GEOTECHNICAL INVESTIGATION OF COAL MINE REFUSE FOR BACKFILLING IN MINES

A THESIS SUBMITTED IN PARTIAL FULFILLMENT OF THE REQUIREMENTS

FOR THE DEGREE OF

BACHELOR OF TECHNOLOGY

IN

MINING ENGINEERING

BY

VISHAL KUMAR AGARWAL

DEPARTMENT OF MINING ENGINEERING
NATIONAL INSTITUTE OF TECHNOLOGY
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C E R T I F I C A T E

This is to certify that the thesis entitled **"Geotechnical Investigation of Coal Mine Refuse for Backfilling in Mines"** submitted by **Sri Vishal Kumar Agarwal** in partial fulfillment of the requirements for the award of Bachelor of Technology degree in Mining Engineering at the National Institute of Technology, Rourkela is an authentic work carried out by him under my supervision and guidance.

To the best of my knowledge, the matter embodied in the thesis has not been submitted to any other University/Institute for the award of any Degree or Diploma.

Date:

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ABSTRACT

Introduction

Coal is the primary sources of energy and there is an increase in demand for coal production to meet the requirements of the industries and the amount of refuse generated is also increasing alarmingly. Increased mining of thinner seams or of inferior coal may even lead to larger amount of coal refuse, resulting in serious environmental and stability problems. This refuse is mostly disposed in the form of refuse piles and behind embankment type remaining structures. At present, with the exception of some small scale underground waste disposal operations in abandoned coal mines, most of this waste is disposed at the surface, which inevitably requires excessive planning and control to minimize the environmental impact of mining. It also results in non-productive use of land, air and water pollution, possible failure of waste embankments, and the loss of aesthetic value of the land.

Experimental Investigation of Fill Properties

When choosing a particular stowing material for a backfilling system, one should examine the material's short and long term mechanical properties and expected behaviour following placement. This will allow an assessment of the fill's ability to act as a ground support material. An analysis of these properties will help to determine whether the design objectives of the fill will be met. To study the physical and mechanical properties of coal mine refuse samples were collected from different mines of Mahanadi Coalfields Ltd (MCL), Hindalco, and South Eastern Coalfields Ltd (SECL).

The different tests were performed in the present work are triaxial test, slake durability test, liquid limit test, standard proctor compaction test and permeability test. All the tests were carried out following standard experimental procedures (Singh and Choudhary, 1994) and the results are as presented in Table 1

Discussion

It would be desirable for coal refuse to be used as stowing material to have at least medium durability, i.e.: Id_1 should be greater than 95 and Id_2 greater than 85. It could be seen from the experimental studies (Table 1) that the refuse from sample nos. 1, 5 and 7 possess the desired characteristics and are suitable for back filling, where as the Id_1 and Id_2 values are from sample nos. 2, 3, 4, 6 indicate that the refuse is susceptible to moderate amount of disintegration when exposed to wetting and drying cycles.

Liquid limits can be qualitatively related to compressibility of soils. Soils with liquid limit from 0-30 should have a low compressibility. The experimental results show that the liquid limit for the samples lies in low to moderately compressible region could be termed very slightly plastic.

Compaction testing of the refuse determines the moisture content which will achieve the maximum dry density for a material that has been compacted with a given compactive effort (Wray, 1986). It could be observed from Table 1 that the maximum dry density varies from 1.901 Mg/m^3 to 2.207 Mg/m^3. When compared with the compacted densities of sand (which varies between 1.7Mg/m^3 and 2.2 Mg/m^3) which has been successfully

used as hydraulic stowing material in India, it could be seen that the refuse from all the mines can be suitably used as a backfilling material.

Table 1: Summarized Experimental Results

Sample No.	Slake Durability		Atterberg Limit		Standard Proctor Compaction		Constant Head Permeabilty	Triaxial Compression	
	Id_1 (%)	Id_2 (%)	L (%)	P (%)	γ (gm/cc)	M (%)	K (mm/sec)	Φ (degrees)	C (KPa)
1.	96.12	93.75	20.29	15.42	2.031	10.56	19.07×10^{-5}	16.0	85
2.	71.4	68.8	30.35	16.92	1.917	11.8	7.88×10^{-7}	20.56	150
3.	64.23	53.84	24.76	15.08	1.995	10.2	9.64×10^{-6}	25.71	47.5
4.	65.14	49.86	25.65	17.01	2.207	11.21	3.64×10^{-5}	22.5	215
5.	95.39	94.03	21.26	12.19	2.041	10.122	2.75×10^{-5}	25.0	105
6.	91.83	86.40	20.87	-	1.901	10.122	20.39×10^{-5}	21.0	50
7.	96.17	92.39	22.10	14.57	1.954	10.53	3.05×10^{-5}	28.5	142.5
8.	88.43	77.80	20.86	-	1.992	10.49	15.13×10^{-5}	22·0	60
9.	81.22	65.20	22.32	19.29	2.029	10.32	3.38×10^{-5}	34.0	70

L = Liquid limit, P = Plastic limit, γ =Maximum dry density, M =Optimum moisture content, k=Constant head permeability constant, Φ = Angle of internal friction, C = Cohesion,

For a material to be considered as suitable for backfilling, the coefficient of permeability should be at least $2.78 * 10^{-5}$ mm/sec. From the experimental results it may be observed that the sample nos. 1, 4, 5, 6, 7, 8, 9 has the required permeability and are suitable for backfilling whereas for sample nos. 2 and 3 permeability is less than $2.78 * 10^{-5}$ m/s and so are not suitable for filling purposes.

Sands with little or no fines generally have a much higher residual angle of friction. It would be desirable for coal refuse to have at least a comparable residual angle of friction, say around 30^o. From the experimental results it may be observed that sample nos. 7 and 9 have the required residual angle of friction and is suitable for backfilling whereas sample nos. 1, 2, 3, 4, 5, 6, 8 have residual angle of friction less than 30^o and so they are not preferred for filling purposes.

Conclusion

It may be concluded that coal refuse of sample no. 1. 5 and 7 can be used for the purpose of backfilling without much treatment. But all other samples need some treatment such as removal of some fine particles, mixing with some amount of cement or some other binding material so that its strength increases and it does not deteriorate when subjected to wetting and drying cycles.

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CONTENTS

INTRODUCTION

At present, the power sector in India is dominated by coal. Coal currently accounts for more than 50% of total primary commercial energy supply in the country and for about 70% of total electricity generation. Coal is likely to remain a key energy source for India, for at least the next few decades, as India has significant domestic coal resources (relative to other fossil fuels) and a large set of existing installed base of coal-based electricity capacity, although recent experiences have thrown into sharp relief the uncertainties and concerns regarding the adequacy of coal supplies to satisfy the growing hunger for power. At the same time, with the growth of the coal-based power, local environmental and social challenges relating to coal mining, processing, and use are becoming more pressing.

India now ranks 3rd amongst the coal producing countries in the world. According to the 2008 BP Statistical Energy Survey, India had end 2007 coal reserves of 56498 million tonnes, 6.66% of the world total. Through a sustained programme of investment and greater thrust on application of modern technologies, it has been possible to raise the production of coal from a level of about 70 million tonnes at the time of nationalisation in the early 1970's to production of 478.18 million tonnes in 2007. Most of the coal production in India comes from open pit mines which contribute over 81% of the total production. A number of large open pit mines of over 10 million tonnes per annum capacity are in operation. Underground mining currently accounts for around 19% of national output.

At present, with the exception of some small scale underground waste disposal operations in abandoned coal mines, most of this waste is disposed at the surface, which inevitably requires excessive planning and control to minimize the environmental impact of mining. It also results in non-productive use of land, air and water pollution, possible failure of waste embankments, and the loss of aesthetic value of the land. It is important to find alternative uses for coal mine refuse and fly ash, the two important byproducts of coal industry, so that their disposal, without adverse environment effects becomes possible. Realizing the economic and environmental consequences, efforts have recently been made to study the physical and engineering properties of stabilized and un-stabilized mixtures of coal refuse and fly ash for construction of highway embankments and base courses.

Opencast coal mining involves the exploitation of shallow seams on a number of small sites in and around the traditional mining areas. For example in U.K., generally, 50-60 sites are in operation at any one time, with a total annual production of 13-15 million tonne of coal. Maximum depth of working is about 100m, with a stripping ratio up to 25:1. Site area varies from 10-800 hectares of mined land. Some 1400 hectares of mined land is converted to areas of opencast backfilling every year. Progressive restoration of sites is carried out as a means of limiting waste handling and permitting an economic and desirable rehabilitation of the mine site. In some cases opencast mining is carried out on areas of derelict land, arising from industrial or old underground workings which are subsequently restored. In India also problems of mining waste are there and hundred of hectares of area around the mine are used for dumping of overburden and low grade ore. The top soil which is very important for agricultural purpose is lost in huge quantity. As per Indian rule the height of the dumped overburden cannot be more than 60m, so, large areas are acquired for dumping of these overburden and low grade ores. So, apart from dumping these wastes during mining period we have a problem of restoration of sites.

So, for safe disposal of the coal mining wastes and to make proper restoration of site, geotechnical investigation of coal mining waste is important.

1.1 WASTES GENERATED BY COAL MINING ACTIVITIES

1.1.1 In Opencast Mining

- In opencast mining in India generally the coal seams are below the surface. So, we need to remove this overburden to excavate or mine coal. Sometimes we have to remove overburden from places below which no coal is there. But this is important in order to get a full access to the deposit.
- Sometimes low grade coal or ores has to be removed and dumped as mining waste.
- Mill tailings or coal with high ash percent, sulphur present are dumped on land.
- Sometimes coal seams have parting between them and these parting are removed and dumped as coal mining waste.
- LEACHED ORE: Processes for extraction of metal from low-grade ores, e.g. heap leaching, pad leaching, etc are being practiced as modem technologies for beneficiation. After leaching / extraction of metal, the spent ore is either left on

the leach pads or removed and dumped elsewhere. For the purpose, highly acidic, corrosive and poisonous materials like sodium cyanide, arsenic, etc are commonly used chemicals. They remain in the residual solution after leaching, and the pulp may contain cyanides and sulphates apart from the spent ore.

1.1.2 In Underground Mining

- Stone bands are major source. These stone bands are present in the coal seam and are removed from excavated coal by the washery.
- Sometimes during blasting the roof of the underground opening gets blasted because of wrong way of drilling the holes for blasting. So, the entire layer of roof has to be dressed down. These dressed down roof and coal sometime get mixed up and are taken to the washery.
- During shaft sinking and drivage of incline (also adit) we have to excavate the waste to make an access to coal.
- Waste material has to be removed for the creation of sumps.
- In underground mines seepage of water is there and disposal is needed to avoid inundation. This water may contain dissolved salts due to leaching and affect the pH value adversely. This may contain toxic substance like heavy metals (Cr, Ni, Cd, Pb, etc), besides domestic effluent containing carbonaceous material and pathogenic bacteria.

1.2 MAJOR PROBLEMS OF COAL MINING WASTES

- The waste materials are dumped on agricultural land. So, huge loss of land is there as height of disposal cannot be more than 60m as per Indian rules.
- Disturbance of the land topography by dumping of debris or stripping of land hence flow pattern of the area is changed.
- Diversion of streams and alteration of drainage is there. There is leaching of debris and thereby pollution of water resources.
- Lowering of water table of area of influence due to zonal drawdown of aquifer.
- There is a disturbance of subsoil water regime due to micro-deformation of the slope.
- Pyrite is a relatively common iron sulphide in some of the coal. They are highly unstable and break quickly because of oxidation and forms Ferrous and Ferric sulphates and sulphuric acid. Oxidation of pyrite within spoil heap waste is

governed by the access of air that in turn depends upon the particle size, distribution, amount of water saturation and degree of compactness.

- Due to spontaneous heating SO_2, CO, CO_2 etc are released.
- For stability of the overburden, compaction of overburden, slope stability etc. are done. A localized slope or foundation failure or they could have a major adverse and long term impact on the environment, such as valley blockage causing a drainage changes and erosion. Performance records indicate a greater occurrence of stability associated problems in cold regions than in areas of moderate climate.
- Sometimes due to rain, there can be thawing which could supply excess of water and may trigger sudden mudflows or rock flows.
- After mining of the coal, we have to do proper restoration of sites.

1.3 NEED OF BACKFILLING IN MINES

To get rid of all the wastes economically and without hampering environment, backfilling in mines can be a viable solution. Backfilling of coal mines can be performed in conjunction with mining (in case of underground mines), or even after a mine has been abandoned. In addition to reducing mining subsidence, other potential benefits that might be gained include increased coal recovery, enhanced ventilation control, reclaiming mined out surface land etc. Coal refuse can also be disposed, provided that it is deemed a stowing material.

It may be possible to increase coal recovery with the proper placement of backfill. If the in situ fill has sufficient strength to provide support to the overburden, or to increase the strength of pillars by providing lateral support, less coal would be required to be left out in pillars.

Ventilation control can be enhanced by improving ground control. Leakage in under ground coal mine is experienced when stoppings, overcasts, etc. experience damage from ground movements. If backfilling helps improve ground control, leakage caused by ground movements would be also reduced.

Backfilling, particularly hydraulic backfilling, can help to minimize underground mine fires (Vorobjev and Deshmukh, 1966). Oxygen is necessary for a mine, and because the amount of oxygen that can come into contact with the exposed coal is reduced, the chance of fire is reduced.

Generally 30-40 % of the mine product sent to preparation plants is rejected as waste, and subsequently disposed of on the surface in waste embankments and settling ponds. Backfilling can be used as an alternative coal refuses disposal method when preparation plant reject is used as a stowing material.

Underground coal refuse disposal may pose a pollution potential as a result of leaching. However this can be rectified by disposal below the water table, where the fill will eventually become saturated. Permanent saturation of the refuse reduces or eliminates the oxidation that might produce acid mine drainage. On the other hand saturated fill may not have the desired strength and stiffness for ground control requirements, so incorporating cementing mixtures into the mine waste may be a more appropriate solution to the pollution problem.

Backfilling methods are classified according to the manner in which the stowing material is placed in the mine void. Stowing methods utilized in active underground mining includes:

- By hand
- Gravity
- Mechanical
- Pneumatic
- Hydraulic

The two most popular stowing methods, both past and present, are pneumatic and hydraulic. When choosing a particular stowing material for backfilling system, one should examine the material's short and long-term mechanical properties, and expected behavior following pattern. Important properties for the in situ fill are strength, deformability; slake durability, its compaction characteristics, ability to dissipate pore pressure etc.

There are two major concerns in using coal mine refuse in engineered construction, viz.,

(i) the possibility of spontaneous combustion, and
(ii) The acidic leachates and other harmful ingredients that may contaminate the ground water.

Studies have shown that well-compacted coal mine refuse is unlikely to show spontaneous combustibility or support combustion from an external source. The effects of acidic leachates from coal refuse can be reduced by addition of fly ash and by maintaining proper drainage control. Although one should try to make as sake a mixture as possible for backfilling so that it automatically takes care of acid mine drainage, by minimizing it as far as possible.

1.4 OBJECTIVE

Geotechnical investigation of coal mine refuse is required to be carried out as it can help to judge and make the area safe for work and development. It is necessary to know the compactness of the material that is being used for backfilling, which can be achieved by geotechnical methods. Stability and performance of the waste dumps on steeply sloping terrain is an important issue, as it is a cause of many accidents. These parameters can be monitored with the help of geotechnical investigation. To have a control over the geo-mechanical process during the formation of quarry slopes we need geotechnical investigation. With the help of geotechnical investigation we can judge how temperature and climate will have affect on the stability of the backfill. This work has therefore been planned to be carried out with the following objectives:

1. Collection of coal mining refuse samples from different coal mines
2. Experimental investigations on by carrying out a number of experiments such as triaxial compression, standard proctor compaction, slake durability, plasticity, permeability and particle size analysis of the collected samples to determine the fill properties.
3. Analysis of the fill properties to assess the suitability of these wastes as filling material based on the experimental investigation.

LITERATURE REVIEW

This chapter presents the important findings of a few researchers regarding backfilling of coal mine wastes brief. It may be mentioned here that the investigations only relevant to the present research work are described in chronological order from the past to the present.

Chugh and Deb (1997) successfully demonstrated that extraction ratio at an Illinois mine can be increased from 56% to about 64% with backfilling done from surface upon completion of all mining activities, by using fine coal processing waste (FCPW) and coal combustion byproducts based paste. This concept has the potential to increase mine productivity, reduce mining costs, manage large volume of CCB's (coal combustion byproducts) and FCPW'S beneficially, and improve miner's heath, safety and environment.

Palarski (2002) has reviewed the method of disposing coal waste on the surface at low costs. His assertion that using hydraulically transported, densified and cemented fills with fly-ash ad flue gas desulphurization by products has a major economic and environmental advantage over normally used method of waste fill by dumped rocks and hydraulically or pneumatically placed crushed rocks specifically in coal mines.

Karfakis et al. (1996) described the physical and mechanical properties of coal mine waste from different sites and the effect of these properties on the duty requirements of fill materials were assessed. For environmentally acceptable and economically viable method of backfilling by coal mine refuse important properties to be considered are strength characteristics, deformability, and ability to dispose pressure and slake durability. On analyzing these properties we can determine if the design objectives of the fill can be met or not. As a result of this testing, it is concluded that if improving ground control is the only reason for backfilling, coal refuse alone does not appear to be a suitable stowing material. If coal-refuse disposal is also a consideration, then it may be more attractive stowing and backfilling material.

Ariouglu (1982) studied the effect of coarse aggregate addition on strength properties of cemented fill. This study reveals laboratory test related design parameters, such as amount of coarse aggregate leading to maximum density, cement content and (water/cement) ratio to cemented fill strength. The results clearly pointed out that strength properties markedly increase with increased cement content and decreased (water/cement) ratio.

Dixit and Raju (1983) carried out investigation of different parameters affecting the stabilization of the cement stabilized fills. The various parameters that they considered were overall height of backfill, lateral dimension of fill, thick layer of low cement fill and thin layer of high cement fill. They suggested that cement stabilized fills can provide the required ground control if their strength distribution functions match with the probability of strength of the wall rocks. They proposed to produce a reinforced cemented backfill that must have strength to match with the probability of the wall rock strength.

Thomas (1980) described a laboratory test study on tailings produced from washeries and metallurgical operations to characterize them for fill purposes. The test tailings were characterized for their size and preparation of fill from these tailings, recoveries ranging from 45 to 65 weight percent. Results show techniques to allow satisfactory fill preparation from tailing material hitherto considered too fine for such utilization. Results show conclusively that the fill product was quite suitable and recoveries quite adequate. Results are considered to be highly significant to fill practice worldwide, in that they indicate techniques to allow satisfactory fill preparation from tailing materials considered too fine for such utilization. A broad, new area of refuse mill tailing fill technology is opened up, with significant implications for both fill preparation and underground disposal of mining wastes.

Knissel and Helms (1980) performed laboratory tests of uniaxial compressive strength of cemented rock fill from washery refuse of coal mines and cement, and showed that the main factors to influence backfilling economy and ground stability are cement contents, water-cement ratio. Results showed that when preparing cemented rock fill from washery refuse of coal mines, the specific water demand to achieve optimum strength of the mixture has to be taken into account. The optimum water-cement ratio can be calculated from the cement content and a water demand factor, which is depending on the aggregate type. It also showed that strength of backfill will decrease, if the moisture content of the moisture is too low or too high. With excessive amounts of water there is the danger of separation between cement lime and rock fill. Another aspect is that often a definite consistence of the mixture will be necessary for transportation and handling. Therefore a minimum of mixing water will be required. Both aspects have to considered, when calculating the composition of a cemented rock fill mixture.

Yilmaz et.al., (2004) carried out evaluation of acid producing sulphidic mine tailings as a paste backfill. In this case effect of presence of different quantities of cement in mill tailing pastes were found out. Tailings paste were formed and mixed with varying quantities of cement and their respective compressive strength were found out.

W. Lee Daniels (2005) studied the effect of cover soil thickness on revegetation of acidic Appalachian coal refuse. The main objective of his research was to evaluate the minimal topsoil depth necessary for the successful reclamation of acid forming coal refuse materials and to determine the effect of lime additions to the refuse/topsoil contact zone upon minimal topsoil depth requirements for successful reclamation of coal waste piles. Three study sites were established, where each site was active coal disposal. Soil depth wedges were made to carry out the investigation.

3. SAMPLING

The methods for sampling of wastes of coal preparation processes, overburden rocks and rejects of refuse dumps are used for evaluation of compositions and properties of coal wastes in the most coalfields. Waste generated by coal mines may be categorized as coarse or fine. Generally, coarse refuse is larger than 0.6 mm, because this is the size at which coarse and fine coals are usually separated during cleaning. Coarse refuse can contain significant amounts of minus 0.6 mm material due to degradation during processing. Fine refuse is generally smaller 0.6mm, and consists mainly of slurry and tailings.

Coarse coal refuse should be sampled from different coal refuse piles (approximately nine to twelve different refuse piles), each pile containing refuse from different seams. But there is a problem of obtaining a representative sample of the entire population before size analysis can be performed. Before beginning particle size analysis, it is necessary to collect a representative sample. Depending on the nature of the materials sample, this task can be relatively easy or extremely difficult. In general, the refuses that are nearly mono-disperse and somewhat cohesive are easier to sample. In poly-disperse materials, size segregation occurs through mechanical vibrations that occur during sample handling and transport. Large particles tend rise to the top, thus scoop sampling from the top of piles or from the top of containers is particularly unsatisfactory. Thus samples can be taken from randomly selected points located on the surface of the pile. While collecting samples, go up to a depth of 0.6m inside the pile from surface. As we have to achieve as representative a sample as possible, a large amount of material should be collected from various locations on each pile and then mixed thoroughly to obtain a representative sample.

3.1 Sample Collection Procedure

At each site an informal grid was made that contained most of the material in the waste pile. The grids consisted of at least 30 evenly-spaced cells for sampling. Approximately 30 sub-samples or increments were randomly collected from each grid cell by sampling the weathered surface material (upper 15 cm) using stainless steel trowels, small hand shovels, and three-prong scrapers. During collection, all fragments > 4 cm were discarded. Increments from each cell were combined in a 1-gallon plastic bucket. After collection, the composite from each cell are placed in a plastic washtub and mixed. A sub-sample of the cell composite was placed in a soil bag and saved as a cell composite. The remaining cell composite material was placed into 5-gallon bucket(s) to be mixed with other cell composites to form the mine-waste pile composite sample.

Each sample was spread out on clean plastic tarps to dry at room temperature. To insure complete drying, samples are completely turned daily with a small plastic scoop until visibly dry (approximately 2 to 3 days depending upon ambient humidity). After drying, each composite is mixed for 5 minutes in a large stainless steel V-Blender to break up friable clods. The composite material is then sieved with a 2 mm screen, with the < 2 mm fraction being recombined and thoroughly homogenized by mixing in the V-Blender for 30 minutes. The > 2 mm fraction is discarded. After homogenization, the < 2 mm composite material is split into 1 gallon cardboard ice cream containers. The tops of the containers were sealed with tape for storage. Table 3.1 gives the names of the coalfields and company from which samples were collected for the study.

Table 3.1: Name of the Mines and Coalfields from which samples were collected

Sample no.	Name of the Mine	Company
1. 2. 3. 4.	Talcher Coalfield	Mahanadi Coalfields Ltd.
5.	Talabira Coalfield	Hindalco, Sambalpur
6. 7. 8.	Ib Valley Coalfield	Mahanadi Coalfields Ltd.
9.	Chirimiri OCP	Chirirmiri Coalfields, SECL

4. GEOTECHNICAL INVESTIGATION OF FILL PROPERTIES

Various tests can be performed to help assess the performance of coal refuse as a backfilling material. The effect of wetting and drying cycles on the durability of the rocks composing the refuse can be determined by the slake durability test. Pertinent qualitative information about the nature of the material can also be obtained from Atterberg tests, sieve analysis, and from the moisture density relationship test. Certain engineering properties can be measured with the permeability and triaxial compression tests.

This project is intended to perform various laboratory tests to help to evaluate the suitability of coal refuse as a backfilling/stowing material. These include the following tests:

Slake Durability Test:– To know the susceptibility of refuse to disintegrate when exposed to two cycles of wetting and drying.

Plasticity:- Since plasticity affects the strength and permeability characteristics of a sample, its assessment are necessary.

Compaction Test:- Compaction can considerably increase sample densities in comparison with the densities of the loose samples.

Permeability Test:- The ability of the in situ fill to dissipate pore pressure is affected by its permeability characteristics which in turn are affected by the percentage of fine particles in the fill.

Triaxial Compression Testing:- Fill strength is largely dependent on the apparent, or measured, angle of friction, whose value is determined by considering all the factors of shearing resistance to displacement for the fill particles.

Particle Size Analysis:- To illustrate the concept of particle size distribution (PSD)
The details about the experimental setups and procedures have been presented here.

4.1 SLAKE DURABILITY TEST

Durability may be defined as the resistance to weakness of weathering. If a rock mass is more durable, it will last longer when put to use. Durability will also depend on the nature of environment it is to be subjected to. This test is carried out to know the susceptibility of refuse to disintegrate when exposed to wetting and drying cycles. It gives us the Slake

Durability Index (I_d) of the sample. The coal mine refuse may experience wetting and drying cycles during and following placement which can degrade the particles and subsequently after the fills mechanical properties. This test qualitatively assesses the resistance offered by the sample to weakening and disintegration when subjected to two standards cycles of wetting and drying. Previous investigation has shown that there exists a relationship between slake Durability and the Atterberg's limit. It was revealed that the rock with low slake durability index should be subjected to plasticity tests to characterize better their potential behavior in the presence of water. The durability classification of different types of soils have been presented in table 4.1.

Table 4.1: Slake Durability Classification

Group	% Retained after one 10 minute cycle	% Retained after two 10 minute cycles
Very high durability	>99	>98
High durability	98-99	95-98
Medium high durability	95-98	85-95
Medium durability	85-95	60-85
Low durability	60-85	30-60
Very low durability	<60	<30

Experimental Setup

The apparatus consists of a motor on base board capable of driving two drums at a speed of 20 rpm. A suitable number of plastic water troughs, each designed to contain a test drum with quick release drive assemblies, permit 1 to 4 drums to be driven at one time. The test drums are supported on water lubricated bearings allowing 40 mm unobstructed clearance below the drum and a trough water level 20 mm below the axis of the drum. Drums comprise of 2 mm wire mesh cylinders of 140 mm dia and 100 mm length. The photograph of the slake durability apparatus is as presented in figure 4.1.

Experimental Procedure

1. 500 g of dry rock was broken into 10 pieces and was put in the drum. The cylindrical wall of the drum is made of sieve mesh of 2 mm openings.
2. The drum along with the sample was turned in a water bath for 10 minutes at a rate of 20 revolutions per minute.
3. The sample in the drum was dried in the muffle furnace and it was weighed.
4. The weight lost in percentage was calculated and then the percentage retained after one 10 minutes cycle was calculated.

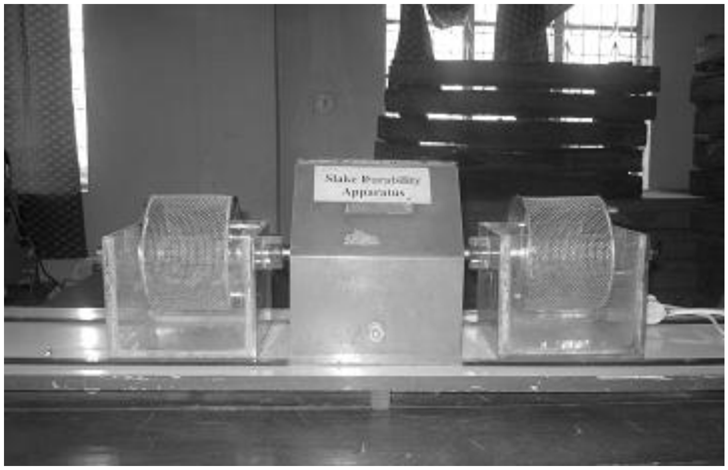


Figure 4.1: Slake Durability Apparatus

5. The sample was again taken in the drum and rotated for another 10 minutes with 20 revolutions per minute.
6. The sample was again dried and percentage retained after second cycle was calculated.

The durability index is calculated as:

$$\mathrm{I_d} = \frac{(W_d) \times 100}{W_i}$$

Where, W_i = initial weight of the sample.

W_d = weight of the sample after the experiment.

I_d = Durability index.

The slake durability index of all the samples after the 1st and 2nd cycles were determined by following the above procedure and the results have been presented in table 4.2.

Table 4.2: Results of Slake Durability Index of Coal Refuse Samples

Sample Name.	**Weigh of sample before cycle I (in gm)**	**Weigh of sample before cycle II (in gm)**	**Weigh of sample after cycle II (in gm)**	**I_{d1}**	**I_{d2}**	**Average Slake Durability Index**
Mine 1	492.0	476.5	455.4	96.85	92.56	93.75
	504.0	480.5	478.5	95.38	94.94	
Mine 2	500.0	357.0	344.0	71.40	68.80	68.80
Mine 3	520.0	334.0	280.0	64.23	53.85	53.85

Mine 4	498.0	323.0	237.0	64.86	47.59	49.86
	504.5	330.0	263.0	65.41	52.13	
Mine 5	498.0	458.0	454.5	91.97	91.26	94.03
	500.0	494.0	484.0	98.80	96.80	
Mine 6	499.0	458.0	429.0	91.78	85.97	86.40
	505.0	464.0	438.5	91.88	86.83	
Mine 7	501.0	488.5	476.5	97.50	95.10	
	494.0	468.5	443.0	94.84	89.68	92.39
Mine 8	506.0	437.0	386.0	86.36	76.28	77.80
	494.0	447.0	392.0	90.49	79.32	
Mine 9	505.5	408.5	322.5	80.81	63.80	65.20
	503.5	411.0	335.0	81.63	66.53	

4.2 LIQUID LIMIT & PLASTIC LIMIT

The Atterbergs limit test is carried out to determine the plasticity. An unconsolidated material which contains little or no clay will exhibit plasticity and is considered non-cohesive. Non-cohesive material is generally considered free draining when used as back fill. Liquid limit can be qualitatively related to the compressibility of the soils.

Soils with liquid limit 0 to 30 have low compressibility. Soils with liquid limit between 31 to 50 should be moderately compressible, whereas soils with liquid limit greater than 50 should be highly compressible..On the basis of experimental evidence it seems reasonable to assign a unique strength to all soils at their liquid limit. Comprehensive study on clays over the range of liquid limit of 30-200% indicate that the range of un-drained shear strength at liquid limits is from 24 to 13kg/cm^2 with a mean value of about 17 kg/cm^2, may be adopted as the present best estimate of the un-drained shear strength of a soil when at the liquid limit.

Plasticity limit may be defined as the change in moisture content producing a 100-fold change in strength of the soil. If the cone penetrometer is fitted with two cones of different masses M_1 and M_2, and the corresponding penetration for the same soil plotted on the semi-log plot, a set of parallel lines is obtained.

Figure 4.2: Liquid Limit Apparatus

Table 4.3: Results of Liquid Limit of Coal Refuse Samples

S. No.	Empty weight of the pan (in gram)	Weight of soil + pan (before heating) (in gram)	Weight of soil + pan (after heating) (in gram)	No. of blows	Moisture percentage	Average moisture percentage
Mine 1						
1.	12.56	36.51	31.65	18*	20.29	20.29
Mine 2						
1.	5.61	22.80	17.73	35	29.65	30.35
2.	5.93	23.33	18.06	28	30.40	
3.	5.45	22.12	17.01	25	30.53	
4.	5.72	23.72	18.22	18	30.55	
5.	5.54	25.53	19.31	15	31.00	
Mine 3						
1.	13.20	31.43	27.12	25	23.63	24.76
2.	13.20	31.26	26.93	23	24.02	
3.	13.20	31.22	26.90	32	24.44	
4.	13.60	33.19	28.30	20	25.51	
5.	9.70	30.37	24.91	15	26.21	
Mine 4						
1.	12.59	34.60	28.99	32	25.48	25.65
2.	13.21	32.85	27.56	30	26.93	
3.	13.32	35.98	30.36	24	24.80	
4.	13.27	36.83	30.92	19	25.08	
5.	13.45	35.82	30.01	16	25.97	
Mine 5						
1.	5.22	18.93	16.11	34	20.56	21.26
2.	5.60	20.25	17.06	24	21.77	

3.	6.12	20.11	17.07	18	21.72	
4.	5.63	22.33	18.80	34	21.13	
5.	5.81	25.50	21.34	27	21.12	
Mine 6						
1.	13.60	67.85	56.53	24*	20.87	20.87
Mine 7						
1.	12.49	27.69	24.42	35	21.51	
2.	13.32	28.32	25.04	44	21.86	
3.	13.27	29.34	25.85	28	21.71	22.10
4.	12.15	29.40	25.69	13	21.50	
5.	12.59	31.72	27.14	20	23.94	
Mine 8						
1.	13.19	63.95	53.36	24*	20.86	20.86
Mine 9						
1.	13.28	35.59	30.61	24*	22.32	22.32

* - Penetration depth in Depth Penetration method

Table 4.4: Results of Plastic Limit of Coal Refuse Samples

S. No.	Empty weight of the pan (in gm)	Weight of soil + pan (before heating) (in gm)	Weight of soil + pan (after heating) (in gm)	Moisture percentage	Average Moisture percentage
Mine 1					
1.	5.19	11.11	10.24	15.25	
2.	5.61	11.32	10.41	15.79	15.42
3.	5.40	10.01	9.31	15.22	
Mine 2					
1.	5.81	11.61	10.38	20.69	
2.	6.12	11.49	10.70	14.81	16.92
3.	5.52	11.39	10.51	15.25	
Mine 3					
1.	5.80	12.76	11.62	15.94	
2.	5.57	11.04	10.23	14.54	15.08
3.	5.55	11.66	10.71	14.75	
Mine 4					
1.	5.24	14.34	12.77	17.25	
2.	5.62	13.39	12.06	17.11	17.01
3.	6.12	14.27	12.91	16.68	
Mine 5					
1.	5.60	9.36	8.92	11.70	
2.	5.47	8.74	8.33	12.53	12.19
3.	5.55	9.46	8.97	12.34	
Mine 7					
1.	5.36	12.49	11.42	15.00	
2.	6.39	14.09	12.65	14.67	14.57
3.	5.22	12.13	11.16	14.03	
Mine 9					
1.	5.22	12.89	11.49	18.25	19.29

2.	5.36	13.58	11.93	20.07
3.	6.28	14.97	13.27	19.56

4.3 STANDARD PROCTOR COMPACTION TEST

This test is conducted to determine the maximum practically achievable density of soils and aggregates and also the moisture content at which it can be achieved. This method is generally used in construction industry as a means of comparing the densities of the soil obtained in the laboratory to those obtained in the field. The results of this tests may be used to compare the compaction values obtained for the potential backfilling materials. This can be done if both the material may be compacted with same effort. This gives an indication, which material may achieve higher densities following compaction, by any means.

Experimental Setup

A cylindrical mould of internal diameter 102 mm and an effective height of 117 mm, with a volume of 0.945 liter is required for the experiment. A detachable collar of 50 mm effective height (60 mm total height) is required to get the sample to the height of original collar. A detachable base plate is also required. A 50 mm diameter metal rammer of weight 2.5 kg and a height fall of 300 mm, moving in a metallic outer sleeve is required for compaction of the sample. The photographic view of Proctor compaction apparatus is as presented in figure 4.3.

Figure 4.3: Proctor Compaction Apparatus

Experimental Procedure

Approximately 4.5kg of air-dried soil was taken in the mixing pan.

Water was added to increase the moisture content by 5%.

Weigh of the empty mould without the base plate and collar was determined, W_1.

The collar and the base plate were fixed.

First portion of the soil was placed in the proctor mould and was compacted applying 25 blows.

Second layer was placed by applying 25 blows, followed by placing of the last portion and by applying another 25 blows.

It was ensured that the final layer of the compacted soil is just above the rim of the compaction mold. The collar was detached carefully, without disturbing the compacted soil inside the mould and using a straight edge the excess soil on the mould was trimmed and leveled.

The weight of mould was determined, with moist soil W_2. The sample was extruded and taken for water content determination preferably from the middle of the specimen.

Weigh of an empty moisture can was noted as W_3 and was noted as W_4 when weighed with the moist soil obtained from the extruded sample.

Rest of the compacted soil sample was broken with hand and water was added to it to increase the moisture content by 2%.

Steps 4 – 11were repeated till we observe decrease in value of W_2 was noticed.

After 24hours, the samples were recovered from the oven and their weights were noted as W_5.

Moisture content and determined using the following formulae

$$M = (W_4-W_5)/(W_4-W_3)$$

$$Y = (W_2-W_1)$$

Where

M = Moisture Content (in %)

Y = Wet Density (Kg/cm^3)

Y_d = Dry Density (Kg/cm^3)

Table 4.5: Results of Proctor Compaction Test of Coal Refuse Samples

W_1	W_2	$W_2 - W_1$	W_3	W_4	W_5	M	Y	Y_d
Mine 1								
2.196	4.198	2.002	13.32	45.08	42.40	8.44	2.040	1.881
2.196	4.307	2.111	12.61	51.02	47.35	9.55	2.151	1.964
2.196	4.399	2.203	12.19	55.28	50.73	10.56	2.245	2.031
2.196	4.352	2.156	12.57	51.95	47.50	11.30	2.197	1.974
Mine 2								
2.385	4.241	1.856	16.7	87.0	81.0	8.53	1.922	1.764
2.385	4.384	1.999	21.5	86.5	80.6	9.07	2.070	1.890
2.385	4.456	2.071	21.7	76.1	69.6	11.90	2.145	1.917
2.385	4.470	2.085	20.9	77.8	70.5	12.83	2.160	1.965
2.385	4.435	2.050	20.5	63.9	57.5	14.70	2.120	1.849
Mine 3								
2.385	4.328	1.943	20.1	79.7	75.7	6.71	2.012	1.886
2.385	4.400	2.015	21.3	85.3	79.7	8.75	2.087	1.919
2.385	4.508	2.123	21.0	75.1	69.6	10.20	2.199	1.995
2.385	4.469	2.084	20.1	76.8	70.6	11.00	2.158	1.940
Mine 4								
2.196	4.104	1.908	13.29	52.36	49.09	8.37	1.944	1.794
2.196	4.199	2.003	13.81	56.89	52.83	9.42	2.041	1.865
2.196	4.304	2.108	12.27	56.32	51.85	10.15	2.148	1.950
2.196	4.409	2.213	12.57	57.69	52.63	11.21	2.255	2.207
2.196	4.356	2.160	13.36	60.39	54.56	12.40	2.201	1.958
Mine 5								
2.233	4.133	1.900	12.19	33.48	31.80	7.891	1.936	1.795
2.233	4.362	2.129	13.61	47.38	44.24	9.298	2.170	1.985
2.233	4.438	2.205	12.36	69.86	64.04	10.122	2.247	2.041
2.233	4.338	2.105	13.32	53.59	49.11	11.125	2.145	1.930
Mine 6								
2.196	3.990	1.794	13.27	44.27	41.99	7.355	1.828	1.703
2.196	4.124	1.928	12.56	49.04	46.03	8.251	1.765	1.815
2.196	4.232	2.036	13.32	62.84	58.32	9.128	2.075	1.901
2.196	4.204	2.008	13.19	61.28	56.20	10.564	2.046	1.851
Mine 7								
2.233	4.063	1.830	13.29	43.09	40.63	8.26	1.865	1.723
2.233	4.176	1.943	12.81	46.37	43.23	9.36	1.980	1.811
2.233	4.352	2.119	12.57	49.33	45.46	10.53	2.159	1.954
2.233	4.304	2.071	13.61	50.38	46.22	11.31	2.111	1.896
Mine 8								
2.233	4.139	1.906	12.83	49.38	46.43	8.07	1.942	1.797
2.233	4.267	2.304	12.19	52.67	48.97	9.14	2.072	1.899

2.233	4.393	2.160	12.59	53.19	48.93	10.49	2.201	1.992
2.233	4.347	2.114	13.36	56.8	51.85	11.44	2.154	1.933
Mine 9								
2.196	4.121	1.925	12.16	43.13	40.90	7.20	1.961	1.830
2.196	4.259	2.063	13.64	73.62	67.79	9.72	2.102	1.916
2.196	4.392	2.196	12.34	75.42	68.91	10.32	2.237	2.029
2.196	4.323	2.127	13.21	76.84	69.52	11.50	2.167	1.944

4.4 PERMEABILITY TEST

Permeability refers to the ability of a porous material to allow a liquid to pass through its pores. Since the pores are connected with each other, the flow of a liquid takes place through the pores if there is difference in head at the two ends of the sample.

Constant Head Test

Constant head is a large diameter glass tube with stopper and air intake tube at top and outlet nozzle at the bottom. The tank is initially filled completely with water, without any air space. As soon as water starts flowing out of the tank and air enters the air intake tube, the pressure at air intake tube becomes atmospheric and it remains so long as water level remains above the bottom of the tube. Constant hydraulic head causing the flow is the elevation difference between the bottom levels in the bottom tank. The head can be increased or decreased by raising or lowering the air intake tube. The photographic view of permeability test apparatus used for the experiments is as presented in figure 4.4.

In an alternate setup, a reservoir with an inlet and overflow serves as a constant head tank. The permeameter has a Lucite mold for containing the specimen. Filters of coarse materials are placed at top and bottom of the specimen to prevent fines being washed out. While testing less previous fines sands and silty sands, more actual results can be obtained by measuring the actual head less over a fixed length in the specimen by inserting piezometric tube. Flow through the specimen is maintained long enough so that the flow rate becomes constant. Once the steady state has been reached, the quantity of flow “Q” during a time interval “t” is collected and measured. The water should be collected only after a steady state of flow has been established.

If Q is the total quantity of water collected in the measuring jar after flowing through the soil in an elapsed time t, from Darcy’s law;

$$q = Q / t = k \times i \times a$$

$$k = \frac{Q}{t} \times \frac{1}{ia} = \frac{Q}{t} \times \frac{l}{aH} = \frac{Ql}{tHa}$$

Where,

K = Darcy's coefficient of permeability

l and a =Length and area of cross section of soil specimen

H = Hydraulic head causing flow

Figure 4.4: Permeability Test Apparatus

Specimen Height (L): 12.8cm

Area of sample mould(A): 78.5cm^2

Table 4.6: Permeability Test Results of Coal Refuse Samples

S. No.	Head h (cm)	Quantity of water Q (cc)	Time t (sec)	Permeability k (x 10^{-5} cm/sec)	Average Permeability k (x 10^{-5} cm/sec)
Mine 1					
1	235.8	245	900	18.824	
2	235.8	502	1800	19.285	19.072
3	235.8	746	2700	19.106	
Mine 2					
1	235.8	25	1800	95.96	
2	235.8	38	2700	97.24	96.37
3	235.8	50	3600	95.96	
Mine 3					
1	235.8	8	7200	7.677	
2	235.8	10	8100	8.292	7.882
3	235.8	10	9000	7.677	
Mine 4					
1	235.8	47	900	3.611	
2	235.8	97	1800	3.727	3.641
3	235.8	140	2700	3.586	
Mine 5					
1	235.8	35	900	2.689	
2	235.8	75	1800	2.881	2.753
3	235.8	105	2700	2.689	
Mine 6					
1	235.8	264	900	20.284	
2	235.8	534	1800	20.515	20.387
3	235.8	795	2700	20.361	
Mine 7					
1	235.8	39	900	2.997	
2	235.8	81	1800	3.112	3.052
3	235.8	119	2700	3.048	
Mine 8					
1	235.8	195	900	14.826	
2	235.8	394	1800	15.136	15.127
3	235.8	596	2700	15.264	
Mine 9					
1	235.8	43	900	3.304	
2	235.8	89	1800	3.432	3.385
3	235.8	134	2700	3.432	

4.5 TRIAXIAL TEST

When a rock mass is subjected to an all round pressure and if further subjected to an additional vertical pressure, then strength exhibited by the rock is known as Triaxial compressive strength. The lateral pressure acting on the rock mass is known as hydrostatic pressure and the additional vertical pressure is known as deviatric stress. In a labortary, strength is evaluated by Triaxial compression test. A sample of L/D ratio varying from 2 to 2.5 is kept in a chamber in which a fluid pressure is applied from all 3 directions. Through plunger, vertical load is applied to it, which causes failure of the sample. The lateral fluid pressure corresponds to the all-round pressure existing in the rock mass in the field.

Experimental Setup

The triaxial apparatus comprises of a high pressure cell to house the specimen provided with connections for introducing high pressure fluid, with arrangements for loading the specimen and with appropriate connections for switching in pressure and strain measuring devices. A loading machine provides the axial loading and a high pressure pump introduces oil for lateral loading of the specimen along with load and strain measuring and recording apparatus. The photographic view of the triaxial cell has been presented in figure 4.5.

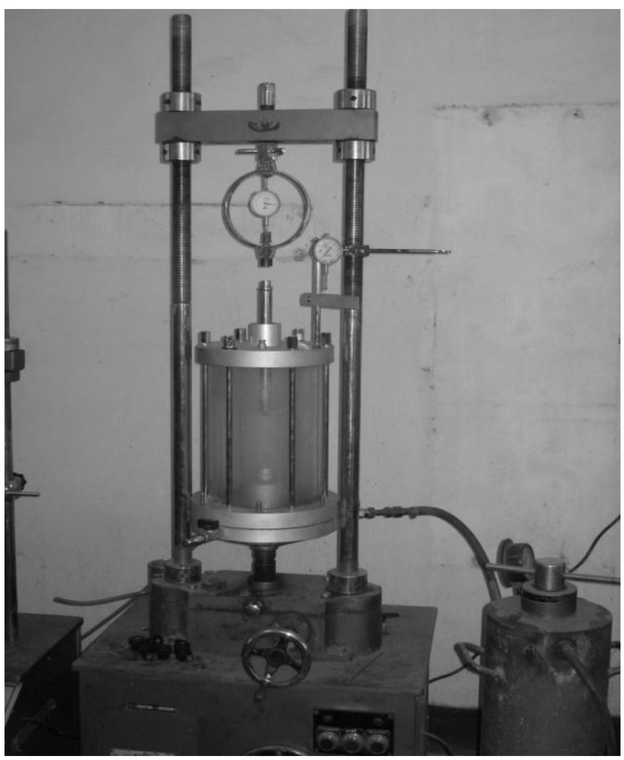


Figure 4.5: Triaxial Cell

Experimental Procedure

The thickness of the membrane was measured by a scale. The thickness is best obtained by measuring the membrane doubled and then halving the measurement.

A porous stone was fixed at the bottom.

A rubber membrane of proper diameter was attached to the bottom platen with rubber “O” rings. It was made sure that the membrane overlaped the platen at least ½”.

Specimen mold was placed around the rubber membrane. The top portion of the membrane was folded down the mold, taking care that the membrane was not twisted or pinched.

Vacuum to pull the membrane to pull the membrane against the side of the mold.

The sand was placed in the membrane and mold by tampering each spoonful of soil, taking care not to pinch the membrane with tamper. Scarify the top of each layer before placing the next one, to reduce stratification.

The dish of soil was weighed again.

A porous stone was placed and the top platen onto the sand. The membrane was rolled onto the top of the platen and was sealed to the platen with rubber “O” ring.

The vacuum was released to the mold.

The vacuum line was attached from the bottom platen.

The specimen mold was removed and the membrane was observed to be free from holes and obvious leaks. If any are found then the sample was rebuilt using a new-membrane.

Four height measurements were obtained approximately 90^o apart. Two diametric readings were taken again 90^o apart. These measurements are accurate to 0.1cm.

The loading frame was turned on and was set to the desired strain rate.

The test was continued till the sample was broken.

The vacuum was released to remove the sample.

Both lateral pressure and axial pressure is noted.
The above procedure is repeated for two other samples.
On a graph paper Mohr’s circles are drawn and common tangent is drawn. The internal angle of friction (Ø) and cohesion (C) is determined from these plots.

The specification of the samples are :

Length of each soil specimen prepared = 8 cm.

Average diameter of the soil specimen = 3.94 cm.

Volume of the soil Specimen = 97.49cm^3

L/D = 2.03

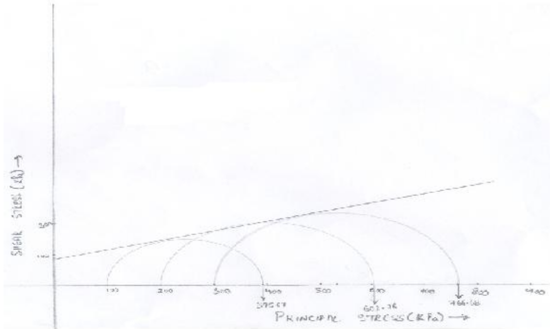
Least count of dial gauge reading = 0.01 mm

A sample reading of triaxial test for one of the samples (mine no.1) has been presented in table 4.7. The same procedure was repeated for all the samples and the Mohr's circles of all the samples are as presented in figures 4.6(a) to4.6(i). The internal angle of friction (Ø) and cohesion (C) is determined from these plots has been presented in table 4.8

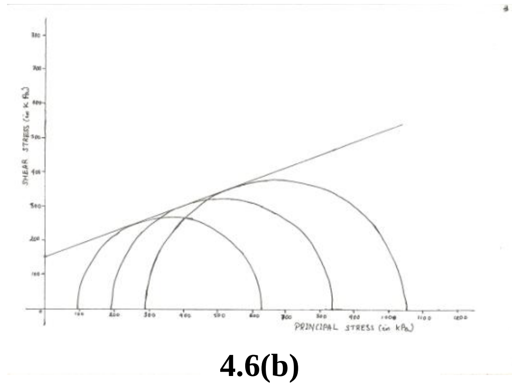
Table 4.7: Triaxial Test Reading of Sample from Mine No. 1

S. NO.	A	B	σ_3 = 100Kpa			σ_3 = 200Kpa			σ_3 = 300Kpa		
			C	D	E	C	D	E	C	D	E
1	0	12.19	0	0	0	0	0	0	0	0	0
2	50	12.25	15	51	41.62	15	51	41.62	19	64.6	52.72
3	100	12.31	25	85	69.03	19	64.6	52.46	22	74.8	60.74
4	150	12.38	35	119	96.156	38	129.2	104.39	39	132.6	107.14
5	200	12.44	40	136	109.33	62	210.8	169.47	67	227.8	183.13
6	250	12.50	57	193.8	155.00	93	316.2	252.90	98	333.2	266.50
7	300	12.57	86	292.4	232.67	109	370.6	294.89	117	397.8	316.54
8	350	12.63	99	336.6	266.46	119	404.6	320.29	139	472.6	374.12
9	400	12.70	108	367.2	289.18	131	445.4	350.76	156	530.4	417.70
10	450	12.76	111	377.4	295.66	148	503.2	394.22	165	561	439.50
11	500	12.83	111	377.4	294.11	152	516.8	402.75	171	581.4	453.10
12	550	12.90	-	-	-	152	516.8	400.63	176	598.4	463.89
13	600	12.97	-	-	-	-	-	-	178	605.2	466.68
14	650	13.04	-	-	-	-	-	-	178	605.2	464.20

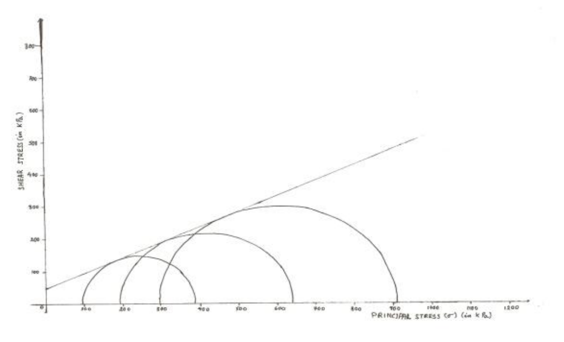
A = Dial Gauge Reading, B = Corrected Area, C = Proving Reading , D = Deviatory Reading, E = Deviatory Stress (KPa)



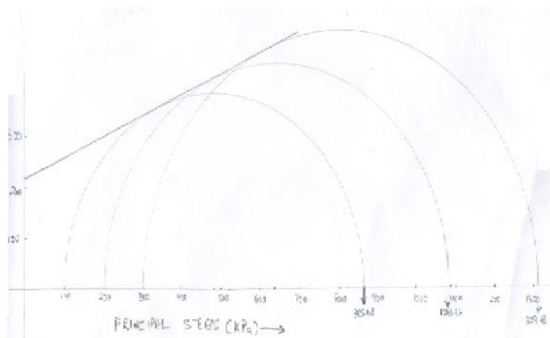
4.6(a)



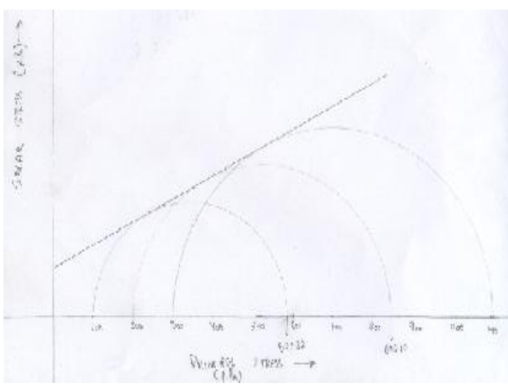
4.6(b)



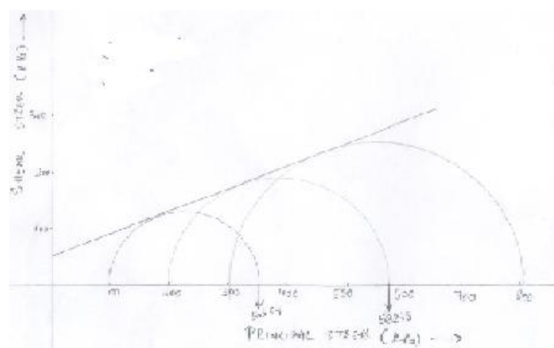
4.6(c)



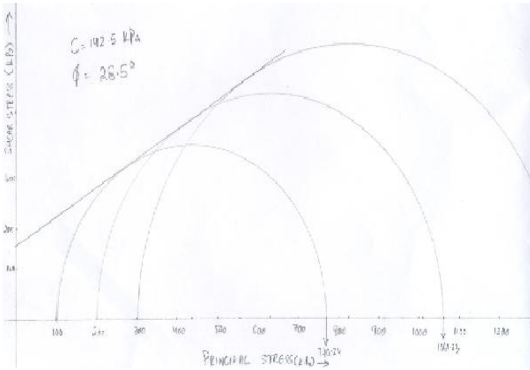
4.6(d)



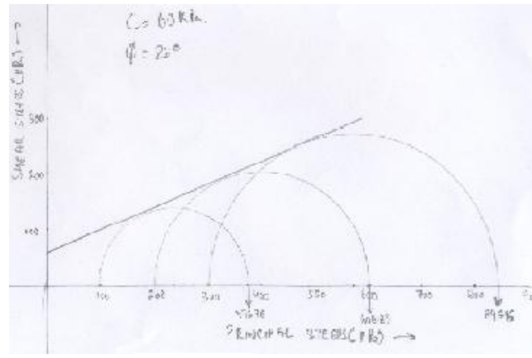
4.6(e)



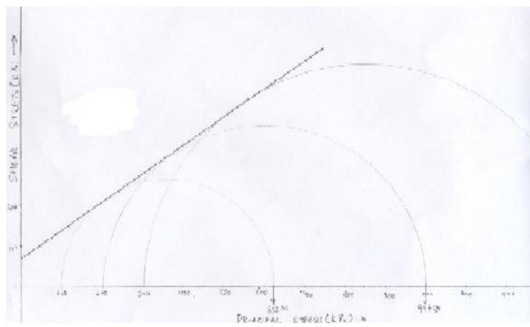
4.6(f)



4.6(g)



4.6(h)



4.6(i)

Table 4.8: Results of Triaxial Tests

Sample No.	Φ(degrees)	C(KPa)
Mine 1	16.0	85
Mine 2	20.56	150
Mine 3	25.71	47.5
Mine 4	22.5	215
Mine 5	25.0	105
Mine 6	21.0	50
Mine 7	28.5	142.5
Mine 8	22·0	60
Mine 9	34.0	70

4.6 PARTICLE SIZE ANALYSIS

Assuming other parameters to be same, different grain sizes during and following backfilling, one can analyze particle sizes to predict how a fill composed of a given material may be expected to behave. A fill with well-graded particles will offer more resistance to displacement and settlement than one with uniformly graded particles.

Particle size distribution means the range of sizes of particles in a soil and percentage of particles which occur within a range. This can be done either by sieve analysis or sedimentation analysis, former being preferred for coarse grained and later for fine grained sols. In many cases combination of both is required. In this experiment we require series of sieves of size, weighing machine and a vibrating machine. The photographic view of the sieves in sieve shaker with sieves has been presented in figure 4.7.

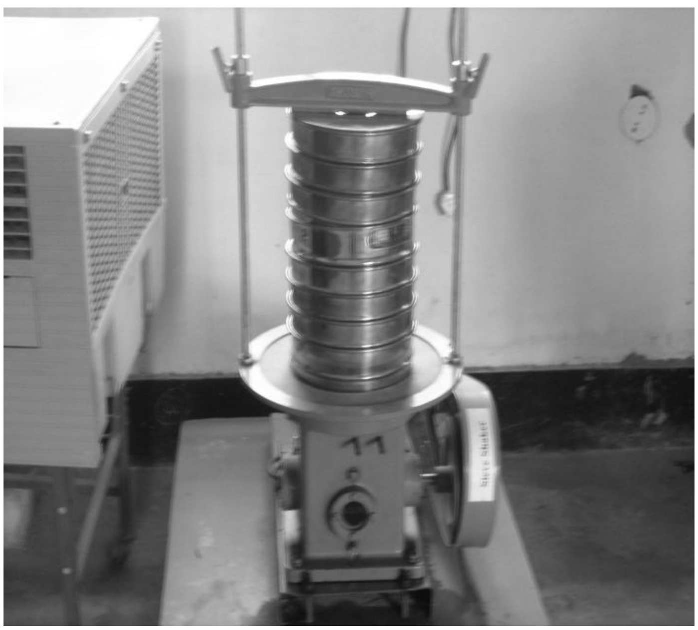


Figure 4.7: Sieve shaker with sieves

Experimental Procedure

A series of sieve of 75mm, 37.5mm, 19mm, 9.5mm, 4.75mm, 2mm and 1mm and 600, 425, 215, 150, 75 were stacked with the larger size over the smaller.

A receiver was kept at the bottom and a cover was kept at the top of the assembly.

The soil sample to tested was dried, clumps were broken if necessary, and the sample was passed through the series of sieve by shaking.

The fraction retained on and passing 2 mm IS sieve was tested separately.

An automatic sieve-shaker, run by an electric motor, may be used; about 10 – 15 minutes of shaking was considered adequate.

Larger particles were caught on the upper sieves, while the smaller ones filter through to be caught on one of the smaller underlying sieve.

The material retained on any particular sieve should naturally include that retained on the sieve on top of it, since sieves are arranged by there aperture size decreasing from top to bottom. The weight of material retained on each sieve was converted to a percentage of the total sample. The percentage material finer than a sieve size may be known by subtracting this from 100.the material passing the bottom most sieve, which is actually the 75 u sieve, is used for conducting sedimentation analysis for the fine fraction.

If the soil is clayey in nature the fine fraction cannot be easily passed through the 75 u sieve in the dry condition. In such a case, material is to be washed through it with water (preferably mixed with 2 g of sodium hexametaphosphate per litre), until the washed water is fairly clean. The material which passes through the sieve is obtained by evaporation. This is called “wet sieve analysis” and may be required in the case of cohesive granular soils.

A particle size distribution curve was plotted on semi-log coordinates, where the sieve size is on a horizontal logarithmic scale, and the percentage by weight of the size smaller than a particular sieve size is on a vertical arithmetic scale.

CALCULATION

From the graph the coefficient of uniformity (C_u) and the shape of the grain size distribution curve called coefficient of curvature (C_c) were determined by using the following relation:

$$C_u=D_{60}/D_{10}$$

and $$C_c=(D_{30})^2/(D_{10}*D_{60})$$

Where

D_{30}=30% finer size

D_{10}=10% finer size

D_{60}=60%finer size

Usually in soil if C_u <5 it is to be of medium uniformity

If C_u =(5-15) it is said to be very non uniform

If C_u >15 it is said to be well graded

C_c should be 1-3 for a well graded soil

Table 4.9 below shows cumulative frequency of Particle size analysis of different samples. The coefficient of uniformity (C_u) and the coefficient of curvature (C_c) was determined for all the samples and the results are as presented in table 4.10.

Table 4.9: Cummulative Percentage of Particle Size Analysis of Refuse Samples

Sieve Size	Mine 1	Mine 2	Mine 3	Mine 4	Mine 5	Mine 6	Mine 7	Mine 8	Mine 9
40mm	00.00	0	0	4.30	0.00	7.65	4.30	0.00	5.90
20mm	35.70	0	0	15.55	8.65	25.25	11.55	12.15	13.50
10mm	43.35	0	0	31.50	20.80	38.70	32.35	21.60	19.25
6.3mm	45.05	0	0	33.75	28.85	40.70	39.75	27.20	23.30
4.75mm	45.45	6.52	2.86	34.90	30.65	41.45	41.35	30.40	26.85
2mm	52.60	28.2	24.22	38.80	45.40	44.75	48.45	48.35	42.80
1mm	58.15	57.38	51.42	44.05	60.35	48.90	54.90	56.35	50.35
600μ	69.30	71.26	61.24	49.55	71.25	53.50	62.75	68.40	62.30
425 μ	82.80	88.14	72.38	59.80	84.35	63.20	82.40	80.60	78.20
212 μ	95.40	97.65	89.04	89.55	97.40	87.95	92.35	94.55	93.90
150 μ	97.35	98.49	94.3	95.50	98.85	93.45	94.35	98.00	97.85
75 μ	99.65	99.87	99.82	98.75	99.85	96.25	94.85	99.10	98.95
Tray	100	100	100	100	100	100	100	100	100

Table 4.10: Coefficient of Curvature and Coefficient of Uniformity of samples

Sample No.	C_u	C_c
Mine 1	60.00	0.0667
Mine 2	3.75	0.6000
Mine 3	7.00	0.7232
Mine 4	8.57	0.3058
Mine 5	8.24	0.3782
Mine 6	45.00	0.0681
Mine 7	26.43	0.1356
Mine 8	10.71	0.3733
Mine 9	10.00	0.4000

5. DISCUSSION AND CONCLUSION

5.1 DISCUSSION

The coal refuse may experience wetting and drying cycles during and following placement which can degrade the particles and subsequently alter the fill's mechanical properties. Slake durability testing qualitatively assesses the resistance offered by weak rocks such as shales, mud stones, silt stones and other clay bearing rocks to weakening and disintegration when subjected to two standard cycles of wetting and drying. Resistance to slaking is significant since it can be expected that degradation of the refuse will reduce its strength and permeability. Therefore, it is felt that it would be desirable for coal refuse to be used as stowing material to have at least medium durability, i.e.: Id_1 should be greater than 95 and Id_2 greater than 85. It could be seen from the experimental studies (Table 2) that the refuse from Mine number 1, 5 and 7 possess the desired characteristics and are suitable for back filling, where as the Id_1 and Id_2 values are from Mine number 2, 3, 4, 6 indicate that the refuse is susceptible to moderate amount of disintegration when exposed to wetting and drying cycles.

According to a slaking study performed by Morgenstern and Eigenbrod (1974), when the liquid limits for the weathered particles was in the range of 20 to 50, rocks exhibit only small amount of slaking and are generally slightly plastic. However, an important aspect of refuse having plastic properties is that when excess pore pressures are there in the fill, it may not be quickly dissipated as the material is subjected to increase in stress. Thus, it would be desirable for coal refuse, considered as a potential backfilling material, to be non-plastic.

Liquid limits can be qualitatively related to compressibility of soils. Soils with liquid limit from 0-30 should have a low compressibility. Soils with liquid limit from 31-50 should be moderately compressible, and soil having a liquid limit over 51 should be highly compressible (Sowers, 1979). The experimental results show that the liquid limit for the sample of Mines 1, 3, 4, 5, 6, 7, 8 and 9 lies in low compressible zone, whereas the sample of Mine 2 lies in the Moderately compressible region. These samples are very low and medium compressible and could be termed very slightly plastic.

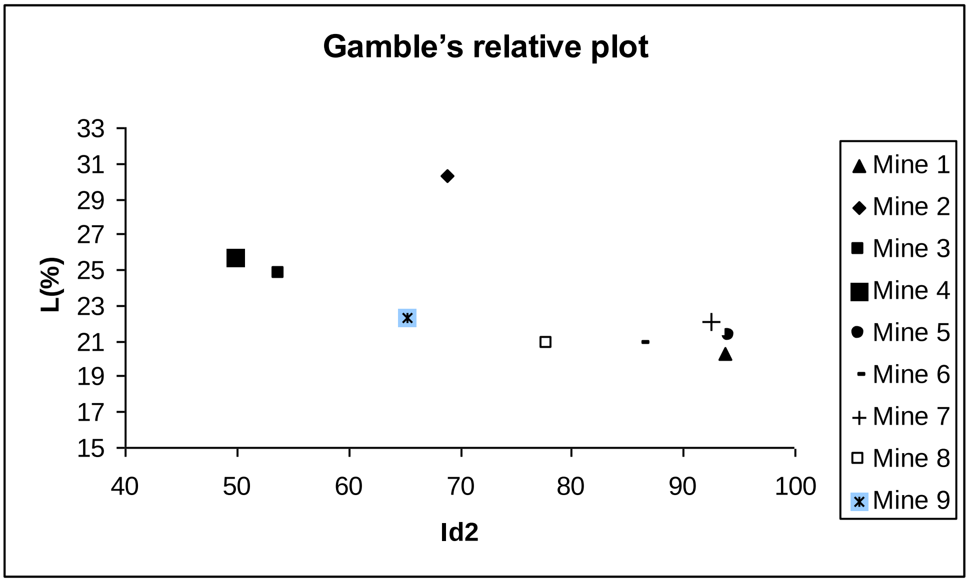


Figure 5.1: Gamble's relative plot

Compaction testing of the refuse determines the moisture content which will achieve the maximum dry density for a material that has been compacted with a given compactive effort (Wray, 1986). It could be observed from the test results that the maximum dry density varies from 1.901 Mg/m^3 to 2.207 Mg/m^3. When compared with the compacted densities of sand (which varies between 1.7Mg/m^3 and 2.2 Mg/m^3) which has been successfully used as hydraulic stowing material in India, it could be seen that the refuse from all the mines can be suitably used as a backfilling material.

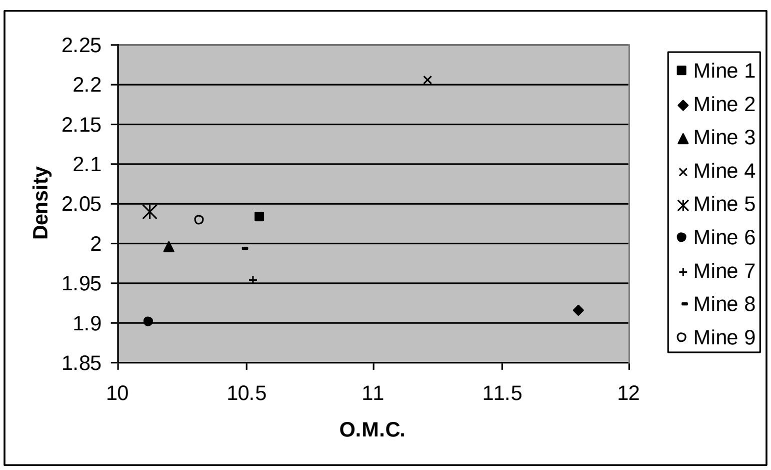


Figure 5.2: Standard Proctor Curve

As stated earlier, the nature of water flow through an unconsolidated material will have a great effect on physical properties of that material. The test performed here measures the permeability of compacted sample. Thus in actual applications, the coefficient of permeability of the backfilled material will probably, at least initially, be higher, since placement density is not going to be as high as compacted density of the refuse in this test. For a material to be considered as suitable for backfilling, the coefficient of permeability should be at least $2.78 * 10^{-5}$ m/s. From the experimental results it may be observed that the samples for mine no. 1, 4, 6, 7, 8, 9 has the required permeability and is suitable for backfilling whereas for mine no. 2, 3, 5, permeability is less than $2.78 * 10^{-5}$ m/s and so is not suitable for filling purposes.

Sands with little or no fines generally have a much higher residual angle of friction. For example, the residual shear strength of fiver sand has been determined to be around 33^{o} for both drained and un-drained conditions. Since river sand has been proven to be successful backfilling material (Sinha,1989), it would be desirable for coal refuse to have at least a comparable residual angle of friction, say around 30^{o}. From the experimental results it may be observed that the samples for mine no. 9 has the required residual angle of friction and is suitable for backfilling whereas for mine no. 1, 2, 3, 4, 5, 6, 7, 8 has residual angle of friction less than 30^{o} and so they are not preferred for filling purposes.

Sands with little or no fines generally have a much higher residual angle of friction. For example, the residual shear strength of fiver sand has been determined to be around 33^{0} for both drained and un-drained conditions. Since river sand has been proven to be successful backfilling material (Sinha,1989), it would be desirable for coal refuse to have at least a comparable residual angle of friction, say around 30o. From the experimental results it may be observed that the samples for Mine No. 9 has the required residual angle of friction and is suitable for backfilling whereas for Mine No. 1, 2, 3, 4, 5, 6, 7, 8 has residual angle of friction less than 30^{0} and so they are not preferred for filling purposes.

Samples with U value less than 5, is said to have medium uniformity. For u value between 5 and 15, the ample is said to have very non-uniform. And for the U value greater than 15, sample is said to be well graded. From the above analysis, it is inferred that mine no.2 has medium uniformity. Samples of mine no. 3, 4, 5, 8 and 9, show very non-uniform nature. Mine no. 1, 6 and 7, show well graded distribution.

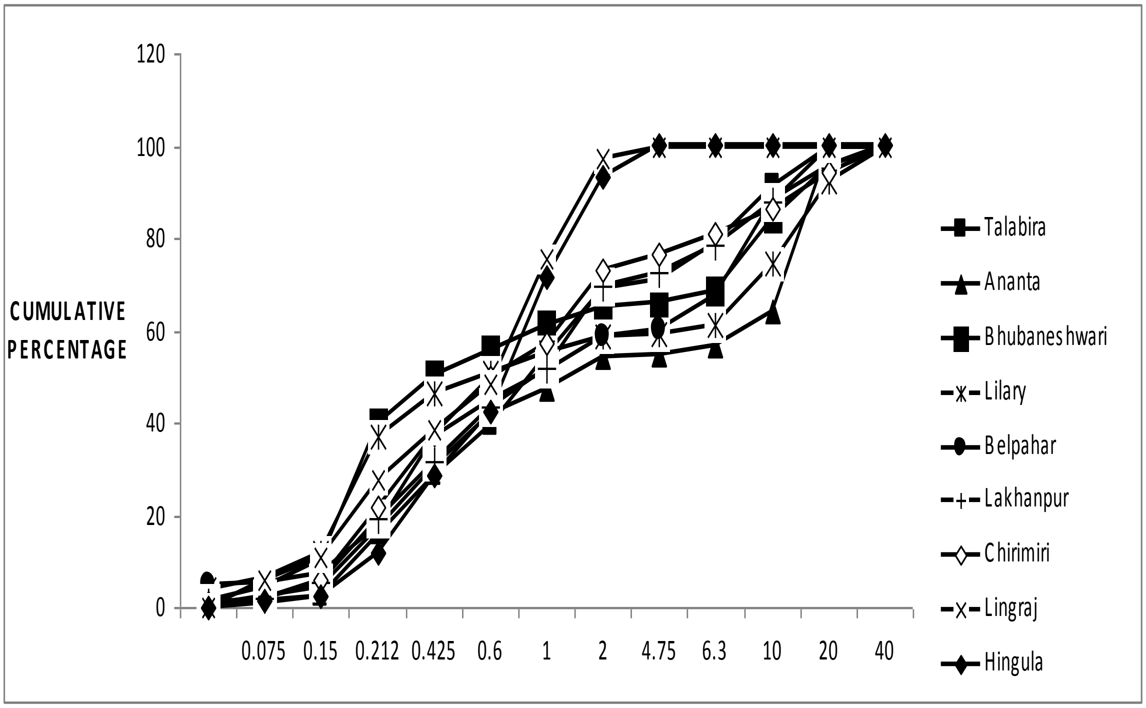


Figure 5.3: Cumulative percentage vs Particle Size graph

5.2 CONCLUSION

Geotechnical tests were performed for the coal mine refuse samples of different mines to evaluate the their suitability as filling material. These test include Slake Durability, Standard Proctor Compaction test, Triaxial test, Permeability test and Atterberg limits test. From the result of these tests it may be concluded that coal refuse of sample no. 1, sample no. 5 and sample no. 7 can be used for the purpose of backfilling without much treatment. But all other samples need some treatment such as removal of some fine particles, mixing with some amount of cement or some other binding material so that its strength increases and it does not deteriorate when subjected to wetting and drying cycles.

5.3 SCOPE FOR FURTHER STUDY

Leachate analysis is an important study in assessing the suitability of coal mine refuse for backfilling. The results of leachate analysis will help in determining the adverse environmental effects.

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