to have enough strength and suitable drainage characteristics to resist failure due to self weight in a backfilled stope and therefore facilitate ore pillar recovery. The resulting voids, due to mining of the ore pillars, will then be filled with ordinary tailings. This process will lead to increased copper ore production and permit extraction of high grade ore with minimum dilution.

Konkola Mine has constructed two backfill plants at its premises (with the third plant called 'Waste Rock Crushing and Milling plant' nearing completion at the time of report writing) to produce and supply the backfill material. These plants have been named KCM 1 BF, KCM 2 BF and KCM 3 BF Plants. Currently, the oldest backfill plant, KCM 1 BF Plant situated at No. No.1 Shaft mine, produces un-cemented classified tailings as Hydraulic Fill material for delivery to areas at No.3 Shaft underground mine which are using Modified Overcut and Bench mining method. The primary function of the fill material is to give increased lateral confinement pressure onto the rock walls or pillars that support the rock mass loads. However, this backfill material cannot provide the required strength characteristics in the proposed future mining operations commencing after the KDMP project. This is due to the fact that the material has low cohesion and its strength relies on the mobilized internal angle of friction.

The KCM 2 BF plant was constructed in early 2002 next to the old one to facilitate the production of Cemented Hydraulic Fill material for use in underground mining operations. This plant is however not operational at the moment because it has been reserved for future underground mining operations that will require cemented backfill material at No.1 Shaft.

The new KCM 3 BF Plant, also called Waste Rock Crushing and Milling Plant, has been constructed to commence production of 70% of the total backfill material requirement at Konkola Mine (as has been determined in this research project) and will be the primary backfill producing facility to meet the demand as KCM ramps up to 7.5 million tonnes production rate. This plant will produce coarse aggregate material by way of crushing the waste rock from the nearby existing waste dumps at No.3 Shaft. Potentially, there will be three types of backfill base products (two types of classified tailings from KCM 1&2 BF plants as well as crushed and ground material from KCM3 BF).

In mining operations utilizing the primary/secondary extraction system mentioned above which will demand high strength backfill material, the hydraulic fill from KCM 1 BF plant will be mixed with the coarse aggregates from KCM 3 BF plant to improve the particle size distribution thereby ensuring a backfill material with high *strength* and good *permeability* characteristics.

This research project was initiated by KCM to develop a suitable backfill material that will meet the required strength and drainage requirements with addition of optimum but economic quantities of cement and/or additives and therefore facilitate the proposed *Primary/Secondary extraction system* at Konkola Mine.

1.2 Location and Description of the Research Site

Konkola Mine is in Chililabombwe, one of the towns on the Copperbelt Province in north central Zambia, bordering the Democratic Republic of Congo. It is the most northerly of the currently operating Copperbelt mines as shown in Figure 1.1.

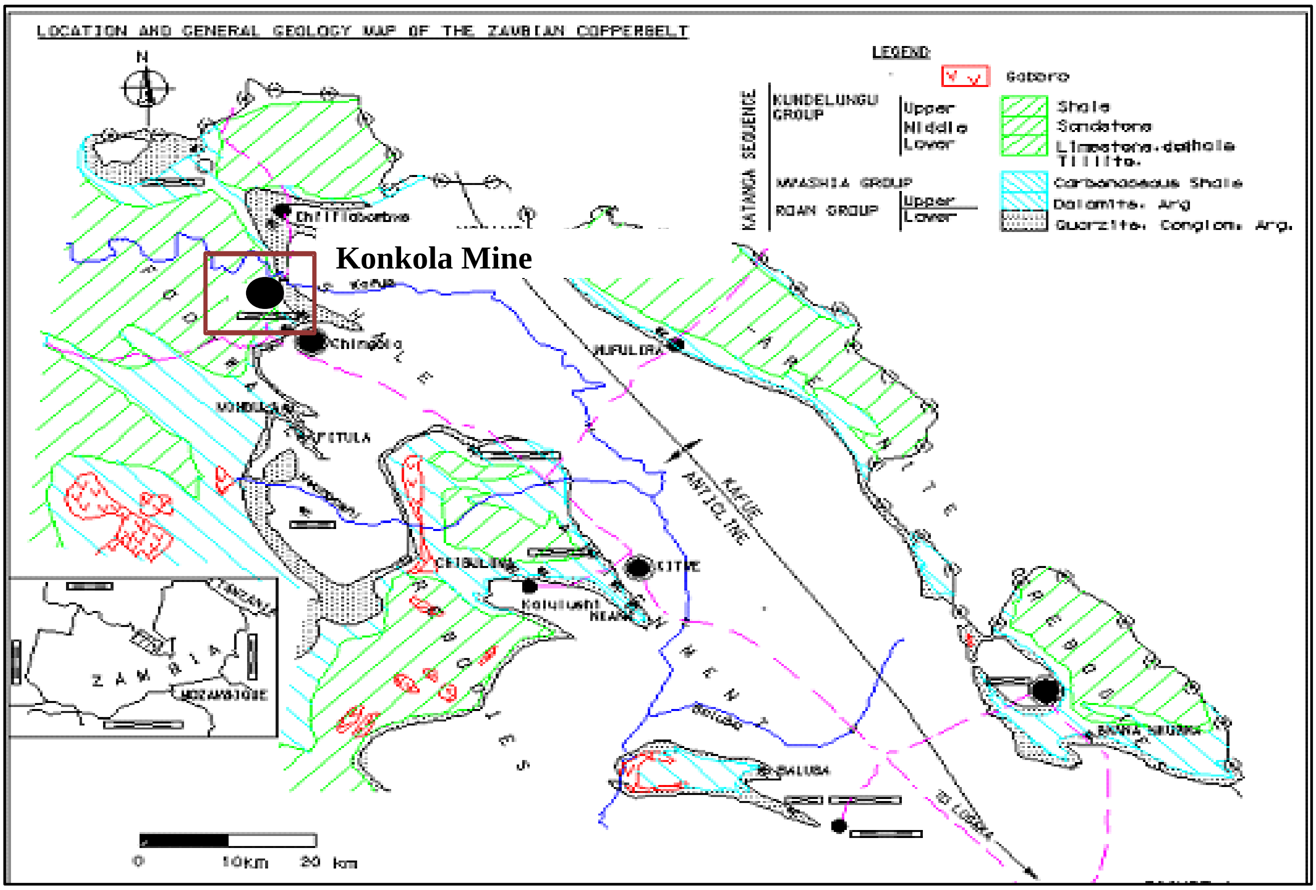


Figure 1.1 Location of the Research Site (KCM Expansion Projects) [11]

1.2.1 General Geology and Hydrogeology of Konkola Mine

At Konkola Mine, the Lower Roan succession lies un-conformably on Schist and Gneiss which have been intruded by Muliashi Porphyry as shown in the stratigraphy in Figure 1.2.

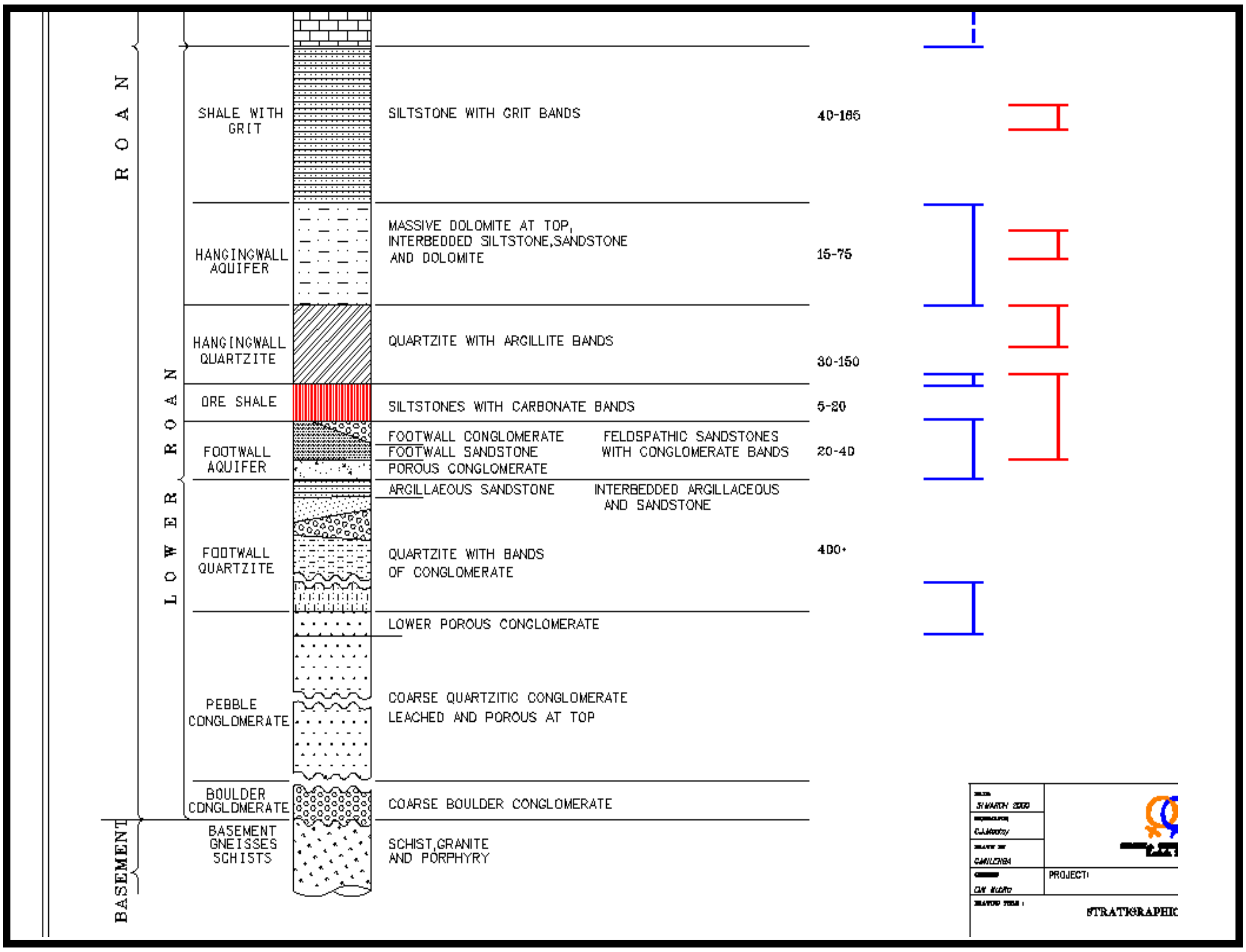


Figure 1.2 Geologic Stratigraphy of Konkola Mine (KCM Expansion Projects) [11]

The lowest formation of the Lower Roan is the Boulder Conglomerate and this is overlain by the Pebble Conglomerate a thick succession of conglomerates with occasional Quartzites. The uppermost unit of the PC, the Lower Porous Conglomerate has been leached and weathered. The basal unit of the overlying Footwall Quartzite Formation is highly jointed and this, together with the LPC, forms a major aquifer. The remainder of the Footwall Quartzite is a strong, occasionally cross-bedded Quartzite with an inter-bedded Quartzite and Argillite, the Argillaceous Sandstone, at the top. The overlying Footwall Aquifer is sandstone between two conglomerates and is the second significant aquifer in the succession.

The Ore Shale formation rests un-conformably on the Footwall Aquifer and is a series of inter-bedded silt-stones, fine sand-stones and Dolomitic sand-stones. It has been lithologically subdivided into five units:

- Unit A, the lowest in the series, is a finely banded impure Dolomitic silt-stone;
- Unit B is a massive grey silt-stone;
- Unit C is a silt-stone with inter-bedded Dolomitic sand-stone;
- Unit D is similar to Unit C but the Dolomitic inter-beds are less regular;
- Unit E is a massive grey silt-stone with lenses of Dolomitic sand-stone and Arkose.

The basal and upper contacts of the Ore Shale with the enclosing rocks are known as the Geological Footwall (GFW) and Geological Hanging wall (GHW) respectively.

The Ore Shale is overlain by a mixed succession of massive Quartzites and Arkoses that form the Hanging wall Quartzite. Above this, the Hanging wall Aquifer is a sequence of inter-bedded fine sand-stones, silt-stones and impure dolomites. The beds at the top of this formation and the basal beds of the overlying Shale-with-Grit are extremely weathered and decomposed and form the third major aquifer at Konkola. The Shale-with-Grit is the uppermost formation of the Lower Roan.

The Upper Roan Group consists of the Upper Roan Dolomite (URD) and is overlain by the Mwashia Group, which comprises the Mwashia Shale. The URD has water bearing zones which form the uppermost aquifer at the mine.

Two dominant features define the structure in the mining license area: The Kirila-Bombwe Anticline in the south-east and the Konkola Dome in the north-west. The metasediments have been draped around these structures in relatively gentle folds with some disruption by faulting.

Due to the complex Geo-hydrological setting and the presence of major aquifers, the Konkola mine experiences a high rate of water inflows. These inflows are further exacerbated by the presence of transmissive structures that cut across the KDMP mineral resources. Over many years about 300,000m^3 of water have been pumped out of the mine daily to ensure that the production workings are dry. There are three main aquifers that have direct impact on the mine planning and operations, namely:

- Footwall Quartzite Aquifer,

- Footwall Aquifer and
- Hanging wall Aquifer.

Pumping infrastructure for KDMP has been planned to ensure adequate dewatering ahead of mining operations.

1.2.2 Underground Mining Operations at No. 1 and No.3 Shafts

Historically, the existing No.1 Shaft and No.3 Shaft systems at Konkola have been managed as separate mines due to their ore reserves being physically divided by the presence of a barren gap in the orebody that extends from surface down to 720m level. Underground haulage connections between the two mines were developed mainly for access and dewatering purposes. Below that level the orebody is continuous along a strike length of approximately 10km. The following mining methods are in use at Konkola Mine.

- Sub level Open Stoping with Gravity
- Sub level Open Stoping with Scraping
- Continuous Retreat Up Dip
- Post Pillar Cut and Fill
- Overcut and Bench

1.3 Objectives of the Konkola Backfill Research Project:

i. To develop a suitable backfill material for Konkola Mine with addition of a binder for increased safety and economic copper ore production.

ii. To determine the optimal amount of binder to be added to the backfill material to achieve a uniaxial compressive strength (UCS) of at least 1 MPa (This requirement has been stipulated by Konkola Mine management)

iii. To develop a suitable backfill material for Konkola Mine with suitable drainage conditions, to facilitate the proposed primary/secondary extraction system.

iv. To assess the environmental impacts of the Konkola mine composite backfill system

v. To evaluate the risks and costs associated with the Konkola mine composite backfill system

1.4 Problem Statement

The mining operations commencing after the KDMP project at Konkola Mine will require extensive use of carefully engineered backfill material with cement and/or additives to facilitate the primary/secondary extraction strategy. However, the current uncemented classified tailings used as hydraulic fill in the MOCB mining method at No.3 Shaft cannot provide the required strength characteristics in the proposed future mining operations due to its low strength of 58.4 kPa. It is proposed, therefore, that addition of binders such as cement and/or additives to fill material is undertaken to provide the needed high strength and good permeability characteristics underground. It was discovered in the old trial tests conducted earlier at KCM that addition of cement to classified tailings reduces the percolation rates to as low as 65mm/hr as opposed to 80mm/hr required at Konkola mine. This *conflicting requirement* will have to be resolved. This research project was initiated by the need to develop a suitable backfill material that will meet the required strength and drainage requirements with addition of optimum but economic quantities of cement and/or additives and therefore facilitate mining operations with the proposed Primary/Secondary extraction sequence at Konkola Mine.

In underground mining processes requiring high strength fill, the classified tailings from No.1 Shaft Backfill Plant will be blended with the aggregates from the new Waste Rock Crushing and Milling Plant at No.3 Shaft to improve the particle size distribution thereby ensuring a backfill material with high *strength* and good *permeability* characteristics.

1.5 Research Questions

In pursuing this research, the following questions were born in mind:

- What are the basic considerations for success of the Konkola Composite Backfill system?
- What are the backfill material engineering characteristics and properties that would be considered to assess the suitability of different materials available at Konkola Mine for use as backfill material for the mining operations commencing after the KDMP project?
- What optimal quantities of cement and/or other additives are required to ensure the attainment of the recommended Uniaxial Compressive Strength of at least 1 MPa?
- The current mining operations at Konkola Mine use classified tailings without cement and mine management requires this material to have a permeability of 80mm/hr. Does

the addition of cement and/or additives reduce or increase this permeability? If so what mitigation measures would be put in place to ensure a suitable drainage environment?

- What characteristic engineering properties of the backfill are affected as the material is pumped from the backfill plant to the underground workings?

1.6 Significance of study

The outcomes of this research project are very vital for the successful implementation of the composite backfill system in mining operations starting after the KDMP Project. This is because the backfill material characteristics that have been developed in this research would facilitate application of the proposed primary/secondary extraction system and thereby ensure increased copper ore production as no ore pillars will be left underground un-mined. In addition, the backfill system will permit extraction of high grade ore with minimum dilution.

The successful implementation of the backfill system at KCM will greatly help in the company's environmental management efforts, as this will mean less surface land usage as less mine tailings will be sent to the existing Lubengele tailings Dam.

1.7 Research Methodology

In order to achieve the above mentioned objectives, the following research methodologies were undertaken:

- Literature survey and review of the latest information available in this area.
- Sampling and testing of the various materials at Konkola mine to assess their suitability for use as backfill (using available laboratory equipment at Nchanga Geotechnical Services Laboratory). These materials included: classified mill tailings, waste rock from No.3 Shaft Waste Dump, lime and furnace slag from the New Copper Smelter at Nchanga. The cement brands used as binder were: (i) Mphamvu from Lafarge Cement Company (ii) Zambezi from Zambezi Cement Company (iii) Fast setting cement from Minova Cement Company in South Africa (used on blended aggregate material)
- Data analysis of the suitable backfill material at Konkola Mine based on research findings

Chapter 2

LITERATURE SURVEY

Backfill is generally described as any material used to refill an excavated area. In underground mining operations, backfill is material or a mixture of materials used to fill underground stopes. The following are fundamental reasons for use of backfill material (www.infomine.com)[15]:

- Ore pillar recovery in open stoping methods. This is achieved by way of giving structural support to the roof and sides of the stopes for improved production and safety
- Ensuring long-term regional stability
- Limiting excavation exposure
- Working platform in particular mining methods such as Cut and Fill.
- Environmental protection and waste disposal. Economic use of surface land has demanded use of underground waste as backfill material.
- Prevention of inundation (water flooding) in some underground mines
- Storage of very explosive Methane gas (CH_4) in Coal mines, and
- Prevention of surface subsidence.

Backfill material is sometimes only used to fill voids and requires sufficient strength to prevent any form of remobilization. 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. In the past, material that was being used as mine fill was placed regardless of its properties. With the adoption of bulk mining methods and increased use of cemented fill lately, the nature and properties of fill have become more important in that fill is really part of a suite of construction materials within a mine. This necessitated greater understanding of the material properties of each of the constituents used. This understanding is required to ensure reliable and consistent fill performance, as well as to optimise costs, particularly where cemented fill is used.

The modulus of elasticity for backfill material is generally less than in-situ rock. Backfill material is not used to transmit the rock stresses, but to reduce the relaxation of the rock mass so the rock itself would still be able to carry a load. This will ensure reduction in the deterioration of ground condition and thereby improve safety. Adding low percentages of Ordinary Portland Cement permits the development of cohesive strength and the ability for the backfill mass to be self supporting when exposed in vertical faces by adjacent pillar mining (Potvin, 2005)[14]. The self supporting nature of the backfill permits higher recovery of pillar ore, which in turn improves the utilization of the mining reserve and the economics of the mining operations.

2.1 Backfill design Rationale

A backfill design procedure is very data-intensive, bringing a wide range of relevant data and knowledge together to support the necessary decision making. It is also very time consuming. Like most engineering problems, backfill design requires substantial practical judgment based on a large amount of information and experience. During the consideration for backfill operations, the following aspects have to be taken into account (Dorricott & Grice, 2000)[2]:

- The Geological and Geotechnical condition of the mine site.
- The backfill techniques currently in operation which are available.
- The historical experiences and commonly accepted rules of backfill design parameters applicable in similar mining conditions. There is a large number of data involved and hence numerous options leading to different backfill operations.
- Specify the mining environment and mining system.
- According to the mining system specified, identify the backfill purposes and based on the backfill purposes and mining condition, define the target properties of backfill materials to serve the backfill purposes.
- Define the operating system to make backfill materials available to meet with the target properties and finally the backfill operation itself, which includes:
    - Backfill material preparation;
    - Backfill material transportation;
    - Backfill material placement;
    - Backfill operation quality control and environment monitoring. The information monitoring in this stage will be feed back to the previous earlier steps to modify the design parameters.
- Economical evaluation of backfill system.
- Documentation and implementation of backfill mining system.

In practical backfill operations, the backfill is made up of three resources: waste rock, natural sand and mill tailings. The constituents of backfill material may include the following items: Water, rock, sand, tailings, cement, slag fly ash and aggregates.

It is important to identify the main backfill purpose to be served by the backfill material in order to optimize the backfill system. The purpose identified will determine the properties of the backfill material. These properties are termed as the *target properties*, with which the backfill material has to comply. The target properties include *hydraulic, mechanical and environmental* properties. The target properties may be predicted on a site-specific basis such as required mechanical strength to avoid failure of a particular backfill face, or based on commonly accepted standard values such as a required hydraulic percolation rate to permit effective drainage. In practical operations, *ground control* and *waste disposal* are the main reasons that backfill is employed. The ground control is further divided into different specific cases to serve the different purposes such as pillar recovery, construction of a working platform etc., which lead to the different strength requirements and material properties. Table 2.1 summarizes the desirable properties of backfill material and their relevance to Backfill design.

Table 2.1 Desirable properties of backfill material and their relevance to backfill application in underground mining

No	Property		Brief Description	Relevance
1	Uniaxial Compressive strength (*UCS*),		Maximum Compressive Strength that is mobilized by backfill material to resist failure	To ascertain whether the backfill material will fail due to self weight in a backfilled stope or not. UCS also determines maximum height that backfill can stand in a stope.
2	Unit weight, γ		Weight of backfill material per unit of volume	Important for determining quantity of fill material required in a stope
3	Permeability, k		Rate of flow of water through backfill material	Important in determining how quickly transport water will drain from the backfill material in a stope.
4,5	Shear Strength Characteristics	Internal angle of friction, φ	Shear resistance mobilized purely by the interlocking of backfill	To determine whether backfill will remain free standing during and after pillar extraction. Also used to determine the bearing capacities of fill if used as a working platform (e.g. in PPCF)
		Cohesion, c	Shear resistance mobilized purely by bonding forces between particles of backfill material	To determine whether backfill will remain free standing during and after pillar extraction. Also used to determine the bearing capacities of fill if used as a working platform (e.g. in PPCF)

6	Elastic modulus, E	The relationship between backfill stress and strain.	To study the deformability characteristics of backfill material under various loading conditions
7	Void ratio, e	The ratio of volume of voids to volume of solids in backfill material	Void ratio has a strong effect on the permeability and strength of backfill material. It is affected by the particle size distribution of the fill.
8	Liquefaction potential	Behavior of backfill when loaded suddenly suffering a transition from a solid state to a liquefied state	To study the mechanical response of backfill material when blasting is conducted during pillar recovery.
9	Particle Size Distribution	The grading of the different particles in the backfill material	The strength and permeability of backfill material is largely affected by the particle size distribution
10	Mineralogical composition and chemical reactions	The different minerals that rock or soil is composed of	To study possible chemical reactions when additives or binders are added to backfill.
11	Atterberg limits	A range of moisture contents over which backfill will exhibit consistence	It is important in order to understand the effect of water addition to a particular backfill medium at different water contents
12	Slump	A measure of the quality of cemented backfill mixture	Important in order to determine the pumpability or transportability characteristics of cemented backfill material

2.2 Mechanics of Backfill Materials

Backfill mechanics aims to describe the relevant intrinsic relationships existing between the constituent materials that comprise backfill and its in-situ response. In order to optimize backfill design for maximum quality, engineers ought to first understand the physiochemical properties of its constituent materials and their interaction. This will give an insight into mechanical response of backfill within a mining stope. If the composition is of tailings, alluvial sand or a coarse rock fill, backfill can be considered to be a special form of soil and therefore many soil mechanical properties and relationships can be applied to it. Mechanical properties of fill placed underground may change remarkably depending upon the fill material's intrinsic properties, method of preparation, placement and the conditions of the mine environment.

Backfill consists of two or three distinct phases that includes solid particles and, trapped in the pore spaces or voids between the solid particles, water and/or air. Figure 2.1 shows a representative phase diagram for a unit of backfill and the relationships between the volume and mass of each phase with one other.

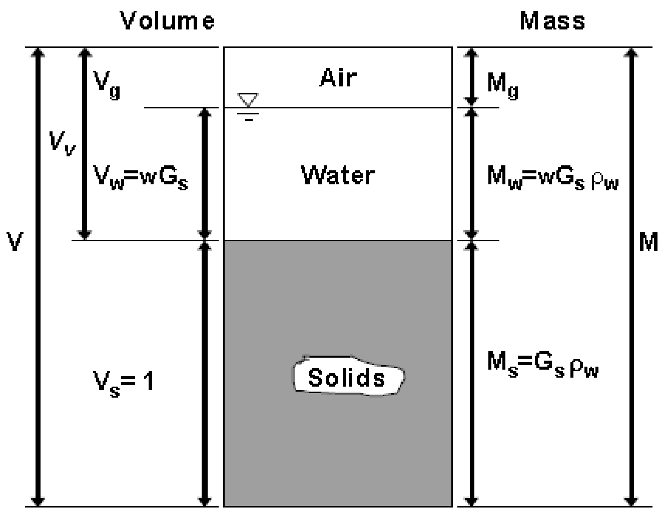


Figure 2.1 Phase diagram of backfill (Craig, 2005)[1]

Where,

V_g, V_w, V_s	= volume of gas, water and solids, respectively, m^3
V_v	= volume of void space, m^3
M_g, M_w, M_s	= mass of gas, water and solids, respectively, kg
W	= water content, %
G_s	= specific gravity of the soil particles, dimensionless
ρ_w	= density of water, kg/m^3

It is important to understand existing stress conditions within the rock or backfill masses in order that the stability of underground excavations in rock, of rock structures, or of backfill materials can be adequately assessed

In underground mining environments the main 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 compression shear failure. Mining stress effects are mobilized three dimensionally and the condition of "principal stress" will result in the formation of three mutually perpendicular normal stress components, or principal stresses, as shown in Figure 2.2.

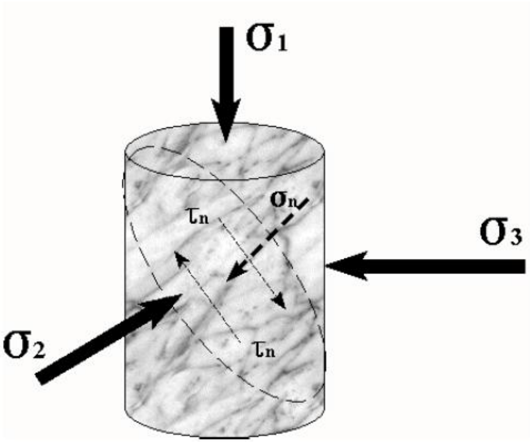


Figure 2.2 Principal stresses acting upon a backfill cylinder and resultant stress conditions upon a potential shear plane surface (Craig, 2005)[1]

σ_1 = major principal stress (largest compression magnitude)

σ_2 = intermediate principal stress

σ_3 = minor principal stress (smallest compression largest tensional stress magnitude)

Should a plane of weakness exist within the cubic element which bisects the plane of stress being evaluated, but oriented other than parallel to one of the reference axial directions, then it becomes important to be able to determine the stress conditions acting on this plane. Shear failure ensues if stress components affecting the plane of weakness exceed the planar strength capabilities (Craig, 2005)[1].

Backfill strength is solely dependent on the following two components:

- *Frictional forces*, proportional to the internal angle of friction (φ), that results from interlocking solid particles and which are independent of moisture content.

Apparent *cohesion* (c), in uncemented fills, results from surface tension forces in pore water that disappears when the fill is fully dry or fully saturated. Portland cement or similar soil binders act to greatly increase actual fill cohesion by coupling the solid particles together. Apparent cohesion results from surface tension in the water held at grain contacts within the fill. Surface tension is inversely proportional to the radius of curvature of the free water surface at contact, and therefore for a given pair of particles increases as the amount of water held between them decreases. Net particle attractive force, however, varies with both the magnitude of surface tension (decreasing with increasing water content) and the area over which it acts (increasing with increasing water content). Therefore a point exists where optimum fill moisture content at which apparent cohesion is a maximum.

The importance of grain interlocking and hence the magnitude of the internal friction angle depends upon grain shape, overall particle size and packing density. The strength capabilities of cohesion-less materials are mobilized solely by the frictional resistance that develops along internal shear plane surfaces as shown in Figure 2.3, within the fill under stress (σ_1, σ_3).

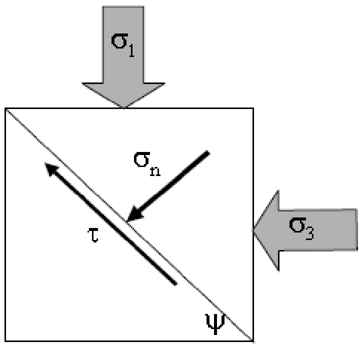


Figure 2.3 Shear plane development within backfill under load (after Craig, 2005)[1].

The level of frictional resistance that can develop is directly proportional to the magnitude of the normal stress (σ_n) that develops perpendicular to such failure planes. Resisting shear stress (τ) develops along the failure plane, based upon the material's coefficient of sliding friction and the normal stress developed across the sliding plane. The coefficient of sliding friction is based upon a physical parameter constant that is dependent upon the measured angle of failure (ψ) that can be determined when backfill samples are failed in compression.

$$\tau_f = \sigma_n \tan \phi \qquad (1)$$

$$\phi = 2(\psi - 45^{\circ}) \qquad (2)$$

(after Craig, 2005)[1]

Where,

τ_f	= frictional shear stress developed along a failure plane, kPa
σ_n	= resolved stress perpendicular to the failure plane, kPa
φ	= internal friction angle of fill material, ^{o}
ψ	= measured failure angle of backfill in compression, ^{o}
σ_1	= major principal compressive stress, kPa
σ_3	= minor principal compressive stress, kPa

This failure mode for backfill can be plotted as shear stress versus normal stress to establish a positively sloped linear Mohr failure locus (envelope) to define the maximum tolerable backfill normal stress (Figure 2.4).

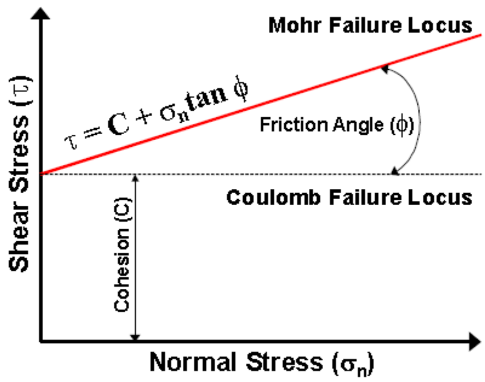


Figure 2.4 Mohr-Coulomb failure locus (after Craig, 2005)[1].

Failure will occur, for purely cohesion materials, along any planar orientation where the maximum generated shear stress exceeds the intrinsic bonding strength between sample grains (cohesive strength of the material). The backfill material is therefore assumed to behave as a frictionless material that will develop failure shear surfaces oriented at a constant angle of 45^{o} to the direction of principal compressive stress (σ_1) application. Typical examples of Coulomb materials are:

- Structural materials such as pelletized backfill components, held in loose confinement,
- Fractured rock having wet, slicken sided or clay-filled apertures, and
- Gelled backfill materials containing very high retained water contents.

$$\tau_c = C \qquad (3)$$

Where,

τ_c = cohesive shear stress developed along a failure plane, kPa

In practice, backfill exhibits strength characteristics that are mobilized both by cohesive and by frictional resistance effects as expressed by the Mohr-Coulomb relationship:

$$\tau = c + \sigma_n tan\Phi \qquad (4)$$

(after Craig, 2005)[1]

Where,

τ= total shear stress developed along a failure plane, kPa

.

The behavior of cemented backfill when exposed during the extraction of the adjacent stope in underground mining operations depends on the shear strength of the fill. Potvin (2005)[14] has concluded that uncemented fill like hydraulic fill cannot form a vertical face. Only cemented fills can form vertical exposures

The design of fill with the required level of shear strength requires knowledge of in-situ stress conditions in the fill mass at the time of exposure. This can be estimated by using simple theoretical models or by using elaborate numerical models (Kuganathan, 2005)[12]. Figure 2.5 shows stress fields within Backfill Material.

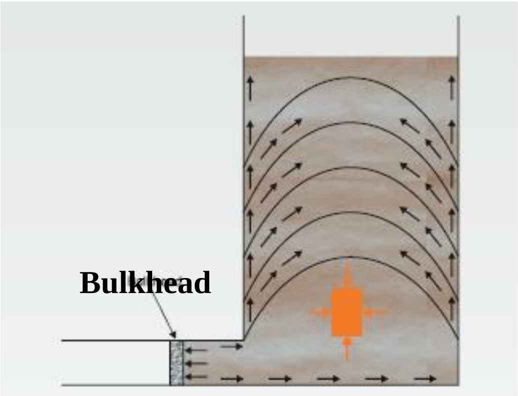


Figure 2.5 Stress field within Backfill Material (after Kuganathan, 2005)[12]

Once the prevailing stress conditions are known at the time of fill exposure, it is necessary to use the appropriate binders to develop the shear strength of the fill. Laboratory testing on appropriate mixes of binders and the fill materials is required to evaluate the strengths of the cemented fill at various curing times and to specify the required amount of binder for the fill.

Backfill material with no binder added needs to be contained and can be used when the material will not be exposed. Granular materials can develop shear strength by two mechanisms, namely sliding resistance and interlocking between the particles. Relative movement between the particles is the most important mechanism of deformation in a granular medium like minefill. When the particles start moving and rearranging to accommodate any changes in loading, the fill undergoes deformation. Resistance to fill deformation is influenced by the shear resistance between the particles, and the interlocking of particles. Interlocking of particles is influenced by the packing density. The sliding resistance and the interlocking resistance are increased when the particles are pushed together perpendicular to the shearing plane. Hence, the sliding and interlocking components of the shear strength are frictional in nature and dependent on the normal stress across the shearing plane. There are some situations when part of the shear resistance between particles is independent of the normal force pushing the particles together.

2.2.1 Terzaghi Arching Theory and Stability Analysis of Backfill Material

The downward movement of the fill mass will be partly resisted by the rock walls through shear resistance. This process results in transferring a part of the weight of the fill mass to the rock walls. The process of vertical stress reduction in the fill mass due to transfer of weight to sidewalls is called *arching.* Because of arching the vertical stresses in the fill masses are not geostatic. Towards the bottom of the fill mass the vertical movement is obstructed by the rock bottom of the stope. Figure 2.6 below shows arching and stress distribution in a fill mass.

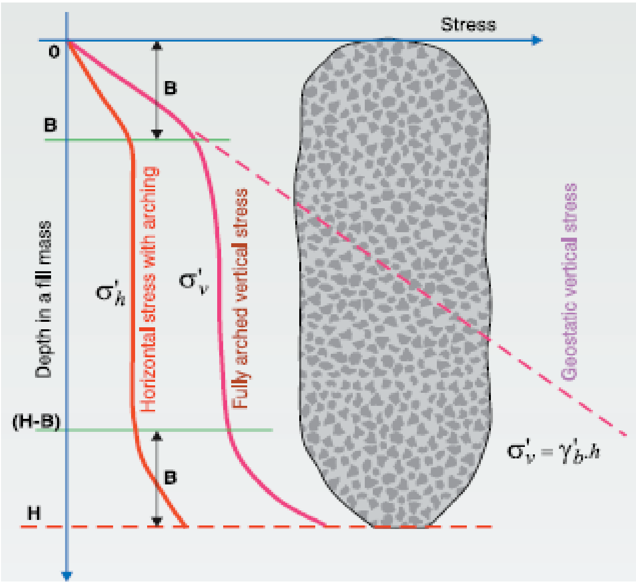


Figure 2.6 Vertical and horizontal stress distributions due to arching in Backfill (after Kuganathan, 2005)[12]

For a depth nearly equal to the width of the stope, the fill will not develop full arching, and this results in the vertical stress σ'*v* increasing towards the bottom. Above this level and up to one width in magnitude below the top surface fill arching would develop, and σ'*v* would stabilize at a steady state. Above this level, σ'*v* would gradually reduce to zero due to reducing height. Arching helps to reduce the vertical stresses in the fill mass. Because of arching a fill with a UCS of 0.5 MPa can withstand an exposure of over 100m in height. If arching were not present, such a condition would require over 2 MPa strength fill, which would in turn require higher cement addition and make the cost of filling very high (Kuganathan, 2005)[12].

Cemented fill is used when complete ore extraction is planned. Some mining systems require rock pillars to be left in place in order to provide rock mechanical support by bearing tributary loads. It is very common to come back and mine pillars following primary mining in order to maximize ore recovery. When this is done, large vertical heights of backfill may be exposed. It is necessary that the fill displays sufficient strength to remain free standing during and after pillar extraction.

The main cost component for the fill is the binder cost. The amount of binder needed at a particular stoping environment depends on the design strength of the fill. The design strength of the fill is estimated by using appropriate methods of stability analysis. At present two types of methods are used:

- Simplified wedge analysis originally.
- Elaborate numerical modelling using commercially available packages.

A common form of fill failure is the result of shear sliding, whereby a planar failure surface forms that dips outwards into the pillar excavation, causing part of the exposed fill mass to move into the new stope as shown in Figures 2.7 and 2.8.

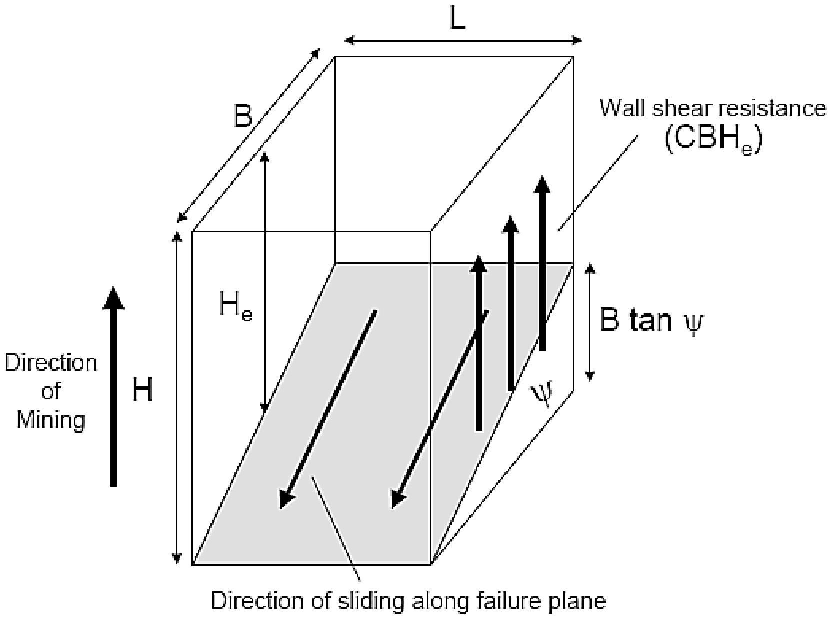


Figure 2.7 Backfill shear failure (after Hassan and Bois,1998)[6]

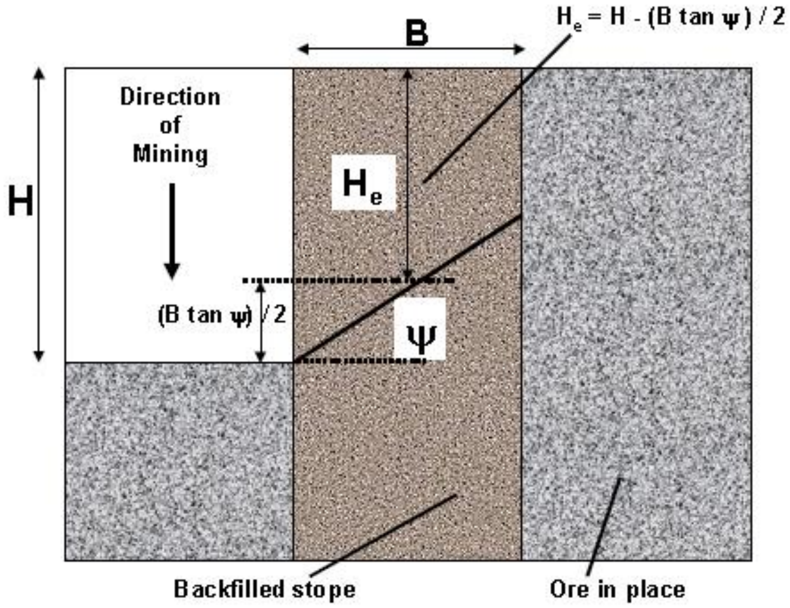


Figure 2.8 Profile of backfill shear failure (after Hassan and Bois,1998)[6]

This occurs when a sufficient height of backfill is exposed to create a fill block of sufficient weight to overcome any resistance to sliding along a critically-oriented failure plane. Resistance to shear sliding results from a combination of cohesion and particle friction both along the failure surface and at the side walls. The driving force towards failure is the net weight of the block. The FOS on the fill block shear surface is determined by the ratio of the forces resisting the block failure and those forces driving failure. The following parameters must be considered when evaluating sliding wedge stability:

Net block weight, W_{NET} = block weight – side wall resistance

= $LBHe\gamma - 2CBHe$

- *Resisting forces:*

  Cohesion on the plane of sliding = $CLB/\cos\psi$

  Sliding plane resistance = $(W_{NET})(\cos\psi)(\tan\varphi)$

- *Driving Forces:*

  Net Block Weight = $(W_{NET})(\sin\psi)$

  Therefore the factor safety FOS can be calculated as:

$$FOS = \frac{(CLB/\cos\psi) + (LBHe\gamma - 2CBHe)(\cos\psi - \tan\varphi}{(LBHe\gamma - 2CBHe).\sin\psi} \qquad (5)$$

(after Hassan and Bois,1998)[6]

Where,

C = fill cohesion through fill mass on failure plane, kN/m^2

L = strike length of backfilled stope, m

H = total height of exposed fill in stope, m

He = effective height of exposed fill in stope, m

B = dip length (width) of backfilled stope, m

φ = backfill internal angle of friction, ^{o}

ψ = failure angle through fill, ^{o}

γ = unit weight of placed backfill, kN/m^3

Equation below can be rewritten so that, given a required FOS (FOS.$\geq$1.0) the maximum allowable exposure height can be calculated:

$$H_F = \left[\frac{CL}{(L\gamma - 2C)(F\sin\psi - \cos\psi\tan\phi)\cos\psi}\right] + \left[\frac{B\tan\psi}{2}\right] \qquad (6)$$

(after Hassan and Bois,1998)[6]

Where,

H_F = Maximum allowable fill height with a Factor of Safety of F, m

FOS = 1.5

Uncemented backfill would never be used for pillar extraction and so, for cemented backfills, UCS is dominated by the added cohesion due to the binder and strength contributed by friction is considered negligible (i.e. $\varphi = 0^o$). Therefore, if a relationship between binder content and cohesion can be determined using triaxial testing methods, the equation below can be rearranged to yield an estimate of backfill strength with respect to the shear failure mode:

$$C = F\,(UCS) = \frac{0.5FL\gamma\,(H - 0.5B)}{L + (H - 0.5B)\,F} \qquad (7)$$

(after Hassan and Bois,1998)[6]

Where, *F (UCS)* = qualified function of UCS with respect to cohesion.

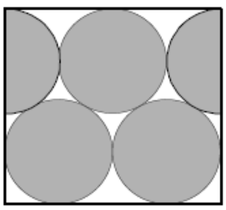
If the friction angle was nullified, cohesion was assumed to be half of the UCS, the maximum height was conservatively assumed to be equal to the effective height and a factor of safety of unity was set, the following expression could be developed:

$$UCS = \frac{\gamma H}{\left(\frac{H}{L+1}\right)} \qquad (8)$$

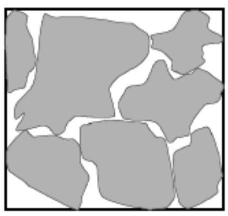
(after Hassan and Bois,1998)[6]

2.2.2 Particle Shape, Size and Distribution of Backfill Material

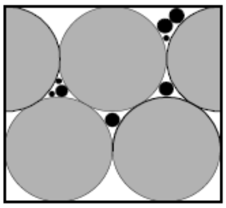
The mechanical properties of the backfill are highly influenced by physical nature of the solid particles, and the way they fit together. The constituent particle's shape, size and size distribution are often due the source of the fill material and the way in which it is processed. Due to the mineral communition process and blasting, most tailing particles and rock fills, respectively, are very angular and rough in nature while alluvial sands, which are weathered by water, tend to be round and smooth (Henderson & Revell, 2005)[9]. Particle shape affects the size of voids and connection paths available for holding and transporting fluids as can be seen in Figure 2.9, and aggregate friction angles and packing density as demonstrated in Table 2.2



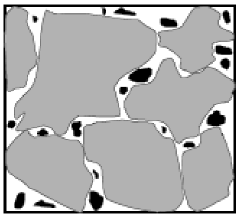
Uniform smooth particles



Uniform angular particles



Graded smooth particles



Graded angular particles

Figure 2.9 Diagrams of smooth and angular, uniform and graded, backfill (after Hassani & Archibald, 1998)[5]

Table 2.2 Effect of particle shape and packing density on friction angle

Shape and grading	Friction Angle (ϕ)	
	Loose Packing	Dense Packing
Rounded, uniform	30°	37°
Rounded, well graded	34°	40°
Angular, uniform	35°	43°
Angular, well graded	39°	45°

Source: (Hassani & Archibald, 1998)[5]

Fine particles in a soil increase the compressive strength of soil by filling in the voids between coarser particles and increasing inter-particle connections. An increase in fine particles also increases the pulp density of backfill during transportation making the slurry less likely to settle and thereby allowing for lower slurry critical velocity. Paste fills takes advantage of the presence of fines for increased strength and reduced settlement transport. However, slimes elimination traditionally has been practiced for slurry backfill in order to improve backfill drainage and to reduce slimes problems in decant systems. As can be seen in Figure 2.9, different backfill materials may be comprised of considerably different particle sizes. In order to compare one backfill to another and appreciate its structural makeup, fill materials are often classified by their particle size distribution. This distribution shows the range and frequency of particle sizes comprising the fill material and acts to define its material structure (www.infomine.com) [15]. Particles of different sizes within the fill material have effects on:

- Porosity,
- Permeability,
- Fill strength and

- Backfill transportation characteristics.

Particle size distributions have traditionally been determined using a system of progressively finer sieves that trap particles of a certain size range. A cyclosizer, which employs gravity separation of particles, may be used for finer particle size ranges as sieves become ineffective over the very fine size range, less than No. 400 mesh (38µm). Laser-based instruments have become more commonly employed recently for a wide range of fine and medium-particle size measurements. The cumulative percent of the fill material passing a certain particle size range is plotted versus the particle size on a log-normal chart. As shown in Figure 2.10, a fill material is described as being well graded if its constituent particles demonstrate a wide range of sizes and poorly graded or uniform if they demonstrate a narrow range.

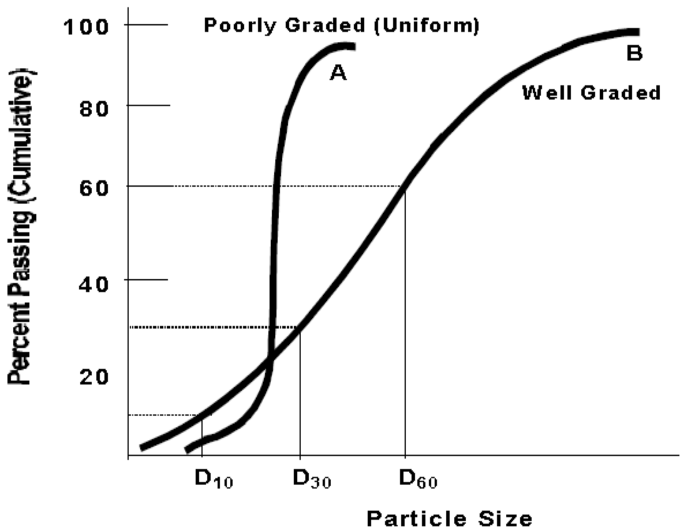


Figure 2.10 Typical particle size distribution curves for poorly graded (A) and well graded (B) aggregate materials (www.infomine.com) [15].

In order to quantify the distribution characteristics, a number of indices have been used to summarize the graphical representation with the most widely used index being the Coefficient of Uniformity (C_u):

$$Cu = (D_{60}/D_{10}) \qquad (9)$$ (after Craig,2005)[1]

The second index is the Coefficient of Curvature (C_c):

$$Cc = (D_{30}^{2})/(D_{60}*D_{10}) \qquad (10)$$

(after Craig, 2005)[1]

Where,

D_{10} = particle diameter at which10% of the material passes, µm

D_{30} = particle diameter at which 30% of the material passes, µm

D_{60} = particle diameter at which 60% of the material passes, µm

A well graded material contains equal representations of all size fractions with C_U values from 4 to 6, and C_C values from 1 to 3, being most desirable (Craig, 2005)[1]. For paste backfill a broad size distribution is often desired. Since paste fill materials comprise mixtures of aggregates and slimes, the coefficient of uniformity can be very high from 2 to 500. The value of D_{50} is often used to represent the average particle size. Figure 2.11 shows typical PSD curves for a cemented rock fill (CR), three cemented sand or alluvial fills (CS) and a cemented tailings fill (CT).

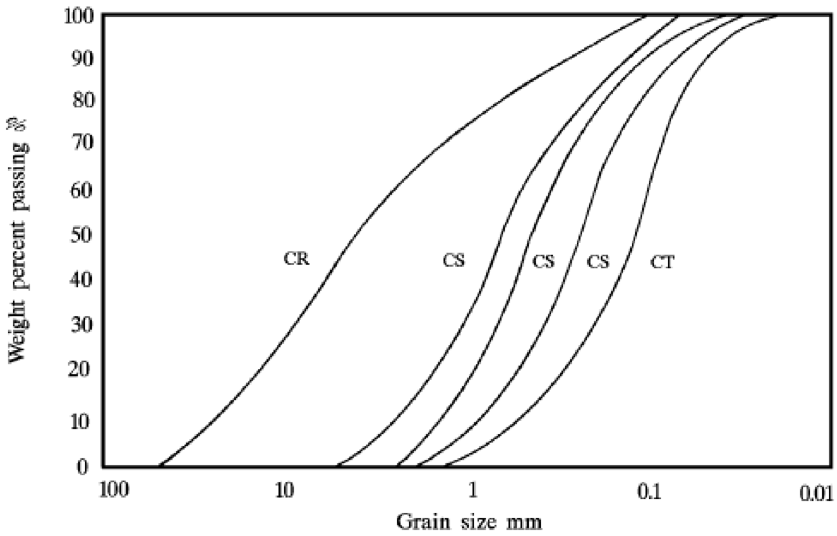


Figure 2.11 Grain size distributions for various backfill types. (www.infomine.com)[15]

The maximum grain size is generally limited in accordance with transport conditions. For pipeline transportation coarse aggregate, such as rock fill, must be crushed to smaller than 1/3 the diameter of the pipe. In the case of transportation by conveyor, the maximum grain size can be much larger.

2.2.3 The Power Law in Particle Size Optimization

The main purpose of backfill particle size optimization is to produce a fill that develops dense packing during placement. The particle size distribution of fill material can be represented by a power law as shown below:

$$P = 100\times [d/dmax]^{n} \qquad (11)$$

(after Kuganathan, 2005)[13]

Where: d = the particle size (mm)

$dmax$ = the maximum particle size (mm)

P = the percentage of rock fill smaller than size d

The above power law equation will generate a family of particle size distributions for a chosen maximum particle size. Such a family of particle size distributions is known as a set of Fuller curves as shown in Figure 2.12.

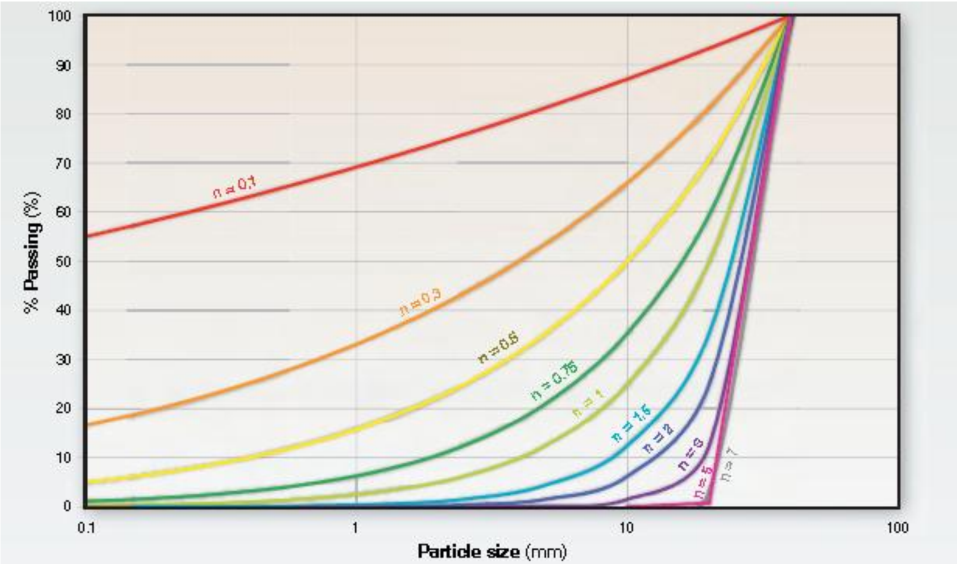


Figure 2.12 Fuller curves (Kuganathan, 2005)[13]

The particle size distribution demonstrates that, as the power law exponent 'n' increases, the particle size distribution becomes increasingly confined to a narrow range of particle sizes. Therefore, the porosity of fills with large power law exponents will be high, and the material will not develop high shear strength even when bound by binding agents (Kuganathan, 2005)[13]. It has been found by experience in Civil Engineering that when the power law exponent is 0.5, the optimum particle size distribution is obtained, resulting in a densely packed fill material. Hence, the most appropriate particle size distribution that would give such dense packing is given by:

$$P = 100\times [d/dmax]^{0.5} \qquad (12)$$

A well graded backfill, when combined with cemented deslimed tailings slurry at appropriate proportions, can result in a non-segregating, readily flowing rocky paste fill. If the fill obtained from development mining or from quarries does not satisfy the requirements for dense packing, then the grading must be adjusted by blending screened products of appropriate sizes at appropriate blending ratios.

In backfill engineering, two or more backfill materials can be blended together in particular ratios to produce a final backfill material with a desirable grading envelope.

2.2.4 Permeability and Percolation Rate

Permeability and percolation rates are extremely important slurry fill properties. A good permeability is required to ensure that the emplaced backfill drains and that there is no potential for fill liquefaction. It is also required to ensure that excess transport water drains rapidly to allow for high placement and curing rates. Traditionally, permeability has not been considered in the design of paste fills as there is usually little excess water to drain, and binders are always added to minimize the potential of liquefaction.

Permeability is an absolute property of a porous medium and describes the flow of any fluid at any temperature and at any hydraulic gradient, provided Darcy's Law remains valid.
Percolation rate applies only to a particular fluid at a particular temperature and for a hydraulic gradient of unity. The universally accepted percolation rate, for mining slurry backfills, is 10cm/hr and is suggested as a good design guideline for backfill placement. Percolation rate is closely related to grain size composition, particularly for fine particles, and the cement content, if any, of the backfill. The percolation rate has a very close relationship to the content of fine particles in tailings. With an increase in the quantity of fine particles in backfill, the percolation rate drops rapidly. This is the principal reason why mines have to de-slime tailings material which is to be utilized as slurry backfill.
The factors influencing the percolation rate are categorized into three parts:

- porosity (particle shape included),
- particle constituency, and
- Composition of minerals.

The definition of permeability is derived from Darcy's law:

$$k = (QL\eta)/hA\gamma^2 \qquad (13)$$

(www.infomine.com)[15]

Where,

k = permeability of porous medium, m^2.

Q = quantity rate of flow of fluid through porous medium, N/s

L = length of porous medium, in direction of flow, m.

h = static pressure differential across porous medium, m.

A = cross-sectional area of porous medium, normal to direction of flow, m^2.

η= absolute viscosity of fluid flowing, Ns/m, and

γ= unit weight of fluid flowing, N/m^3.

This expression for permeability is valid only while Darcy's law is valid, as is usually the case during testing and dewatering of fill materials. Darcy's Law inherently improves in range of validity as fill fineness increases. Percolation rate is defined by the equation:

$$V_p = q/A \qquad (14)$$

(www.infomine.com)[15]

Where,

V_p = percolation rate, m/s

q = quantity rate of flow, m^3/s, and

A = cross-sectional area of sample normal to flow direction, m^2.

Permeability, with units of m^2, has no inherent physical significance, so it is not surprising that percolation rate, with its readily comprehensible units of m/s, has been adopted throughout the mining industry as an indicator of backfill drainage suitability. Percolation rate and permeability are in fact directly related, as described below:

$$k = (qL\eta)/hA\gamma \qquad (15)$$

(www.infomine.com)[15]

For a unique solution at constant temperature, η/γ is constant and the equation becomes:

$$k = (qL)/hA \qquad (16)$$

(www.infomine.com)[15]

Where,

k = permeability coefficient, m/s

q= quantity rate of flow, m^3/s

L= length of porous medium, in direction of flow, m

h= static pressure differential across porous medium, m

A= cross-sectional area of porous medium, normal to direction of flow, m^2

η= absolute viscosity of fluid flowing, Ns/m, and

γ= unit weight of fluid flowing, N/m^3

It should be noted that permeability coefficient units are the same as those for the percolation rate. Accurate measurement of permeability (or for that matter percolation rate) is one of the most difficult experimental aspects of fill technology. The main problem is the inability to establish a constant flow rate. There are three methods available for measuring the permeability of fill materials:

- The Constant Head Permeameter,
- The Falling Head Permeameter and
- Percolation tube.

The first two methods are used with materials that exhibit very low permeability, while the latter is more relevant to the mining industry where high permeability backfill materials are commonly encountered.

Some studies on actual mine fill materials have concluded that percolation rates can be calculated using the following empirical equation.

$$ln(25.4P_{20}) = 11.3915 + 2.85342ln(eD_{10}) + 0.1747436eCu - 178.8039D_{10}D_{50} + 311.703D^{3}_{50} \quad (17)$$

(www.infomine.com)[15]

Where,

P_{20} = percolation rate at 20 °C, mm/h

e = void ratio

D_{10} = particle diameter where 10% of the material passes, mm

D_{50} = particle diameter where 50% of the material passes, mm

C_U = Coefficient of Uniformity

Equation 17 remains valid provided that test samples satisfy the following modified conditions:

- e, between 0.430 and 1.080,
- D_{10} between 0.003 and 0.105mm,
- D_{60} between 0.0053 and 0.240mm
- C_u between 1.77 and 22.0

An expansion of the equation to remove the above limitations was done and the following empirical relationship was developed.

$$P = 5000\left[\frac{D_{10}^{2}}{e^{\left(\frac{0.45-n}{0.16}\right)}}\right] \quad (18)$$

(www.infomine.com)[15]

Where,

P = percolation rate, cm/h

n = porosity, %

Addition of Portland cement greatly reduces the permeability of hydraulic fill (Grice, 2007)[4]. Measurements using the falling head Permeameter procedure have demonstrated that, as cement content increases, the permeability decreases. It has been reported that the addition of small quantities of cement to classified hydraulic backfill would not alter the initial porosity significantly. Cementation will, however, decrease the percolation rate due to the formation of cement gel in the void spaces with cure time. The effect of cement content on percolation can be summarized by the following:

- Increased cement content decreases the percolation rate.
- For given cement content, the percentage decrease in percolation rate is greater in finer-sized aggregate materials.
- For a given cement content, the decrease in percolation rate also appears to be dependent on the type of tailings and pulp density of the pour
- In most cases where slimes (minus 0.02 mm material) are included in the classified tailings, decant systems will be required for cemented backfills.

Percolation rate is influenced strongly by the content of fine particles in tailings (Grice, 2007)[4]. With an increase in the content of fine particles the percolation rate drops quickly as shown in Figure 2.13.

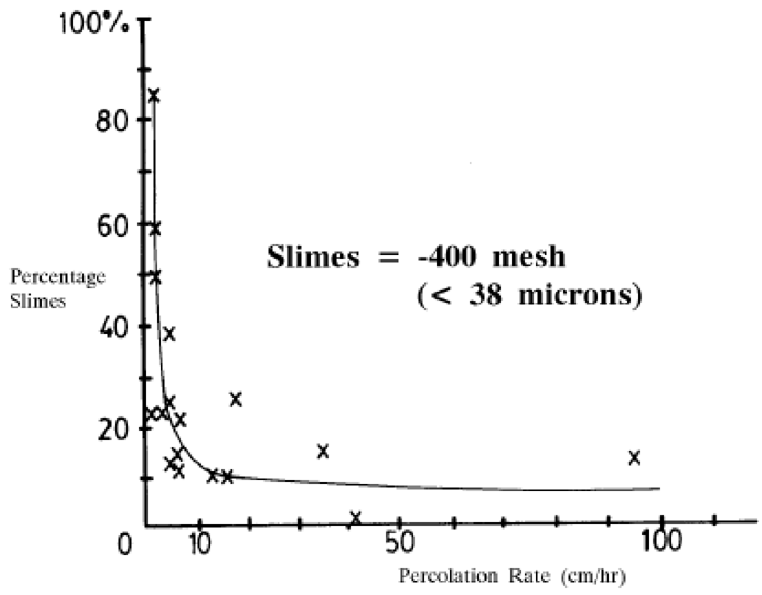


Figure 2.13 Percolation rate versus percentage of fines in Backfill (after Grice, 2007)[4].

This is the primary reason why mines have traditionally deslimed (classified) mill tailings backfill aggregate. The factors influencing the percolation rate are: porosity (particle shape included); particle consistency and material composition (important in thickness of watering film)

2.2.5 Consolidation and Compaction

Consolidation in engineering terms refers to a cemented or consolidated mass. However, in classical soil mechanics, consolidation refers to the densification (i.e. closer particle packing) of a soil at a rate controlled by the flow of water from the soil. When a saturated soil is subjected to some external load, the pore water pressure will increase to resist the load and prevent particle rearrangement. If pore pressures dissipate very rapidly, as they do in sands, pore pressures may be ignored except when considering very rapid loading, such as during blasting. In practice, consolidation of initially loose, moist, cohesionless slurry fills under low confining loads occurs very quickly with consolidation frequently being 90% complete within approximately three seconds following load application (Craig, 2005)[1].

Paste fill, however, is much finer and its ability to dissipate pore pressures may be insufficient under rapid loading conditions when uncemented (Grice, 2007)[4]. Compaction is similar to consolidation, but is not normally controlled by the expulsion of water. A partially saturated or drained fill can undergo compaction as void space is reduced due to the action of external forces, such as mobile equipment moving across it.

2.2.6 Stiffness

Stiffness (κ) relates a material's resistance to deformation, either in compression or tension, and is expressed in terms of load resistance per unit of deformation, as given:

$$\kappa = P/\Delta L \qquad (19)$$

(www.infomine.com)[15]

Where,

κ = stiffness, N/m

P = applied force, N

ΔL = deformation induced by the applied force, m

Similarly, axial strain provides a measure of material deformability, and is expressed as the deformation per unit length when a stress is applied to the material. Axial strain relates a material's resistance to deformation or compression and is expressed as the deformation per unit length:

$$\varepsilon = \Delta L/L \qquad (20)$$

(www.infomine.com)[15]

Where,

ε = Axial compressive strain, %

ΔL = deformation, m

L = unit length, m

Stress and strain are intrinsically linked and can be related by the following expression:

$$\sigma = \varepsilon \cdot E \qquad (21)$$

Where,

σ= Axial compressive stress, MPa

E = Modulus of Elasticity (Young's Modulus), MPa (www.infomine.com)[15]

From strength perspective, it is always important to design a backfill with as high stiffness or Modulus of Elasticity, as possible. Deformability of a material is proportional to its modulus of Elasticity or Deformation. The modulus of a backfill is generally less than that of the in-situ rock. Table 2.3 below shows typical values of Young's Modulus (E):

Table 2.3 Typical values of E for common mine materials

Material	**Young's Modulus, E (GPa)**
Intact Rock	70
Rock fill	5
Slurry Backfill	0.5

Source: (www.infomine.com)[15]

Stiffness is important where the backfill is intended to limit ground convergence and/or bear some tertiary stress from adjacent pillars. The addition of a binder to backfill significantly increases its stiffness. However, it should be noted that a stiff material is also a brittle material. This makes stiff backfill more prone to cracking under blast loading and to rupturing under local rock deformation. It is recommended that forces, displacements and energy dissipation be considered in backfill design. Addition of cement to backfill must be moderate so as to enable exposed fill yield without rupturing (www.infomine.com) [25].

The operational use to which backfill will be subjected has significant impact on the required mechanical and cure response needed. In mining systems such as open stoping operations, the fill must be stable when free standing wall faces are exposed during pillar recovery. Depending upon the mining schedule, high strengths for such materials may not be required in the short term. In mining systems such as in cut and fill stoping, the fill must work as a platform for mining equipment and personnel and typically requires high strength development in the short term. For optimum backfill strength and cure response, slurry fills require proper drainage to allow excess transport water to drain from the stope, and the backfill must be sufficiently strong to fulfill its primary function with or without a binding agent. These functions all focus on strength (i.e. mechanical properties) and availability (i.e. strength at a desirable time). Backfill strength is directly attributable to the physiochemical properties discussed previously and, again, many soil mechanics relationships can be drawn on and expanded upon to reflect the backfill mechanical environment of mining.

Backfill is relatively soft compared to the rock mass that surrounds it after placement and does not provide significant tributary support for the rock mass. The main stabilizing effect from the fill is to give increased lateral confinement pressure onto the rock walls or pillars that support the rock masses load (Hassan and Bois, 1998)[6]. The strength of a rock pillar, in confinement, may be increased above its unconfined compressive strength, as represented by:

$$S_C = \sigma_1 + K\sigma_3 \qquad (22)$$

$$\sigma_3 = \sigma_V(\tan^2(45° - \phi/2)) \qquad (23)$$

$$\sigma_V = \gamma_{FILL} \cdot H_{FILL} \qquad (24)$$

$$K = \frac{(1+\sin\phi)}{(1-\sin\phi)} \qquad (25)$$

(Hassan and Bois, 1998)[6]

Where,

S_C = confined pillar strength, MPa

σ_v = vertical stress (i.e. induced within emplaced fill by gravity), MPa

σ_3 = active fill pressure, MPa

K = friction factor (dimensionless)

Φ = backfill friction angle, ^{o}

γ_{FILL} = bulk density of emplaced backfill, N/m^3

H_{FILL} = average depth of backfill in stope, m

2.2.7 Liquefaction

Liquefaction describes the behaviour of soils when there is a rock burst or is loaded suddenly. It suffers a transition from a solid state to a liquefied state, or having the consistency of a heavy liquid. Liquefaction is more likely to occur in loose to moderately saturated granular soils with poor drainage, such as silty sands or sands and gravels capped or containing seams of impermeable sediments. During loading, usually cyclic un-drained loading, e.g. earthquake loading, loose sands tend to decrease in volume, which produces an increase in their pore water pressures and consequently a decrease in shear strength, i.e. reduction in effective stress (www.wikipedia.org)[16].

Liquefaction can occur in saturated tailings, sands, silts and quick clays. When liquefaction occurs in backfill materials, the backfill body will behave as a fluid with a mass greater than that of water, resulting in very high hydrostatic pressures on retaining walls and great potential danger should the backfill flow. On liquefaction the backfill mass undergoes very large unidirectional shear strain it appears to flow until the shear stresses are as low as, or lower, than reduced shear resistance. Where a fill mass sustains static shear stress than the steady state un-drained shear strength, it is susceptible to liquefaction.

Liquefaction is primarily a concern for unconsolidated fills and is not possible with consolidated fill which has cured sufficiently to develop even a small amount of true internal cohesion. However liquefaction may be a concern for all types of hydraulic fill in the event of cyclic loading. Monatomic loading is not a concern for slurry fill due to its high permeability (Helinski, 2007)[7]. The lower permeability of paste fills should be considered in assessing liquefaction potential. An explosive charge may cause liquefaction of saturated unconsolidated tailings backfill. However, if the fill has a high percolation rate so that the backfill is kept in an unsaturated state, liquefaction of the fill will not occur. Practice and experimental results have indicated that dilative soils are not susceptible to liquefaction. However, at the lower effective consolidation pressures and lower tailings to cement ratios, paste backfill is weakly dilatant (Helinski, 2007)[7]. It has been reported by Potvin (2005)[14] that a minimum UCS of 100 kPa is required in consolidated backfill to prevent liquefaction.

2.2.8 Backfill Segregation

When finer particle fractions of hydraulic backfill slurry settle at considerably slower rates than the coarse fractions, the phenomenon is called Segregation or stratification. Backfill material in the stope will show grading if fine fractions are carried away in the decant water. Segregation causes the fill mass to be non-homogenous and may result in an overall reduction in fill strength due to the creation of planes of weakness and areas of low binder content.

The strength of cemented hydraulic backfill is influenced by the orientation of segregation planes. Segregation may also be directly influenced by the position of the backfill slurry discharge point within a stope. By adjusting the discharge point, the orientation of the planes of segregation in a stope can be controlled and the formation of weaker zones that dip in the direction of faces to be exposed can be avoided (Hassani and Bois, 1998)[6].

2.3 Backfill Mineralogical and Chemical Composition

The mineralogy of fill material influences a number of backfill characteristics such as:

- water retention,
- strength,
- settling characteristics and
- abrasive action.

Different ore types with different mineral assemblages can result in different fill properties. The mineralogy can also affect the final strength of paste fill by influencing chemical reactions that promote strength or produce additional chemical reactions. Sericite has been identified as retaining water (and hence reducing strength) in paste fill due to the mineral layers absorbing water. Other minerals that commonly exhibit similar behaviour are micas and clay minerals (Henderson & Revell, 2005)[9]. Other examples of mineralogy affecting fill properties include:

- Specific Gravity of minerals which will be one determinant of the density of the fill,
- Silica minerals (particularly quartz) as it can be very abrasive and result in high pipeline wear

- Sulphides have the tendency to oxidize due to their reactivity and could become a dangerous source of hydrogen sulphide gas that can form the ignition point for underground fires. Mixtures of sulphides can increase reactivity. Sulphide oxidation can produce several different effects within the mine fill environment such as:
    - Time dependent strength gain in uncemented fills,
    - Time dependent strength deterioration in cemented fills.

  Studies have shown that if a mine fill contains a large enough quantity of sulphides and the moisture conditions and oxygen supply are ideal, accelerated oxidation of the sulphides can result in:
    - Heat and sulphur dioxide fumes,
    - reduction in oxygen supply underground and
    - Acid mine drainage (www.infomine.com)[15]

  Processing of sulphide ore results in generation of significant sulphates in the form of gypsum. This would lead to strength deterioration by way of facilitating secondary expansion of cement. This type of strength deterioration could be halted by the partial substitution of cement with ground waste glass. In general, studies within the concrete industry have shown that pozzolans and slags improve the resistance to sulphate attack on cements (www.infomine.com) [15].

Chemical admixtures exist which are used to either accelerate or retard the set of cement and/or its early strength performance when added to backfill. The following lists the most common reasons for such requirements (Hassani and Bois, 1998)[6].

- Unacceptably low early strengths due to application of blended cement, resulting in long cycle times.
- Unacceptably low early strength due to low in-situ temperatures.
- Need to prevent normal cement set due to delays in backfill emplacement (commonly required for truck-delivered batched backfill operations)

Accelerator additions designed to decrease backfill cycle time have been the principal focus of additive research.

2.3.1 Flocculants and Dispersants

During drainage in slurry backfilling operations, fine particles are lost. To reduce on this, flocculants can be used. Ideally a flocculant should not only prevent the loss of cement and

fines from the fill and improve the overall drainage, but also decrease segregation of cement within the fill mass and thereby improve strength.
Dispersants have also been suggested as a means of improving the performance of slurry fills. The primary effect of a dispersant is to improve the drainage properties of the tailings which would be particularly useful in uncemented backfilling applications or for increasing the storage capacity of a tailings pond. Effectiveness of dispersants or flocculants is strongly influenced by tailings mineralogy (www.infomine.com)[15].

2.4 Reinforcement Elements

A range of reinforcing techniques, have been investigated as a means of increasing backfill stabilization and include fibre, mesh and geotextile-sheet reinforcing elements as shown in Figure 2.14.

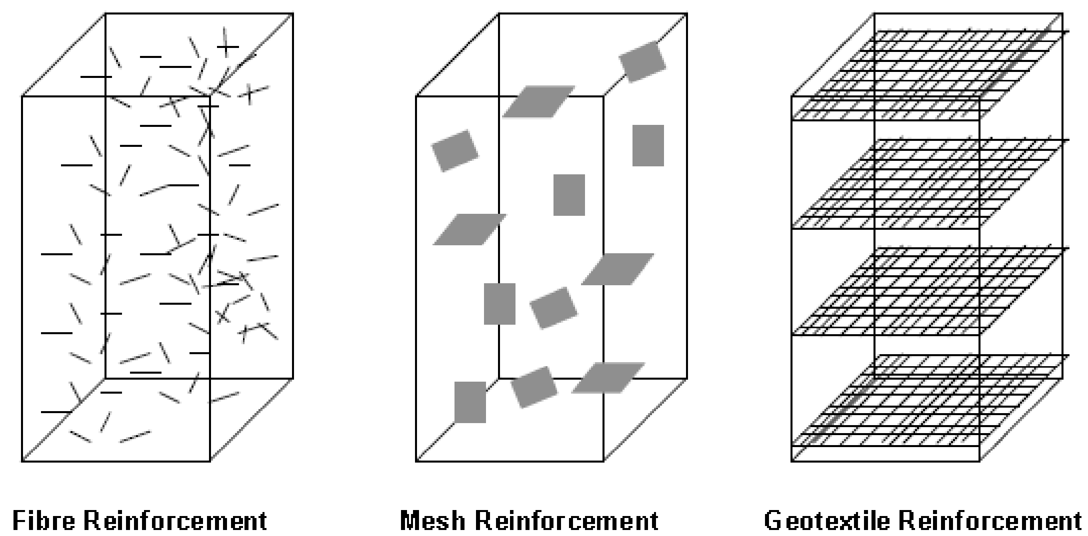


Figure 2.14 Reinforcement elements used in backfill.

Mesh elements are essentially sheets of screen of varying dimensions and aperture widths. Geotextile sheets, placed throughout the fill mass aid in maintaining backfill strength integrity, but tend to be more complicated to install. However, one possibility for installation would involve anchoring the Geotextile sheets with nails to the rock sidewalls followed by a pour session. Elements may be incorporated with backfill prior to distribution or at the stope; however, larger elements may plug the distribution pipelines.

Fibres can increase the ultimate strength of cemented backfill by up to 50% in a backfill with 6% cement. Catastrophic failure is inhibited as the fibres tended to maintain some strength integrity even after fill failure (www.infomine .com) [15].

2.5 Backfill Binders

The inclusion of cement into mine backfill has given rise to new systems and extraction strategies to be employed, including increased stope dimensions. This allowed for the development of bulk mining methods that lead to improvement in mine safety and decrease in mine costs. There are two distinct, but similar, mechanisms utilized in binder chemical reactions. The popular modern binder in use is Normal Portland cement and its tailored derivatives for special cases (i.e. sulphate resistance). Portland cement is a crystalline material that, upon hydration, forms cementitious compounds.

A pozzolan is "a siliceous or siliceous and aluminous material which in itself possesses little or no cementitious value but will, in finely divided form and in the presence of moisture, chemically react with calcium hydroxide (lime) at ordinary temperature to form compounds possessing cementitious properties." In olden days, natural pozzolans and lime mixtures were used as binders. Pozzolans may consist of small amounts of crystalline material but tend to be amorphous in nature; upon hydration they form cementitious compounds similar to those of Portland cement. Portland cement is said to be cementitious since it contains large quantities of silica and lime – the primary building blocks of binders. Most pozzolans are deficient in lime and require its addition to become chemically active (Henderson & Revell, 2005)[9]. Modern pozzolans can be classified into five representative groups:

- Cementitious (quenched blast furnace slag),
- Cementitious and pozzolonic (high calcium fly ash),
- Highly pozzolonic (condensed silica fume),
- Normal pozzolonic (low calcium fly ash), and
- Weakly pozzolonic (slow cooled blast furnace slag).

One source of pozzolonic material is blast furnace slag which is a by-product that results from the fusion of fluxing limestone with coke ash and siliceous and aluminous residues remaining after the separation of the metal from ore. Blast furnace slag tends to be quite consistent in physical and chemical composition which assists in assuring reproducible product quality. As a result of selective cooling, four distinct blast furnace slag ‘products’ exist, these being: air-cooled, foamed, granulated, and pelletized (Henderson & Revell, 2005)[9].

2.6 Types of Backfill Systems and Associated Flow or Transport Regimes

2.6.1 Slurry Backfill

Slurry backfill, also known as hydraulic backfill, is a classified, highly permeable, low solids density mixture of aggregate and water (averaging between 60% to69% solids, by weight) that must be transported in pipelines at high velocities to maintain turbulent flow in order to maintain its solid phase in suspension. Substantial transport water is needed to flush the backfill through the mine distribution system. As a result, there is significant drainage water that seeps from stopes, in which it is placed, that must be returned to surface. Slurry backfills normally consist of either classified mill tailings (tailings fill), alluvial sand (sand fill), from a naturally occurring deposit, or some combination of the two. It may or may not incorporate a binder, depending on the nature of its function. This type of backfill may incorporate an even greater solids concentration (ranging between 70% to 79% solids, by weight) to form a dense slurry. Dense slurry backfill is distinct from paste fill in its intrinsic composition and transport regime (Grice, 2007)[4].

Solid materials chosen for slurry backfill must be prepared according to the needed technical properties of the fill and functional needs. The fundamental criteria for solid preparation are (Hassani and Bois, 1998)[6]:

- Fill should be prepared in accordance with mechanical property requirements generated from underground operation, and preparation cost should be optimized.
- The solids concentration of slurry must be optimum for its transportation and satisfactory backfill performance. The results of practical studies indicate that the optimum solids concentration of a slurry ranges between 65% to 75% solids (by weight), depending on the specific gravity of the solids.
- Fine particle content tailings or sand must be minimized to render the resultant fill inherently safe with respect to possible liquefaction after placement and to allow it to drain sufficiently rapidly after placement.
- The percentage recovery of fill from tailings or waste disposal should be maximized, to ensure an adequate supply of fill with reduced environmental hazard. The preparation flow sheet for a cemented classified tailing is illustrated in Figure 2.15.

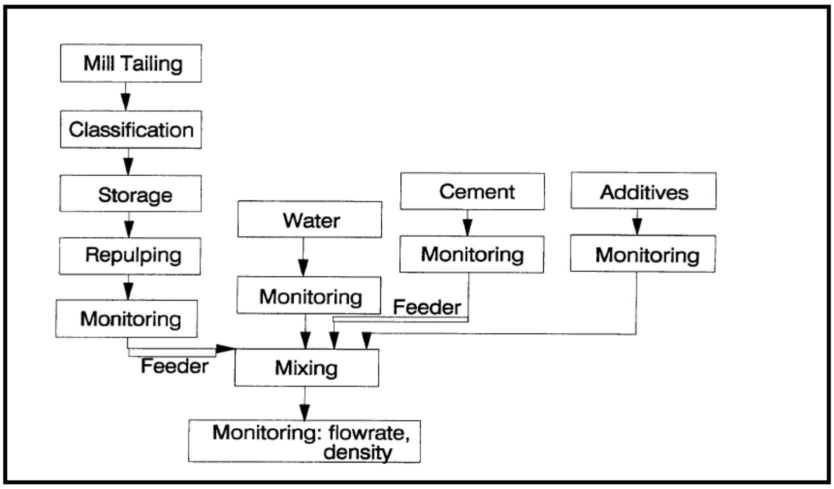


Figure 2.15 Tailings fill preparation flow sheet (Hassani and Bois, 1998)[6].

Mill tailings consist of a fine solid aggregate material with an average particle size of about 100μm. They can be used as deslimed tailings or full tailings depending upon the percolation rate required. Tailings fill can be divided into six categories based on fine particle content (minus 10μm) as illustrated in Table 2.4.

Table 2.4 Tailings fill classification.

Mass Fraction (minus 10 μm), %	Classification
0 to 10	Very coarse
10 to 20	Coarse
20 to 30	Medium
30 to 40	Fine
40 to 50	Very fine
> 50	Extremely fine

Source: (Hassani and Bois, 1998)[6]

Classified tailings containing fines of less than 20% (by weight) 10μm are commonly utilized in backfill that has a percolation rate above 10cm/hr. Tailings are used only for paste fill, where pulp densities may exceed 80% solids (by weight).

2.6.1.1 De-sliming and Centrifugal Dewatering

Ore processing produces tailings which generally contain a significant portion of fine particles of size less than or equal to 10μm. This brings down the effectiveness of stope drainage for hydraulic fill and may lead to liquefaction. Tailings de-sliming must therefore be undertaken always, to reduce the presence of fine solids in the tailings to less than 20% by weight. The de-sliming is usually performed using hydrocyclones, which may operate in various circuit combinations. A classifier (thickener) could also be used depending on the size limitation and requirements of tailing recovery (Grice, 2007)[4].

Hydrocyclones are probably the most popular mechanical classifying equipment used in mining today. The hydrocyclone is physically small and mechanically simple with relatively low capital and operating costs. The tailings are pumped directly from the mill plant or disposal pond to the hydrocyclone in slurry form with a density of between 30% to 40% solids (by weight) (www.wikipedia.org) [16]. Figure 2.16 illustrates the operations of the cyclone.

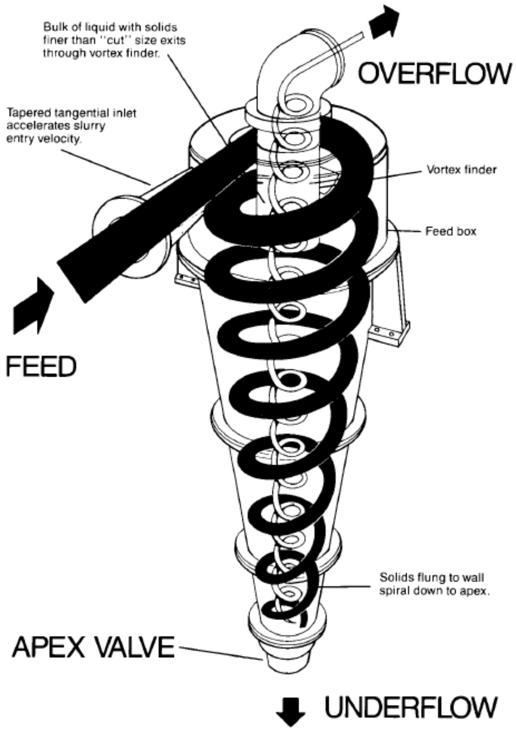


Figure 2.16 Centrifugal Methods of Dewatering and De-sliming of Tailings (Hassani and Bois, 1998)[6].

The solids are thrown to the outside of the cyclone and, as the primary vortex travels down the conically shaped cyclone, they are throttled and the solids escape through the bottom. A secondary vortex carries the clarified slurry in a tighter and faster flow up through the center of the cyclone. This acts as a second filtering system which increases the particle recovery. The fine particles are rejected as overflow and the coarser particles are discharged as underflow with a density of above 50% solids (by weight).

Cut point is proportional to the fourth root of the pressure drop across the cyclone. Hence fine separation requires small hydrocyclones and it is found that in commercial applications the capacity dictates that many cyclones must be connected in parallel (www.wikipedia.org)[16].
The principal disadvantage of hydrocyclones is that they are relatively inefficient in terms of accurate particle sizing, particularly in allowing fine slimes to be misplaced into the coarse underflow stream. For fill preparation, where high throughput de-sliming with a sharp separation is required, this leads to the necessity to use a large number of very small cyclones operating in parallel from a common feed manifold. Consequently it causes supervising problems in ensuring that each cyclone in the cluster receives the same feed in terms of pulp density and size distribution (Hassani and Bois, 1998)[6].

2.6.1.2 Mixing Cement and Tailings

In the production of cemented backfill, re-pulped tailings flow down to a mix tank, where tailings are mixed with a binding agent and/or additives from the cement silo. Storage of cement in the Silos is generally brief and in small quantities that can be delivered by a screw feeder or conveyer to the mixing tank. The cement supply is measured by a measuring device such as a weight-meter and controlled mechanically or automatically by changing the speed of the discharging device installed at the bottom of the silo. Production of consistent slurry required for backfill, is achieved by addition of a specific quantity of water to the mixer. Addition of additives is through a special container installed at the fill station that consistently supplies additives at a preset rate. If there are insufficient tailings, a combination of tailings with sand, gravel or crushed rock may be utilized (Hassani and Bois, 1998)[6].

2.6.1.3 Stope Preparation

Stope preparation prior to backfilling differs according to the type of stope to be filled. This in turn is affected by the mining method employed. There are two aspects that must be considered in stope preparation. The first one involves preparation in cyclic stope production prior to backfilling, which requires an initial or basic preparation followed by cyclic adjustments and maintenance. The second one involves preparation in non-cyclic or open stopes prior to backfilling, in which the preparation activity is delayed until stope depletion. Backfilling is then commenced as one continuous operation to completely fill the stope. Generally, stope preparation involves installation of stope and backfill dewatering systems and installation of bulkhead (Grice, 2007)[4].

Primary methods which permit controlled and effective stope drainage are drainage pipes and decant towers or raises. Mechanical pumping is sometimes used but is difficult to employ during backfilling and also requires considerable maintenance due to the presence of solid particles in the stope water. In longer ore bodies, such as those over 25m, sand fill ramps and barricades are built to separate production activities, and plastic sheeting is laid down along the inner slope to prevent excessive seepage. Percolation water is either pumped into the sublevel drainage system or allowed to gravitate to the main drainage level via drain holes (Hassani and Bois, 1998)[6]. Figure 2.17 shows a schematic illustration of drainage pipes in a stope.

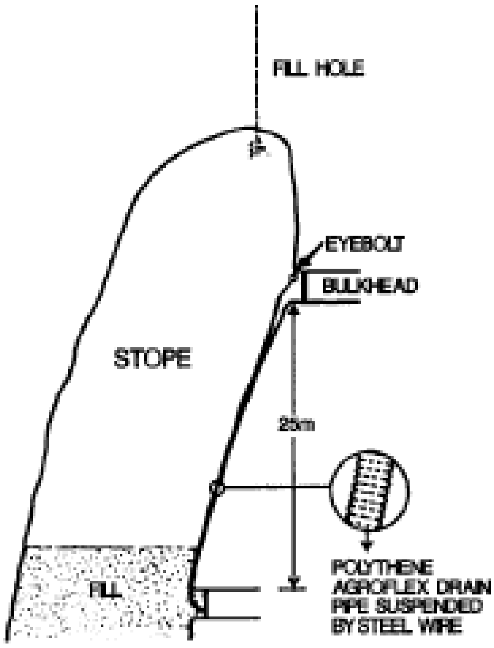


Figure 2.17 Percolation Pipes in Open Stope. (Hassani and Bois, 1998)[6].

2.6.1.4 Bulk Heads

A bulkhead is defined as a structure which contains fill materials in the stope as illustrated in Figure 2.18.

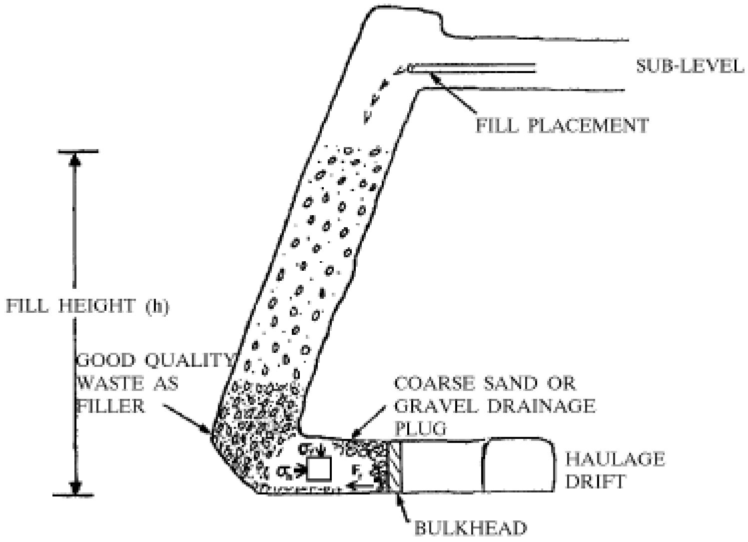


Figure 2.18 Schematic diagram of a stope being filled (after Hassani and Bois, 1998)[6].

Bulkheads are installed in the draw points of the topes or on remnants of sub-levels (Helinski, 2007)[7]. Bulkheads for retaining dry fill do not require any special consideration unless water flows into the stope. Rock fill containing appreciably large amounts of fines, when saturated, could generate mud flows into adjacent working areas, hence a bulkhead is required. Bulkheads to retain slurry fill require special considerations such as a drainage system, solids retaining devices, and so forth. There are many types of standard bulkhead and the most common will be described briefly below. Costs associated with bulkhead construction are site specific and can differ appreciably depending on the mining method employed and the backfill used.

A vertical bulkhead, constructed some distance from a stope face, as shown in Figure 2.18 can be represented in the general equation as shown below.

$$P = \sigma_h + \sigma_v - F_r = U + P_a - F_r \tag{26}$$

(after Hassani and Bois, 1998)[6]

Where:

P = Total maximum pressure on bulkhead, kPa.

σ_h = Horizontal fill pressure, kPa.

σ_v = Vertical pressure due to the fill and water, kPa.

$= \gamma h$

γ = Average weight per unit volume of the fill, kN/m3

h = Fill height, m

F_r = Frictional resistance of the fill material, kPa (0.1P to 0.4P)

U = Hydrostatic water pressure in the fill, kPa ($\gamma_w h$)

Pa = Active fill pressure acting horizontally on bulkhead, kPa

φ = Internal friction angle of the fill, ^{o}

$$P_a = \sigma_V \tan^2\left(45 - \frac{\phi}{2}\right) = \gamma h \tan^2\left(45 - \frac{\phi}{2}\right) \qquad (27)$$

(after Hassani and Bois, 1998)[6]

Liquefaction involves a reduction of the effective horizontal stress within the mass of granular fill. This situation exists if energy is imparted to the mass by occurrences such as rock bursting, blasting or earthquake activity. Under such conditions, the fill behaves as a liquid with a density nearly double that of water (www.wikipedia.org) [40]. The result is a sudden pressure increase on the bulkhead equivalent to the hydrostatic head of the fill slurry. The total pressure on the bulkhead is:

$$P = U + Pa - Fr \qquad (28)$$

$$P = \gamma_{LF} h + 0 - 0$$

Where:

γ_{LF} = weight per unit volume of the liquefied fill material, kN/m^3

2.6.1.5 Transportation of Slurry

Three basic configurations have been identified for moving fill material from a point on surface to stopes underground as shown in Figure 2.19 (Hassani and Bois, 1998)[6].

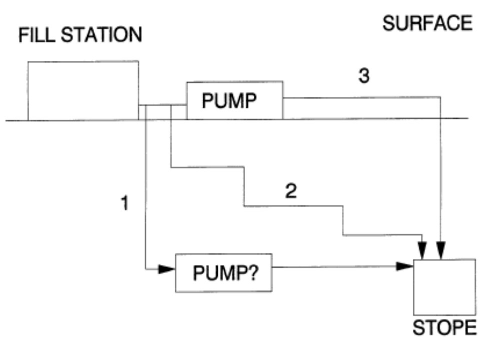


Figure 2.19 Basic configurations for fill placement Systems.

No. 01

The number 1 configuration has the advantage of being totally restricted underground. This does not cause any disturbance to surface operations. Additionally, the ratio of the vertical to horizontal distance is generally favorable and little pumping energy is essential. Actually measures to limit the flow velocity are occasionally taken to control the wear rate of the pipeline system.

The disadvantages of this circuit comes in when the ratio of the vertical to horizontal distance is relatively large or small, such as 20:1 or 2:1, respectively. This case comes in especially in deep mines where the stope to be filled is close to the vertical drop section of the pipeline resulting in very high pressure at take-off points, and a burst line may disrupt the Shaft level or main level operations. This could be secondly a pump could be required to transfer the fill in the horizontal section of the pipeline leading to additional energy and maintenance costs. The pressure at take off points could be as much as 20 MPa for a high ratio scenario and 120 kPa for a low ratio scenario. (Hassani and Bois, 1998)[6].

No.02

The number 2 configuration has the benefit of making the vertical head to horizontal pressure progressive. In this case shorter and lighter pipes are used. The pressures at take-off points are moderate and line failures, if they occur, do not interrupt the main Shaft or main level of activities. The circuit develops as the mine expands. The disadvantages of this configuration are in terms of increased maintenance costs resulting from the stepwise pipeline paths (Hassani and Bois, 1998)[6].

No.3

An advantage of number 3 configuration is the ease of installation, inspection and maintenance. This means no special underground level requirements and no disruption of the main Shaft. The disadvantage of this system is that it makes the filling operation dependent upon a pumping operation and requires a long borehole to place fill underground. This results in high pressure take-off point. Additionally, disruption to surface activities is possible (Hassani and Bois, 1998)[6].

The Bernoulli equation for the conservation of energy as applied to continuous incompressible fill flow in pipelines may be stated as follows:

$$\frac{P_1}{\rho g}+\frac{V_1^2}{2g}+Z_1+H_P-H_F=\frac{P_2}{\rho g}+\frac{V_2^2}{2g}+Z_2 \tag{29}$$

(www.wikipedia.org)[16]

Where,

$P/\rho g$ = static head (pressure), at points 1 and 2, respectively

$V^2/2g$ = velocity head, at points 1 and 2, respectively

Z = elevation head, at points 1 and 2, respectively

H_P = head added by pump

H_F = head lost to friction

The assumption of the incompressibility of the fluid is usually valid; however, the continuity condition may not always be satisfied, especially in the free fall sections of the vertical pipes or boreholes. Free fall in pipes causes' inlet static pressures below atmospheric and hence draw air into the fill line and must be avoided. Furthermore 'pipe hammer' may result from impact and, as the flow joins the full flow section of the pipe, accelerated pipeline wear may occur (www.wikipedia.org)[16].

2.6.1.6 Basic Design Principles

To minimize the volume of water required for fill transportation, it is recommended to use a high solids concentration when designing a slurry fill system. Furthermore to minimize pipe wear, operating flow velocity should be as low as possible but high enough to keep coarser particles in suspension (Grice, 2005)[3]. Critical velocity is usually introduced as a lower bound for the operating velocity below which deposition of solid particles forms a stationary bed indicating imminent plugging of the pipeline. As the solids concentration increases, critical velocity becomes less relevant due to the hindered settling tendency of the particles. In order to reduce the effect of free fall on the flow behaviour of the slurry, the pipe diameter in the free fall region may be reduced or some other method of restricting the flow engineered (Hassani and Archibald, 1998)[5].

Generally, four flow regimes are recognized in the transport of solid-liquid (i.e. slurry) mixtures as illustrated in Figures 2.20 and 2.21.

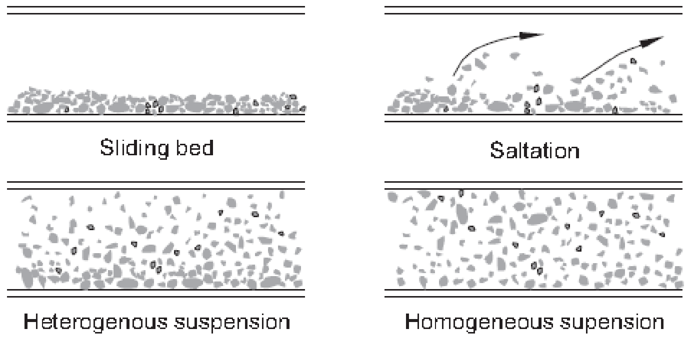


Figure 2.20 Four regimes of flow for settling Slurries in horizontal pipelines.

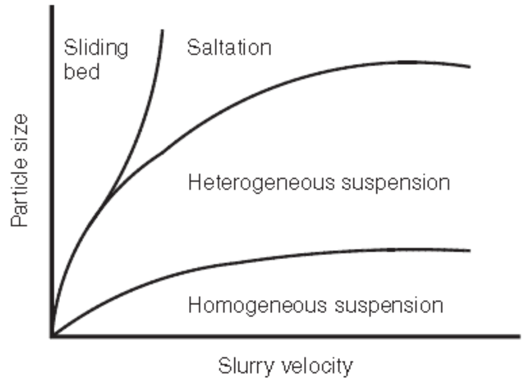


Figure 2.21 Representation of the boundaries between flow regimes for settling slurries in horizontal pipelines

In practice, a state of mixed regime flow is usually established where two or more flow regimes occur simultaneously at different zones of the pipe cross-section. The prevalent flow regime depends mainly on the solids concentration, the operating flow velocity, and the particle size distribution. The pressure gradient along the pipeline is significantly affected by these parameters for a given pipe diameter and particle size distribution of the solids. In the context of slurry fill transportation, the high pulp density required dictates that the flow system operates in the pseudo-homogeneous regime where particle settling is appreciably reduced by inter-particle contact (King, 2002)[10].

The transportation of mine waste in slurry form often requires pumps. The type of pump required will depend on the concentration, weight, and chemical properties of the slurry in question. Centrifugal and positive displacement pumps are the two types of pumps commonly used for slurry transportation (Grice, 2005)[3].

2.6.2 Paste fill

The quality control of paste backfill is of paramount importance. As shown in Figure 2.22, when concrete becomes under saturated, its flow resistance increases dramatically by an order of magnitude over a narrow water-to-cement ratio transition zone of between 0.4 to 0.5.

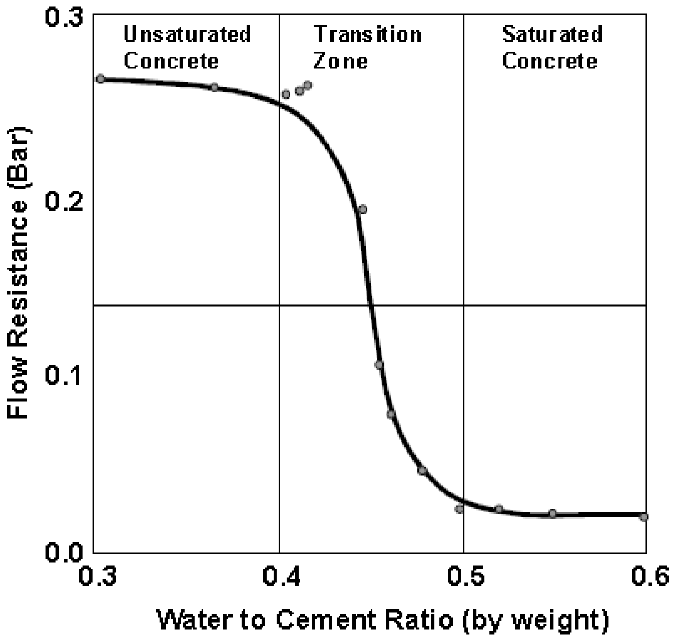


Figure 2.22 Effect of paste saturation on pipeline pressure.

A small variation in the solids concentration of paste fill has a substantial effect on backfill consistency (King, 2002)[10].

The quality of paste fill is ascertained mostly by its ability to remain standing, when exposed by adjacent mining, provide a working floor, and to provide ground support through confinement - hence by its strength and stiffness response. As much as it is important, from a tailings management perspective, to backfill as much of the mill tailings underground as possible, studies have shown that such poorly graded, higher porosity material can generate a greater strength and stiffness response when blended with alluvial sands. Little strength advances are made when using total tailings as compared with more conventional slurry fill. Henderson, et al. (2005)[8] have documented that tailings materials containing a larger fraction of coarser particles demonstrate an increased strength response from fine to medium to coarse tailings. Blended alluvial paste fills demonstrate better stiffness results than the higher porosity, poorly graded total tailings material. A practical balance between the desire for total tailings disposal, paste rheology and backfill quality must be engineered to optimize backfilling at a mine site.

Due to the need for highly efficient dewatering technology and concrete pumps, paste backfill has often been associated with high capital cost requirements. More recently, new technologies, such as Fluidization offer simpler dewatering techniques at reduced capital investments. Paste fill systems also tend to have lower operating costs and therefore, over time, may allow for higher initial capital investments. While every mine will have different needs, the basic components of a paste backfill system include tailings dewatering facilities, cement and mixing plant, borehole/pipelines, as well as advanced computer control technology (Grice, 2007)[4].

2.6.3 Rock fill

Waste rock from underground developments or surface quarry is often dumped into raises then distributed by trucks or conveyors to the stopes. When cemented rockfill is needed, the cement slurry can be introduced by a pipeline and mixed with the waste rock prior to placement or can be used to post-consolidate placed rockfill. Rockfill has significant advantages in mines, especially when development waste is utilized since it is virtually cost free. Mines with an open pit on site or nearby see rockfill as an excellent method of waste disposal. While slurry fill or paste fill can be used to fill voids and achieve tight filling within stopes, rockfill continues to provide the best strength support and for this reason it is unlikely to be totally replaced by other types of fill (www.infomine.com) [15]. The following are technical advantages of rockfill:

- Waste rock is used to backfill underground openings, reducing waste disposal on surface, especially if there is an open pit associated with the mine.
- Simple preparation system.
- Relatively high strengths can be attained when waste rock is consolidated with cement.
- Stope dewatering can be avoided.

The following are technical disadvantages of rock fill:

- Quarried rock will require crushing thus transportation; surface production and haulage costs can be significant.
- To ensure the rockfill competence, the voids in rockfill may require fine material and cementing agents to be added (e.g. through a slurry fill).

- Coning of placed rockfill when placed in the stope results in the segregation of coarser material to stope sides and reduces the ability to tight fill stopes.
- Any tailings produced are only partially utilized and surface disposal must be considered.

A comparison between the properties of slurry fill, paste fill and rockfill is presented in Table 2.5.

Table 2.5 Comparison of different backfill systems

Properties	Slurry Fill	Paste Fill	Rockfill
Placement State	60% to 75% solids (by weight)	75% to 85% solids (by weight)	Dry
Underground Transport System	Borehole/pipeline via gravity	Borehole/pipeline via gravity, can be pumped	Raise, mobile equipment, separate cement system
Binder Application	Cemented or uncemented	Cemented only	Cemented or uncemented
Water to Cement Ratio (w/c)	High w/c ratio, low binder strength	Low to high w/c ratio. Low to high binder strength	Low w/c ratio, high binder strength
Placement Rate	100 to 200 Tonne/hr	50 to 200 Tonne/hr	100 to 400 Tonne/hr
Segregation	Slurry settlement and segregation, low strength development	No segregation	Stockpile and placement segregation, reduced strength and stiffness
Stiffness	Low stiffness	Low or high stiffness	High stiffness if placed correctly
Tight Filling	Cannot tight fill	Easy to tight fill	Difficult to tight fill
Binder Quantity	Requires large quantity of binder	Usually lower quantity of binder required	Moderate binder quantities
Barricades	Expensive	Inexpensive	Not necessary
Water Runoff	Excessive water runoff	Negligible water runoff	No water runoff
Capital Costs	Low capital costs	High than for slurry fill	Moderate capital costs
Operating Costs	Low distribution costs; lowest cost for an uncemented fill	Lowest cost for a cemented fill	High operating costs

Source: (Hassani & Archibald, 1998)[5]

Chapter 3

RESEARCH METHODOLOGY

3.1 Preamble

This chapter discusses the various research methodologies that were undertaken in this Backfill Research Project in order to achieve the objectives outlined above.

3.2 Site Visits

Several site visits were undertaken to Konkola Mine in Chililabombwe, to familiarise with the operations of the mine. Different departments of the mine such as Geology and Mining were visited to appreciate the geological setup of the mine and the mining operations. The New Konkola Concentrator, with the capacity to handle 6 million tonnes of copper ore per annum, was also visited in order to understand how the copper ore is processed into concentrates and subsequent production of classified mill tailings (also called hydraulic fill). The hydraulic fill material is produced at the two backfill plants (KCM BF 1 and 2) situated at the end of the Konkola Concentrator process circuit as shown in Picture 3.1.

Picture 3.1 Backfill Plants at No.1 Shaft

The visit to the two backfill plants was also meant to get an insight of the associated pumping arrangements of the material to No.3 Shaft which is 3.56 km North West of No.1 Shaft mine. Currently, there are no backfilling operations at No.1 Shaft. The use of hydraulic fill at No.3 Shaft has been necessitated by very poor ground conditions at No.3 Shaft underground, especially in the famous 'nose' area.

3.3 Literature Review

An extensive literature survey was undertaken of the existing Konkola Backfill system including past reports of several attempts by the mine to introduce cemented backfill at the mine. Information on latest backfill systems and best world practices was studied.

3.4 Sampling and Testing

In order to develop the suitable backfill material for safe and economic production, a number of tests were conducted at the BSI Certified Nchanga Geotechnical Services laboratory on the materials available at Konkola Mine, shown in Picture 3.2 to assess their suitability for use as backfill material in underground mining operations.

Picture 3.2 Proposed base materials for use as Backfill material

The proposed base materials were:

- Classified Mill tailings from both KCM 1 and 2 Backfill plants situated at No.1 Shaft
- Waste rock material from No.3 Shaft Waste Dump which will be crushed to produce aggregates at KCM 3 Backfill plant

- Lime and waste furnace slag from the New Smelter at Nchanga Mine

For binder, the following materials shown in Picture 3.3 were used:

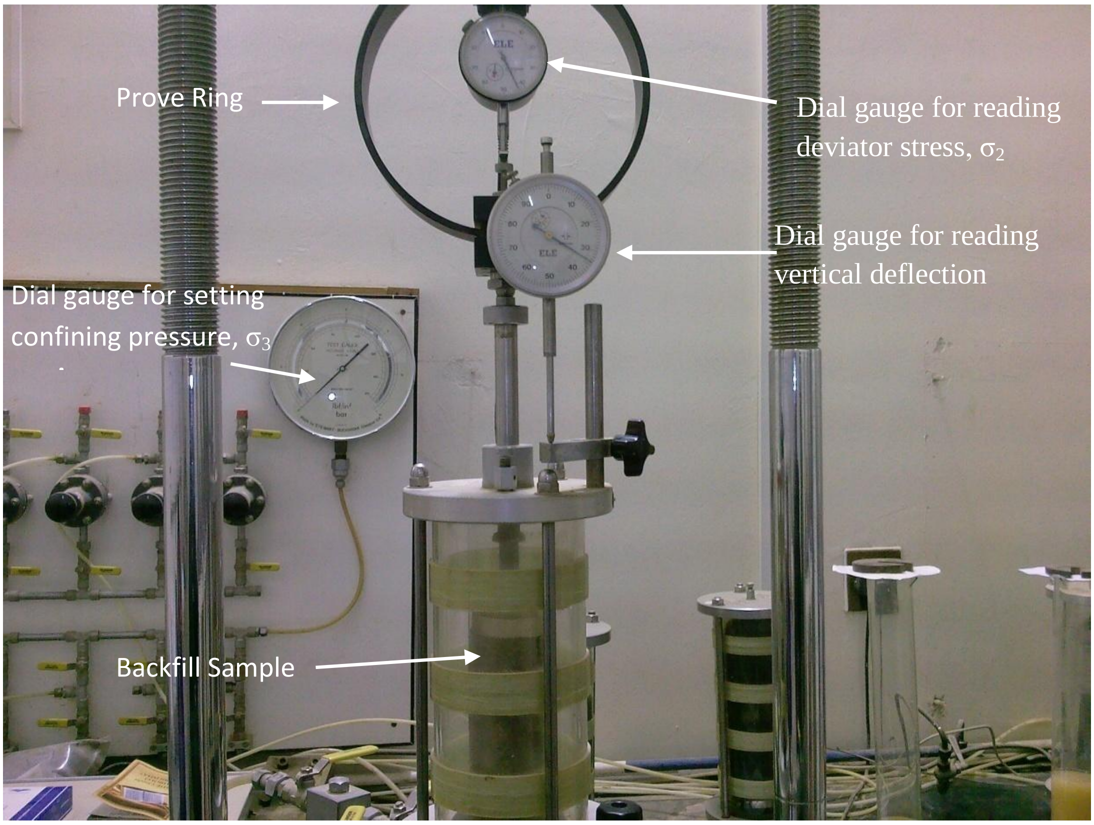
Picture 3.3 Binder materials used in the experiments

- Locally produced cement from both Lafarge and Zambezi Cement Companies, called Mphamvu and Zambezi cement, respectively.
- Special Fast Set Cement from Minova Cement Company in South Africa.

The following is a description the different Geotechnical tests that were carried out on the materials mentioned above using available equipment at the lab.

3.4.1 Triaxial Tests

Triaxial tests were conducted in order to determine the *shear strength characteristics* of the proposed fill materials, either singly or in combination. This was necessary in order to determine whether backfill would remain free standing during and after pillar extraction in a stope and not undergo shear failure. The test samples were prepared using plastic cylindrical moulds made of PVC of length to diameter (L/D) ratio of 2. The length of each sample was 75mm and the diameter was 37.5mm. The Triaxial testing machine shown in Picture 3.4 was used, set to a strain rate of 0.6096.



Picture 3.4 Machine for testing shear strength of backfill material

Confining pressures ranging from 50 kPa to 800 kPa were used on the samples. The results from the tests were used to plot the Mohr circles from where the *internal angle of friction, φ,* and the apparent *cohesion, c,* were derived.

3.4.2 Uniaxial Compressive Strength (UCS) test

Uniaxial Compressive Strength tests *(UCS)* were conducted on the proposed backfill material to ascertain whether the material will fail due to self weight in a backfilled stope or not. The UCS also determines maximum height that backfill can stand in a stope. The test samples were prepared using plastic cylindrical moulds made of PVC of length to diameter (L/D) ratio of 2.5. The length of each sample was 125mm and the diameter was 75mm. The same Triaxial testing machine in Picture 3.4, was used. However, no confining pressure was added this time. The results of these laboratory tests were used to plot Mohr circles at zero confining pressure from where the UCS was derived. The normal confining stress situation can be represented by the following relationship:

The stress strain relationships were developed from the UCS tests from where the *moduli of elasticity, E,* values were determined. The other parameters that were obtained from the results included, *void ratios*, *Bulk and Dry densities* and *unit weights*,

3.4.3 Particle Size Distribution (PSD)

The above mentioned test was conducted in order to determine the sizes of the solid particles comprising fill materials and their relative proportions. PSD in backfill materials affects shear strength, settlement and permeability. In cemented backfills, PSD affects strength development. A set of sieves with progressive sizes of apertures was used for fill material sizes up to 0.075mm. Thereafter, Hydrometer analysis was used for sizes less than 0.075mm.

3.4.4 Atterberg Limits

Atterberg Limits describe a range of moisture contents over which backfill will exhibit consistence. The above tests were conducted to understand the effect of water addition to a particular backfill medium at different water contents. The limits are as follows:

- *Liquid Limit,* denoted by LL, is the upper limit of the range of water content over which backfill material exhibits plastic behaviour. This test determined using the Casagrande apparatus is a dynamic method and gives indirect information on the fluidity of the backfill material.
- *Plastic Limit* denoted by PL, is the lowest limit of the range of water content over which backfill material exhibits plastic behaviour. The moisture content at which a 3mm thread of backfill material will be rolled without breaking is the Plastic Limit.
- *Shrinkage Limit*, denoted by SL, is the water content at which the volume of backfill material reaches its lowest volume as it dries out. The SL is measured in apparatus called Shrinkage Moulds.

3.4.5 Slump

Slump tests in Concrete Technology are used to measure the workability, consistence and quality of a concrete mixture. In Cemented Backfill Technology, the tests are important in order to determine the pumpability and transportability or fluidity of cemented backfill material. Slump tests were conducted on the cemented backfill material using a 300mm Slump cone. Figure 3.1 demonstrates slump measurement.

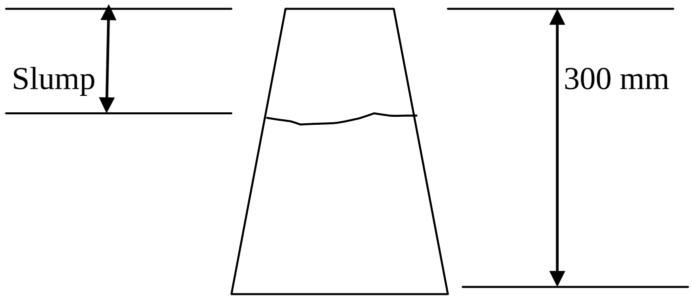
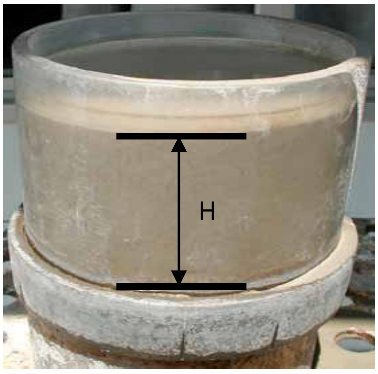


Figure 3.1 Slump measurements using a Slump cone

3.4.6 Percolation tests

Percolation tests were conducted on the Konkola Hydraulic fill material to verify whether the material was able to meet the mining specified percolation limit of 80mm/hr. The tests were conducted at Nchanga Geotechnical Services laboratory using the Backfill Quality Tester (BQT) calibrated specifically for Konkola HF, shown in Picture 3.5.



Picture 3.5 Percolation test in Progress

A typical percolation test begins by pouring the hydraulic fill in the tube and immediately noting the time, *t*, when water begins to drain from the tube up to the time it stops. The percolation rate in the tube is governed by a differential equation given below.

$$P = 3600(H / t) \ln (100/H) \qquad (30)$$

Where

P = percolation rate in mm/hr

H = Final height of backfill material in mm measured in the apparatus

t = time in seconds taken for the water to completely percolate through the backfill material

3.4.7 Permeability tests

Permeability tests were conducted on cemented backfill material to determine how quickly water used for transporting the material would drain from a stope. A Constant Head Permeameter apparatus, available at the lab was used for this exercise.

The tests mentioned above were undertaken in accordance with BS 1377 tests standards.

3.5 Mineralogical Examinations of the Konkola HF Material.

Samples of the available fill material at Konkola Mine were taken to the KCM Mineralogical Laboratory to examine their mineralogical composition. This was in order to assess their suitability for use as backfill material in underground mining operations. As mentioned earlier, mineralogical composition of fill material affects water retention, strength, settling and abrasive characteristics.

Chapter 4

CURRENT BACKFILL OPERATIONS AT KONKOLA MINE

4.1 Preamble

This chapter describes the current uncemented Backfill System at Konkola Mine. It also presents results of site visits and research activities undertaken at Konkola Mine including the Backfill plants and excavated areas at No.3 Shaft underground where the current un-cemented classified tailings are pumped.

4.2 Current Hydraulic Backfill System at Konkola Mine

Several site visits were undertaken to the two Konkola Backfill Plants at No.1 Shaft to familiarize with the operations. This was also to get an insight of the actual production of the Hydraulic Fill (HF) material and the associated pumping arrangements of the material to No.3 Shaft mine which is 3.56 km North West of No.1 Shaft mine. Currently, there are no backfilling operations at No.1 Shaft. The use of hydraulic fill at No.3 Shaft underground has been necessitated by very poor ground conditions, especially in an area called the 'nose' shown in Figure 4.1.

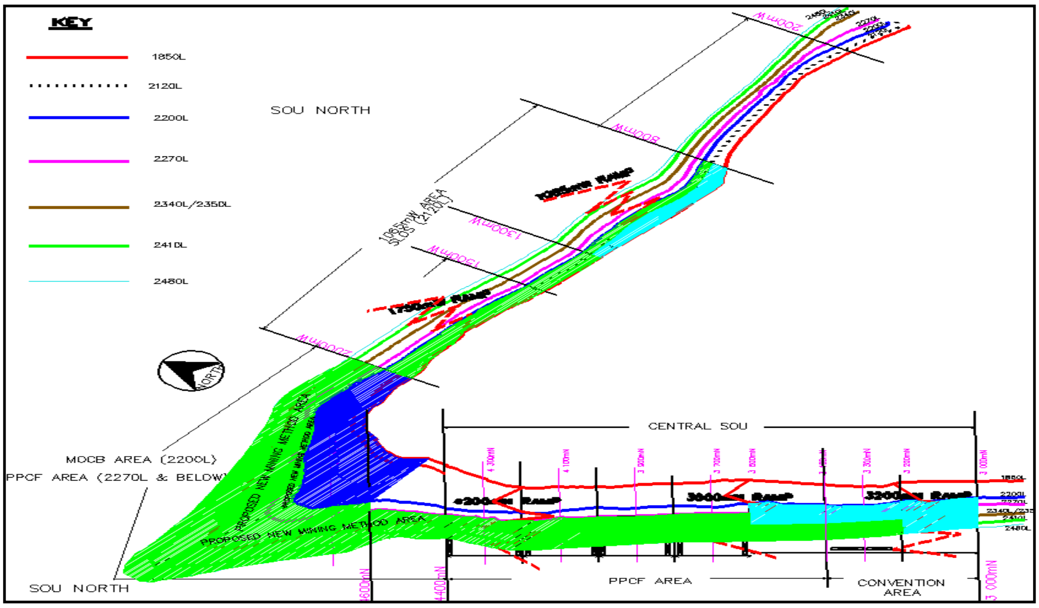


Figure 4.1 Konkola Mine, No.3 Shaft Underground plan view: 'Nose' area (KCM Expansion Projects) [11]

The old Hydraulic Backfill Plant, called KCM BF 1, shown in Picture 4.1 below, is designed to produce 60,000 tpm or 43,000 m^3 per month of classified uncemented tailings with a target permeability of 80mm/hr.

Picture 4.1 Old Hydraulic Fill Plant at No.1 Shaft (KCM BF1)

This material is produced at a Relative Density of 1.70 at 65% solids fraction (cw). The plant is located 20m from the new Konkola Concentrator. It has been designed to operate throughout the 360 days per year at 90% availability. Full tailings are supplied from the concentrator and are pumped either directly to the Lubengele dam for disposal or to the HF Plant for processing. At the designed milling rate of 2.4 Mtpa of ore, the full tailings (slurry) contains around 242 tph of solids delivered at a volumetric slurry rate of 753m^3/hr. The SG of the tailings is 2.60 and the slurry density is around 27% cw or RD of 1.20. The tailings slurry is delivered to a 30m^3 feed tank and then pumped to a 24 Multotech 165mm cyclone cluster. The stream of full tailings is classified by the cluster of cyclones into *under flow* and *over flow* material. The plant was originally designed to achieve a minimum of 40% recovery to underflow at 97 tph or 110m^3/hr at a Relative Density of 1.68. In practice recoveries have been as low as 32% or 77 tph due to the strict percolation requirements of the HF specified by underground production personnel.

In the production of copper concentrates at Konkola Concentrator, higher recoveries are achieved with very highly pulverized ore. However, this seriously affects the quantity of hydraulic backfill production due to strict percolation requirements of 80mm/hr at No.3 Shaft underground. The higher, the fines content, the lower the percolation.

The classified tailings are stored in one of two 550m^3 air agitated tanks or can be diverted to the residue disposal tank. Figure 4.2 shows the block diagram for the Hydraulic Fill Plant at No.1 Shaft.

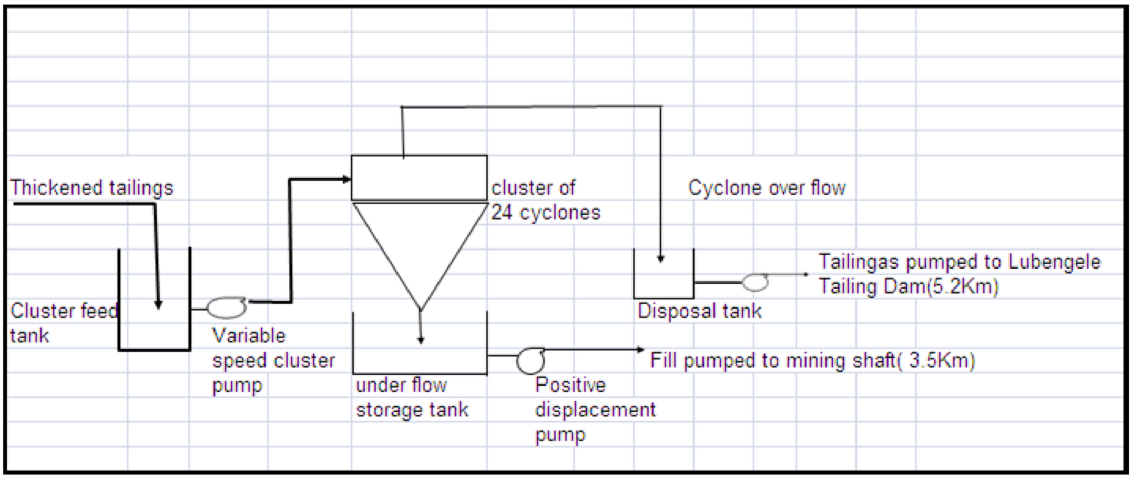
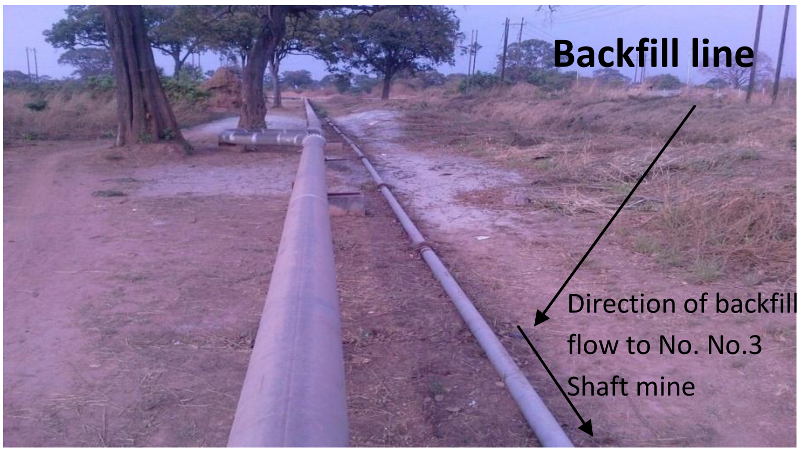


Figure 4.2 Block Diagram for Hydraulic Fill plant at Konkola Mine

The Hydraulic Fill is then pumped via a Warman centrifugal pump to the three-chamber Abel positive displacement pump rated at 90m^3/hr at a maximum feed pressure of 4.5 MPa. The Abel pump is a hydraulic membrane unit which delivers Hydraulic Fill slurry to the holding tanks at No.3 Shaft at a rate of 90 m^3/hr through a 125mm (Schedule 40) pipeline, shown in Picture 4.2.



Picture 4.2 Pipe delivering Hydraulic Fill material to No.3 Shaft underground Mine

The pipeline is 3.540km in length and has a 6 mm polyurethane lining. It is rated at 20.2 MPa, well in excess of the 4.5 MPa pump rating.

4.3 Backfill Underground

Hydraulic fill material is dispatched at $120m^3$/hr from the holding tank at No.3 Shaft surface and delivered by gravity to the underground workings. The backfill holding tanks are located next to the Shaft bank as shown in Picture 4.3.

Picture 4.3 Holding tanks at No.3 Shaft

The current reticulation route is illustrated schematically in Figure 4.3 and has been designed to be operated in full pipe *turbulent flow* mode.

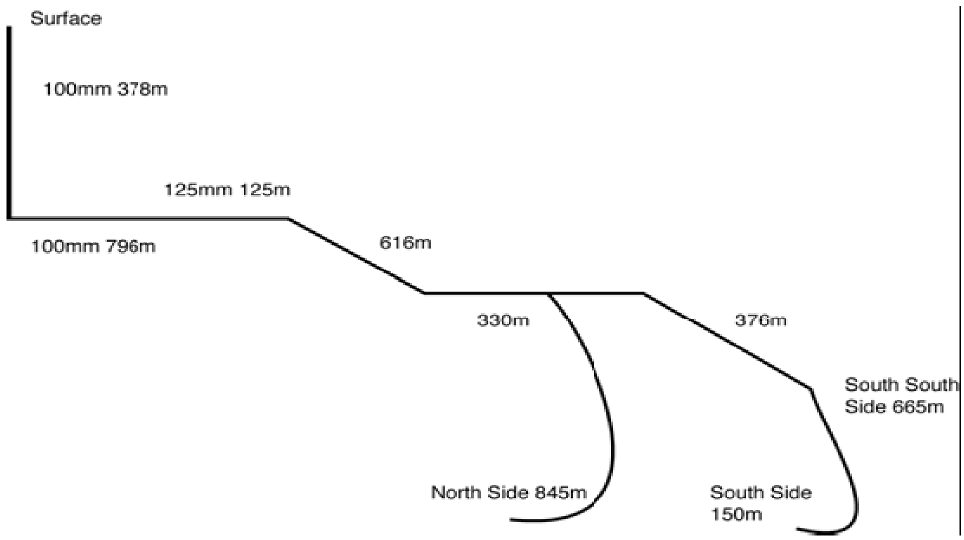


Figure 4.3 Current Hydraulic Fill Reticulation Route at No.3 Shaft underground (KCM Expansion Projects) [11]

Filling operations only take place when personnel are on shift underground to observe placement for the three eight hour shifts each day.

Picture 4.4 shows hydraulic fill placed into a modified overcut bench stope (MOCB). This fill run had been completed approximately 48 hours previously and the top surface of the fill was only moist and visibly unsaturated.

Picture 4.4 Hydraulic Fill in an MOCB Stope

This indicated good drainage and hence good permeability of the hydraulic fill.

Picture 4.5 shows a small sample of the hydraulic fill scooped off the surface of the pour.

Picture 4.5 Sample of fresh Hydraulic Fill – 2 days old and well drained

The grains are free of any sticky clay and appear as a good quality hydraulic fill. The function of the hydraulic fill is to provide local and regional ground support to the mining operations, both in the room and pillar and the MOCB mining areas.

Figure 4.4 shows the current backfill system at Konkola Mine all the way up to the stopes at No.3 Shaft underground.

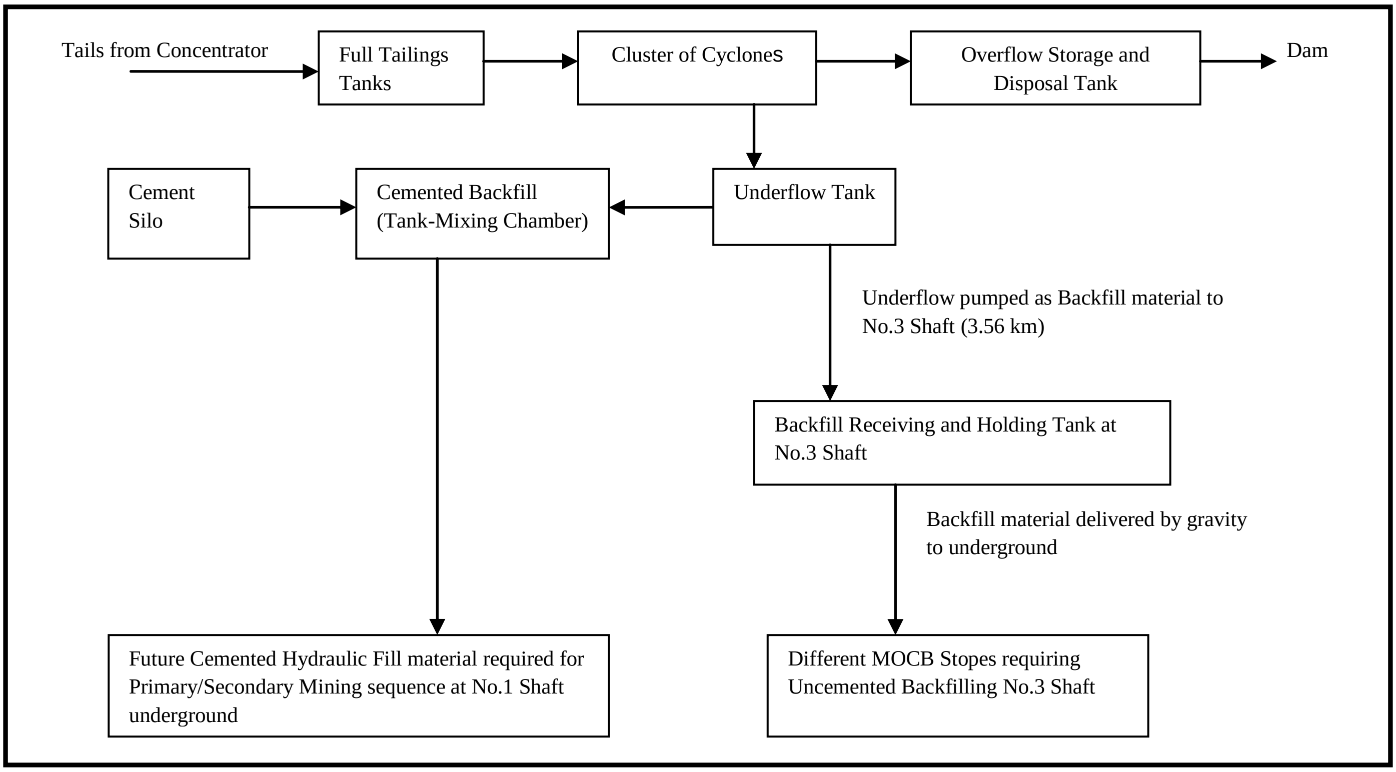


Figure 4.4 Konkola Hydraulic Backfill System Flow Diagram

4.4 Flushing of the Reticulation pipe to prevent Blocking

Flushing or cleaning of the reticulation pipes is undertaken by the No.1 Shaft feeding Backfill Crew and the No.3 Shaft receiving Backfill crew before and after each fill pumping operation. This exercise is very critical as it ensures that there is no jamming or clogging of pipes with the hydraulic fill. The excess water from the pipes is pumped through the drain drives from No. 3 to No.1 Shaft underground.

4.5 Fill Bulk Heads

Konkola Mine constructs large poured concrete bulkheads to retain the hydraulic fill in the stopes. The walls are designed to withstand hydrostatic loading conditions and are fitted with numerous drainage pipes to enable drainage. One of these walls is shown in Picture 4.6.

Picture 4.6 Fill Bulk Head at No.3 Shaft underground

This also shows the magnitude of the construction effort undertaken to build these walls. The walls are planar in shape, 2m thick at the base and 1m wide at the top.

Figure 4.5 shows a plan view of the Konkola Fill Bulk Head

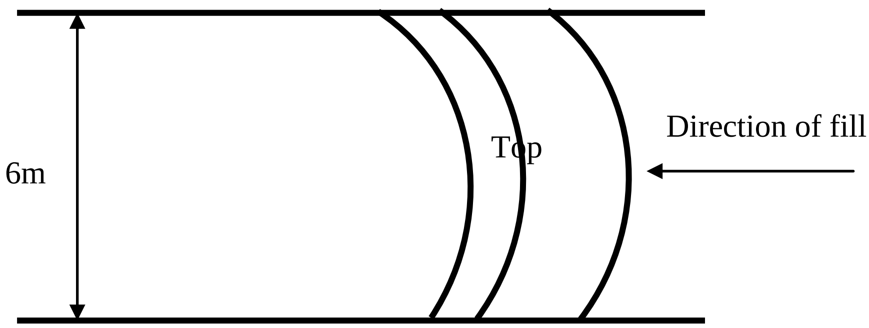


Figure 4.5 Plan view of the Konkola Bulk Head

Chapter 5

LABORATORY TESTS AND ANALYSIS OF RESULTS

5.1 Preamble

This chapter describes the different laboratory tests that were undertaken at Nchanga Geotechnical Services laboratory on the materials available at Konkola Mine to assess their suitability for use as backfill material in underground mining operations. The tests were done in line with the desirable properties of backfill material and their relevance to backfill design outlined in Table 2.1 (Chapter 2, page 13)

5.2 Laboratory tests on Uncemented Hydraulic Fill (HF) material in use currently at Konkola Mine

In cognizance of the main research objective which is to develop a suitable Backfill Material for safe and economic ore production at Konkola Mine, the tests were conducted. The first available material to be analyzed was the uncemented classified tailings used currently as hydraulic fill material at No.3 Shaft underground mine.

5.2.1 Uniaxial Compressive Strength (UCS) of Konkola Hydraulic Fill Material

UCS tests were done on the samples collected from the holding tanks at No.3 Shaft mine Konkola Hydraulic fill material to ascertain the insitu unconfined compressive strength characteristics. Six samples were collected using plastic cylindrical moulds made of PVC of length to diameter ratio of 2.5. The length of each sample was 125mm and the diameter was 50mm. The Triaxial testing machine given in Picture 3.4 in Chapter 3, was used for this exercise without the confining pressure, σ_3. During the test, it was difficult to stand the samples upright due to the observed weak strength.

The results of these laboratory tests were used to plot Mohr circles at zero confining pressure shown in Figure 5.1.

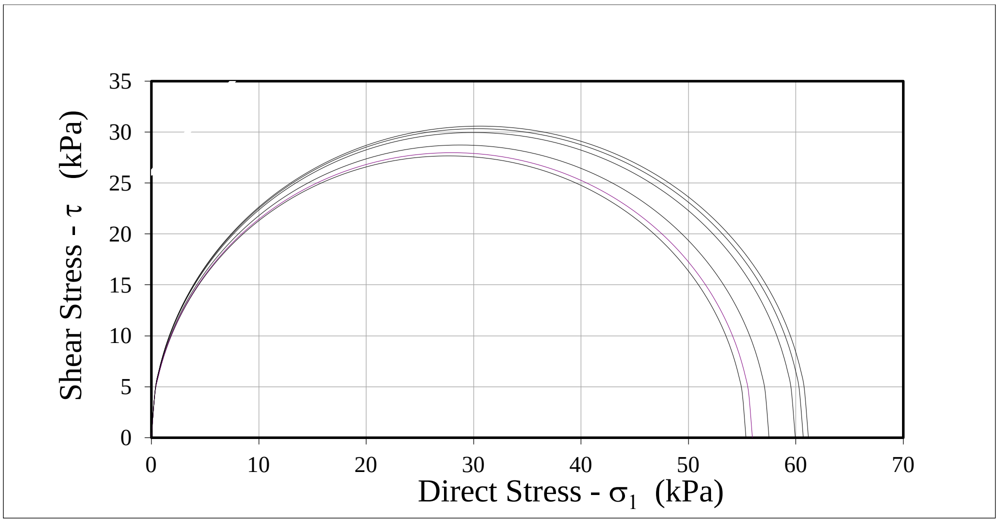


Figure 5.1 UCS tests results for Konkola Hydraulic fill material

The results from the above UCS tests were *statistically* analyzed to assess the *reliability* of the data, the summary of which is given below in Table 5.1

Table 5.1 Statistical Analysis of UCS test results for Konkola Hydraulic Fill Material

Konkola Hydraulic Fill	Sample No						Statistical Analysis		
	1	2	3	4	5	6	Mean	Standard Deviation	Coefficient of Variation
	60.7 kPa	55.3 kPa	59.9 kPa	57.5 kPa	56.0 kPa	61 kPa	58.4 kPa	2.5 kPa	4.3%

The average Uniaxial Compressive Strength obtained for the Konkola Hydraulic Fill material is 58.4 kPa.

The stress strain graphs from the above UCS tests were plotted and used to derive the Modulus of Elasticity, E. A typical example of a stress-strain graph is shown in Figure 5.2.

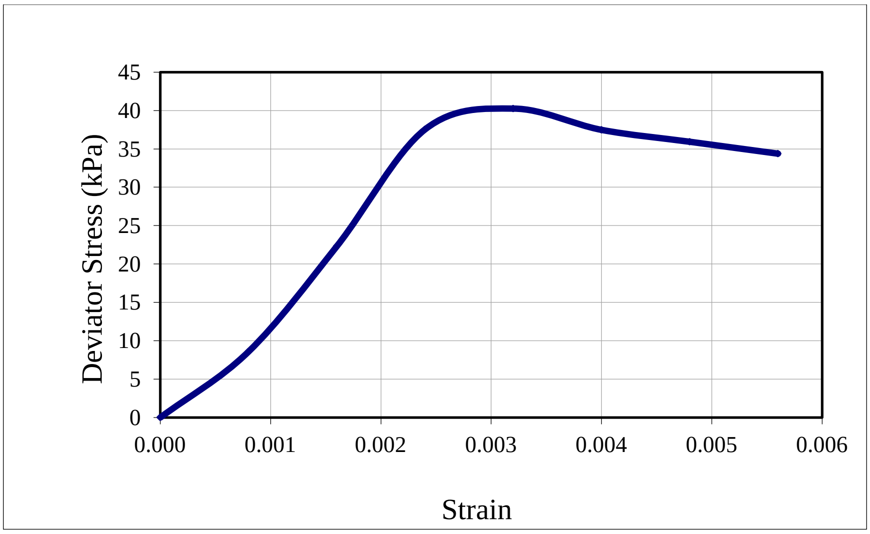


Figure 5.2 Stress – Strain relationship for the Konkola Hydraulic Fill material

5.2.2 Triaxial Tests on Konkola Hydraulic Fill Material

Triaxial tests were conducted on the Hydraulic Fill material sampled from the holding tanks at No.3 Shaft mine to determine shear strength characteristics. The samples were collected using plastic cylindrical moulds of size 37.5mm diameter and 75mm height. During testing, the Triaxial testing machine was set to a strain rate of 0.6096 mm/min and the confining pressures of 50 kPa, 100 kPa, 200 kPa, 400 kPa and 600 kPa.

The results of the triaxial tests were used to plot the Mohr circle envelopes as shown in Figure 5.3 from where the internal angle of friction, φ, and the apparent cohesion, c, were derived.

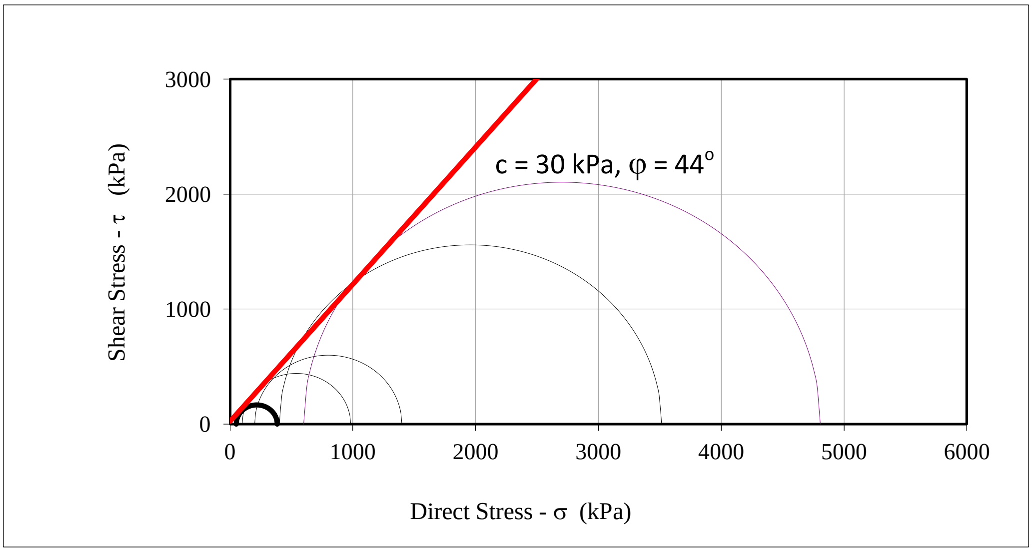


Figure 5.3 Mohr - Coulomb Failure Envelope for Konkola Hydraulic Fill material

Results of the test are presented together with other test results on HF in Table 5.3.

5.2.3 Percolation tests of Hydraulic Fill Material

Percolation tests, which are part of the daily routine at the Backfill plant at No.1 Shaft, were sampled from the holding tanks at No.3 Shaft. Thereafter, the tests were conducted using the BQT apparatus. The results are shown below.

- Time, t = 642 seconds
- H = 83mm

Hence Percolation, P = *3600(H / t) ln (100/H)*

= *86.72mm/hr*

The results indicate that the material had suitable percolation and would facilitate drainage of transport water from the backfilled stope at No.3 Shaft.

5.2.4 Particle Size Distribution Tests on Konkola Hydraulic Fill Material

It was observed during the several field visits to both No.1 and No.3 Shafts that the percolation rates at the backfill plant where the hydraulic fill material is produced at No.1 Shaft were higher than those at the receiving tanks at No.3 Shaft. This was attributed to the attrition or abrasion that the material undergoes as it is pumped from No.1 Shaft to No.3 Shaft, a distance of 3.56 km. This process increases the fines content of the material. The higher the fine content the lower the percolation and vice versa. The fill flow is characteristically *turbulent* as constant agitation is required to prevent the material from settling in the reticulation pipes and tanks.

To verify this behaviour, particle size distribution (PSD) was conducted on the fill material from both No.1 and No.3 Shafts. Appendix 01 and 02 show the PSD test results. The two graphs indicate an increase in the proportion of fines content as material moves in the pipes to No.3 Shaft holding tanks. The percolation at No.1 Shaft Backfill plant and the holding tanks was found to be 90.2 mm/hr and 86.72mm/hr, respectively.

5.2.5 Atterberg Limits tests on Konkola Hydraulic Fill Material

Atterberg Limits tests were undertaken on the Konkola Hydraulic Fill Material to determine the range of moisture contents over which the material would exhibit consistence. The tests were undertaken in accordance with BS 1377. The results of the tests presented in Table 5.2 show that the Hydraulic Fill material has no plasticity.

Table 5.2 Results of Atterberg limits Tests on Konkola Hydraulic Fill Material

Liquid Limit, LL	0
Plastic Limit, PL.	0
Shrinkage Limit, SL	0

The significance of these results is that the material would easily be reticulated in the pipes as there would be no yield stresses to overcome. The summary of results for all the geotechnical tests conducted on the hydraulic fill material is shown in Table 5.3.

Table 5.3 Summary of Test Results of Hydraulic Fill Properties at Konkola Mine

<table>
<tr><th colspan="2">Property</th><th>Symbol</th><th>Brief Description</th><th colspan="2">Average Values from Test Results</th></tr>
<tr><td colspan="2" rowspan="2">Unit weight</td><td rowspan="2">γ</td><td rowspan="2">Weight of backfill material per unit of volume derived from Triaxial Tests</td><td>Unit Weight</td><td>20 kN/m^3</td></tr>
<tr><td>Dry Unit weight</td><td>15.5 kN/m^3</td></tr>
<tr><td colspan="2">Percolation</td><td>p</td><td>Rate at which water drains from the Konkola Backfill material underground, tested using the Percolation Tube</td><td colspan="2">87mm/hr</td></tr>
<tr><td rowspan="2">Shear Strength characte ristics</td><td>Internal angle of friction,</td><td>φ</td><td>Shear resistance mobilized by the Konkola Backfill material purely by inter-particle frictional forces, derived from Triaxial Tests</td><td colspan="2">44^o</td></tr>
<tr><td>Cohesion</td><td>c</td><td>Shear resistance mobilized purely by strong bonding forces between particles of backfill material, derived from Triaxial Tests</td><td colspan="2">30 kPa</td></tr>
<tr><td colspan="2">Uniaxial Compressive strength</td><td>UCS</td><td>Maximum axial stress that Hydraulic Konkola Backfill material would withstand before failure, tested using the Triaxial</td><td colspan="2">58.4 kPa</td></tr>
</table>

		Machine at Zero Confining pressure	
Modulus of Elasticity,	*E*	Relationship between stress and Strain of Konkola Hydraulic Backfill material derived from UCS test results	13,800 kPa
Void ratio	*e*	The ratio of volume of voids to volume of solids in settled Backfill material, derived from Triaxial Tests	0.7
Specific Gravity	S.G	Useful in determining quantity of Konkola Hydraulic Backfill material required underground, tested using the Vacuum Pump S.G Machine	2.6
Atterberg limits:	LL, Pl,SL	Important in order to understand the effect of water addition to a particular backfill medium at different water contents	No plasticity in the HF material

5.2.6 Mineralogical Examinations of the Konkola Hydraulic Fill Material.

Samples of the Konkola Hydraulic Fill material were taken to the KCM Mineralogical Laboratory to examine their mineralogical composition. The results of the tests are shown in Table 5.4 below.

Table 5.4 Konkola Backfill Mineralogical Examination Results

Mineral	Percentage Content in the Hydraulic Fill Material (%)
Argillite	74.0
Quartz	18.4
Mica	3.3
Carbonates	2.7
Iron Oxide	1.0
Accessories	0.6

5.3 Laboratory tests on Cemented Hydraulic Fill (CHF) material

The current Konkola Hydraulic Fill material which is produced at the old backfill plant and analyzed in the preceding section would not provide the required strength characteristics in the proposed future mining operations that will follow after completion of the KDMP project. This is due to the fact that it has very low Uniaxial Compressive Strength and cohesion and its strength only relies on the mobilized internal angle of friction, φ. The future mining operations will require a properly engineered backfill material that will meet the required strength of at least 1 MPa with suitable drainage characteristics to facilitate the proposed *Primary/Secondary* extraction system mentioned earlier.

Therefore to develop the material that meets the required specifications, the hydraulic fill material was sampled from the holding tanks at No.3 Shaft and mixed with different proportions of the Cement brands from Lafarge and Zambezi Portland Cement Companies to prepare samples for tests. The tests were done in line with the desirable properties of backfill material outlined in Table 2.1 in Chapter 2, page 13.

Cemented Hydraulic Fill samples were prepared for UCS testing using cylindrical sampling moulds of size 50mm diameter and 125mm height made of PVC. Another set of samples was prepared for Triaxial Testing using cylindrical sampling moulds of size 37.5 mm diameter and 75mm height made of the same material. The fill material was prepared by varying cement quantities and allowed to cure at different time periods. Table 5.5 shows cement mixing schedules that were followed for both Mphamvu and Zambezi cement brands.

Table 5.5 Konkola Hydraulic Fill Material - Cement Addition Schedule
(Mphamvu and Zambezi Brands prepared separately)

Container No	**Mass of Hydraulic Fill material (g)**	**Cement Added**	**Mass of Cement (g)**	**Number of Cylindrical Samples tested at the indicated curing periods**			
				7 days	**14 days**	**21 days**	**28 days**
1	19,095	2%	390	6	6	6	6
2	16,574	4%	691	6	6	6	6
3	18,959	6%	1,210	6	6	6	6
4	19,389	8%	1,686	6	6	6	6
5	16,389	10%	1,821	6	6	6	6

The laboratory tests were done with efforts made to simulate the prevailing underground humid conditions at No.3 Shaft. Water was not allowed to drain out of the samples after cement mixing to prevent loss of cement fines which would otherwise jeopardized the hydration process. The cemented Hydraulic fill material was cured in the humid room at Geotechnical Services Laboratory shown in Picture 5.1.

Picture 5.1 CHF samples prepared for UCS testing at Geotechnical Services Laboratory

After 3 to 5 days the supernatant water was drained off after ascertaining that cement setting had taken place

5.3.1 Uniaxial Compressive Strength Tests on Cemented Konkola Hydraulic Fill Material

The Cemented Konkola Hydraulic Fill material samples were taken out of the humid room and tested for UCS at the curing periods indicated in Table 5.5. The Triaxial testing machine was used with no confining pressure. The samples could not be tested in the Concrete Cube Testing Machine at the lab because some of the cemented hydraulic fill samples, especially those with low cement content, were too weak for the peak failure loads to be picked and be registered by the machine. During testing the machine was fitted with the Prove Ring sensitive to soft rocks or soils. Results of the Unconfined Compressive Strength tests were used to plot the Mohr circle envelopes at zero confining pressures from where the UCS values where read. Figure 5.4 is a typical example of the Mohr circle envelope plotted at zero confining pressure for Hydraulic Fill material with 4% Mphamvu cement added and cured for 28 days.

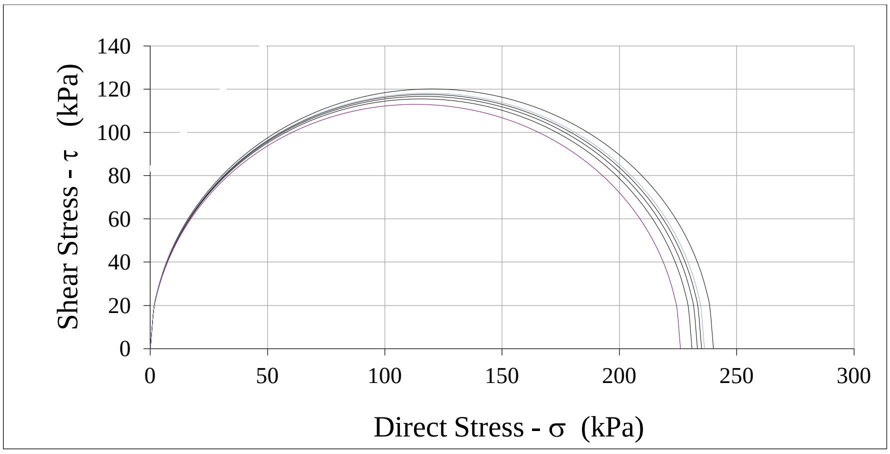


Figure 5.4 UCS test results for Cemented Konkola Hydraulic Fill (4% Mphamvu Cement after 28 days)

Overall results of the UCS test for Mphamvu and Zambezi Cements brands (with associated statistical analysis to assess reliability of data) are given in Tables 5.6 and 5.7.

Table 5.6 Statistical Analysis of UCS test results for Konkola Cemented Hydraulic Fill Material – Mphamvu Cement

Cement Content	**Curing Period (Days)**	**UCS (kPa) for CHF samples**						**Statistical Analysis**		
		1	**2**	**3**	**4**	**5**	**6**	**Mean (kPa)**	**Standard Deviation (kPa)**	**Coefficient of Variation (%)**
2%	**7**	50	56	54	51.6	49	48.9	51.6	2.9	5.6
	14	60	65	69	60	66	70	65	4.3	6.6
	21	74	72	70	69	69	60	69	4.8	7.0
	28	73	75	70	70	79	83	75	5.2	6.9
4%	**7**	91	80	79	80	90	84	84	5.3	6.3
	14	120	130	135	139	125	131	130	6.8	5.2
	21	160	165	176	177	172	170	170	6.5	3.8
	28	233	240	235	231	226	236	234	4.9	2.1
6%	**7**	95	95	95	98	90	97	95	2.8	2.9
	14	154	170	166	161	169	140	160	11.4	7.1
	21	247	257	255	250	250	239	250	6.4	2.6
	28	345	360	366	340	350	339	350	11.0	3.1
8%	**7**	119	112	126	127	118	118	120	5.6	4.7
	14	245	255	240	249	257	254	250	6.6	2.6
	21	339	329	327	340	335	310	330	11.1	3.4
	28	470	487	490	469	476	488	480	9.5	2.0
10%	**7**	222	230	233	220	215	230	225	7.0	3.1
	14	295	295	295	310	315	314	304	10.0	3.3
	21	485	499	478	505	498	475	490	12.4	2.5
	28	645	655	660	648	651	641	650	6.9	1.1

Table 5.7 Statistical Analysis of UCS test results for Konkola Cemented Hydraulic Fill Material – Zambezi Cement

Cement Content	Curing Period (Days)	UCS (kPa) for CHF samples						Statistical Analysis		
		1	2	3	4	5	6	Mean (kPa)	Standard Deviation (kPa)	Coefficient of Variation (%)
2%	7	60	69	70	63	71	63	66	4.56	6.91
	14	77	89	83	90	90	75	84	6.75	8.04
	21	90	87	80	97	91	89	89	5.55	6.24
	28	104	106	93	97	92	90	96	6.35	6.64
4%	7	119	103	100	116	117	105	110	8.25	7.50
	14	177	180	170	170	165	158	170	7.97	4.69
	21	222	219	231	215	227	230	224	6.39	2.85
	28	281	290	299	291	294	285	290	6.39	2.20
6%	7	119	120	127	128	126	124	124	3.74	3.02
	14	214	213	219	206	204	204	210	6.23	2.97
	21	260	271	265	268	277	285	271	8.92	3.29
	28	465	470	471	460	466	446	463	9.21	1.99
8%	7	149	148	157	160	166	162	157	7.21	4.59
	14	320	331	337	331	340	321	330	8.15	2.47
	21	426	436	436	444	430	444	436	7.27	1.67
	28	639	641	644	630	625	631	635	7.40	1.17
10%	7	309	300	290	293	300	290	297	7.43	2.50
	14	399	389	406	401	397	402	399	5.76	1.44
	21	655	635	649	650	640	659	648	9.03	1.39
	28	870	871	855	850	861	859	861	9.18	1.07

Laboratory test data is given in Appendices 03 and 04, for Mphamvu and Zambezi cement, respectively. Figures 5.5 and 5.6 below are graphical representations of the average UCS test results for the Mphamvu and Zambezi cement brands, respectively, plotted at different curing periods.

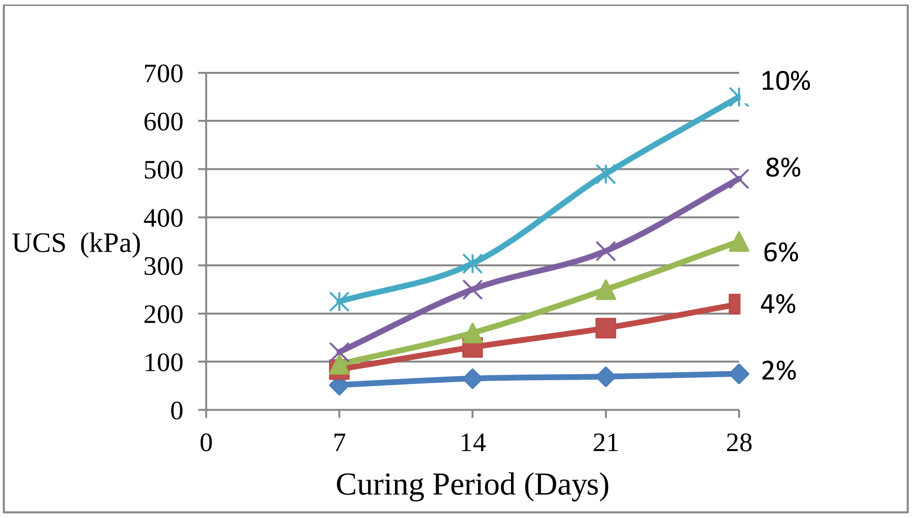


Figure 5.5 Graph of Konkola Cemented Hydraulic Fill UCS Test Results using Mphamvu Cement

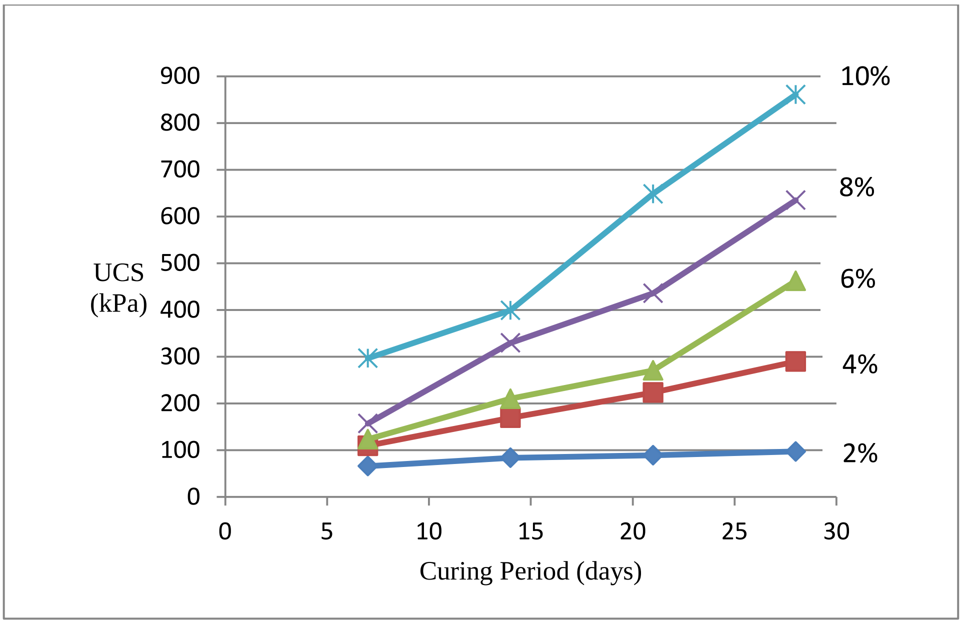


Figure 5.6 Graph of Konkola Cemented Hydraulic Fill UCS Test Results using Zambezi Cement

The other test results that were derived from the UCS test works of the Cemented Hydraulic Fill material above include: Bulk and dry densities, void ratios and Young's Moduli (from stress/strain relationships).

5.3.2 Triaxial Tests on Cemented Konkola Hydraulic Fill Material

The Cemented Konkola Hydraulic Fill Material samples were taken out of the humid room and subjected to triaxial tests in order to determine the shear strength characteristics of the material. The cylindrical samples of length to diameter ratio of 2 were tested using the Triaxial testing machine set to a strain rate of 0.6096mm/min and the confining pressures of 200 kPa, 400 kPa and 600 kPa. The actual length and diameter of each sample was 75mm and 37.5mm diameter, respectively.

Mohr circle envelopes were plotted from the results of the triaxial tests for both Mphamvu and Zambezi brands from where the internal angles of friction, φ, and the apparent cohesion, c, were derived. Figure 5.7 is an example of the Mohr circle envelope plotted for Hydraulic Fill material with 2% Zambezi cement added and cured for 14 days.

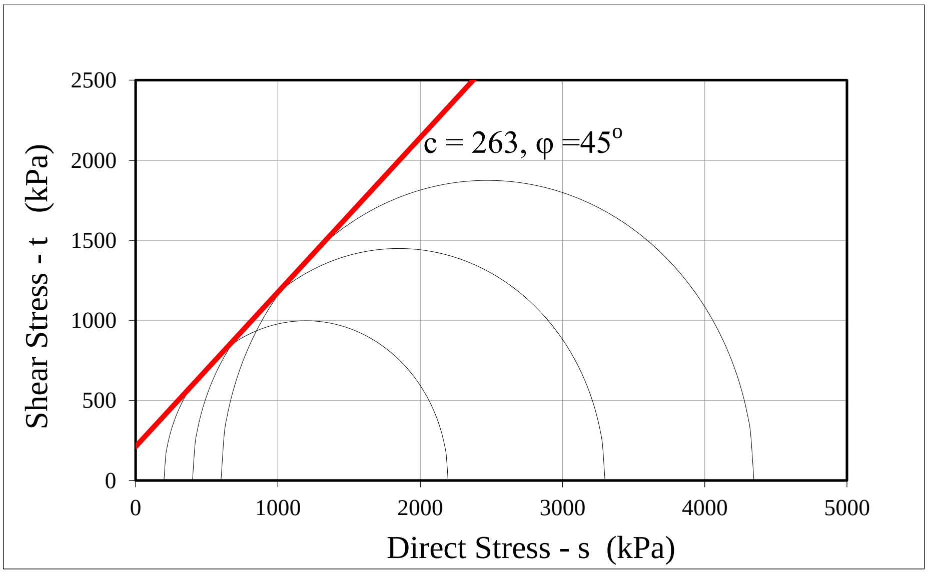


Figure 5.7 Mohr circles plotted for Hydraulic Fill material with 2% Zambezi cement added and cured for 14 days.

The summary of results for the shear parameters together with results for other tests on CHF are presented in Tables 5.8 and 5.9 for Mphamvu and Zambezi cements, respectively.

5.3.3 Permeability Tests on Cemented Konkola Hydraulic Fill Material

Permeability tests were conducted on the Konkola Cemented Hydraulic Fill material prepared from both Mphamvu and Zambezi cement brands, to determine how quickly water used for transporting the material to underground (and also supernatant water after cement hydration) would drain from a backfilled stope. Previous laboratory tests conducted at Konkola Mine have indicated that addition of cement to the classified tailings reduces the percolation rate. One of the objectives of this research project was to determine the optimal amount of cement to be added to the backfill material to not only provide the required UCS of 1 MPa but also to establish suitable drainage conditions in the mine, as stipulated by mine management. A Constant Head Permeameter apparatus, shown in Picture 5.2, available at the lab was used for this exercise.

Picture 5.2 Constant Head Permeameter

The tests were done immediately after cement addition before cement setting could commence. The equation below was used in computing the permeabilities for the materials at varying cement quantities.

$$k = QL/HAt \qquad (31)$$

Where, k = coefficient of permeability in m/s,

Q = total quantity of water collected in a measuring Cylinder after time t, (m^3)

L = height of backfill sample, (m)

H = driving water head, (m)

A = top surface area of backfill sample through which flow occurs, (m^2)

t = time taken for total quantity of water, Q, to collect in a measuring cylinder, (s)

Below is an example of Permeability calculations on Cemented Hydraulic Fill Material with 10 % Zambezi cement added

Q	$8 \times 10^{-5} m^3$
L	0.268m
H	1.660m

A	Diameter of Sample = 0.075m
t	2257.56 s

$$k = (8 \times 10^{-5} \times 0.268)/ (4.4179 \times 10^{-3} \times 2257.56)$$

$$= \underline{\mathbf{1.3 \times 10^{-6}\ m/s}}$$

The rest of the permeability values for the varying quantities of both Mphamvu and Zambezi cement brands are shown in Tables 5.8 and 5.9 together with other results of tests done on CHF.

5.3.4 Slump Tests on Cemented Konkola Hydraulic Fill Material

Slump tests in Concrete Technology are used to measure the workability, consistence and quality of a concrete mixture. In Cemented Backfill Technology, the tests are important in order to determine the pumpability and transportability or fluid characteristics of cemented backfill material. The slump tests on Cemented Hydraulic Fill samples prepared from varying amounts of both Mphamvu and Zambezi cement brands were done immediately after cement addition using a 300mm cylindrical cone. Results of the slump tests for both Mphamvu and Zambezi cement brands are presented in Tables 5.8 and 5.9, respectively, together with results from other tests on the CHF.

5.3.5 Atterberg limits Tests on Cemented Konkola Hydraulic Fill Material

Atterberg Limits tests were undertaken on the Konkola Cemented Hydraulic Fill material in order to determine the range of moisture contents over which the material would exhibit consistence. The tests performed using a Casagrande apparatus were important in order to understand the effect of water addition to the consistence of the cement mixtures. It was observed in all the tests that the cemented material for both Mphamvu and Zambezi cement brands had no plasticity.

Table 5.8 Konkola Cemented Hydraulic Fill Material Geotechnical Test Results for Backfill Desirable Properties-Mphamvu Cement

Cement Content	**Curing Period (Days)**	**Uniaxial Compressive strength, UCS (kPa)**	**Density (kg/m^3)**		**Shear Strength Parameters**		**Young's Modulus, E, (MPa)**	**Void ratio, e**	**Permeability** Immediately after Cement mixing (m/s)	**Slump** Immediately after Cement mixing (mm)
			Bulk	Dry	Cohesion, c (kPa)	Internal angle of friction, φ				
2%	**7**	51.6	1970.1	1475.6	211	44^o	15.2	0.76	**4.5 x 10^{-6}**	**167**
	14	65	1960.6	1486.5	257	44^o	25	0.75		
	21	69	1956.5	1456.6	136	39^o	30	0.71		
	28	75	1974.0	1522.5	214	28^o	30	0.71		
4%	**7**	84	1886.6	1365.2	300	36^o	15.5	0.90	**4.0 x 10^{-6}**	**165**
	14	130	1886.6	1365.2	229	34^o	25	0.90		
	21	170	1882.1	1365.2	310	41^o	70	0.90		
	28	234	1924.9	1404.4	172	35^o	100	0.87		
6%	**7**	95	1995.6	1516.1	150	34^o	20	0.72	**3.6 x 10^{-6}**	**158**
	14	160	1968.5	1488.9	132	35^o	70	0.75		
	21	250	1995.6	1516.1	109	35^o	120	0.72		
	28	350	1944.4	1464.5	145	34^o	150	0.78		
8%	**7**	120	1944.4	1464.5	331	34^o	25	0.78	**2.5 x 10^{-6}**	**169**
	14	250	1971.6	1491.6	225	34^o	100	0.74		
	21	330	1944.4	1464.5	306	43^o	150	0.78		
	28	480	2084.0	1874.9	172	35^o	200	0.30		
10%	**7**	225	1952.0	1676.1	303	45^o	70	0.58	**1.5 x 10^{-6}**	**157**
	14	304	1950.7	1404.4	309	45^o	150	0.85		
	21	490	1943.9	1418.0	311	49^o	190	0.83		
	28	650	2097.6	1978.8	319	49^o	260	0.31		

Table 5.9 Konkola Cemented Hydraulic Fill Material Geotechnical Test Results for Backfill Desirable Properties-Zambezi Cement

Cement Content	**Curing Period (Days)**	**Uniaxial Compressive strength, UCS (kPa)**	**Density (kg/m^3)**		**Shear Strength Parameters**		**Young's Modulus, E (MPa)**	**Void ratio, e**	**Permeability** Immediately after Cement mixing (m/s)	**Slump** Immediately after Cement mixing (mm)
			Bulk	Dry	Cohesion, c, (kPa)	Internal angle of friction, φ				
2%	**7**	66	2001.3	1505.1	241	43^o	20	0.73	**4.8 x 10^{-6}**	**165**
	14	84	2001.7	1500.0	263	45^o	33	0.73		
	21	89	2001.3	1500.0	234	33^o	40	0.73		
	28	97	2017.8	1983.0	217	29^o	40	0.31		
4%	**7**	110	1923.2	1410.0	300	37^o	21	0.85	**4.2 x 10^{-6}**	**163**
	14	170	1927.3	1406.0	212	37^o	33	0.85		
	21	224	1925.0	1411.6	314	43^o	93	0.85		
	28	290	1965.3	1418.0	107	38^o	133	0.83		
6%	**7**	124	2010.2	1532.4	134	34^o	27	0.70	**3.4 x 10^{-6}**	**156**
	14	210	2009.3	1529.3	173	33^o	93	0.70		
	21	271	2006.5	1532.4	230	39^o	159	0.70		
	28	463	1985.0	1505.3	212	39^o	199	0.73		
8%	**7**	157	1981.1	1502.5	222	33^o	33	0.73	**2.1 x 10^{-6}**	**168**
	14	330	1985.2	1505.2	236	38^o	133	0.73		
	21	436	1980.1	1513.4	243	37^o	199	0.73		
	28	635	2111.3	2091.0	212	30^o	266	0.25		
10%	**7**	297	1965.6	1428.9	233	43^o	93	0.83	**1.3 x 10^{-6}**	**155**
	14	399	1965.6	1418.0	251	36^o	199	0.83		
	21	648	1961.5	1431.6	230	45^o	252	0.83		
	28	861	2111.3	2091.0	265	49^o	345	0.25		

The tests above on Cemented Konkola Hydraulic Fill show that the only maximum Uniaxial Compressive Strength was 861 kPa (0.861 MPa) attained by addition of 10% Zambezi Cement cured for a maximum period of 28 days. This is clearly below the recommended strength of 1 MPa required to facilitate the Primary/secondary extraction system for KDMP mining operations. Preliminary assessments indicate that very uneconomic quantities of cement would need to be added to the hydraulic fill to achieve the UCS of 1 MPa. This is owing to excessive fines in the material which tend to consume a lot of cement due to the high surface area of the particles.

Henderson and Revell (2005)[9] have recognized that addition of coarse material to tailings helps to improve the strength characteristics and reduces the cement consumption. Fill grading is an extremely important control on strength, and the objective in grading the aggregate is to minimise void ratio and maximize the placed density. An excess of coarse particles will result in a loose, open material susceptible to blast damage as it relies largely on point to point contact. Too few fines may also allow excess water to percolate through the fill mass, washing out cement, and may not allow the mixture to be fluid enough to flow during placement.

The relationship between fill particle size distribution, cement content and strength is very complex and non linear. Various variables also impact on strength development of cemented backfill such as mineralogical, chemical composition and attrition potential of the base fill material. Therefore it is very critical to conduct site specific laboratory testing to develop suitable backfill materials for safe and economic ore production.

The following section presents and discusses further laboratory tests and findings that were undertaken at Nchanga Geotechnical Services Laboratory in the continued pursuit to develop a suitable backfill material to meet the required strength characteristics at Konkola Mine.

5.4 Particle Size Distribution and Optimization tests

In view of the findings by Henderson and Revell (2005)[9] above, rock samples were picked from around the No.3 Shaft waste rock dump and taken to Nchanga Geotechnical Services Laboratory for crushing into aggregates. At the lab, a study of the optimal blending and resulting grading envelope between crushed waste rock and the hydraulic fill (classified mill tailings) was commenced. Different mixtures were developed between the crushed waste rock and the hydraulic fill in order to arrive at the blending that was going to improve the well graded-ness of the blend whilst at the same time improving the density of the backfill material as shown in Picture 5.3.

Picture 5.3 Blending of crushed waste rock from No.3 Shaft waste rock dump and the classified tailings

In order to quantify and assess the particle size distribution characteristics of the blended trial materials, the grading indices namely, the Coefficient of Uniformity (C_u) and Coefficient of Curvature (C_c) were used.

The power law equation was also utilized in this exercise to determine whether the blended fill material satisfied the appropriate particle size distribution which ensures well graded-ness and dense packing of particles.

During the crushing and blending trials at Nchanga Geotechnical Services laboratory, it was discovered that an optimum grading envelope that would be ideal is a mixture of 70% of crushed waste rock graded at 100% passing No. 2.83mm sieve and 30% Hydraulic fill produced at both KCM 1 and 2 Backfill plants, respectively. The limitations imposed by the existing backfill reticulation system at Konkola Mine, such as pipe sizes and pump capacities were taken into consideration during the trial crushing and blending exercise. The maximum particle size of 2.83mm for crushed waste could be transported in the existing 100mm pipes. Figure 5.8 shows the grading envelope of the blended material arrived at after the PSD test.

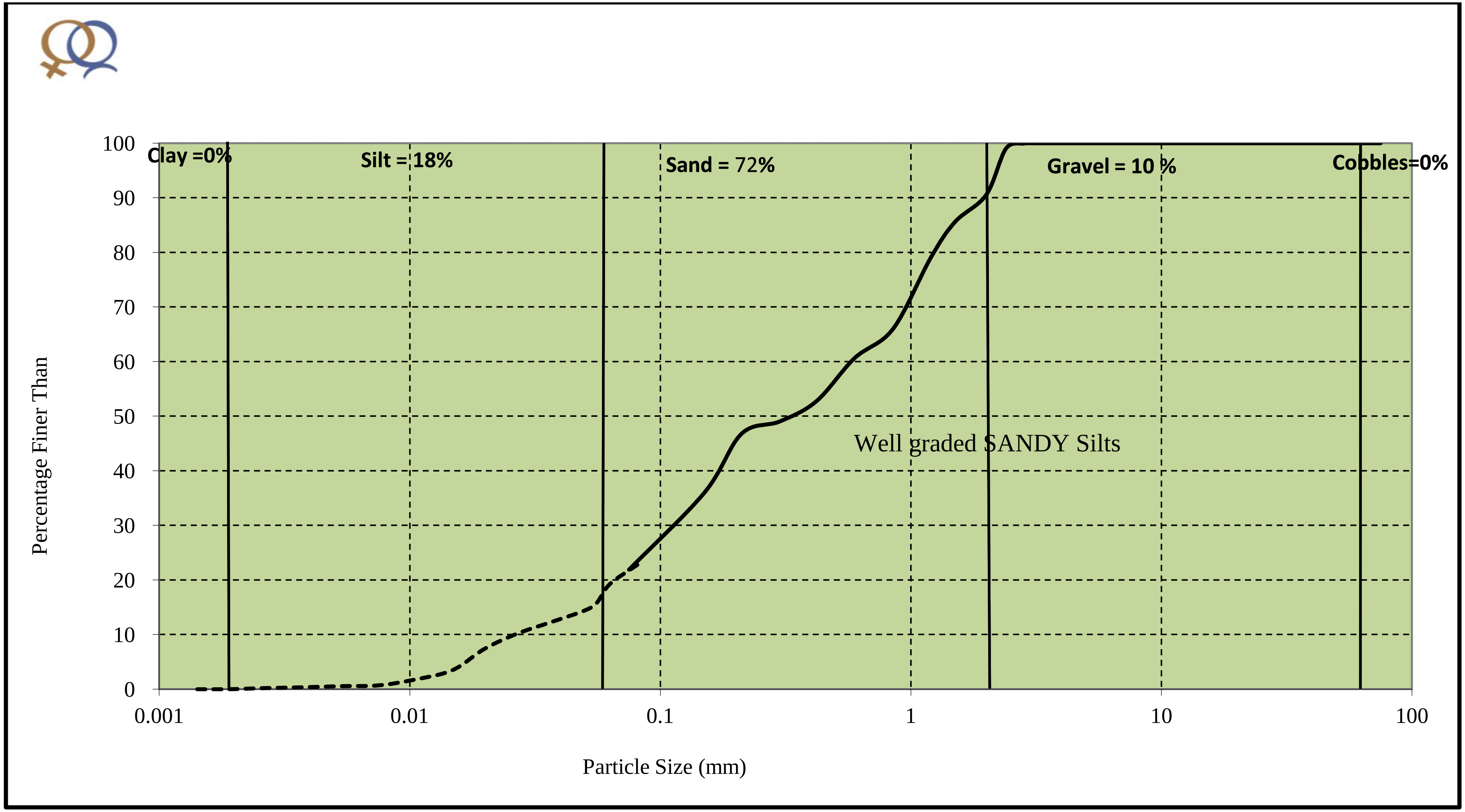


Figure 5.8 PSD for 70% Crushed Waste Rock and 30% Hydraulic Fill Material

Table 5.10 shows readings from the graph in Figure 5.12 used to determine the grading indices for use in deriving the distribution characteristics of the blended fill material.

Table 5.10 Graph Readings for PSD Analysis

Graph Readings	D_{10}	D_{30}	D_{60}
Size	0.0268	0.13	0.6

- Coefficient of Uniformity, $Cu = D_{60}/ D_{10} = 22$
- Coefficient of Curvature, $Cc = D^2_{30} / D_{60} \times D_{10} = 1.05$

The C_u and C_c values above indicate that the blending of crushed waste rock and hydraulic fill material was carefully selected to produce a well graded mixture with a long range of particle sizes.

The blended fill material above was further subjected to the power law equation ($\boldsymbol{P = 100 \times [d/dmax]^{0.5}}$) to determine whether the material satisfied the appropriate particle size distribution which ensures well graded-ness and dense packing of particles. Table 5.11 below shows the sieve sizes in the blending trials and the resulting passing percentages.

Table 5.11 Particle Size Distribution of the Blended Material

Sieve	**Sieve Opening (mm)**	**Mass of soil retained (g)**	**% Retained**	**% Passing**
8	2.4	80	8	92
10	2	90	9	83
12	1.5	55	6	77
14	1.2	82	8	69
20	0.85	110	11	58
30	0.595	95	10	49
40	0.42	78	8	41
52	0.3	63	6	35
70	0.211	56	6	29
100	0.15	46	5	24
200	0.075	72	7	17
-200		173	17	

Total mass before washing Dry mass after washing Mass washed through 75mm sieve	1000 grams 827 grams 173 grams

The n values for each of the passing percentages of the blended material were calculated from the following reversed power law equation:

$$n = log(P/100)/\,log\,(d/dmax) \qquad (32)$$

Where, *d* = the particle size (mm)

dmax = the maximum particle size (mm)

P = the percentage of blended material smaller than size *d*

*Example of calculation of the n value for 83% of the blended material passing through sieve size 2.00 mm:

P	***d***	***dmax***
83%,	2.00mm,	2.83mm

Hence, $\boldsymbol{n}= \log(83/100)/\,\log\,(2/2.83)$

= <u>0.54</u>

The rest of the values were compiled and are presented in Table 5.12.

Table 5.12 Power Law n values for the Blended material

***P* (Passing %)**	***dmax* (mm)**	***d* (mm)**	$\boldsymbol{n = log(P/100)/\,log\,(d/dmax)}$	
92	2.83	2.4		0.51
83	2.83	2		0.54
77.46	2.83	1.5		0.4
69.28	2.83	1.2		0.43
58.31	2.83	0.85		0.45
48.79	2.83	0.6		0.46
40.99	2.83	0.42		0.47
34.64	2.83	0.3		0.47
29.05	2.83	0.21		0.48
24.49	2.83	0.15		0.48
17.32	2.83	0.08		0.48
Statistical Analysis to asses reliability of the n values			Mean	0.5
			Standard Deviation	0.04
			Coefficient of Variation	7.9 %

From the average n value obtained as 0.5 above, and low coefficient of variation of 7.9%, it was concluded that the results of the blending exercise were reliable and hence the aggregate fill material was densely packed and well graded.

5.5 Cemented Aggregate Fill (CAF) Material

The blended aggregate material described above was mixed with different proportions of the Mphamvu, Zambezi and the Fast Set Minova cement brands (by dry weight) and water to prepare samples for tests. Table 5.13 shows the cement mixing schedule that was followed to prepare the samples in separate batches.

Table 5.13 Cemented Aggregate Fill Material - Cement Addition Schedule (Mphamvu, Zambezi and Minova Fast Set Cement Brands batched separately)

Container No	Mass of 70% Crushed Aggregate (g)	Mass of 30% Hydraulic Fill material (g)	Total Mass of Blended Material (g)	% Cement Added • Mphamvu • Zambezi • Minova (Prepared in separate batches)	Mass of Cement (g)	Total Mass of Cemented Aggregate Fill (CAF) Material (g)
1	34,300	14,700	49,000	2%	1000	50,000
2	33,600	14,400	48,000	4%	2000	50,000
3	32,900	14,100	47,000	6%	3000	50,000
4	32,200	13,800	46,000	8%	4000	50,000
5	31,500	13,500	45,000	10%	5000	50,000

This material, which is referred to as Cemented Aggregate Fill (CAF), was prepared for Unconfined Compressive Strength (UCS) testing using cylindrical sampling moulds of size 50mm diameter and 125mm height made of PVC. Another set of samples was prepared for Triaxial Testing using cylindrical sampling moulds of size 37.5mm diameter and 75mm height made of the same material. The excess materials after batching were subjected to other engineering tests in line with the desirable properties of backfill material outlined in Table 2.1 in Chapter 3, page 13.

The CAF material samples prepared with the Mphamvu and Zambezi cement brands were cured and tested at 7, 14, 21 and 28 days whilst those prepared with the Minova Fast Set cement were cured and tested at 1 hour, 4 hours, 1 day, 3 days, 7 days, 14 days, 21 days and 28 days. It was necessary to include shorter testing times for the Minova Fast Set cement owing to its rapid setting and curing characteristics. The laboratory tests were done with efforts made to simulate the prevailing underground humid conditions at No.3 Shaft. Water was not allowed to drain out of the samples after cement mixing to prevent loss of cement fines which would otherwise jeopardized the hydration process. The Cemented Aggregate Fill material was cured in the humid room at Geotechnical Services Laboratory.

5.5.1 Uniaxial Compressive Strengths tests on Cemented Aggregate Fill

The Cemented Aggregate Fill Material samples were taken out of the humid room and tested for UCS at the indicated curing periods mentioned above for all the three cement brands. The same Triaxial testing machine was used with no confining pressure. Picture 5.4 shows the typical failure characteristics of the cylindrical Cemented Aggregate Fill samples after UCS testing.

Picture 5.4 Typical failure characteristics of the cylindrical CAF samples after UCS testing

Results of the Unconfined Compressive Strength tests were used to plot the Mohr circles at zero confining pressure for different cement contents at different curing periods. Figure 5.9 is an example of the Mohr circle plotted for the UCS testing of the CAF with 4% Zambezi cemented added and cured for 7 Days.

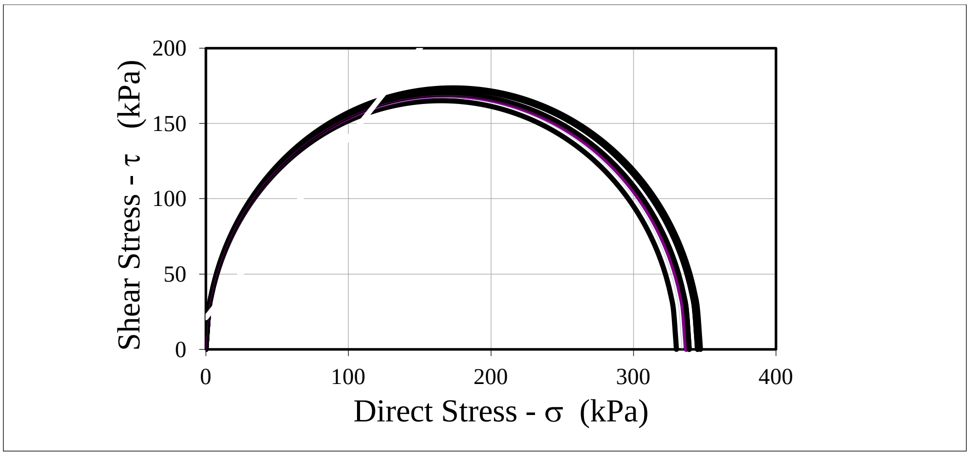


Figure 5.9 UCS Test Results for Cemented Aggregate Fill (4% Zambezi Cement at 7 days)

UCS test results for all the three cement brands; Mphamvu, Zambezi and Minova Fast Set Cement (with associated statistical analysis to assess reliability of data) are given in Tables 5.14, 5.15 and 5.16.

Table 5.14 Statistical Analysis of UCS test results for CAF Material - Mphamvu Cement

Cement Content	Curing Period (Days)	UCS (kPa) for CAF samples						Statistical Analysis		
		1	2	3	4	5	6	Mean (kPa	Standard Deviation (kPa)	Coefficient of Variation (%)
2%	7	87	80	85	81	77	88	83	4.34	5.22
	14	179	175	170	185	173	180	177	5.40	3.05
	21	180	183	176	189	186	166	180	8.22	4.57
	28	200	210	208	211	199	220	208	7.77	3.74
4%	7	257	260	266	249	256	254	257	5.73	2.23
	14	314	312	310	325	320	321	317	5.87	1.85
	21	444	440	439	457	450	452	447	7.16	1.60
	28	760	757	770	765	753	764	762	6.09	0.80
6%	7	423	437	440	431	439	428	433	6.78	1.57
	14	518	530	539	526	531	518	527	8.15	1.55
	21	760	740	758	738	748	744	748	9.21	1.23
	28	1030	1015	999	1021	997	1040	1017	16.98	1.67
8%	7	800	825	830	810	800	831	816	14.49	1.78
	14	919	920	937	930	917	951	929	13.22	1.42
	21	1021	1038	1043	1030	1023	1025	1030	8.81	0.86
	28	1400	1399	1370	1376	1380	1367	1382	14.30	1.03
10%	7	981	999	989	976	971	964	980	12.62	1.29
	14	1130	1120	1111	1100	1125	1104	1115	11.93	1.07
	21	1333	1327	1330	1321	1329	1256	1316	29.66	2.25
	28	1959	1945	1966	1939	1927	1976	1952	18.24	0.93

Table 5.15 Statistical Analysis of UCS test results for CAF Material – Zambezi Cement

Cement Content	Curing Period (Days)	UCS (kPa) for CAF samples						Statistical Analysis		
		1	2	3	4	5	6	Mean (kPa)	Standard Deviation (kPa)	Coefficient of Variation (%)
2%	7	99	100	110	111	102	108	105	5.29	5.04
	14	225	239	230	235	222	241	232	7.64	3.29
	21	230	238	239	235	233	247	237	5.90	2.49
	28	271	270	266	280	277	280	274	5.83	2.13
4%	7	330	339	345	347	337	334	339	6.47	1.91
	14	415	420	427	430	433	407	422	9.88	2.34
	21	699	692	700	700	722	699	702	10.26	1.46
	28	1013	1020	1015	1020	999	993	1010	11.35	1.12
6%	7	569	580	588	573	573	555	573	11.08	1.93
	14	700	700	711	690	698	689	698	8.02	1.15
	21	1000	998	991	989	990	978	991	7.80	0.79
	28	1330	1359	1333	1359	1339	1368	1348	15.95	1.18
8%	7	1050	1100	1090	1070	1060	1116	1081	25.26	2.34
	14	1200	1232	1244	1250	1245	1221	1232	18.88	1.53
	21	1353	1379	1366	1360	1350	1388	1366	14.93	1.09
	28	1850	1852	1820	1845	1800	1819	1831	21.09	1.15
10%	7	1200	1350	1333	1320	1345	1252	1300	60.53	4.66
	14	1411	1379	1350	1390	1400	1344	1379	27.03	1.96
	21	1703	1722	1729	1760	1770	1786	1745	31.87	1.83
	28	2250	2245	2200	2202	2269	2046	2202	30.74	1.40

Table 5.16 Statistical Analysis of UCS test results for CAF Material – Minova Fast Set Cement

Cement Content	Curing Period (Days)	UCS (kPa) for CAF samples						Statistical Analysis		
		1	2	3	4	5	6	Mean (kPa)	Standard Deviation (kPa)	Coefficient of Variation (%)
2%	0.04	110	115	117	114	105	117	113	4.69	4.15
	0.17	235	240	220	233	232	226	231	7.04	3.05
	1	251	246	256	249	241	257	250	6.07	2.43
	3	261	269	273	263	264	284	269	8.56	3.18
	7	306	309	293	295	300	303	301	6.23	2.07
	14	340	349	355	351	347	334	346	7.69	2.22
	21	366	370	359	356	361	360	362	5.10	1.41
	28	400	391	401	380	392	382	391	8.76	2.24
4%	0.04	349	333	336	331	345	346	340	7.59	2.23
	0.17	419	418	431	433	422	427	425	6.29	1.48
	1	712	721	698	708	709	718	711	8.15	1.15
	3	1023	996	1011	1021	1015	988	1009	14.07	1.39
	7	1270	1265	1330	1350	1279	1378	1312	47.30	3.61
	14	1499	1520	1533	1450	1495	1437	1489	38.10	2.56
	21	1602	1609	1550	1570	1639	1564	1589	33.39	2.10
	28	1611	1598	1670	1652	1653	1674	1643	31.37	1.91
6%	0.04	560	566	589	570	571	606	577	17.20	2.98
	0.17	700	701	702	700	669	602	679	39.81	5.86
	1	965	989	1001	1003	961	997	986	18.49	1.88
	3	1346	1351	1362	1286	1288	1365	1333	36.31	2.72
	7	1401	1489	1465	1414	1436	1507	1452	42.10	2.90
	14	1551	1501	1503	1523	1568	1660	1551	59.60	3.84
	21	1685	1755	1701	1733	1674	1658	1701	36.81	2.16
	28	1811	1761	1835	1823	1763	1759	1792	34.82	1.94
8%	0.04	703	711	715	683	683	693	698	13.84	1.98
	0.17	1102	1179	1122	1122	1163	1248	1156	53.42	4.62
	1	1311	1376	1401	1401	1365	1354	1368	33.75	2.47
	3	1821	1851	1826	1792	1845	1815	1825	21.36	1.17
	7	2111	2125	2230	2211	2231	2262	2195	62.00	2.82
	14	2389	2462	2410	2501	2399	2557	2453	66.43	2.71
	21	2556	2533	2501	2511	2492	2413	2501	48.94	1.96
	28	2689	2556	2633	2687	2641	2466	2612	86.36	3.31
10%	0.04	983	1000	956	945	963	1009	976	25.47	2.61
	0.17	1236	1156	1265	1286	1266	1207	1236	47.92	3.88
	1	1813	1811	1756	1742	1717	1619	1743	71.76	4.12
	3	2151	2162	2211	2233	2212	2237	2201	36.23	1.65
	7	2463	2451	2501	2381	2399	2283	2413	77.21	3.20
	14	2612	2545	2511	2623	2499	2522	2552	53.07	2.08
	21	2711	2799	2656	2633	2651	2744	2699	64.31	2.38
	28	2999	3045	3021	2965	2977	3047	3009	34.48	1.15

Laboratory test data for the above three cement brands, Mphamvu, Zambezi and Minova Fast Set cement is given in Appendices 05, 06 and 07, respectively.

Figures 5.10, 5.11 and 5.12 are graphical representations of the average UCS test results for the Mphamvu, Zambezi and Minova Fast Set cement brands, respectively, plotted at different curing periods.

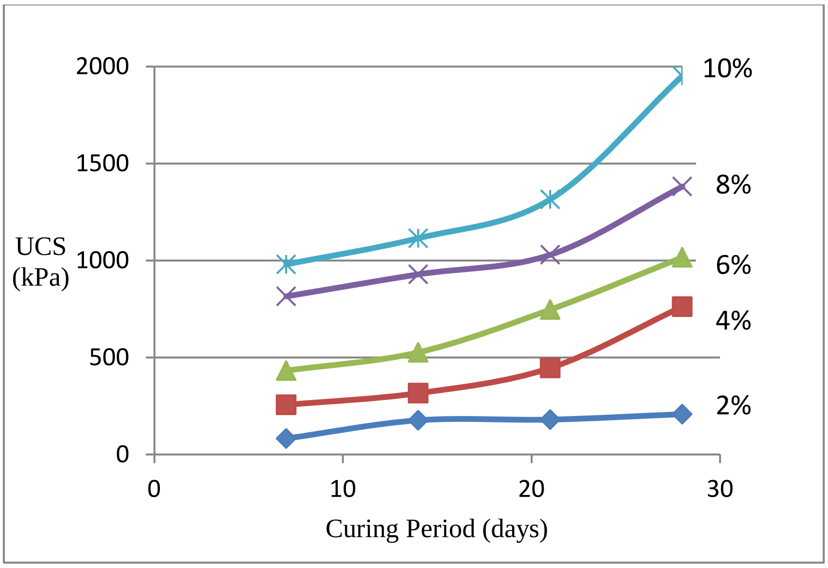


Figure 5.10 Graph of Cemented Aggregate Fill UCS Test Results using Mphamvu Cement

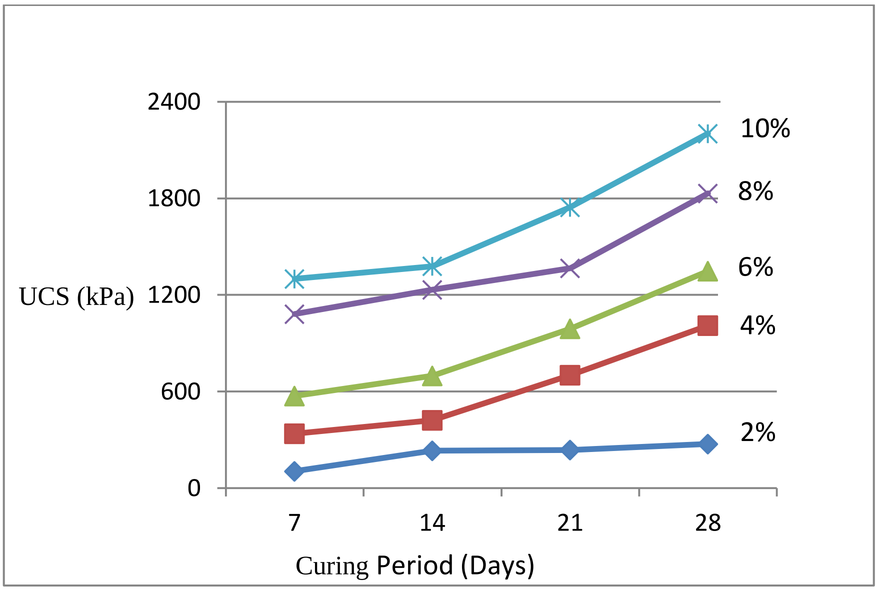


Figure 5.11 Graph of Cemented Aggregate Fill UCS Test Results using Zambezi Cement

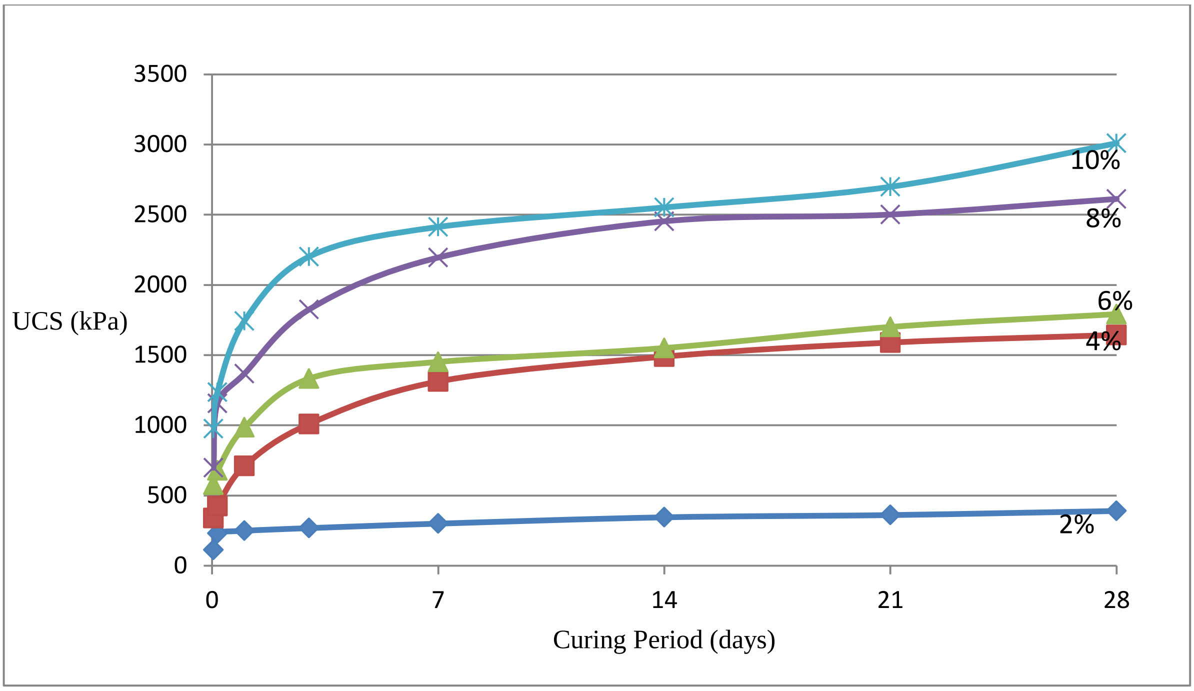


Figure 5.12 Graphs of Cemented Aggregate Fill UCS Test Results - Minova Fast Set Cement

5.5.2 Triaxial Tests on Cemented Aggregate Fill (CAF) material

Triaxial tests were undertaken for the Cemented Aggregate Fill samples prepared using the three cement brands described earlier to determine the shear strength characteristics at the indicated curing periods. Figure 5.13 is a typical example of the Mohr circle envelope for the blended material with 2% Zambezi Cement added and cured for 14 days from where the internal angle of friction, φ, and the cohesion, c, were derived.

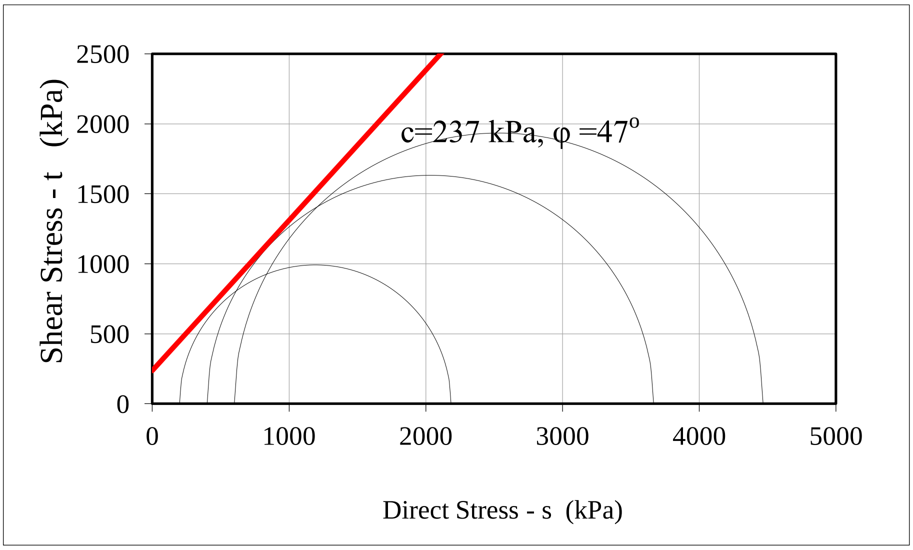


Figure 5.13 Mohr circles for the CAF material with 2% Zambezi cement cured for 14 days.

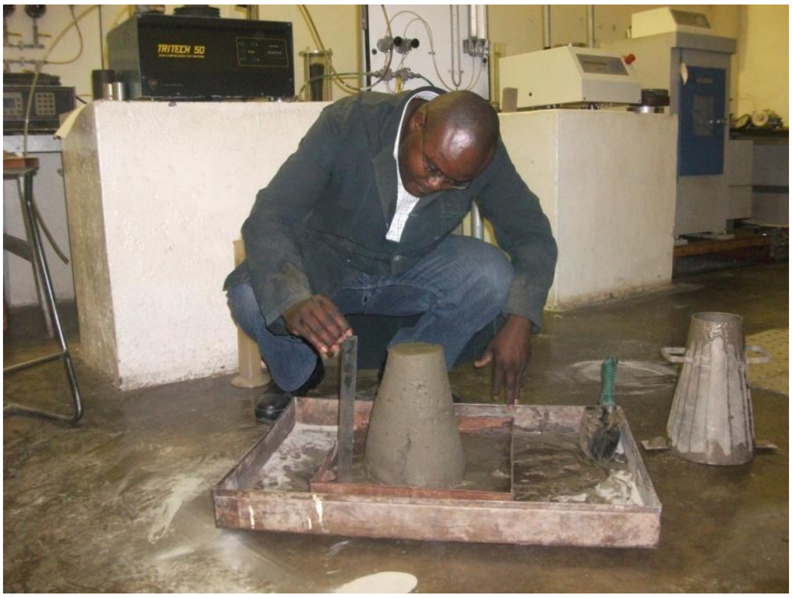
Results of the rest of the triaxial tests together with results from other tests done on CAF samples are summarized in Tables 5.17, 5.18 and 5.19 for samples prepared from Mphamvu, Zambezi and Minova Fast Set Cement, respectively.

5.5.3 Permeability Tests on Cemented Konkola Hydraulic Fill Material

The Cemented Aggregate Fill material was subjected to permeability tests so as to ascertain the drainage of transport water that would be used to deliver the material to undergrounds stopes. The tests were done immediately after cement addition. Results of the permeability tests are presented in Tables 5.17, 5.18 and 5.19 and for all the three cement brands

5.5.4 Slump Tests on Cemented Aggregate Fill Material

Slump tests were conducted on Cemented Aggregate Fill Material to determine the pumpability and transportability or fluidity characteristics of cemented backfill material. The slump tests were done immediately after cement addition using a 300mm cylindrical cone as demonstrated in Picture 5.5.



Picture 5.5 Slump test measurement

Results of the Slump tests are presented in Tables 5.17, 5.18 and 5.19 for all the three cement brands

5.5.5 Atterberg limits Tests on Cemented Konkola Hydraulic Fill Material

Atterberg Limits tests were undertaken on the Cemented Aggregate Fill Material in order to determine the range of moisture contents over which the material would exhibit consistence. It was observed in all the tests that the cemented material for both Mphamvu and Zambezi cement brands had no plasticity.

The results for all the tests conducted on the Cemented Hydraulic Fill material, against the desirable characteristics, for Mphamvu, Zambezi and Minova Fast Set cement brands are presented in Tables 5.17, 5.18 and 5.19, respectively

Table 5.17 Cemented Aggregate Fill Material Geotechnical Test Results for Backfill Desirable Properties-Mphamvu Cement

Cement Content	**Curing Period (Days)**	**Uniaxial Compressive strength UCS (kPa)**	**Density (kg/m^3)**		**Unit weight (kN/m^3)**		**Shear Strength Parameters**		**Young's Modulus, E, (MPa)**	**Void ratio, e**	**Permeability** Immediately after Cement mixing (m/s)	**Slump** Immediately after Cement mixing (mm)
			Bulk	Dry	Bulk	Dry	Cohesion, c (kPa)	Internal angle of friction, φ				
2%	**7**	83	2017.8	1983.0	19.8	19.5	90	45^o	39	0.31	**2.6 x 10^{-5}**	**159**
	14	177	2099.0	1987.0	20.6	19.5	263	44^o	69	0.31		
	21	180	2138.7	2058.8	21.0	20.2	234	26^o	67	0.26		
	28	208	2141.3	2015.3	21.0	19.8	196	30^o	90	0.29		
4%	**7**	257	2155.3	2055.0	21.1	20.2	296	36^o	50	0.27	**2.3 x 10^{-5}**	**163**
	14	317	2139.3	2048.0	21.0	20.1	233	37	70	0.27		
	21	447	2109.3	1980.7	20.7	19.4	263	44^o	70	0.32		
	28	762	2197.3	2084.0	21.6	20.4	107	38^o	106	0.25		
6%	**7**	433	2117.3	2043.3	20.8	20.0	97	38^o	65	0.73	**1.2 x 10^{-5}**	**157**
	14	527	2111.3	2091.0	20.7	20.5	107	29^o	77	0.25		
	21	748	2161.7	2108.0	21.2	20.7	120	33^o	93	0.23		
	28	1017	2147.7	2050.0	21.1	20.1	128	40^o	110	0.27		
8%	**7**	816	2213.0	2078.0	21.7	20.4	143	29^o	67	0.25	**7.0 x 10^{-6}**	**153**
	14	929	2144.7	2063.0	21.0	20.2	120	35^o	100	0.26		
	21	1030	2179.7	2082.7	21.4	20.4	133	38^o	150	0.25		
	28	1382	2257.7	2154.0	22.1	21.1	115	39^o	200	0.21		
10%	**7**	980	2129.0	2010.7	20.9	19.7	234	41^o	109	0.30	**6.4 x 10^{-6}**	**149**
	14	1115	2148.7	2054.3	21.1	20.2	243	42^o	122	0.27		
	21	1316	2135.0	2011.3	20.9	19.7	255	39^o	193	0.30		
	28	1952	2147.3	2053.3	21.1	20.1	278	49^o	250	0.27		

Table 5.18 Cemented Aggregate Fill Material Geotechnical Test Results for Backfill Desirable Properties-Zambezi Cement

Cement Content	**Curing Period (Days)**	**Uniaxial Compressive strength, UCS, (kPa)**	**Density (kg/m^3)**		**Unit weight (kN/m^3)**		**Shear Strength Parameters**		**Young's Modulus, E, MPa**	**Void ratio, e**	**Permeability** Immediately after Cement mixing (m/s)	**Slump** Immediately after Cement mixing (mm)
			Bulk	Dry	Bulk	Dry	Cohesion, c	Internal angle of friction, φ				
2%	**7**	**105**	2016.1	1984.2	19.8	19.5	102	46	60	0.31	**2.5×10^{-5}**	**155**
	14	**232**	2092.2	1981.5	20.5	19.4	237	47	91	0.32		
	21	**237**	2134.3	2054.8	20.9	20.2	180	40	89	0.27		
	28	**274**	2137.0	2011.4	21.0	19.7	76	39	120	0.29		
4%	**7**	**339**	2151.3	2054.8	21.1	20.2	254	40	65	0.27	**2.1×10^{-5}**	**160**
	14	**422**	2121.4	2030.4	20.8	19.9	187	40	93	0.28		
	21	**702**	2102.4	1975.4	20.6	19.4	342	44^o	93	0.32		
	28	**1010**	2192.0	2084.0	21.5	20.4	139	38^o	141	0.25		
6%	**7**	**573**	2121.4	2043.6	20.8	20.0	126	38^o	86	0.27	**1.3×10^{-5}**	**156**
	14	**698**	2105.8	2078.6	20.7	20.4	139	29^o	103	0.25		
	21	**991**	2165.9	2117.0	21.2	20.8	156	33^o	124	0.23		
	28	**1348**	2142.3	2033.8	21.0	20.0	166	40^o	146	0.28		
8%	**7**	**1081**	2207.6	2074.5	21.7	20.4	186	29^o	89	0.25	**8.9×10^{-6}**	**150**
	14	**1232**	2136.3	2056.2	21.0	20.2	156	35^o	133	0.27		
	21	**1366**	2177.1	2074.5	21.4	20.4	173	38^o	199	0.26		
	28	**1831**	2255.2	2147.2	22.1	21.1	150	39^o	266	0.21		
10%	**7**	**1300**	2135.6	2009.3	21.0	19.7	304	41^o	145	0.30	**6.4×10^{-6}**	**145**
	14	**1379**	2150.0	2048.9	21.1	20.1	316	42^o	162	0.27		
	21	**1745**	2130.9	2019.5	20.9	19.8	332	39^o	256	0.29		
	28	**2202**	2143.1	2053.5	21.0	20.1	361	49^o	332	0.27		

Table 5.19 Cemented Aggregate Fill Material Geotechnical Test Results for Backfill Desirable Properties- Minova Fast Set Cement

Cement Content	**Curing Period (Days)**	**Uniaxial Compressive strength, UCS, (kPa)**	**Density (kg/m^3)**		**Shear Strength Parameters**		**Young's Modulus, E, MPa**	**Void ratio, e**	**Permeability** Immediately after Cement mixing (m/s)	**Slump** Immediately after Cement mixing (mm)
			Bulk	Dry	Cohesion, c	Internal angle of friction, φ				
2%	0.04	113	2239.0	1899.0	21	25	65	0.45	**2.1×10^{-5}**	**163**
	0.17	231	2329.0	1923.0	27	26	95	0.41		
	1.00	250	2289.0	1978.0	27	31	150	0.36		
	3.00	269	2354.0	2001.0	26	29	158	0.32		
	7.00	301	2230.0	1989.0	33	32	199	0.38		
	14.00	346	2199.0	1966.0	35	31	186	0.37		
	21.00	362	2239.0	1911.0	31	34	178	0.35		
	28.00	391	2199.0	1897.0	45	32	187	0.22		
4%	0.04	340	2289.0	1945.6	32	27	187	0.39	**1.1×10^{-5}**	**155**
	0.17	425	2295.5	1951.6	43	31	210	0.33		
	1.00	711	2308.1	1998.5	46	38	321	0.30		
	3.00	1009	2259.2	1951.6	74	48	333	0.33		
	7.00	1312	2256.0	1967.0	123	48	341	0.36		
	14.00	1489	2311.0	2000.0	231	49	346	0.41		
	21.00	1589	2233.0	1933.0	167	47	378	0.30		
	28.00	1643	2302.0	2011.0	233	49	383	0.25		

Table 5.19 Cemented Aggregate Fill Material Geotechnical Test Results for Backfill Desirable Properties- Minova Fast Set Cement (Continued)

6%	0.04	577	2256.0	1965.0	37	25	245	0.39	**2.3 x 10^{-6}**	**142**
	0.17	679	2157.0	1936.0	39	33	311	0.25		
	1.00	986	2275.0	1901.0	66	36	323	0.22		
	3.00	1333	2304.0	1954.0	72	35	334	0.21		
	7.00	1452	2241.0	1972.0	167	38	351	0.39		
	14.00	1551	2233.0	1932.0	233	42	369	0.35		
	21.00	1701	2312.0	2002.0	242	43	378	0.33		
	28.00	1792	2271.0	1932.0	253	48	393	0.21		
8%	0.04	698	2239.0	1899.0	73	33	313	0.46	**1.6 x 10^{-6}**	**136**
	0.17	1156	2329.0	1923.0	111	36	320	0.41		
	1.00	1368	2289.0	1978.0	123	34	345	0.38		
	3.00	1825	2354.0	1999.0	87	36	411	0.32		
	7.00	2195	2230.0	1989.0	143	41	429	0.31		
	14.00	2453	2244.0	1963.0	265	44	444	0.29		
	21.00	2501	2239.0	1911.0	267	40	449	0.4		
	28.00	2612	2233.0	1986.0	311	49	465	0.23		
10%	0.04	976	2201.0	1913.0	78	34	331	0.39	**1.1 x 10^{-5}**	**123**
	0.17	1236	2285.7	1929.2	145	38	341	0.35		
	1.00	1743	2304.1	1961.8	151	38	387	0.33		
	3.00	2201	2338.7	1959.8	165	38	452	0.33		
	7.00	2413	2311.0	1966.0	178	43	456	0.39		
	14.00	2552	2278.0	1971.0	192	41	503	0.25		
	21.00	2699	2312.0	1981.0	256	43	534	0.22		
	28.00	3009	2256.0	1996.0	321	50	566	0.21		

5.6 Slag from New Smelter at Nchanga Mine

Samples of the granulated slag from the New Copper Smelter at Nchanga Mine were taken to Geotechnical Services Laboratory for tests to assess their suitability for use as backfill material. The slag is expected to be pozzolonic and would form compounds possessing cementitious properties if it chemically reacts with calcium hydroxide (lime) at ordinary temperature in the presence of moisture.

Mixtures of the slag and lime were prepared in the ratio 2:1 by dry weight, as demonstrated in Picture 5.6 for tests in line with the desirable properties of backfill material.

Picture 5.6 Preparing slag/lime mixture

It was dangerous to handle the slag/lime mixture initially due to a lot of heat that was generated due to the highly exothermic reactions.

The material was prepared for Unconfined Compressive Strength (UCS) tests in the same way as before using cylindrical sampling moulds of size 50mm diameter and 125mm height made of PVC. Another set of samples was prepared for Triaxial Testing using cylindrical sampling moulds of size 37.5mm diameter and 75mm height made of the same material. The excess material was subjected to the other tests namely, Permeability, Atterberg limit tests, and Slump tests following the testing procedures outlined above. The slag/lime mixture was cured and tested after 7, 14, 21 and 28 days.

5.6.1 Uniaxial Compressive Strengths tests on the Slag/Lime Mixture

The slag/lime mixture was subjected to Unconfined Compressive Strength testing at the indicated curing periods using the Triaxial Testing Machine again with no confining pressure.

Results of the tests were used to plot the Mohr circles at zero confining pressure from where the UCS test results were read. Table 5.20 shows all the UCS results and the associated statistical analysis to assess reliability of data.

Table 5.20 Statistical Analysis of UCS test results for the slag/lime Mixture

Slag/Lime 2:1	Curing Period (Days)	UCS (kPa) for the Slag/Lime Mixture						Statistical Analysis		
		1	2	3	4	5	6	Mean (kPa)	Standard Deviation (kPa)	Coefficient of Variation (%)
	7	90	80	84	82	77	88	84	4.89	5.85
	14	180	173	188	170	190	160	177	11.43	6.46
	21	199	181	187	191	165	166	182	13.71	7.55
	28	220	201	200	215	221	199	209	10.44	4.99

The laboratory test data for the slag/lime mixture is given in Appendix 08.

Figure 5.14 shows the graphical representations of the UCS test results for the slag/lime mixture in the given curing periods.

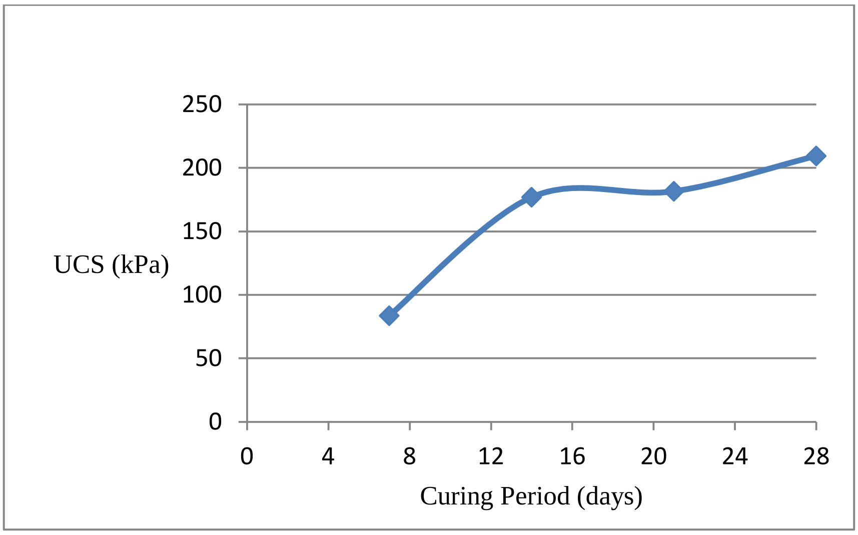


Figure 5.14 Graph of UCS tests results for slag-lime mixture

The other test results that were derived from the UCS test works above included, bulk and dry densities, void ratios and Young's Moduli (from where the stress/strain relationships were derived). The results are given in Table 5.21 together with results of other tests performed on these samples.

5.6.2 Triaxial Tests on the Slag/Lime Mixture

The slag/lime mixture samples prepared for triaxial testing were taken out of the humid room and tested to ascertain the shear strength characteristics. The same test procedure as before was used in the exercise. The results are given in Table 5.21 together with results of other tests performed on these samples.

5.6.3 Permeability Tests on the Slag/Lime Mixture

The slag/lime mixture was subjected to permeability tests so as to ascertain the drainage of transport water that would be used to deliver the material to undergrounds stopes. The tests were done in line with the earlier stated procedure, immediately after cement addition so as to be able to capture the true permeability before any strength develops. The results are given in Table 5.21 together with results of other tests performed on these samples.

5.6.4 Slump Tests on the Slag/Lime Mixture

Slump tests were conducted on the slag/lime material to determine the pumpability and transportability or fluid characteristics of the mixture. The slump tests were done immediately after cement addition using a 300mm cylindrical cone. The results are given in Table 5.21 together with results of other tests performed on these samples.

5.6.5 Atterberg limits Tests on Slag/Lime Mixture

Atterberg Limits tests were undertaken on the slag/lime mixture in order to determine the range of moisture contents over which the material would exhibit consistence. It was observed in all the tests that the slag/lime mixture had no plasticity.

Table 5.21 Geotechnical Test Results of the Slag/Lime Mixture for Backfill Desirable Properties

Curing Period (Days)	**Uniaxial Compressive strength, UCS (kPa)**	**Density (kg/m^3)**		**Unit weight (kN/m^3)**		**Shear Strength Parameters**		**Young's Modulus, E, (MPa)**	**Void ratio, e**	**Permeability** Immediately after mixing (m/s)	**Slump** Immediately after mixing (mm)
		Bulk	Dry	Bulk	Dry	Cohesion, c, (kPa)	Internal angle of friction, φ				
7	84	1764.2	1389.4	17.3	13.6	53	25	41	0.31	**2.4×10^{-6}**	**151**
14	177	1889.1	1456.7	18.5	14.3	98	29	65	0.31		
21	182	1754.0	1463.3	17.2	14.4	122	31	69	0.26		
28	209	1818.5	1415.2	17.8	13.9	150	30	93	0.29		

Chapter 6

DISCUSSION OF RESULTS

6.1 Preamble

The Geotechnical tests and all the associated backfill research activities were carried out at KCM Nchanga Geotechnical Services Laboratory, a British System International (BSI) certified site. A copy of the certificate is shown in Appendix 09. All the Geotechnical test procedures were done in accordance with the British Standard Methods of Tests for soils for Civil Engineering purposes, BS 1377.

The main objective of this research project at Konkola Mine was to develop a suitable backfill material for safe and increased economic copper ore production from the current 2.4 million tonnes to 7.5 million tonnes per annum. To achieve this production milestone, the company embarked on the KDMP project (which was still underway at the time of report writing) which would access the orebody containing over 250 million tonnes copper ore at 4% contained copper, shown in Figure 6.1.

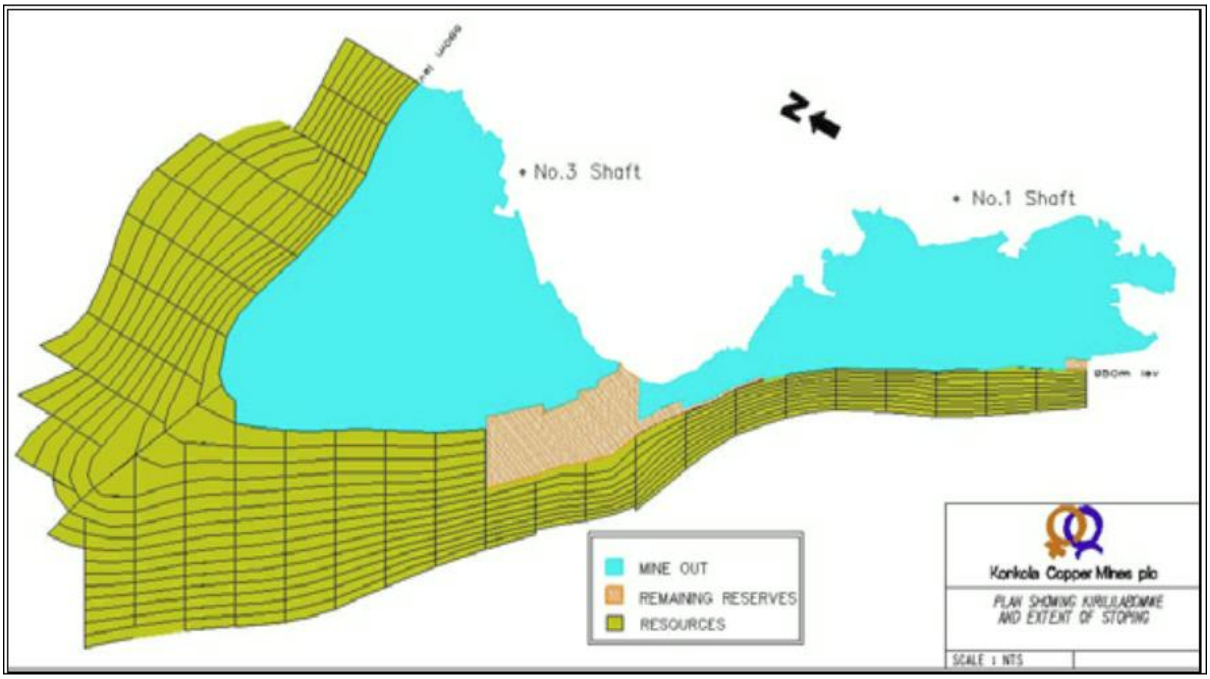


Figure 6.1 KDMP Ore Resources at Konkola Mine

The KDMP resources are below the current production levels of 950m at number No.1 Shaft. The success of the mining operations after the KDMP project will depend on the successful implementation of the composite Backfill System at Konkola Mine.

The increase in copper ore production referred to above will be at the premise of mining out all the copper ore from underground without leaving any pillars of ore standing for structural support, as is the case currently. The structural support will instead be provided by the engineered backfill material. In order to achieve this, a systematic ore extraction system has been proposed which will comprise two phases of extraction namely primary and secondary. In the primary phase, ore will be extracted leaving adjacent ore pillars to give structural support. In the secondary extraction phase, the voids created during the primary phase (called primary stopes) will be systematically filled with the engineered backfill material which would need to have enough strength and suitable drainage characteristics to resist failure due to self weight in the backfilled primary stope and therefore facilitate adjacent ore pillar recovery. The resulting voids due to mining of the adjacent ore pillars (called secondary stopes) will then be filled with the Hydraulic Fill material currently being produced at the old backfill plant. This process is expected to lead to increased copper ore production and permit extraction of high grade ore with minimum dilution. Figure 6.2 shows part of the plan of No.3 Shaft underground mine that is earmarked for the Primary/Secondary extraction system whilst Figure 6.3 is a section along a secondary Stope.

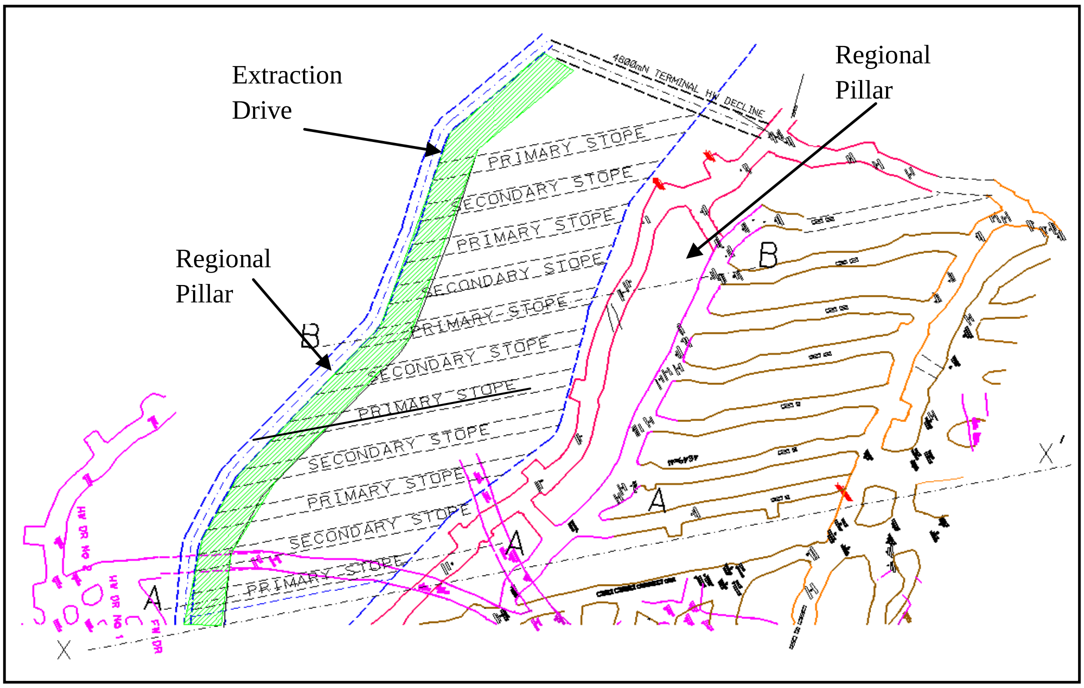


Figure 6.2 Plan of stopes to be mined out by Primary/Secondary extraction System

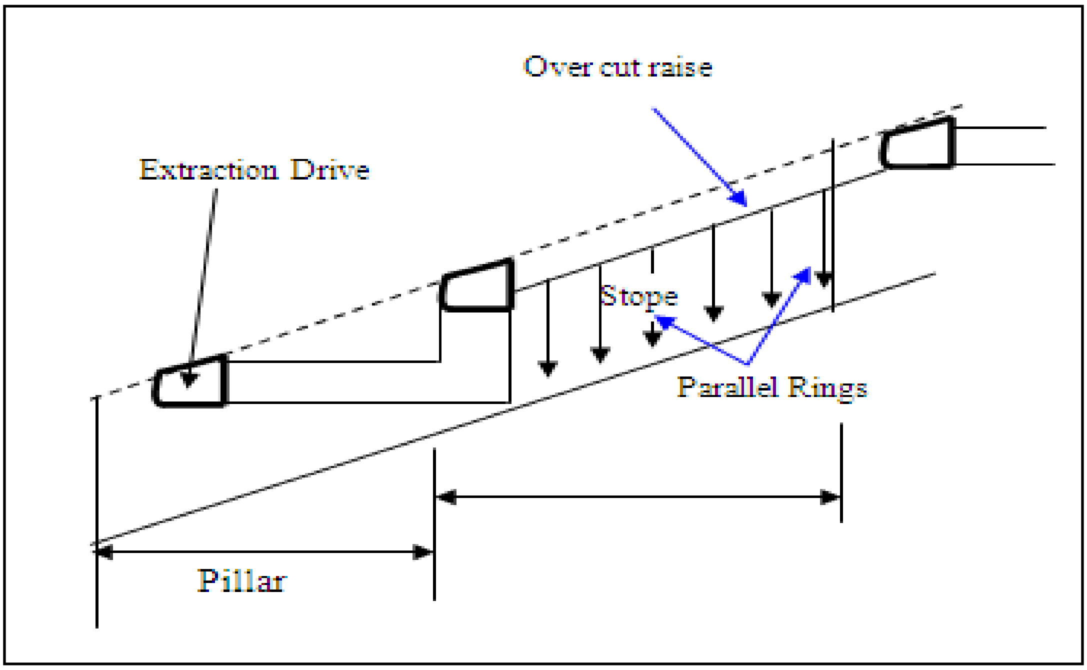


Figure 6.3 Section A-A along secondary stope

The secondary extraction phase will be done against backfill which would need to be vertical and not fail due to self weight and blasting operations. So in order to ensure that the backfill material is stable, a minimum Unconfined Compressive Strength of 1 MPa was specified for the material at a FOS of 3. The UCS was determined by Konkola Geotechnical Department using the Limit Equilibrium formula (33) and taking average height, *H*, and width, *B* of Konkola stopes as 20m and 100m, respectively.

$$UCS = FOS\,\gamma\, B \,/\, (1 + B/H) \qquad (33)$$

(Kuganathan, 2005)[12]

Where, γ = unit weight of backfill

H = Height of Stope

B= width of stope

Hence, $UCS = (3 \times 19.62 \times 100) / 6$

$= 0.981$

$=$ **<u>1 MPa</u>**

The following were key factors that mine management considered for adopting the high Factor of Safety (FOS):

- The cemented backfill would be applied using existing infrastructure at Konkola mine for hydraulic fill system (with some retrofitting) including all the associated reticulation system to underground stopes. It was difficult to predict with accuracy whether the quality of cemented fill material mixed on surface in the backfill plant will be consistent up to the stopes.
- Due to inexperience with cemented backfill system at Konkola mine, a lot of caution would have to be exercised during cement mixing in the chambers at the Backfill plant, hence the high FOS.
- Huge quantities of water will be used to transport the cemented backfill material to underground. This is in order to ensure that the material is kept fluid and does not block the 100mm pipe work. However, the existing pumps at the backfill plants are only able to pump material up to a maximum bulk density of $1.8 g/cm^3$. Hence the cemented backfill material must be carried in dilute mode up to the underground stopes. However, too much water affects the cement hydration process due to increased water to cement ratio.
- Possibility of some fine cement being carried off in the decant water from the cemented backfill material exists at the bulk heads. Therefore, to compensate for this inevitable loss, a high FOS was chosen.
- To prevent liquefaction of the cemented backfill material during blasting operations, a high factor of safety was selected. The minimum UCS required to prevent liquefaction is 100 kPa (Potvin, 2005)[14]. However, the fill material should not be very brittle as to be ruptured during blasting. The choice of 1 MPa was suitable as the Cemented Backfill material would not be ruptured during blasting.
- The high FOS and choice of 1 MPa will ensure cemented backfill material that would not be susceptible to re-saturation due to ingress of water.

It has been recognized by Kuganathan (2005)[12], that an uncemented backfill cannot form a vertical face and therefore would never be used for pillar recovery. The UCS test results for the Konkola Hydraulic Fill obtained as 58.4 kPa, is way below the required UCS of 1 MPa and could not form a vertical face and hence would never be used in the primary/secondary extraction system at Konkola Mine.

One of the key objectives in this research project was to determine the optimum amount of cement to be added to the backfill material to achieve a Uniaxial Compressive Strength (UCS) of at least 1 MPa and suitable drainage conditions, to facilitate the proposed Primary/Secondary extraction system.

6.2 Suitability of Uncemented Hydraulic Fill at No.3 Shaft Underground to provide Confinement.

The introduction of underground backfilling operations with uncemented classified mill tailings has led to a dramatic improvement of the geotechnical environment and increased ore production at No.3 Shaft mine. Prior to 1987, the mine was considered uneconomical due to poor ground and highly unsafe working environments, especially in the famous ‘nose’ area until the possibility of using backfill system was proposed in 2001.
A large extent of No.3 Shaft underground rock mass falls within *fair* to *poor* ground with average Rock Mass Rating of 30 to 55. Summary of the Ore shale Rock mass properties (including hanging-wall and the foot-wall formations) for the geological zones at number No.3 Shaft, obtained from face mapping and borehole logging, are given in Appendix 10. The poor ground situation has been attributed to the Kirila-Bombwe Anticline. The orebody at No.3 Shaft lies across the axis of the Kirila-Bombwe Anticline. The metasediments are draped around these structures in relatively gentle folds with some disruption by faulting and these have given rise to the poor ground conditions discussed above.

The value of **φ** obtained for the classified mill tailings during triaxial tests at Geotechnical Services Laboratory, presented in Table 5.3 was very high (44^o). This value was found responsible for the improvement of the confining effect of the fill at No.3 Shaft. This ultimately, led to the improved strength characteristics of rock mass at No.3 Shaft underground, especially in the MOCB mining operations in the ‘nose’ area.

The high value of **φ** was attributed to the ore blasting underground and communition that the material undergoes in the metallurgical process at Konkola Concentrator which tends to produce very angular and rough slurry surfaces. The mechanical properties of the fill material such as particle shape, size and size distribution are highly influenced by the physical nature of the solid particles, and the way they fit together. Ore output has increased since the introduction of hydraulic fill in 2001 due to the increased possibility of leaving slender pillars of ore for structural support. The increased load carrying capacity of the slender pillars has been due to the lateral confining effect of uncemented hydraulic fill.

6.3 Suitability of Uncemented Classified Mill Tailings as working Platform at No.3 Shaft Underground

In mining operations at No.3 Shaft underground requiring Cut and Fill mining methods and its associated variations such as PPCF. The hydraulic fill material is required to be used as a working platform for mining equipment and personnel. The bearing capacity of the drained hydraulic fill material placed at No.3 Shaft was adequate to support the LHD trucks underground. This scenario is easily understood by using the Terzaghi bearing capacity theories related to design of shallow foundations summarized in the bearing capacity equation given below:

$$q_f = 0.4\gamma B N_\gamma + 1.2 C N_c + \gamma D N_q \quad (33)$$

(Craig, 2005)[1]

Where,

q_f = surface bearing capacity, kPa

C = cohesive strength of backfill, kPa

γ = bulk density of backfill, kN/m3

B = width of square footing at surface contact position, m

D = depth of footing penetration into fill, m

N_c = cohesion bearing capacity factor, dimensionless

N_γ = unit weight bearing capacity factor, dimensionless

N_q = surcharge bearing capacity factor, dimensionless

Figure 6.4 is a typical lay out of the PPCF mining method at Konkola Mine.

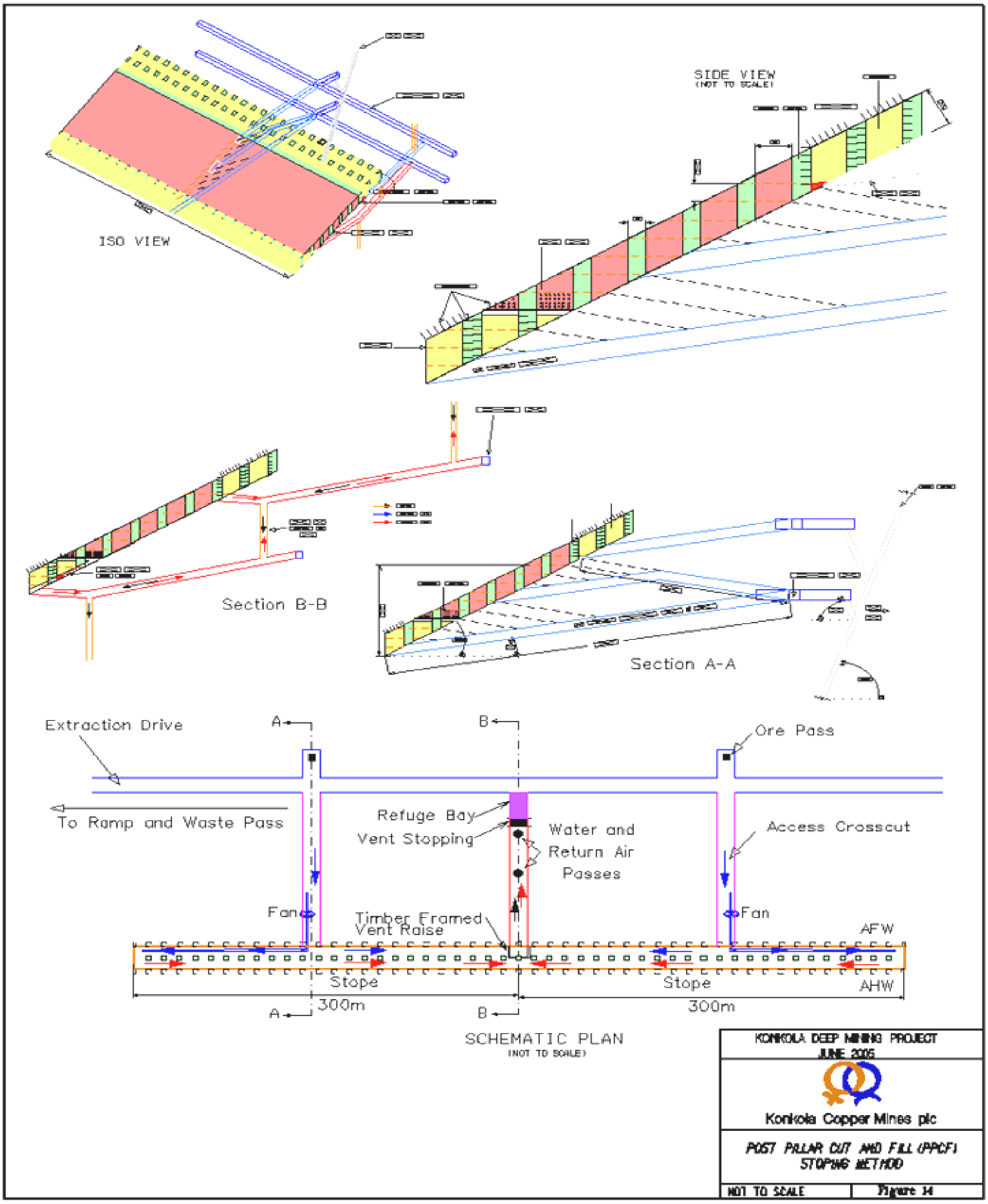


Figure 6.4 Post Pillar Cut and Fill (PPCF) mining method at No.3 Shaft underground mine

The above equation assumes that the backfill is supporting an object having a square footing, which is a reasonable representation of the footprint of a mine vehicle tyre, such as the LHD trucks. N_q, N_c and N_γ are bearing capacity factors that depend only on the measured angle of internal friction (φ) of the backfill material. These dimensionless coefficients characterize the bearing capacity of the fill. The values of the bearing capacity factors may be determined using the following expressions from Craig (2005)[1].

$$N_q = \exp(\pi \tan\phi)\tan^2\left(45^\circ + \frac{\phi}{2}\right) \tag{34}$$

$$N_C = \left(N_q - 1\right)\cot\phi \tag{35}$$

$$N_\gamma = 1.80\left(N_q - 1\right)\tan\phi \quad \text{or} \tag{36}$$

$$N_\gamma = \left(N_q - 1\right)\tan(1.4\phi) \tag{37}$$

Where,

φ = backfill internal angle of friction,

The tyre contact width, B, can be determined as:

$$B = \sqrt{\frac{L_T}{P_T}} \tag{38}$$

Where,

L_T = tyre loading force, kN

P_T = tyre air pressure, kPa

From the above equations, it is important to note that the higher the backfill friction angle, **φ**, the higher the bearing capacity of the hydraulic fill material.

The following bearing capacity calculations demonstrate the adequacy of the hydraulic fill material to support the LHD trucks at No.3 Shaft.

- Friction angle, ***φ*,** for HF material = 44^o
- Cohesion, *c*, = 30 kPa
- γ_{dry} = 14.4 kN
- P_T (LHD tyre air pressure) = 345 kPa (Data obtained from No.3 Shaft Mechanical Engineering Department)
- L_T, tyre loading force, the unit's greatest weight resides on the two tyres nearest the bucket, at 91.63 kN overall (Data obtained from No.3 Shaft Mechanical Engineering Department)

- D = 0.00m (assuming no depth penetration by the LHD tyres)
- B = 0.36m
- N_q = 120
- N_c = 120
- N_γ = 210

Using the Terzaghi bearing capacity equation,

$$q_f = 0.4\gamma B N_\gamma + 1.2 C N_c + \gamma D N_q$$

q_f = 0.4 x 14.4 x 0.36 x 210 + 1.2 x 30 x 120 + 14.4 x 0.0 x 210

= 4,755 kPa

Taking Factor of Safety as 3, the Ultimate bearing Capacity, qf = **1,585 kPa**

Therefore, the hydraulic fill material could support a stress of **1,585** kPa. Since this is considerably more than the maximum force exerted by the equipment tyres (i.e. 345 kPa << 2,824 kPa) no failure could occur. The high friction angle, **φ,** determined in the lab is responsible for the observed ability of the material to support the load stresses from the LHD trucks.

6.4 Percolation of Uncemented Classified Mill Tailings (Hydraulic Fill)

It was discovered during the research that percolation rates at No.1 Shaft were higher than those at No.3 Shaft owing to the attrition that the material undergoes as it is pumped from No.1 Shaft to No.3. This was due to the increase in fines as was verified by the particle size analysis of the fill material from both No.1 Shaft and No.3 Shafts. The average values of percolation for hydraulic fill material at No.3 Shaft, however, are within the global standards of about 100mm/hr. For efficient production schedules at No.3 Shaft, the ideal percolation rate for the hydraulic fill material in the stopes was found as 80mm /hr. The consequence of lower permeability or percolation rates would be restrictions on the placement and drainage of the placed fill and susceptibility to long phases of saturation which would put pressure on the Bulk heads underground. This way, bulkheads would be expensive to construct as they would require more material and cement for them to resist high hydraulic pressures from the fill within a stope.

Percolation tests are routinely conducted on the hydraulic fill material at both the backfill plant at No.1 Shaft and the backfill holding tanks at No.3 Shaft to ensure correct quality of fill material.

6.5 Developed Backfill Material

Table 6.1 shows the six backfill materials that were developed at Geotechnical Services Laboratory assessed against the desirable backfill properties given in Table 2.1 in Chapter 3 page 13.

Table 6.1 Developed Backfill Materials at Geotechnical Services Laboratory

No	Developed Backfill Materials
1	***Cemented Hydraulic Fill*** material developed using ***Mphamvu*** Cement from Lafarge Cement Company.
2	***Cemented Hydraulic Fill*** material developed using ***Zambezi*** Cement from Zambezi Cement Company
3	***Cemented Aggregate Fill*** material developed using ***Mphamvu*** Cement from Lafarge Cement Company.
4	***Cemented Aggregate Fill*** material developed using ***Zambezi*** Cement from Zambezi Cement Company
5	***Cemented Aggregate Fill*** material developed using ***Minova Fast Set*** Cement from South Africa
6	***Slag and Lime Mixture*** with self cementing properties

In addition to the desirable properties of backfill, the developed backfill materials above were also assessed for suitability to fit into the production schedules at No.3 Shaft underground. A typical production panel at No.3 Shaft underground, shown in Figure 6.5 would be depleted of copper ore in 30 days using the MOCB mining method.

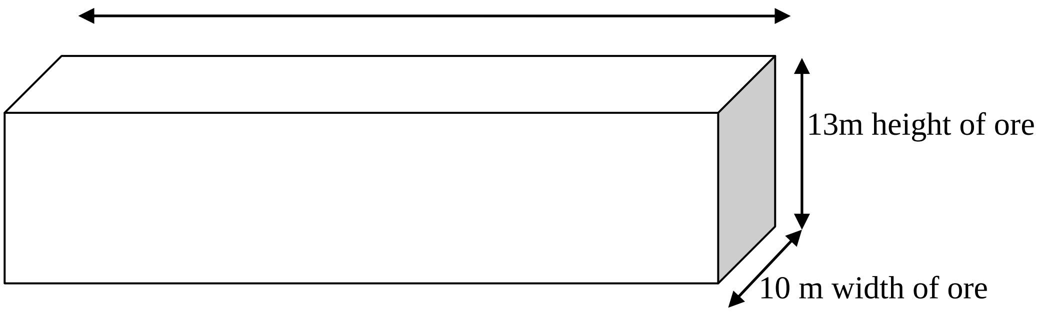


Figure 6.5 Typical Production Panel at No.3 Shaft underground

This is the panel that will become the primary stope and would be filled with the cemented Backfill material afterwards.

6.5.1 Uniaxial Compressive Strength and Associated Cost of Cement in the Developed Backfill Material

Table 6.2 shows a detailed comparative analysis of the cement contents in the different developed backfill material required to attain the prescribed UCS of 1 MPa and the associated cement costs.

Table 6.2 Cement Contents in the Backfill Material and the Associated Costs

No	Developed Backfill Materials	Cement Content required to attain 1 MPa	Mass of Cement required to fill a primary stope (tonnes)	Cost of Cement	Cost of Cement in a Typical Primary Stope
1	CHF - Mphamvu Cement	14%	3570	K 1.08 million/ton	K 3,856 million
2	CHF - Zambezi Cement	12%	3107	K 1.1 million/ton	K 3,418 million
3	CAF - Mphamvu Cement	6%	1673	K 1.08 million/ton	K 1,807 million
4	CAF - Zambezi Cement	4%	1064	K 1.1million/ton	K 1,170 million
5	CAF - Minova Fast Set Cement	4%	1181	K 7 million/ton	K 8,267 million
6	Slag /Lime mixture	Self Cementing			

*Cement Price from KCM Commercial Department, April 2011

The cement contents in the Cemented Hydraulic Fill prepared from both Mphamvu and Zambezi cement brands were deduced from UCS graphs given in Figures 5.7 and 5.8, respectively. The UCS test results obtained after testing the 6 developed Cemented Backfill materials were subjected to statistical analysis to assess reliability of laboratory test data.

In all the test results, the Coefficients of Variation were less than 10%. As a rule, Coefficients of Variation which are less than 10% are considered to be low; therefore the UCS test results were deemed to be very reliable.

Sample calculation of cost of cement required for filling a primary stope with Cemented Hydraulic Fill using Mphamvu Cement

- Volume of a typical Primary Stope to be backfilled =Length x Height x Width

  =100 m x 13 m x 10 m

  =13,000 m^3

- Average Bulk Density of Cemented Hydraulic Fill with Mphamvu = 1961.7 kg/m^3
- Mass of Cemented Hydraulic Fill with Mphamvu = 13,000 x 1961.7 = 25,502.1 tonnes
- Since 14% would be required in the material to attain 1 MPa, this translates to ***3,570*** tonnes of Mphamvu cement.
- The current cost of Mphamvu cement K 1.08 million per ton. Therefore the cost of cement for a complete stope is *K3, 856 million*

The mixture of slag and Lime only attained a maximum of 209 kPa after 28 days. Long periods of time would be needed (over 130 days) for the mixture to attain a UCS of 1 MPa. This is clearly beyond the normal production schedules of 30 days at No.3 Shaft. The mass of lime required in the primary stope is 7,827 tonnes. The current cost of lime is K 500,000/ton. So the total cost of backfilling with the Slag/lime mixture would be K3, 913.5 million. The cost of reclaiming the slag from the Slag Dump will be borne by the existing operational budget at Konkola Mine.

Minova Fast Set Cement has the advantage of attaining 1 MPa in 3 days. However, according to the sequencing of mining of typical production panels and the required production scheduling of 1 month, there is no benefit to be gained if the material cures quickly. For maximum economic benefit, the depletion of ore in a primary ore panel must coincide with the curing of cemented backfill in the primary stope in another production block. Due to the fast setting characteristics of Minova Cement, the Backfill reticulation system and its associated net work of pipes, risks blocking with the backfill material prepared from Minova Cement. Furthermore, the cost of Minova Cement from South Africa is very prohibitive.

The required UCS value of 1 MPa stipulated for Konkola backfill material was attained by the Cemented Aggregate Fill material at comparatively low cement content of 4% using the Zambezi cement brand. The Zambezi Cement brand was discovered to offer slightly better strength gain (about 1.5 times) compared to Mphamvu cement.

The high UCS values exhibited by CAF was as a result of blending 30% hydraulic fill with 70% crushed waste rock from around No.3 Shaft waste dump.

The maximum size of the crushed material of 2.83mm used to produce the Cemented Aggregate fill material was a very suitable choice due to the following reasons:

- The Cemented Aggregate Fill material was found to be densely packed (average Bulk Density of over 2100 kg/m^3) and well graded. The well gradedness was reflected in the n value of 0.5 that was found after subjecting the fill material to the power law equation, $\boldsymbol{P = 100 \times [d/dmax]^{0.5}}$**.** It has been found by experience in the concrete industry that when the power law exponent is 0.5, the optimum particle size distribution is obtained, resulting in a densely packed fill material
- The blended fill material was subjected to a particle size distribution test and results from the plotted graph showed that the Coefficient of Uniformity (C_u) and Coefficient of Curvature (C_c) values were 22 and 1.05, respectively. This again demonstrated that the fill material was carefully selected to produce a well graded mixture with a long range of particle sizes. In soil mechanics principles, materials with a larger C_u, form a well-packed deposit and can develop high strength with relatively smaller amount of binder. A smaller C_u represents a narrower spread and uniform material with large void spaces between the particles requiring higher binder addition to develop similar strength. A value of C_c between 1 and 3 shows a well graded material.

6.5.2 Shear Strength

The Cemented Aggregate Fill exhibited high shear strength characteristics as seen from the high average values of friction angle, **φ,** and cohesion, c, obtained during triaxial tests. The addition of binders to fill helps to improve shear strength by developing cohesion and also altering the frictional characteristics of the particle surfaces. Cementation changes the surface texture of particles and hence increases frictional resistance.

6.5.3 Permeability

The permeability of Cemented Aggregate Fill immediately when 4% Zambezi cement is mixed to give a UCS of 1 MPa was found as 2.3 x 10^{-5} m/s (82.8mm/hr). This was generally within acceptable limits required at No.3 Shaft underground to ensure drainage of transport water from the backfilled stopes. However care would need to be exercised to ensure that minimal amounts of cement fines are not washed out together with the draining water.

Laboratory test results conducted in the past at KCM revealed that addition of cement to hydraulic fill reduces the permeability to as low 1.3 x 10^{-6} m/s from 2.4 x 10^{-5} m/s in the un-cemented hydraulic fill. The consequence of lower permeability or percolation rates would be restrictions on the placement and drainage of the placed fill and susceptibility to long phases of saturation which would put huge hydraulic pressures on the Bulk heads underground.

6.5.4 Slump

The value of slump for Cemented Aggregate Fill material immediately when 4 % Zambezi cement is mixed to give a UCS of 1 MPa was found as 160mm. In Civil Engineering concrete mix design, this would be considered material with very high workability. In Cemented Backfill Technology, the material falls within a consistence that would easily be pumped. The slump of cemented backfill is related to the Yield stress (Yield stress is the critical shear stress of backfill material that must be exceeded before irreversible deformation and flow can occur). It was very vital to ensure that pumps with the correct specifications are selected and purchased at the backfill plants at Konkola Mine which must overcome the yield stresses in the backfill materials. The *ABEL Positive Displacement Pump*, rated at 90m^3/hr at a maximum feed pressure of 4.5 MPa, shown in Picture 6.1 was found suitable for the purpose.

Picture 6.1 ABEL Positive Displacement Pump for pumping Cemented Aggregate Fill Material

The results of the slump tests above have indicated that the mix design ratio of 3:7 of hydraulic fill and crushed aggregates, respectively, was appropriate. This is because, despite the high water content (and subsequently high water to cement ratio) required to bring about high fluidity of the cemented fill material (so as to facilitate pumpability and transportability) the material was still able to attain the required UCS of 1 MPa.

6.5.5 Mineralogical Examination

One of the variables that impact on Strength development is presence of sulphates in the fill that tends to attack Portland cement. However, the mineralogical tests conducted on both the waste material at No.3 Shaft used to produce aggregates and the classified tailings show very negligible amounts of sulphates (less than 0.6%) and these would not appreciably affect the hydration process of cement.

6.6 Economic Benefits of using Cemented Aggregate Fill at Konkola Mine

Economic benefits which would arise from use of Cemented Backfill at Konkola Mine were thoroughly analyzed. No.3 Shaft underground mine was considered because the cemented backfilling operations will be started from there and rolled out to the rest of the underground operations at Konkola Mine. Table 6.3 shows the current full breakdown of cost elements at No.3 Shaft from where the Cost of Production of $ *1,764*/tonne was calculated for April 2011 at 80% extraction ratio. These costs are for mining operations with uncemented backfill.

Table 6.3 No.3 Shaft Cost Report – April 2011

Production	Actual	Budget	Variance
Ore Delivered to Mill (t)	80553	90000	(9447)
Total Contained Copper Tcu (t)	1891	2261	(369)
Details	**Actual Costs($)**	**Target Costs($)**	**Variance($)**
7100010 Consultant Fees	0	5,000	5,000
7100020 Equipment Hire	114,686	143,100	28,414
7100030 Labour Hire	(367)	0	367
7500010 Currency Gains/Los	(375)	0	375
7300000 Electricity -	121,726	156,729	35,003
7600000 Civil Stores And Stores	32,772	26,861	(5,911)
7600010 Electrical Stores	28,783	83,078	54,295
7600020 Heavy Vehicles Stores	169,574	94,595	(74,979)
7600030 Instrumentation Stores	6,125	1,836	(4,289)
7600040 Mechanical Stores	182,644	60,685	(121,959)
7600050 Laboratory Equipment	2,498	893	(1,605)
7600060 Operating Consumable	672	6,451	5,779
7600090 Maintenance Service	272,289	207,000	(65,289)
7600110 Cash Purchase-Spares	74	0	(74)
7700000 COP A Contractor	34,163	72,500	38,337
7800000 Explosive& Accessory	34,606	22,823	(11,783)
7800030 Chemicals	0	1,421	1,421
7800050 Diesel & Petrol	115,582	43,560	(72,022)
7800060 Drilling Material	20,253	21,565	1,312
7800080 Lime	0	0	0
7800100 Pump And Pump Spar	18,849	8,185	(10,664)
7800120 Tyres	184,554	58,807	(125,747)
7800130 Water And Sewerage	36,452	54,215	17,763
7800150 Freight	49	0	(49)
7800190 Mine Development	978,885	1,424,687	445,802
7800220 Grease & Lubricant	152,327	31,744	(120,583)
8100020 Foodstuffs	31,225	19,503	(11,722)
8100060 Printing And Station	641	2,792	2,151
8100070 Protective Clothing	0	0	0
8100090 Safety & Security	22,308	23,267	959
8100100 Admin Non-Production	12,098	29,692	17,594
8200150 Sundry Expenses	10,148	300	(9,848)
8200090 Entertainment	0	0	0
8200170 Training	13,277	10,000	(3,277)
8200180 Travel Expenses	4,046	3,000	(1,046)
8300000 Allowances	36,483	119,670	83,187
8300010 Basic Pay	795,733	612,662	(183,070)
8300020 Bonus	86,080	61,999	(24,081)
8300040 Employee House All	356,120	281,821	(74,300)
8300050 Fixed Term Labour	(149,067)	0	149,067
8300080 Market Supplement	697	0	(697)
8300090 O/Heads	159,323	115,500	(43,823)
8300100 O/Time - Normal	288	0	(288)

8300110 O/Time - Sunday	19,007	96,251	77,244
8300120 Shift Differential	49,389	44,487	(4,901)
8300130 Sick Pay	7,308	0	(7,308)
8300140 Standby Allowance	10,421	14,825	4,404
8300160 Transport Allowance	362	25,430	25,068
8500010 It Services	0	0	0
Total	**3,972,708**	**3,986,934**	**14,225**
Per tonne of Ore Delivered to Mill ($)	49	44	(5)
Per tonne of Cu in Ore Delivered To Mill ($)	2,101	1,764	(337)
c/pound of Cu in Ore Delivered To Mill	**95.29**	**80.00**	**(15)**

Source: No.3 Shaft Mining Engineering Department and KCM Commercial Department

The above cost profile was re-analyzed to include a projection of costs associated with mining with Cemented Aggregate Fill (with 4% Zambezi Cement added) as shown in Table 6.4. This was done with the help of KCM Commercial Department. The aim of mining with Cemented Backfill is to attempt increasing extraction ratio from 80% to 100%

Table 6.4 No.3 Shaft Re-calculated Cost Report taking into account the Cemented Backfilling Operations

Production	**Current Operations**	**Operations with Cemented Backfill**
Ore Delivered to Mill (t)	90000	112500
Total Contained Copper Tcu (t)	2261	2826
Details	**Current Costs ($)**	**Costs after introduction of Cemented Backfill ($)**
7100010 Consultant Fees	5,000	6,000
7100020 Equipment Hire	143,100	180,000
7100030 Labour Hire	0	0
7500010 Currency Gains/Los	0	0
7300000 Electricity	156,729	200,000
7600000 Civil Stores And S	26,861	26,861
7600010 Electrical Stores	83,078	100,000
7600020 Heavy Vehicles Stores	94,595	100,000
7600030 Instrumentation Stores	1,836	2,000
7600040 Mechanical Stores	60,685	70,000
7600050 Laboratory Equipment	893	893
7600060 Operating Consumables	6,451	10,000
7600090 Maintenance Service	207,000	300,000
7600110 Cash Purchase-Spar	0	0
7700000 Cop A Contractor	72,500	72,500
7800000 Explosive& Accessories	22,823	33,100
7800030 Chemicals	1,421	1,421

7800050 Diesel & Petrol	43,560	51,000
7800060 Drilling Material	21,565	21,565
7800080 Lime	0	0
7800100 Pump And Pump Spar	8,185	10,101
7800120 Tyres	58,807	65,000
7800130 Water And Sewerage	54,215	57,500
7800150 Freight	0	0
7800190 Mine Development	1,424,687	1,424,687
7800220 Grease & Lubricant	31,744	33,200
8100020 Foodstuffs	19,503	20,200
8100060 Printing And Stationery	2,792	2,792
8100070 Protective Clothing	0	0
8100090 Safety & Security	23,267	23,267
8100100 Admin Non-Production	29,692	29,692
8200150 Sundry Expenses	300	300
8200090 Entertainment	0	0
8200170 Training	10,000	10,000
8200180 Travel Expenses	3,000	3,000
8300000 Allowances	119,670	119,670
8300010 Basic Pay	612,662	650,000
8300020 Bonus	61,999	61,999
8300040 Employee House All	281,821	281,821
8300050 Fixed Term Labour	0	0
8300080 Market Supplement	0	0
8300090 O/Heads	115,500	113,000
8300100 O/Time - Normal	0	0
8300110 O/Time - Sunday	96,251	98,000
8300120 Shift Differential	44,487	44,487
8300130 Sick Pay	0	0
8300140 Standby Allowance	14,825	14,825
8300160 Transport Allowance	25,430	25,430
8500010 It Services	0	0
TBA Cost Of Cement For Backfilling	0	243,750
TOTAL	**3,986,934**	**4,508,061**
Per tonne of Ore Delivered To Mill ($)	44	40
Per tonne of Cu in Ore Del To Mill ($)	1,764	1,595
c/pound of Cu in Ore Del To Mill	**80.00**	**72.36**

Copper ore production would go up to 112,500 tpm if extraction ratio increases to 100% from 80%. The total cost of operations including cost of Zambezi Cement, would go up to $4,508,061 from the current $3,986,934. The additional cost of cement would represent **5.4%** of the total operational costs.

It is clear from the results in Table 6.4 that mining operations with Cemented Aggregate Fill would reduce the Cost of Production by **10%** at No.3 Shaft assuming 100% extraction.

6.7 Environmental Aspects of Konkola Backfill System

The KDMP mining operations will have minimal effects on subsidence and surface collapse induced by under-mining in the future due to the new mining methods with backfill to be employed and the increased depth of mining. The proposed use of backfill in the project, in particular, should reduce surface subsidence significantly because the orebody is thin and flat. A lot of areas within the Konkola Mine licence area, especially around the nose area, have been barricaded off due to caving that has occurred in the past.

The impact of backfill operations at Konkola mine on both surface and ground water cannot be underestimated. Currently, overflow tailings from the cycloning activities at the backfill plants are pumped to Lubengele Dam a distance of 10km, shown in Picture 6.2.

Picture 6.2 Lubengele Tailings Dam

The effluent from the dam which flows to Kafue River a distance of 4.5km south of Konkola Mine is routinely checked for hazardous materials such as TSS and chemicals which come from the metallurgical plants, shown in Table 6.5.

Table 6.5 Hazardous Materials from Lubengele Tailings

Parameter	Aug 2010	Sept 2010	Oct 2010	Nov 2010	Dec 2010	Jan 2011	Feb 2011	Mar 2011	Apr 2011	Statutory Limit in ppm
TSS	20	32	20	32	21	20	26	62	143	100
TDS	4,850	4,913	2,982	493	2,936	3,434	2,898	2,624	3,535	3,000
pH	6.0	7.6	7.5	7.9	7.6	7.5	7.7	7.9	7.4	6.5 - 9.0
Mg	322	445	423	348	222	256	230	224	497	500
Ca	533	578	1298	578	569	645	234	442	1176	100

Transport water flowing from either the uncemented or cemented backfilled stopes will be pumped to the settling ponds at 691m level at No.3 Shaft underground and later transferred to the other settling ponds at 960m level at No.1 Shaft underground in order to remove the suspended solids before it is pumped out of the mine.

6.7.1 Potential Environmental Impacts and Mitigation of New Waste Rock Crushing and Milling Plant at Konkola Mine

The new Waste Rock Crushing and Milling Plant also called KCM 3 BF Plant will produce 70% of the total backfill material requirement at Konkola Mine and will be the primary backfill producing facility to meet the demand as KCM ramps up to 7.5 million tonnes production rate. This plant, which was still under construction at the time of report writing, shown in Picture 6.3 will produce coarse aggregate material by way of crushing the waste rock from the nearby existing waste dumps at No.3 Shaft.

Picture 6.3 New Waste Rock Crushing and Milling Plant

The Particle Size Distribution (PSD) of the aggregate material that must come out of the plant in Figure 6.7 above was recommended by this research scholar after conducting laboratory optimization tests mentioned earlier.

The following are potential Environmental impacts and mitigation measures of the plant on various environmental elements.

1) Air Quality

It is unlikely that the operations at the plant will have a noticeable impact on air quality beyond the project boundaries. However, appropriate mitigation measures have been developed to protect employees from being affected by the dust such as use of dust masks.

- **Dust from crusher and grinding processes**

Jaw Crushers will be used for crushing of waste rock into smaller pieces down to the size of 2.83mm. This process would release dust particles in surrounding areas, and thus provisions are needed for effective removal of these pollutants locally.

<u>Mitigation</u>

The control of dust emissions from these processes is mainly by the use of dust suppression systems. Crushers will be totally contained or fitted with a water suppression system over the crusher aperture and inter-linked with water flow detectors so that they cannot operate unless water supply is operational.

- **Dust from Stockpiles**

Crushed material before being fed into the next process will be temporally stored. This, if not controlled, would be a potential source of fugitive dust around the plant.

<u>Mitigation</u>

To control dust emissions from stockpiles, storage bays will be used and water sprays will be fixed. This will ensure that stockpiles are wetted to minimise dust emissions.

- **Conveyors for material**

Conveyers will be used to transport crushed material from one stage of the process to the other, and dust will be generated in the process.

<u>Mitigation</u>

Conveyors will be designed to minimise dust emissions at discharge points and will be of sufficient capacity to handle maximum loads without spillage. Conveyors will be provided with adequate protection against wind whipping.

- **Dust from haul roads**

Transportation of raw material from No.3 Shaft waste rock dump would result in dust generation and this will have an effect on the people around the plant. The hauling distance will be very short and thus its effect will be kept to a minimum. However, the greatest long-term health hazard of dust generated from hauling operations will be due to inhalation by haul trucks' operator.

Mitigation

Dust generation from the movement of haul trucks and other heavy equipment on the mine dumps will be suppressed by routine spraying with water. Operators of the hauling trucks will be provided with the dust masks and also subjected to regular medical checks at Silicosis Medical Centre in Kitwe.

- **Water Resources**

Since Konkola Mine is one of the wettest mine in the world, underground dewatering discharge will be used for production of backfill. For cooling purposes in milling and other processes, domestic water will be supplied by a local utility company, Mulonga Water and Sewerage Company. Domestic supply is sourced from both the Kafue River and the underground dewatering discharge.

Mitigation

Water conservation measures will be promoted so as to contribute to the sustainable management of water resources. Water meters will be installed on all pipelines carrying water to the plant. Clear monitoring procedures for pipe line leakages will be introduced to reduce wastage through leakages.

- **Soil Contamination**

Poor handling, storage and transport of oil, diesel and grease may lead to spillages which may contaminate exposed soils. Breakdowns of process plant equipment will occur occasionally and spillages of oil and lubricants may contaminate soils. Accidental spills of material from leaks in the backfill delivery pipeline may contaminate soil.

Contamination of soils may occur due to the blowing of dust from the crushing and grinding circuits, material stockpiles and conveyors. Contaminated soils may impact on surface water, flora and fauna.

Mitigation

Localized soil contamination resulting from the accidental spill of oil, diesel and grease will be treated by the removal of contaminated soil from affected areas to an appropriate disposal facility already in place. To prevent soil contamination new and used batteries will be stored in accordance with specifications in the company's existing Materials Handling Procedure. Furthermore, oil will only be stored and handled in designated areas and oil storage areas will be equipped with impervious surfacing and containment.

- **Waste rock erosion Control**

The No.3 Shaft Waste Rock Dump located adjacent to the No.3 Shaft, covers an area of roughly 11 000m^2 and until recently received approximately 400 tpd in addition to the 4.3 million tonnes already dumped. Currently approximately 125 tpd of waste rock is being taken to the dump.

The dump overlies caving ground but appears not to have been seriously affected by subsidence. The top of the dump is generally well leveled and drained, although minor ponding occurs in places. The embankments are stable and in good condition. However, areas being re-claimed will be exposed which might lead to run-off carrying minor silt. There are no signs of erosion from the dump.

According to the approved Final Environmental Management Plan produced for KCM, no evidence of acid drainage or acidic conditions has been identified from any of the waste dumps on site to date. Acid base accounting (ABA) on a limited set of samples has indicated that waste rock from No.3 Shafts is not acid generating and would be expected to produce neutral to alkaline drainage.

6.8 Risk Assessments and Safety Aspects of Konkola Backfill System

A thorough risk assessment was undertaken for the composite Konkola Backfill System starting from the backfill production plants all the way to the backfilled stopes underground, including the new Waste Rock Crushing and Milling plant. Several safety aspects where identified and these are listed below.

- **Surface Backfill Pipe line**

The surface pipe line is operated at high pressure, exceeding 4 MPa at the pumping end and as such, presents a hazard to persons who gain unauthorized access to the mine in case of pipe burst. The route takes the line close to the township area and it is crossed by a number of well used foot paths. The line is fully accessible and at risk of fire, theft, and in places, vehicle damage; with high pressure risk to any people involved or in the immediate area.

<u>Mitigation</u>

Steps must be taken to keep people away and this may involve fencing. Due to high risk of being stolen, the pipes must be clamped with specialized bolts. The line is very close to the road at the exit point to the number No.1 Shaft area and is in need of protection from vehicle damage.

- **Accidental Spillage**

The operations at the backfill plants may sometimes spill either the underflow or overflow. Adequate catchment ponds have been provided. Spillage from the backfill pipe lines should not run into the adjacent drains and any spillage must be swiftly detected and addressed.

- **Communication**

Communication between the crews on surface at the backfill plants and underground must be enhanced to ensure smooth running of the backfilling operations. It would also help in preventing excessive delays in the commencement of filling a stope. This will also allow adequate drainage time during the stope filling cycle. Failure to establish a good communication system would lead to accidents which would be fatal and expensive to mitigate.

- **Bulk Head**

- A lot of operational care and safety must be exercised to prevent accidental leakage of fill material from Bulk Heads or its immediate surroundings (including rock bolt hole and drill holes). Any Bulk Head failure may lead to injuries or fatalities.
- Care must be taken to allow placement of fill that complies with specifications (e.g. placement pulp density, particle grading curve and in-situ strength. Violation of the above specifications may lead to long phases of saturation which may endanger the stability of the bulk head.
- Fill delivery must not be behind schedule. It may lead to the catastrophic failure of the rib and crown pillars.
- Water must not be allowed to pond on top of the fill for longer than a specified period. It may lead to fill inrush.

The three backfill plants constructed at Konkola Mine will work in synergy to ensure realization of an efficient composite backfill system which is poised to facilitate the primary secondary extraction system and deliver the much anticipated increased ore production from the current 2.4 million tonnes to 7.5 million tonnes. A conceptual layout of the Konkola Composite Backfill System is given in Figure 6.6.

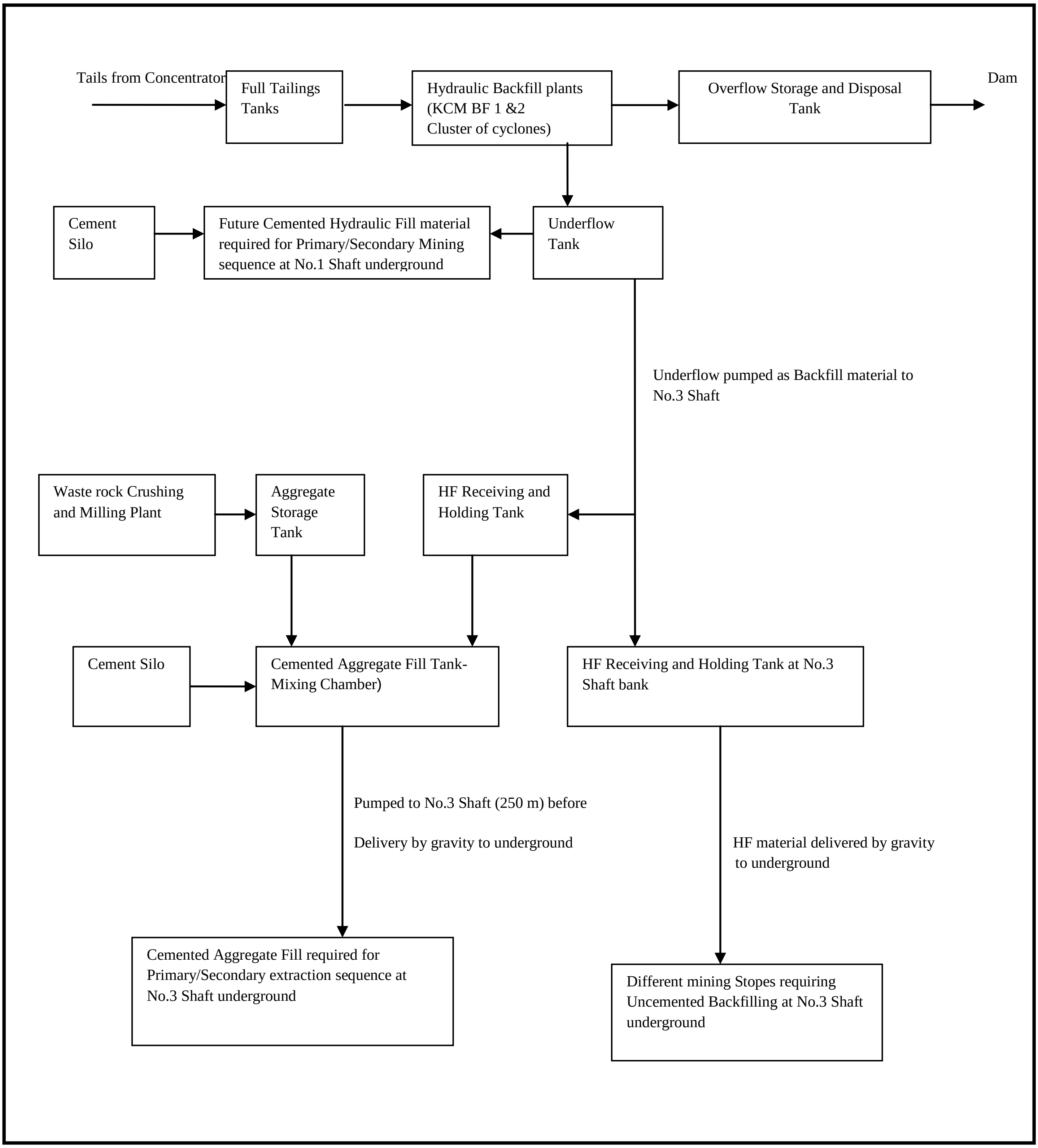


Figure 6.6 Konkola Composite Backfill System - Conceptual Flow Diagram

Chapter 7

CONCLUSIONS AND RECOMMENDATIONS

The objectives of the backfill research project were to develop a suitable backfill material for safe and economic ore production at Konkola Mine and have been achieved. The conclusions have been established on the basis of geotechnical, economic and environmental test results and risk assessments related to the Konkola backfill system.

The suitable backfill material needed to facilitate total ore pillar recovery from underground in the proposed primary/secondary extraction system is the blend between the classified mill tailings currently used as hydraulic fill (produced at 65% solid fraction with an average size of 0.08mm at the old backfill plant) and waste rock crushed to 2.83mm aggregates from No.3 Shaft dump. The blend ratio must be 3:7 by weight of the hydraulic fill and the aggregates, respectively, with addition of 4% Zambezi Portland cement. This material, mixed in the mixing chambers at the new Waste Rock Crushing and Milling Plant on surface, must be pumped to the various stopes underground at Konkola Mine and allowed to cure for not less than 28 days to attain the recommended uniaxial compressive strength of 1 MPa.

Due to the good confining effect that the current uncemented hydraulic fill has demonstrated, it was concluded that the material will be very useful in filling the secondary stopes after the primary stopes have been filled with the engineered backfill material in the future primary/secondary extraction system proposed for mining operations commencing after the KDMP project. The uncemented hydraulic fill material would also be used as a working platform in mining methods such as Cut and Fill or its variations such as Post Pillar Cut and Fill (PPCF).

Old trial tests conducted earlier at KCM revealed that addition of cement to classified tailings reduces the percolation rates to as low as 65mm/hr as opposed to 80mm/hr required at Konkola mine to facilitate production. The Cemented Aggregate Fill material with 4% Zambezi Portland cement has a permeability of 83mm which is acceptable for underground mining operations. However, care would need to be exercised to ensure minimal loss of cement fines in the drainage water.

The sampling and crushing of waste rock material from No.3 Shaft waste dump and blending with the classified mill tailings and subsequent cement addition and curing at the lab was done in simulation of the future backfilling operations at Konkola Mine.

The conclusions drawn based on the research findings are summarized below:

1. The current hydraulic fill material with 65% solid fraction should be blended with aggregate fill material with maximum size of 2.83mm in the ratio of 3:7, respectively, to produce a well graded dense backfill and material.
2. 4% Zambezi cement should be added to the blended material and allowed to cure for not less than 28 days if average uniaxial compressive strengths of over 1 MPa are to be achieved.
3. The introduction of cemented backfilling at No.3 Shaft underground mine will increase the extraction ratio from 80% to 100% and reduce the Cost of Production by **10%**.
4. Due to the good confining effect that the current Hydraulic Fill has demonstrated, the research scholar concluded that the material will be very useful in filling the secondary stopes after the primary stopes have been filled with the engineered backfill material in the future Primary / Secondary extraction system proposed for mining operations commencing after the KDMP project.
5. In the mixing tanks, water should first be added to cement and then later mixed with the developed blended backfill material if maximum strength results are to be achieved. An alternative system of cement mixing on underground site could be tried.
6. The holding and mixing tanks at the backfill plants have constant agitators to prevent the material from settling. Constant flushing of the reticulation system must be done. The flow of the Cemented Aggregate material in the reticulation system must be turbulent to prevent any chances of material settling in the pipes.
7. The underground Backfill Crew, must ensure that cement losses from the fill material in the decant water is minimized. Only excess and supernatant water should flow out of the drainage pipes in the Bulk head. This could be achieved by unblocking the drainage pipes in the bulkheads only after ensuring that the Cemented Aggregate Fill material has settled in the stope. However, the material must be kept saturated for not less than 28 days to ensure full hydration process of the material.

8. Backfill fines and cement particles which are important components of the backfill strength could sometimes be carried off in the drainage water. To prevent this,

flocculants such as ‘Core Shell (TM) 71302’ could be added to the cemented backfill to help minimize loss of particles in the drainage water. The flocculants will not only prevent the loss of cement, but will improve the overall drainage by reducing the fines content.

9. Possibility of reducing cement consumption exists if different stope configurations are considered. Recent advances in numerical modeling can assist in designing back fill for various stope configurations.
10. To ensure effectiveness of both the Cemented Aggregate Fill and uncemented hydraulic fill in the voids underground, the backfill operators must ensure that the material is completely filled in the void.
11. More strength tests should be carried out with the slag material from the New Nchanga Smelter (which is believed to have very good pozzolonic properties) as it has the potential to completely reduce cement consumption in the developed backfill material and thereby reduce the costs of cemented backfilling at Konkola mine.
12. Dewatering of production blocks should be done way ahead of mining operations to avoid collapse of the backfill materials in the stopes due to pore pressures arising from water ingress.
13. A conceptual flow diagram (Figure 6.9) developed based on this research, will ensure an efficient composite backfill system.

The following recommendations have been made for successful implementation of backfilling operations at Konkola Mine.

1. Training of the backfill personnel at No.3 Shaft to operate the new cemented backfill system must be looked into by mine management, especially the quality aspects of the cemented aggregate fill material.
2. To increase backfill stabilization of the cement aggregate fill, mine management could consider usage of geotextile fabrics interspaced in the material.
3. To monitor the performance of fill materials in the stopes, a routined inspection and monitoring program must be devised coupled with instrumentation monitoring systems.

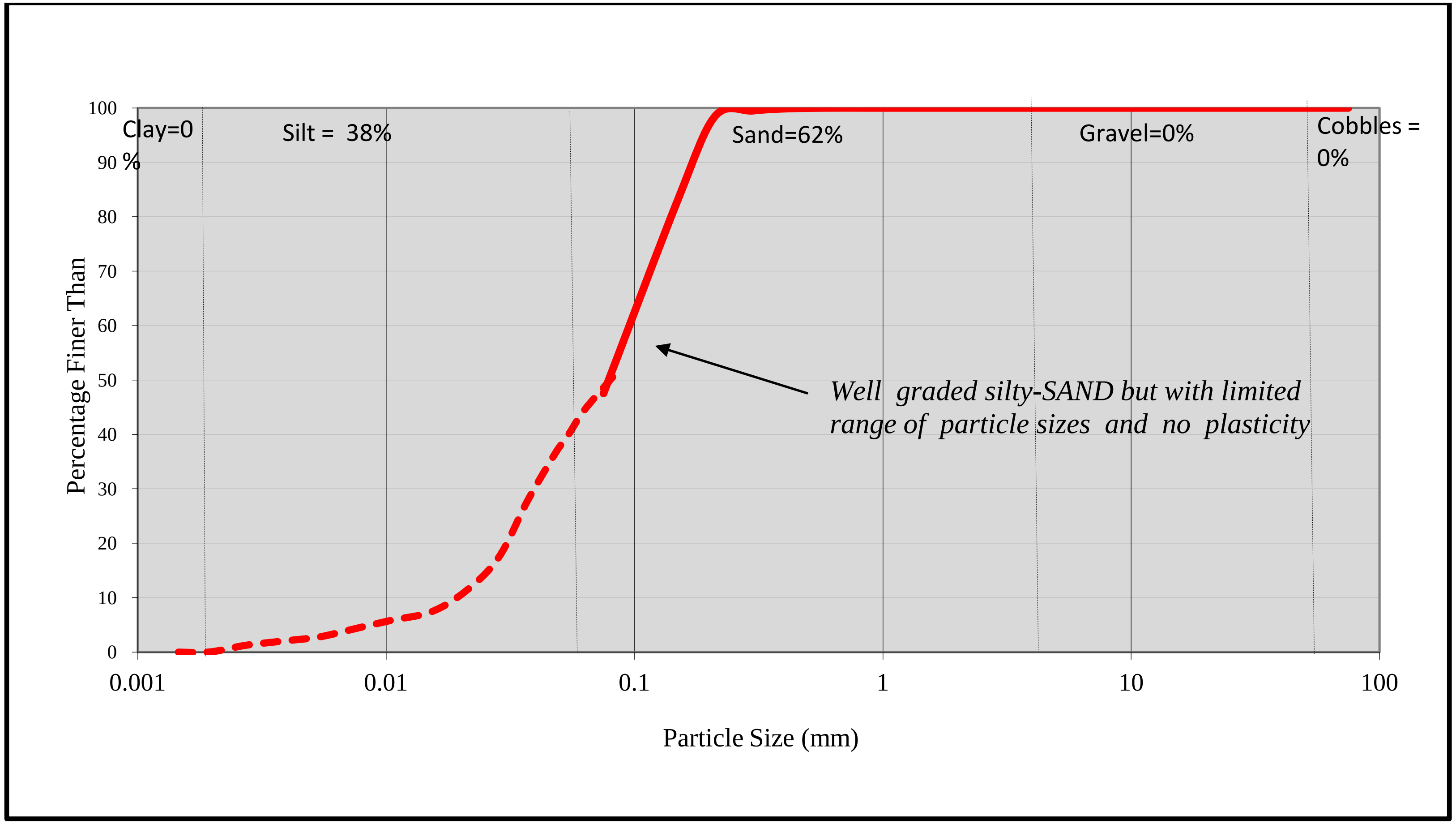
REFERENCES

1. Craig, R.F., 2005. Craig's Soil Mechanics. [e-book] New York: Taylor & Francis. Available at: British Library/Google books/e-books http://www.books.google.zm/ [Accessed 5 September 2009].
2. Dorricott, M., & Grice, A., 2000. Impact of Stope Geometry on Backfill Systems for Bulk Mining. In: MassMin 2000. Brisbane, Australia, 29th October to 2nd November 2000. The Australasian Institute Of Mining And Metallurgy: Brunswick
3. Grice, A.G., 2005. Fluid Mechanics of Mine Fill. In: Y. Potvin, E.G. Thomas and A.B Fourie, eds. 2005. Handbook on Minefill. Western Australia: Australian Centre for Geomechanics (ACG), pp. 52-61.
4. Grice, T., 2007. Introduction to Mining with Backfill; Rock/Aggregate Fill Systems; Fill Delivery. In: Australian Centre for Geomechanics (ACG), Hydraulic and Paste Backfill. Perth, Australia, 5th November 2007, ACG: Perth, Western Australia.
5. Hassani and Archibald, 1998. Portland Cement Mixtures for improving Slurry and Paste Backfill Support Performance. Canadian Institute of Mining, Metallurgy and Petroleum Journal Annual General Meeting Technical Presentation, 1, pp. 5-10.
6. Hassani and Bois, 1998. Slurry and Paste Backfill. Canadian Institute of Mining, Metallurgy and Petroleum Journal Annual General Meeting Technical Presentation, 1, pp. 1-10.
7. Helinski, M., 2007. Mine Fill Geomechanics. In: Australian Centre for Geomechanics (ACG), Hydraulic and Paste Backfill. Perth, Australia 5th November 2007. ACG: Perth, Western Australia.

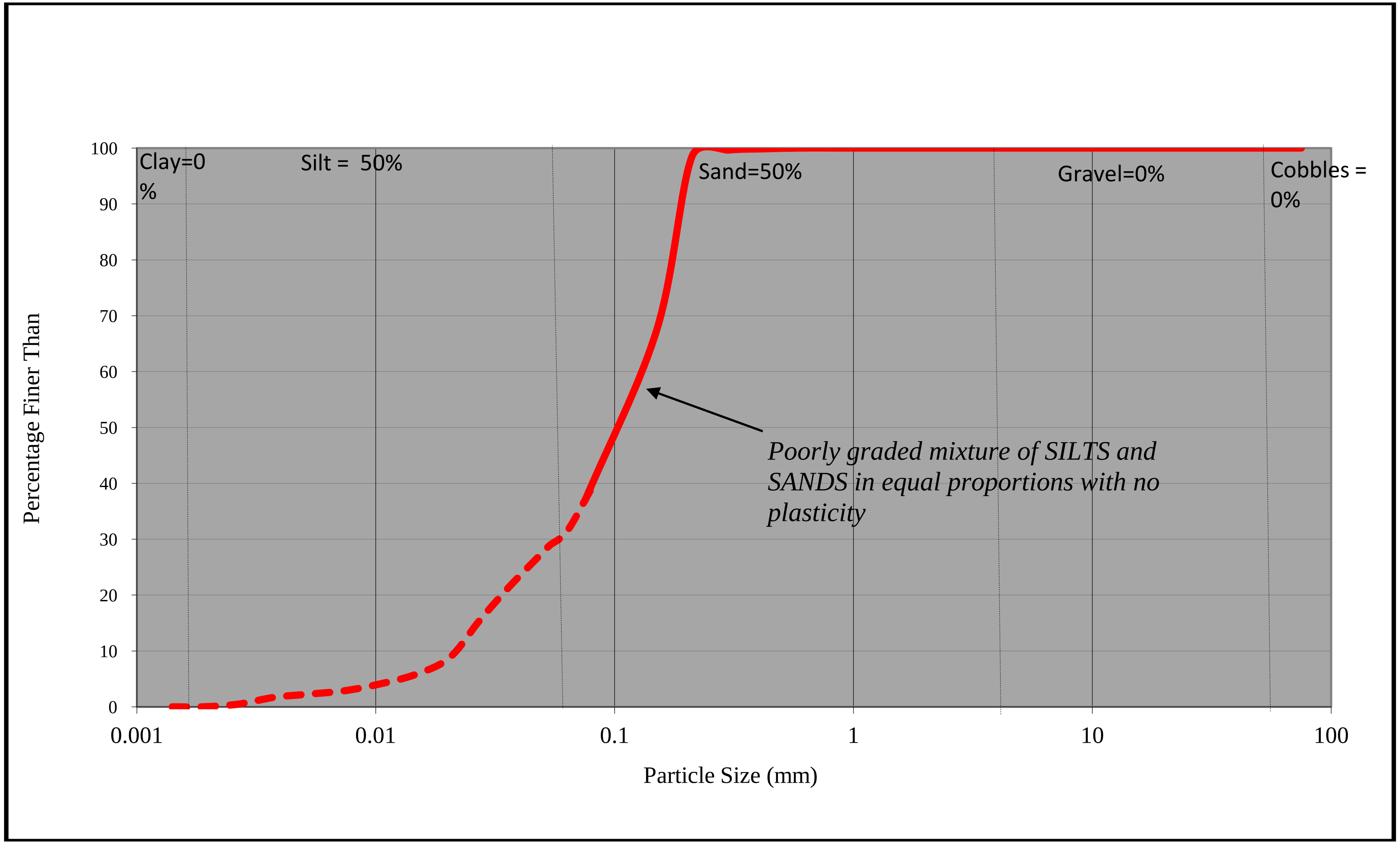
8. Henderson, A. et al., 2005. Paste Fill. In: Y. Potvin, E.G. Thomas and A.B Fourie, eds. 2005. Handbook on Minefill. Western Australia: Australian Centre for Geomechanics (ACG), pp. 83-94.
9. Henderson, A., & Revell, M.B., 2005. Basic Mine Fill Materials. In: Y. Potvin, E.G. Thomas and A.B Fourie, eds. 2005. Handbook on Minefill. Western Australia: Australian Centre for Geomechanics (ACG), pp. 13-19.
10. King, R.P, 2002. Introduction to Practical Fluid Flow. 1st ed. Oxford: Butterworth-Heinemann
11. KCM Expansion Projects. Konkola Copper Mines Plc, 2009. [Online] (Updated 16 Sept 2009)
Available at: http://www.kcm.co.zm/expansion_projects.php
[Accessed 10 Oct 2009].
12. Kuganathan, K., 2005. Geomechanics of Minefill. In: Y. Potvin, E.G. Thomas and A.B Fourie, eds. 2005. Handbook on Minefill. Western Australia: Australian Centre for Geomechanics (ACG), pp.24-46.
13. Kuganathan, K., 2005. Rock Fill in Mine Fill. In: Y. Potvin, E.G. Thomas and A.B Fourie, eds. 2005. Handbook on Minefill. Western Australia: Australian Centre for Geomechanics (ACG), pp. 103-111.
14. Potvin, Y., 2005. Introduction. In: Y. Potvin, E.G. Thomas and A.B Fourie, eds. 2005. Handbook on Minefill. Western Australia: Australian Centre for Geomechanics (ACG), pp.3-9.
15. www.infomine.com
16. www.wikipedia.org

APPENDICES

APPENDIX 01 - PSD for Hydraulic Fill material being produced at No.1 Shaft before being pumped to No.3 Shaft Holding Tanks.



APPENDIX 02 Figure 5.6 PSD for Hydraulic Fill material reporting at No.3 Shaft Holding Tanks



APPENDIX 03 - UCS test results for Konkola Cemented Hydraulic Fill Material using - Mphamvu Cement

Cement Content	Curing Period (Days)	UCS (kPa) for CHF samples					
		1	2	3	4	5	6
2%	7	50	56	54	51.6	49	48.9
	14	60	65	69	60	66	70
	21	74	72	70	69	69	60
	28	73	75	70	70	79	83
4%	7	91	80	79	80	90	84
	14	120	130	135	139	125	131
	21	160	165	176	177	172	170
	28	233	240	235	231	226	236
6%	7	95	95	95	98	90	97
	14	154	170	166	161	169	140
	21	247	257	255	250	250	239
	28	345	360	366	340	350	339
8%	7	119	112	126	127	118	118
	14	245	255	240	249	257	254
	21	339	329	327	340	335	310
	28	470	487	490	469	476	488
10%	7	222	230	233	220	215	230
	14	295	295	295	310	315	314
	21	485	499	478	505	498	475
	28	645	655	660	648	651	641

APPENDIX 04 - UCS test results for Konkola Cemented Hydraulic Fill Material using Zambezi Cement

Cement Content	Curing Period (Days)	UCS (kPa) for CHF samples					
		1	**2**	**3**	**4**	**5**	**6**
2%	**7**	60	69	70	63	71	63
	14	77	89	83	90	90	75
	21	90	87	80	97	91	89
	28	104	106	93	97	92	90
4%	**7**	119	103	100	116	117	105
	14	177	180	170	170	165	158
	21	222	219	231	215	227	230
	28	281	290	299	291	294	285
6%	**7**	119	120	127	128	126	124
	14	214	213	219	206	204	204
	21	260	271	265	268	277	285
	28	465	470	471	460	466	446
8%	**7**	149	148	157	160	166	162
	14	320	331	337	331	340	321
	21	426	436	436	444	430	444
	28	639	641	644	630	625	631
10%	**7**	309	300	290	293	300	290
	14	399	389	406	401	397	402
	21	655	635	649	650	640	659
	28	870	871	855	850	861	859

APPENDIX 05 - UCS test results for Konkola Cemented Aggregate Fill Material using Mphamvu Cement

Cement Content	**Curing Period (Days)**	**UCS (kPa) for CAF samples**					
		1	**2**	**3**	**4**	**5**	**6**
2%	**7**	87	80	85	81	77	88
	14	179	175	170	185	173	180
	21	180	183	176	189	186	166
	28	200	210	208	211	199	220
4%	**7**	257	260	266	249	256	254
	14	314	312	310	325	320	321
	21	444	440	439	457	450	452
	28	760	757	770	765	753	764
6%	**7**	423	437	440	431	439	428
	14	518	530	539	526	531	518
	21	760	740	758	738	748	744
	28	1030	1015	999	1021	997	1040
8%	**7**	800	825	830	810	800	831
	14	919	920	937	930	917	951
	21	1021	1038	1043	1030	1023	1025
	28	1400	1399	1370	1376	1380	1367
10%	**7**	981	999	989	976	971	964
	14	1130	1120	1111	1100	1125	1104
	21	1333	1327	1330	1321	1329	1256
	28	1959	1945	1966	1939	1927	1976

APPENDIX 06 - UCS test results for Konkola Cemented Aggregate Fill Material using Zambezi Cement

Cement Content	Curing Period (Days)	UCS (kPa) for CAF samples					
		1	2	3	4	5	6
2%	7	99	100	110	111	102	108
	14	225	239	230	235	222	241
	21	230	238	239	235	233	247
	28	271	270	266	280	277	280
4%	7	330	339	345	347	337	334
	14	415	420	427	430	433	407
	21	699	692	700	700	722	699
	28	1013	1020	1015	1020	999	993
6%	7	569	580	588	573	573	555
	14	700	700	711	690	698	689
	21	1000	998	991	989	990	978
	28	1330	1359	1333	1359	1339	1368
8%	7	1050	1100	1090	1070	1060	1116
	14	1200	1232	1244	1250	1245	1221
	21	1353	1379	1366	1360	1350	1388
	28	1850	1852	1820	1845	1800	1819
10%	7	1200	1350	1333	1320	1345	1252
	14	1411	1379	1350	1390	1400	1344
	21	1703	1722	1729	1760	1770	1786
	28	2250	2245	2200	2202	2269	2046

APPENDIX 07 - UCS test results for Konkola Cemented Aggregate Fill Material using Minova Fast Set Cement

Cement Content	Curing Period (Days)	UCS (kPa) for CAF samples					
		1	2	3	4	5	6
2%	0.04	110	115	117	114	105	117
	0.17	235	240	220	233	232	226
	1	251	246	256	249	241	257
	3	261	269	273	263	264	284
	7	306	309	293	295	300	303
	14	340	349	355	351	347	334
	21	366	370	359	356	361	360
	28	400	391	401	380	392	382
4%	0.04	349	333	336	331	345	346
	0.17	419	418	431	433	422	427
	1	712	721	698	708	709	718
	3	1023	996	1011	1021	1015	988
	7	1270	1265	1330	1350	1279	1378
	14	1499	1520	1533	1450	1495	1437
	21	1602	1609	1550	1570	1639	1564
	28	1611	1598	1670	1652	1653	1674
6%	0.04	560	566	589	570	571	606
	0.17	700	701	702	700	669	602
	1	965	989	1001	1003	961	997
	3	1346	1351	1362	1286	1288	1365
	7	1401	1489	1465	1414	1436	1507
	14	1551	1501	1503	1523	1568	1660
	21	1685	1755	1701	1733	1674	1658
	28	1811	1761	1835	1823	1763	1759
8%	0.04	703	711	715	683	683	693
	0.17	1102	1179	1122	1122	1163	1248
	1	1311	1376	1401	1401	1365	1354
	3	1821	1851	1826	1792	1845	1815
	7	2111	2125	2230	2211	2231	2262
	14	2389	2462	2410	2501	2399	2557
	21	2556	2533	2501	2511	2492	2413
	28	2689	2556	2633	2687	2641	2466
10%	0.04	983	1000	956	945	963	1009
	0.17	1236	1156	1265	1286	1266	1207
	1	1813	1811	1756	1742	1717	1619
	3	2151	2162	2211	2233	2212	2237
	7	2463	2451	2501	2381	2399	2283
	14	2612	2545	2511	2623	2499	2522
	21	2711	2799	2656	2633	2651	2744
	28	2999	3045	3021	2965	2977	3047

APPENDIX 08 - UCS test results for the Slag/Lime Mixture

	Curing Period (Days)	UCS (kPa) for the Slag/Lime Mixture					
		1	2	3	4	5	6
Slag/Lime 2:1	**7**	90	80	84	82	77	88
	14	180	173	188	170	190	160
	21	199	181	187	191	165	166
	28	220	201	200	215	221	199

APPENDIX 09 - BSI Certificate for Geotechnical Services Laboratory.

Certificate of Registration

QUALITY MANAGEMENT SYSTEM - ISO 9001:2008

This is to certify that:

Konkola Copper Mines Plc
Corporate Head Office
Private Bag KCM (C) 2000
Chingola
Copperbelt Province
Zambia

Holds Certificate No: FM 56573

and operates a Quality Management System which complies with the requirements of ISO 9001:2008 for the following scope:

Supply of Rokana Electrolytic Copper Cathodes (REC) and Kabundi Copper Cathodes (KBC).

For and on behalf of BSI:

Managing Director, BSI EMEA

Originally registered: 01/11/2000 Latest Issue: 21/02/2010 Expiry Date: 29/01/2013

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APPENDIX 10 Summary of the Ore shale Rock mass properties for the geological zones at number No.3 Shaft

	ROCK TYPE	Q - Value	RMR	DESCRIPTION
1	Hanging wall Quartzite	1.1-29.9	45-75	Fair to Good
2	Ore Shale	0.2-1.2	30-46	Fair to Poor
3	Footwall Conglomerate	0.8-14.8	41-67	Fair to Good
4	Footwall Sandstone	5.2	59	Fair
5	Porous Conglomerate	2.4	52	Fair
6	Argillaceous Sandstone	0.9-15.5	43-69	Fair to Good
7	Footwall Quartzite	72	83	Very Good