Geotechnical and Geological Engineering, 1996, **14** 129–150

Characterization of coal-mine refuse as backfilling material

M. G. KARFAKIS*, C. H. BOWMAN† and E. TOPUZ*

**Department of Mining and Minerals Engineering, Virginia Tech, Blacksburg, VA 24061, USA, †1310 Small Court, Gillette, WY 82716, USA*

Summary

With increased production and more stringent regulations for air, water and ground pollution control, the safe and environmentally acceptable disposal of coal-mine refuse is becoming ever more demanding. Backfilling may provide an environmentally acceptable method for the disposal of waste materials. Increased resource recovery, enhanced ventilation control, and minimizing surface subsidence, underground coal-mine fires and spontaneous combustion of coal are potential advantages of backfilling. In this paper the physical and mechanical properties of coal-mine waste from different sites are described and the effects of these properties on the duty requirements of fill material are assessed. As a result of 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 as a stowing material.

Keywords: mine refuse, backfilling

Introduction

Coal mining produces large amounts of refuse material. The amount of waste generated by US coal mines amounts to more than 150 million tons annually. At present, with the exception of a few small-scale underground waste-disposal operations in abandoned coal mines, most of this waste is disposed of at the surface, which inevitably requires extensive planning and control to minimize the environmental impact of mining. It also results in non-productive use of the land, the potential for air and water pollution, possible failure of waste embankments, and the loss of aesthetic value of the land. These problems can be alleviated by disposing of the refuse underground.

Backfilling of coal mines can be performed in conjunction with mining, or even after a mine has been abandoned. In addition to reducing mine subsidence, other potential benefits that might be gained include increased coal recovery, enhanced ventilation control, and minimizing underground mine fires. Coal refuse can also be disposed, provided that it is deemed a suitable 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 remnant pillars by providing lateral support, less coal would be required in pillars. Backfilling in active European coal mines has been popular, particularly during the late 1950s when demand for coal was high. To mine as much coal as possible, it was decided that longwall mining and extracting contiguous coal seams was needed. In order to restrict surface subsidence in heavily populated areas and to facilitate multiple-seam mining, some form of backfilling was deemed necessary (National Academy of Sciences, 1975).

Ventilation control can be enhanced by improving ground control. Leakage in underground coal-mine ventilation systems 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 fire, and because the amount of oxygen that can come into contact with the exposed coal is reduced, the chance of fire is reduced.

Backfilling can be used as an alternative coal refuse disposal method when preparation plant reject is used as stowing material. In US coal mines, around 30% of the mined product sent to preparation plants is rejected as waste, and subsequently disposed of on the surface in waste embankments and settling ponds (Rose *et al.*, 1984).

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 (Vick, 1983). On the other hand, saturated fill may not have the desired strength and stiffness for ground-control requirements, so incorporating cementing admixtures 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 mines include hand, gravity, mechanical, pneumatic and hydraulic. The two most popular stowing methods, both past and present, are pneumatic and hydraulic.

The stowing material used with the different backfilling systems may consist of new or old mine refuse, sand, crushed rock from a quarry, or other types of waste rock such as fly ash. It may also contain cementing agents and other additives. The use of a particular stowing material depends on its availability, cost, and the stowing system being employed. Also important are its *in situ* mechanical properties following placement.

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. Important properties for the *in situ* fill are strength, deformability, ability to dissipate pore pressure, primary and secondary consolidation characteristics, and slake durability. An analysis of these properties will help to determine if the design objectives of the fill will be met.

Mechanical properties affecting fill performance

In actual stowing applications, it may be difficult to place backfill so that it occupies the entire mined out area and has complete contact with the mine roof. In this situation, fill will act as a passive support, only taking load after deformation of the roof and/or pillars. Even when the mine void is completely backfilled, the fill provides little initial support if its stiffness is less than the mine roof and pillars surrounding it. A low-modulus component positioned between two high-modulus components accepts less load than is indicated by their relative cross-sectional areas. The load is distributed around the low-modulus component into the high-modulus components by arching. The modulus of deformation of a well-confined, but undensified, fill will increase as the failing roof and pillars compact the material. Therefore, support potential will increase only after there has been a certain amount of downwards displacement of overlying roof layers, or outwards displacement of the failing pillars.

The void ratio following fill placement is possibly the most important single parameter affecting strength and deformability. In general, the lower the void ratio, the higher the relative density, and the more the fill will be able to withstand deformation when stressed. A dense fill will exhibit an apparent cohesion during shear as grains are forced up, over, and around adjacent confining grains. A loose fill will experience a large amount of plastic deformation and subsequent reduction in volume when stressed due to grain crushing at highly stressed points of contact and grain repositioning.

Fill shear strength is largely dependent on the apparent, or measured, angle of friction (ϕ_m), whose value is determined by considering all the factors of shearing resistance to displacement for the fill particles (Murphy, 1987).

Shear displacement may be caused by distortion, crushing, shifting, rolling and sliding of individual grains. Movement of grains can be reduced by using material composed of minerals that have a rough surface and offer a great amount of resistance to sliding. Resistance to grain displacement can also be increased by using angular, well-graded fill particles that have a large amount of grain-to-grain contact interlocking when stressed (Sowers, 1979).

Shear strength will be greatly affected by the presence of pore water pressure between the fill particles. The shearing resistance of the fill depends on friction, and the amount of friction between individual grains is reduced by pore water pressure. If pore pressure is increased sufficiently to reduce the effective normal stress acting on the fill, the fill will be subjected to liquefaction. This condition can lead to failure if the amount of confinement provided to the fill is not adequate. The ability of the placed fill to dissipate pore pressure is largely dependent on its coefficient of permeability (k). The coefficient of permeability is greatly affected by the percentage of fine particles in the fill material. Adsorption and capillarity restrain water flow, and the influence of both increases with the percentage of small grain sizes. Adsorption is the attraction of water molecules to the surface of clay minerals, which have large electrostatic surface charges. Capillarity can be described as surface tension caused by the attraction of water for solid particles. In a wet (but not saturated) fill, this surface tension will cause a meniscus of water to be formed between the individual solid particles. The tensile stress caused by this meniscus is inversely proportional to the distance between each solid particle, and directly proportional to the cosine of the angle of contact (α) between the meniscus and the solids. Therefore, capillarity effects disappear if the fill is saturated ($\alpha = 90°$). If the fill is not saturated, the presence of fines will result in less distance between the solid particles, thus increasing the tensile stresses between the solids, and reducing water flow through the fill.

Mine operators who have practiced backfilling have reported that fill permeability can be drastically reduced by the presence of even a small percentage of fine particles in the stowing material (Blight, 1979). In order to increase *in situ* permeability, cut-and-fill mine operators generally remove some portion of minus 0.075 mm material before stowing (Wayment, 1978).

The ability of the *in situ* fill to dissipate pore pressure can also be affected by the area in which it is placed. If a fill is placed on an area with a low permeability (such as clay) and/or there is a large amount of ground water flowing into the fill, it may become saturated. Any sudden loads applied to the fill (such as roof failure) may cause pore pressure to rise sufficiently to cause liquefaction.

A fill that does not drain freely, or is composed of weak particles, may experience deformation due to both primary and secondary consolidations (Charles, 1984). Primary consolidation, which is related to the permeability of the fill, is caused by excess pore pressure that slowly expels water trapped between the solid particles of the fill, thus causing compression of the solids. Secondary consolidation (creep) is compression occurring after primary consolidation is complete, and may be experienced when the applied stresses are maintained near the fill's strength for a long period of time. Because of this, creep is most likely to occur in fills that contain weak material, such as clays.

If a backfill material is composed of weak rocks such as shales, alternate wetting and drying cycles may cause deterioration of fill particles due to slaking. Clay-bearing rocks are the most susceptible to slaking. Slaking increases the percentage of fines (particularly clays), which will likely reduce the fill's permeability and ability to dissipate pore pressure, thus reducing the shear strength of the fill.

The mechanical behaviour of an *in situ* fill is greatly affected by placed void ratio, water content and composition of the backfill material. One can have some degree of control over these physical properties by choosing the proper backfilling system and stowing material.

Coal-mine refuse has been used in the US for area backfilling in abandoned coal mines. Since coal refuse is generally a readily available material at most active underground coal mines, it should be examined as a potential stowing material when considering a backfilling system.

Waste generated by coal preparation plants 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 than 0.6 mm, and consists mainly of slurry and tailings. Slurry is the fines remaining in suspension in the processing water after washing. Tailings are the fine rejected material from the froth flotation process used for cleaning fine coal. Fine refuse typically accounts for around 10% by weight of all coal mine refuse (Nandy and Szwilski, 1987).

The mineralogical composition of coal mine refuse is generally clays (illite and kaolinite), quartz, pyrite, haematite, and carbonaceous material. In a study of the relative mineralogy of coal refuse (minus 2 mm fraction) from the Virginia Appalachian Coal Basin (Stewart, 1990), quartz was found to be the dominant mineral.

The susceptibility of coal refuse to weathering can be attributed to the fact that it is largely composed of clay minerals, which, when exposed to moisture, may take up water and swell, causing particle breakdown along laminations and bedding planes. The formation of sulphuric acid due to the oxidation of pyrite can also contribute to the breakdown of refuse particles (Bishop and Simon, 1976).

The quartz content of coal refuse will be an important indicator of its strength and abrasiveness (Franklin and Dusseault, 1989). Generally, increasing amounts of quartz should enhance the strength of the refuse, while increasing its abrasiveness. The abrasiveness will help determine whether preventive measures need to be taken in the design of a backfilling system to avoid undue wear of system components.

The specific gravity of coal refuse is usually in the range of 1.6 to 2.7, with a typical value of 2.2 (Nandy and Szwilski, 1987). The lower values of specific gravity generally correspond to fine coal refuse, which can contain significant amounts of coal.

Since coal-mine refuse is largely composed of minerals with low friction coefficients (clays), it may not have adequate strength to act as a stand-alone ground-control material for all situations. In certain cases, stabilization with some cementitous material to increase its strength and stiffness may be appropriate. In addition to potentially increasing fill strength, cementation should also help reduce weathering of weak particles in the refuse, as well as decrease the oxidation of any pyrite.

There may, however, be problems using coal refuse with cementing agents, because it contains significant amounts of deleterious materials (clays, coal and pyrite) that are known to have adverse effects on concrete strength. Coal and expansive clays located near the surface of the cemented mix may cause pop-outs, leaving behind voids and new surfaces which are exposed to weathering (Bland *et al.*, 1976). Pyrite located near the surface of the mix can become oxidized in the presence of air and water, forming ferrous sulphate and sulphuric acid. The ferrous sulphate will oxidize further to ferric sulphate. In the presence of alkaline-earth compounds (clay minerals), most of the sulphate ions released by the oxidation of pyrite will become the sulphates of calcium, magnesium, sodium and potassium. Sulphates will react with the principle components of Portland cement to form new, insoluble compounds. Crystallization of the new compounds is accompanied by an increase in molecular volume, causing expansion and subsequent disintegration of the cemented mass. Magnesium, sodium, and potassium sulphates are more water soluble than calcium sulphate, and therefore are more detrimental to the cement bonds (Sleeman, 1987).

Obviously, coal refuse is not an ideal material to use as aggregate, and cementing admixtures may not be readily available or affordable to mine operators. Furthermore, uncemented coal refuse may provide sufficient ground support, particularly in providing passive support to remnant mine pillars. Thus, it would be wise to examine its potential use as a fill material.

Fill characterization tests

Various tests can be performed to help assess the potential performance of coal refuse as a stowing material. The effect of wetting and drying cycles on the durability of the rocks composing the refuse can be determined with the slake durability test. Pertinent qualitative information about the nature of the material can also be obtained from Atterberg tests (plasticity), sieve analysis, and from the moisture–density relationship test. Certain

Table 1. Desired properties of coal refuse as a backfill material

Test	Desired Properties
Slake durability	I_{d1} greater than 95, I_{d2} greater than 85.
Plasticity	Refuse should be non plastic.
Grain size analysis	Minus 0.075 mm material should not exceed 10% for hydraulic stowing systems, the minus 3 mm material should not exceed 20% for pneumatic stowing systems, C_u greater than 4, C_z between 1 and 3.
Permeability	Coefficient of permeability should be at least 2.78×10^{-5} m/s.
Triaxial compression	Residual angle of internal friction should be at least 30°, confined modulus of deformation comparable to modulus of deformation of a failing coal pillar.

engineering properties can be measured with the permeability and triaxial compression tests. Table 1 provides a summary of the desired properties based on both placement and engineering performance considerations.

Coal mine refuse samples

Coarse coal refuse was sampled from nine coal refuse piles in southwest Virginia, each pile containing refuse from different seams.

Each sample was taken from randomly selected points on the surface of the pile, at a depth of approximately 0.6 m. To achieve as representative a sample as possible, a large amount of material was collected from four locations on each pile and then mixed to obtain a representative sample. A separate sample of large particles suitable for slake durability testing was also taken. Table 2 provides information on each sample, including

Table 2. Information on coal refuse samples

Sample number	Date pile started	Name of coal seams disposing in pile	Formation
1	1980	Hagy[a], Pond Creek, Blair, Elkhorn, Millar, Amburgy.	Breathitt
2	1976	Hagy[a], Pond Creek, Blair, Elkhorn, Millar, Amburgy.	Breathitt
3	1978	Splash Dam[a], Blair, Pond Creek.	Breathitt
4	1968	Dorchester[a], Taggart, Imboden, Pardee, Low Splint, Wilson.	Wise, Norton
5	1968	Dorchester[a], Taggart, Imboden, Pardee, Low Splint, Wilson.	Wise, Norton
6	1984	Taggart[a], Low Splint, High Splint, Imboden.	Wise
7	1968	Taggart[a], Low Splint, High Splint, Imboden.	Wise
8	1983	Norton[a], Upper Banner, Dorchester, Clintwood.	Norton, Wise
9	1983	Norton[a], Upper Banner, Dorchester, Clintwood.	Norton Wise

[a]Main seam.

the coal seams and formations from which the refuse came.

As can be seen from the Table, each refuse pile consists of material from numerous seams. Samples 1–3 are from the Breathitt formation, which has been described (Huddle *et al.*, 1963) as consisting of mostly shale, sandstone, siltstone, and coal. The rocks below the coal beds are mainly shale and clay.

Samples 4–9 are from the Wise and Norton formations. The Norton formation consists chiefly of beds of shale and siltstone, with lesser amounts of sandstone (Miller, 1973). Miller described the sandstones of the formation as being clayey and silty, usually micaceous and weakly cemented. The Wise formation is composed of shale, sandstone and 20 or more named coal beds (Brown *et al.*, 1952). The sandstone is described as arkosic and containing a large number of feldspar grains.

Stewart (1990) also tested coarse coal refuse from the same general locations as were tested in this investigation. Stewart noted that the coal refuse which he sampled was mostly composed of shale, with a small amount of sandstone. This was consistent with the composition of the refuse for this investigation.

Slake durability test

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, mudstones, siltstones and other clay-bearing rocks, to weakening and disintegration when subjected to two standard cycles of wetting and drying. The test is an index test, and can be used to compare the slaking properties of one material versus another (ASTM D 4644-87).

Previous investigations have shown that there is a relationship between slake durability and the Atterberg limits. Gamble (1971) suggested that rocks with a low slake durability index should be subjected to plasticity tests to characterize better their potential behaviour in the presence of water. He proposed a classification which is based on the results of the slake durability test as well as the Atterberg limit test. By determining the slake durability index for the first cycle (I_{d1}) and the second cycle (I_{d2}), one can give each sample a general durability ranking based on Gamble's (1971) slake durability ratings.

The slake durability test was performed on 31 replicates of the nine samples, with each sample having from three to four replicates. The I_{d1} rating for all the replicates was between 84 and 99, and the I_{d2} rating between 72 and 98. Sample averages for the I_{d1} and I_{d2} values are shown in Table 3.

Atterberg limits

The Atterberg limit tests are used to determine plasticity. An unconsolidated material which contains little or no clay (such as gravels and clean sands) will not exhibit plasticity, and is considered noncohesive (Sowers, 1979). Noncohesive material is generally considered free draining when used as backfill (Dunn *et al.*, 1980).

Morgenstern and Eigenbrod (1974) reported that there is a relationship between liquid limit and the amount of slaking of argillaceous rocks. They suggested that rocks which have a high liquid limit (above 90) will experience a large degree of slaking when exposed to wetting and drying cycles, while rocks with lower liquid limits will experience less.

Table 3. Slake durability test summary

Sample no.	I_{d1} (%)	I_{d2} (%)	I_{d1} Rating[a]	I_{d2} rating[a]
1	91	83	MD	MD
2	90	82	MD	MD
3	94	90	MD	MHD
4	95	82	MD	MD
5	94	91	MD	MHD
6	97	93	MHD	MHD
7	97	93	MHD	MHD
8	98	96	MHD	HD
9	92	86	MD	MHD

LD = low durability, MD = medium durability, MHD = medium high durability, HD = high durability.
[a]Gamble (1971).

Liquid limits can be qualitatively related to the compressibility of soils. Soils with a liquid limit from 0 to 30 should have a low compressibility. Soils with a liquid limit between 31 and 50 should be moderately compressible, and soil having a liquid limit over 51 should be highly compressible (Sowers, 1979).

Liquid limit, plastic limit, and plasticity index were determined by dry screening a portion of the samples to provide about 300 g of material passing the 0.425 mm sieve.

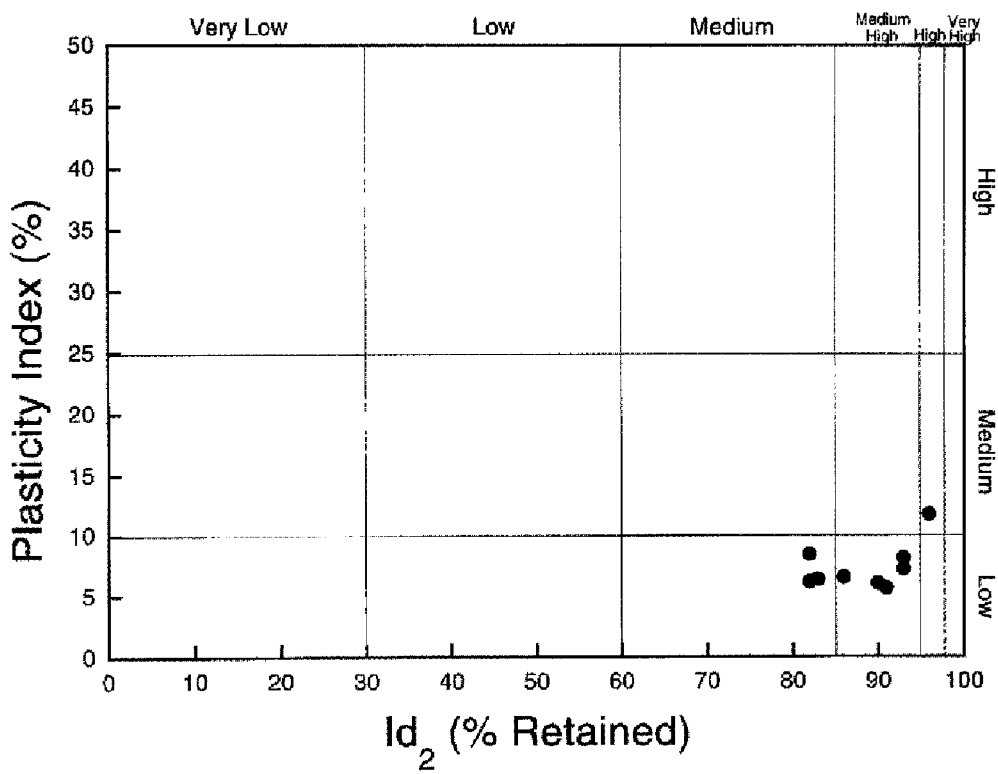

Fig. 1. Gamble's (1971) classification for the coal-mine refuse tested.

Table 4. Plasticity test results summary

Sample no.	LL (%)	PL (%)	PI (%)
1	28.9	22.6	6.3
2	31.9	23.5	8.4
3	29.4	23.4	6.0
4	26.7	20.6	6.1
5	29.0	23.4	5.6
6	26.0	18.8	7.2
7	31.3	23.2	8.1
8	38.5	26.8	11.7
9	28.4	21.9	6.5

Table 4 summarizes the results of the plasticity tests. Figure 1 displays Gamble's classification based on slake durability and plasticity testing. The points plotted on the Figure represent data from the slake durability and plasticity tests from this investigation.

Grain size analysis

Assuming that there is not significant segregation of the different grain sizes during and following stowing, one can analyse particle sizes to help predict, in a qualitative way, how a fill composed of a given material may be expected to behave with respect to strength, settlement, and permeability (Coates, 1981). A fill which contains well-graded particles should offer more resistance to displacement and settlement than one with uniformly graded particles, all other factors being equal. It has been shown (Senyur, 1989) that the amount of water flow through coal-mine refuse can be empirically related to its gradation and void ratio. Permeability decreases with a decrease in the effective size of the material, which is the 10% passing size according to the grain-size distribution. Therefore, one would expect a fill with a high percentage of minus 0.075 mm material to have a low permeability. A size analysis is also required to determine the suitability of a material for a particular stowing system, and whether or not crushing and screening of the material is needed. An indication of the gradation of the refuse can be computed from a grain-size distribution curve for grain sizes larger than the 0.075 mm sieve using the coefficient of uniformity (C_u) and coefficient of curvature (C_u). Both values can be calculated provided that not more than 10% of the grain sizes are less than 0.075 mm (Bowles, 1979).

Material with a C_u less than 4 is said to have a uniform distribution of grain sizes (poorly graded), while material with a C_u greater than 4 is said to have a wide range of grain sizes (well graded) provided the grain-size distribution curve is smooth and reasonably symmetrical. The coefficient of curvature should be calculated to determine if the gradation curve has proper shape and symmetry. Well graded material should have a C_z value between 1 and 3, while C_z for a poorly graded material should be either less than 1 or more than 3 (Dunn *et al.*, 1980).

To determine the gradation of the samples, the samples were first air dried, and then dry sieved through a 25.4 mm and a 19.1 mm sieve to separate the particles larger than 19.1 mm. The remaining sample was then wet screened using 12.7 mm, 4.75 mm, 2.00 mm, 0.850 mm, 0.425 mm, 0.250 mm, 0.105 mm, and 0.075 mm sieves.

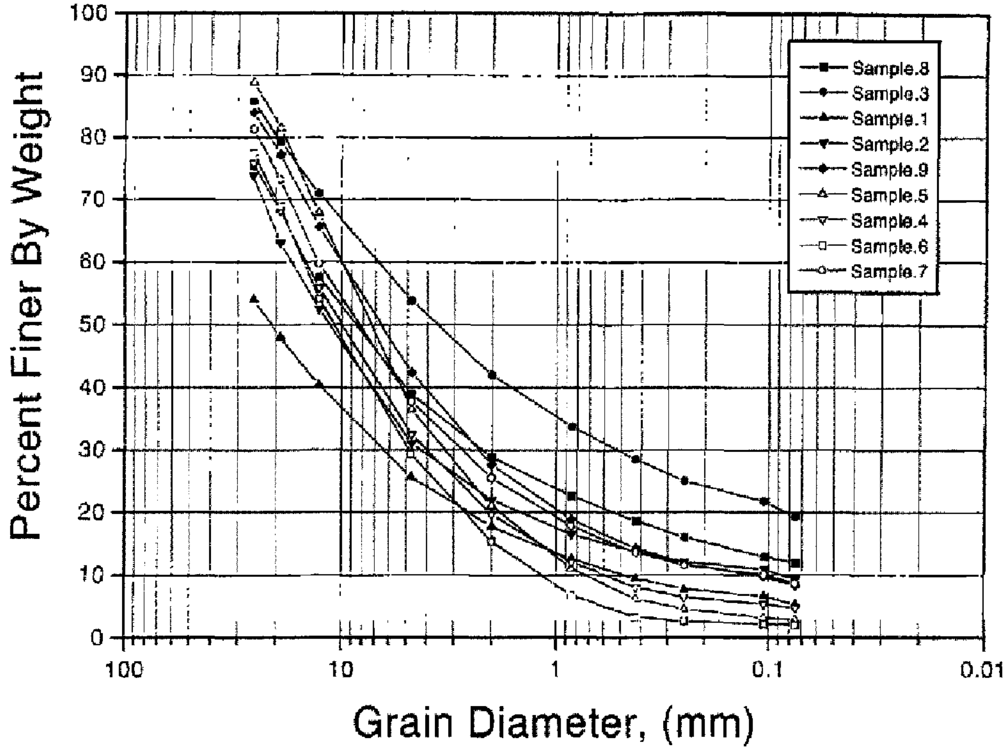

Fig. 2. Sieve analysis for the coal-mine refuse tested.

Figure 2 shows the gradation curves for all the samples. The coefficient of uniformity was calculated for seven samples, with values ranging from 12 to 169, meaning that there was a wide range of particle sizes (Table 5). A value could not be calculated for the other two samples, since they both contained more than 10% of material finer than 0.075 mm. The coefficient of curvature for these seven samples ranged from 1.33 to 11.97, with four samples having values between 1 and 3 (i.e. well graded). Since the other three samples had a C_z value greater than 3, it could be said that they had a wide range of particle sizes, but a disproportionate amount of material in the coarse size classes.

Table 5. Coefficient of uniformity and coefficient of curvature from sieve analysis

Sample no.	Coefficient of uniformity	Coefficient of curvature
1	60	2.96
2	169	11.97
4	24	1.72
5	13	1.53
6	12	1.33
7	100	4.84
9	80	4.05

Standard proctor compaction test

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). This test is generally performed in the construction industry as a means of comparing the densities of soil samples compacted in the laboratory with those obtained in the field. Although the compaction methods which have been used in construction may not be suitable for use in underground coal mines, the results of this test can be used for comparison with compaction values obtained for other potential stowing materials. This can be done if both materials tested have been compacted with the same compactive effort. This gives an indication of which material may achieve higher densities following compaction by whatever means, including by failing mine roof and pillars.

Compaction testing was performed on the refuse particles passing the 19.1 mm sieve, since all of the samples had more than 7% by weight of material retained on the 4.75 mm sieve (Head, 1982). A 2.5 kg hammer with a 305 mm drop (standard Proctor test) were used in the test.

Compaction curves which were obtained from the tests are displayed in Fig. 3. Maximum dry density for the nine samples varies from 1.66 to 2.00 Mg/m^3. Optimum moisture content ranges from 7.2% to 11.4% (Table 6). For comparison, the densities of the samples were also determined in a loose state. Each sample was loosely placed in the

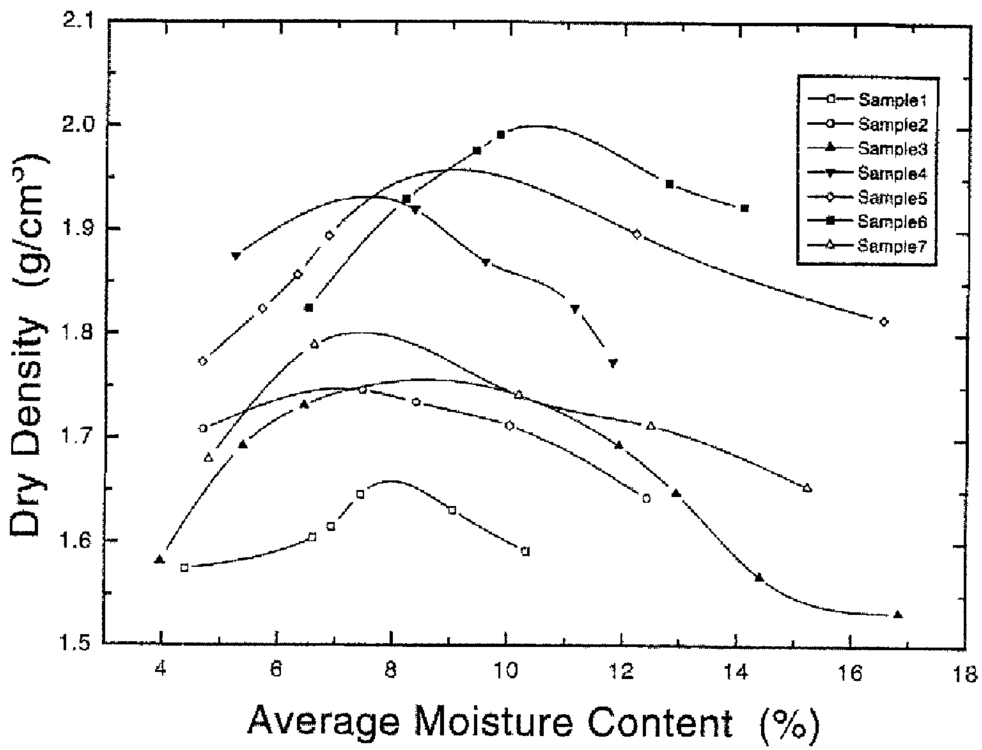

Fig. 3. Compaction curves for the coal-mine refuse tested.

Table 6. Standard Proctor compaction test results summary

Sample no.	Maximum dry density (Mg/m^3)	Optimum water content (%)
1	1.66	8.0
2	1.75	7.2
3	1.77	8.5
4	1.93	7.8
5	1.96	9.0
6	2.00	10.5
7	1.80	7.4
8	1.77	8.7
9	1.99	11.4

compaction mould, and then weighed to determine wet density. An average water content was determined for the sample, and the dry density was calculated. Table 7 provides the results for this test.

Falling head permeability test

The nature of water flow through an unconsolidated material will have a great effect on its physical properties. Coefficient of permeability was determined using a falling head test, because the refuse was expected to have an intermediate to low permeability of less than 10^{-4} m/s (Head, 1982). For this test, distilled water was first added to the dry refuse particles to achieve the optimum moisture content determined during the compaction test. The sample was then compacted using standard Proctor compaction.

For each sample, permeability tests were performed twice a day until the coefficient of permeability became nearly constant from one day to the next. At this point the sample was believed to have reached its maximum saturation, and k values would not be affected by air trapped between the solid particles.

Table 8 summarizes the sample averages for the coefficient of permeability of the refuse compacted at the optimum moisture content. The average for all the samples is 5.72×10^{-6} m/s.

Table 7. Loose densities of mine refuse samples

Sample no.	Wet density (Mg/m^3)	Average water content (%)	Dry density (Mg/m^3)
1	1.10	12.15	0.98
2	1.15	11.74	1.03
3	1.08	11.43	0.97
4	1.30	11.29	1.16
5	1.07	11.34	0.96
6	1.29	13.86	1.13
7	1.26	12.68	1.12
8	1.15	11.95	1.03
9	1.21	12.22	1.08

Table 8. Average falling head permeability for mine-refuse samples

Sample no.	Coefficient of permeability (m/s)
1	3.78×10^{-6}
2	4.74×10^{-6}
3	1.59×10^{-6}
4	3.75×10^{-6}
5	3.69×10^{-6}
6	1.70×10^{-5}
7	7.92×10^{-6}
8	1.19×10^{-6}
9	7.73×10^{-6}

Triaxial compression test

Triaxial compression testing can be used to assess the failure criterion of the refuse at different confining pressures and moisture contents. Shear strength parameters can be used in the estimation of both active and passive lateral support that the fill provides to remnant mine pillars. Drainage during the compression test can be controlled to simulate conditions which exist in actual mining applications.

It is expected that the fill will be placed in a loose state, and that there will be a large amount of grain displacement as the fill is compacted by failing mine roof and pillars. After a slip movement has occurred in the fill, the slip surface forms a permanent plane of weakness. The shear strength that can be mobilized along this plane corresponds to the residual shear strength of the material, which may be significantly lower than the peak strength (Sleeman, 1987). In this situation, an appropriate triaxial compression test would be one which measures residual shear strength parameters. This can be accomplished by 'failing' the sample. increasing the confining pressure, and then bringing the sample to failure again. The test can also be used to determine changes in the modulus of deformation (E_c) of the refuse.

For triaxial compression testing, the coarse material separated during dry screening was first crushed to minus 12.7 mm. The remaining sample was screened to remove the plus 12.7 mm material, which was also crushed. Crushing was performed because, if the material were to be used in an actual backfilling system (particularly pneumatic or hydraulic), a reduction in coarse sizes would probably be needed for transport.

The crushed material was then dry screened to remove any plus 12.7 mm material still remaining. The proper percentage (determined from the sieve analysis) of the crushed minus 12.7 mm material was then added to the remaining sample. Triaxial testing was performed on both dry and wet refuse, the wet refuse having moisture contents of both 10% and 15%. Undrained, unconsolidated triaxial tests were performed.

The testing equipment used consisted of an MTS servo-controlled loading machine, which applies the axial load to the loading piston at the top of the triaxial cell. The triaxial cell contains a flexible membrane filled with hydraulic oil. A hydraulic pump is used to provide pressure to the confining fluid inside the triaxial cell's flexible membrane. Sample and triaxial chamber diameter were the same, 50.8 mm. The length of chamber was 133.9 mm. The triaxial cell was almost completely filled with refuse, so that sample length would be well over 101.6 mm, twice the sample diameter. The sample was then compacted by three blows from the 2.5 kg hammer (total compactive effort is 2.2875 J). For a typical sample length of 106.7 mm, the compactive effort per volume of refuse would be 108 kJ/m^3, considerably less than what was used during the compaction test (605 kJ/m^3). The small amount of compaction is performed only to ensure that the sample top is relatively flat, and length measurements can be taken. Testing the sample at this relatively undensified state should more closely approximate actual fill density following the stowing operation.

The loading piston was then inserted into the top of the triaxial chamber, and the cell placed into the MTS testing machine. The hydraulic pump was coupled to the triaxial cell. Approximately 890 N of axial load was applied to the sample initially, and then a confining pressure of 690 kPa was applied.

The samples were tested using displacement control with a displacement rate of 0.381 mm/min. Undrained, triaxial compression tests for soils are typically conducted with displacement control at a rate of about 2% of the specimen length per minute, although displacement rates between 0.1 and 10.0% of the sample length per minute make little difference in test results (Head, 1982). For all the samples tested in this investigation, the displacement rate was between 0.3 and 0.4% of the specimen length per minute.

The choice of the failure point is rather arbitrary, since failure for unconsolidated materials is not clearly defined. Generally for soils, failure is defined arbitrarily at some predetermined level of axial strain (typically 29%). For this investigation, testing at each confining pressure continued until either the load-versus-displacement curve flattened out, or the confining pressure began to rise due to the outward displacement of the sample. At this point, it was assumed that the samples were failing.

Confining pressure was increased in increments of 690 kPa for the dry samples, and in increments of 345 kPa for the wet samples. The sample was then loaded to failure again. This procedure was continued up to confining pressures of 3450 kPa for the dry samples, and 3795 kPa for the wet samples.

The failure criterion (total stress analysis) for each sample was determined with the aid of a Mohr-Coulomb Failure Criterion computer program, and then plotted (Fig. 4). A test summary for the shear strength parameters is provided in Table 9.

Figure 5 shows the effect of water content and confining pressure on the modulus of deformation of the samples.

Table 9. Summary of results of the triaxial tests (total stress analysis)

Sample number	Cohesion (kPa)			Angle of internal friction (degrees)		
	Dry	w = 10%	w = 15%	Dry	w = 10%	w = 15%
1	138	296	234	16.8	11.2	7.0
2	124	269	200	17.4	9.5	5.6
3	234	276	372	17.1	14.0	9.4
4	97	338	324	17.8	9.7	7.0
5	165	303	283	14.8	10.0	7.2
6	172	290	296	14.8	11.9	7.4
7	124	317	221	15.4	8.8	6.4
8	110	228	359	17.2	14.6	8.7
9	159	352	165	13.9	11.5	5.6

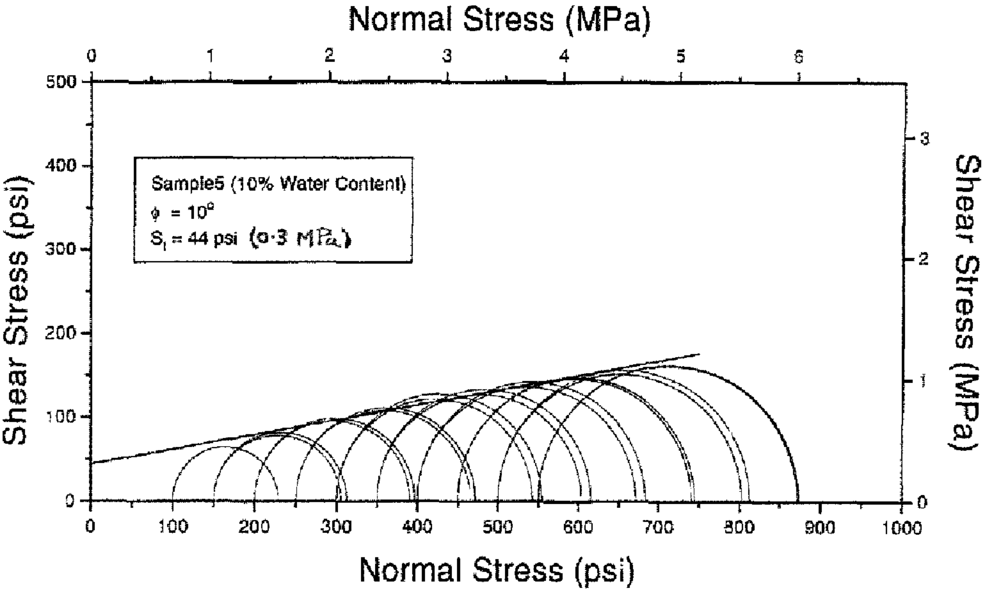

Fig. 4. Typical triaxial test results for coal-mine refuse.

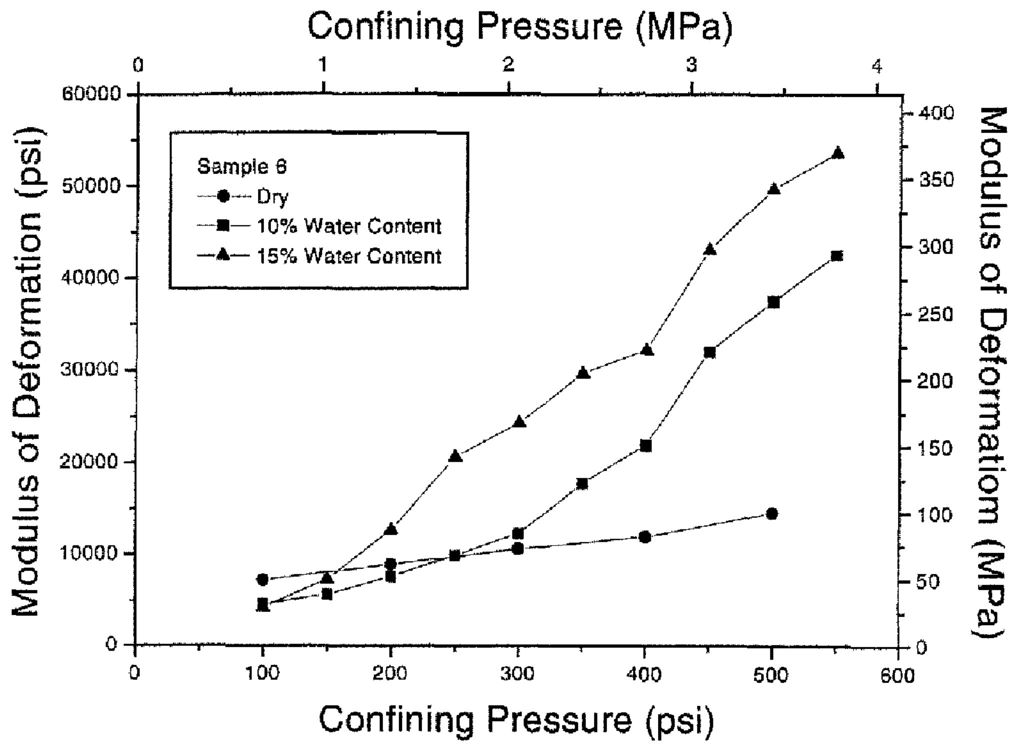

Fig. 5. Typical variations in the modulus of deformation of coal-mine refuse with confining pressure at different water contents.

Discussion

When a comparison is made with rocks of similar composition (shales and claystones) from the study by Gamble (1971), the coal refuse which was tested has at least medium durability when subjected to wetting and drying cycles. 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 high durability, based on Gamble's relative rankings. All of the samples which were tested in this investigation meet or very nearly meet this criterion.

Based on the results of the plasticity test, the refuse particles passing the 0.425 mm sieve can be classified as slightly plastic (Sowers, 1979). Even though the refuse is mostly coarse-grained, its strength and permeability may be adversely affected by the plasticity of the fine particles. For conventional soils, the effect of plasticity can be significant enough to alter these properties even when the soil has as little as 5% material (by weight) passing the 0.075 mm sieve (Bowles, 1979).

An important aspect of refuse having plastic properties is that it cannot be considered free draining. This means that excess pore pressures in the fill may not be quickly dissipated as the material is subjected to increases in stress. Thus, it would be desirable for coal refuse, considered as a potential backfill material, to be non-plastic.

A comparison of the results of the liquid limit test (liquid limit between 26.0 and 38.5) with the results of the slake durability test (which ranked all of the sample averages as having at least medium durability) would seem to support a slaking study performed by Morgenstern and Eigenbrod (1974). In the investigation, the liquid limit was determined for particles which were weathered by alternate wetting and drying cycles. When the liquid limit for the weathered particles was in the range of 20 to 50, the rocks exhibited only small amounts of slaking.

According to the sieve analysis, four of the samples are well graded. The gradation of the other samples could be improved by either crushing a portion of the coarser sizes or removing a portion of the fines (which may also be necessary for placement as backfill).

Based on placement considerations, it might be advisable to remove a portion of the fines if the material is to be used with a pneumatic stowing system. Previous experience with pneumatic stowing of coal refuse (Munjeri, 1987) indicates that the minus 3 mm fraction should not exceed 20% (by weight) to avoid jamming in the pipelines. The samples which were tested had 20–50% minus 3 mm material. For hydraulic systems, it is recommended that the minus 0.075 mm fraction not exceed 10% to ensure proper drainage. Two samples had more than 10% of this size fraction.

The results of the sieve analysis would seem to indicate that the samples which were tested have similar grain-size distributions when compared to coarse refuse tested in other studies. In a study by Busch *et al.* (1974) on coarse coal refuse sampled from eight sites in West Virginia, analysis indicated that the refuse had a wide range of grain sizes, with a large amount of coarse material. For the study, coefficient of uniformity values varied between 14 and 250.

An observation from the results of the standard Proctor compaction test is that compaction can considerably increase sample densities in comparison with the densities of the loose samples. The compacted densities of the samples were comparable to the compacted densities of sand, which has been successfully used as hydraulic stowing material in India

(Sinha, 1989). Sand typically has a compacted dry unit weight between 1.7 and 2.2 Mg/m^3 (Bowles, 1979).

Unfortunately, mechanical compaction of fills in underground mines has been reported to be economically unfeasible (McNay and Corson, 1975). Therefore, one would expect the in-place density of a fill of coarse coal refuse, at least initially, to be more comparable to the loose densities of the samples which are reported in Table 6.

The procedure used in determining the coefficient of permeability for the refuse is conservative. In actual applications, the coefficient of permeability of the stowed material will probably, at least initially, be higher, since placement density is not likely to be as high as the compacted density of the refuse in this test. The results of this test might be more applicable to the *in situ* case after the stowed refuse has been compacted by failing mine roof and pillars.

Based on previous backfilling experiences (Thomas, 1976), a fill should have a coefficient of permeability of at least 100 mm/h (2.78×10^{-5} m/s). Based on this criterion, none of the compacted samples have adequate permeability.

A relationship does seem to exist between amount of fines (minus 0.075 mm material) and coefficient of permeability for the samples. Sample 6 has the least fines (1.96% by weight) and has the highest coefficient of permeability (1.70×10^{-5} m/s). Samples 8 and 3 have the most fines (11.87% and 19.25% by weight) and have the two lowest values for coefficient of permeability (1.19×10^{-6} and 1.59×10^{-6} m/s).

D'Appolonia (1975) reports that typical k values for compacted coarse coal refuse tested in the lab generally vary from 1.0×10^{-4} m/s to 1.0×10^{-8} m/s. It is interesting to note that compacted conventional soils, which have similar size distributions (sand with 5–12% of minus 0.075 mm material), typically have permeabilities in the range of 1.0×10^{-5} to 1.0×10^{-8} m/s (Mitchell, 1983).

Based on the results of the Atterberg limit tests, particle size analysis, moisture density relationship test, and falling head permeability test, the coal refuse which was tested in this investigation appears to be quite typical when compared with that which has been tested in other investigations (Busch *et al.*, 1974; Moulton *et al.*, 1974; D'Appolonia, 1975; Rose *et al.*, 1984).

The failure criterion used for determining shear strength parameters for the wet samples is based on a total stress analysis. Generally, in the construction of most earth structures, a total stress analysis for the unconsolidated material comprising the structure is only considered important immediately after it has been placed, up until construction of the structure has ceased, before drainage can occur. However, this assumption cannot be made in a mine environment, because stresses acting on the fill will constantly be changing. Therefore, a total stress analysis may not be appropriate.

Because this test is a multi-stage triaxial test, ϕ can be considered to approach the residual angle of internal friction, since the sample is being subjected to multiple failures. This type of test was chosen over the single triaxial test, which uses a different sample for each state of stress, because it more closely approximates anticipated *in situ* conditions. It is expected that the fill particles will be subjected to large strains as they are compacted by overlying strata.

Compacted coarse coal refuse, tested using an effective stress analysis and single stage triaxial test, typically has a peak effective ϕ of between 25 to 41° (Bishop and Simon, 1976). However, it should be noted that these values have been obtained at normal stresses under 517.5 kPa. Since the shear-strength parameters of coal refuse have been determined for stability analysis of waste embankments, only low normal stresses have been used during previous investigations. Testing done at normal stresses up to 1380 kPa by the National Coal Board in the UK have shown that the failure envelope begins to flatten out at the higher stresses. In one case, the angle of effective internal friction decreased from 38° at the beginning of the envelope to 20° at a normal stress of 1035 kPa (D'Appolonia, 1975).

No documented multistage triaxial testing on coal refuse has been found, so the results of this test cannot be directly compared with any previous investigation.

Drnevich *et al.* (1976) performed a single-stage triaxial test on coarse coal refuse compacted at optimum moisture content according to AASHO T 99-70 method D (AASHO, 1970). After compaction, the samples were saturated. Confining stresses used during the test were up to 400 kPa, and peak normal stresses did not exceed 800 kPa. Total stress analysis showed that ϕ varied from 18.6 to 20.2°. Cohesion ranged from 60 to 99.4 kPa. For the effective stress analysis, ϕ was between 28.6 and 31.5°. Effective cohesion varied from 9 to 20 kPa.

An unconsolidated-undrained triaxial shear test by Wahler and Associates (1973) was performed on coarse coal refuse sampled in West Virginia. Based on a limited number of tests, a total stress analysis determined a ϕ as low as 9° with cohesion at 27.6 kPa. For the same tests, effective stress analysis determined a ϕ of 38° with zero cohesion.

Various tests measuring the residual shear strength of similar unconsolidated materials have produced shear-strength parameters comparable to the results of this investigation. For sand, silt, and clay mixtures, the residual angle of internal friction falls within the range of 12 to 24° (Hunt, 1986).

Sands with little or no fines (silt and clay) generally have a much higher residual angle of friction. For example, the residual shear strength of river sand has been determined to be around 33° for both drained and undrained conditions (Bishop, 1971). Since river sand has been proven to be a successful backfill material in underground coal mines (Sinha, 1989), it would be desirable for coal refuse to have at least a comparable residual angle of friction, say around 30°. Unfortunately, the results of this investigation suggest that the residual angle of friction for coal refuse is much lower than river sand, although since the results are based on a total stress analysis these conclusions may not be justified. Other studies indicate that materials having a ϕ value as low as 9° in total stress terms can have effective ϕ' values of 28° or greater.

Analysis of the data shows that the modulus of deformation of the refuse increases with confinement and axial strain. The data shows that the modulus of deformation is also sensitive to water content and confining pressure (Fig. 5). When the confining pressure is under 1250 kPa, the modulus of deformation for the wet samples is slightly less or about the same as that of the dry samples. However, as the confining pressure increases stiffness dramatically increases in the wet samples.

The confined modulus of deformation for the refuse is much less than the confined modulus of deformation for coal. Ko and Gerstle (1974) performed triaxial compression tests with confining pressures between 0.7 and 4.1 MPa on coal from the Pittsburgh coal seam. Modulus of deformation varied between 2690 and 3448 MPa.

Although the refuse has a lower confined stiffness modulus than intact coal, its modulus of deformation is comparable to sand. Triaxial compression tests on dry river sand with confining pressures between 2040 and 2720 kPa have yielded moduli of deformation values between 106 and 136 MPa (Vesic and Clough, 1968).

An observation from all the tests is that there are no significant differences, in terms of geotechnical properties, between newly placed and older samples originating from the same mines. This may be explained by considering that weathering only affects the surface layer of coal refuse piles (Charles, 1984), and the samples were taken at shallow depth (0.3 to 0.6 m).

Summary

Six tests were performed to help evaluate the suitability of coal refuse as a stowing material. These included slake durability, Atterberg limits, grain-size analysis, moisture-density relationship, falling head permeability and triaxial compression tests.

From the results of these tests, the following conclusions can be drawn:

(1) The slake durability test indicates that the refuse is susceptible to a moderate amount of disintegration when exposed to wetting and drying cycles.
(2) The fine fractions (minus 0.425 mm) of all the samples do exhibit some degree of plasticity.
(3) The grain-size analysis indicates that the refuse tested in this investigation, owing to the high amount of minus 3 mm material, would be likely to cause jamming in pneumatic stowing pipelines based on prior experiences with backfilling systems (Munjeri, 1987). The grain-size analysis also indicates that some portion of the samples would probably require crushing to be used with pneumatic or hydraulic stowing systems.
(4) The dry density of the refuse can be considerably increased by compaction.
(5) The coefficient of permeability for all the samples is less than what is desirable (2.78×10^{-5} m/s) for a backfill material.
(6) The results of the triaxial compression test indicate that the total stress angle of friction for the tested coal refuse is between 13.9 and 17.8° for the dry samples, between 8.8 and 14.6° for the samples containing 10% moisture, and between 5.6 and 9.4° for the samples containing 15% moisture. These values for angle of friction are much lower than river sand, which has been successfully used as backfill material in underground mines. However, since these results are obtained from a total stress analysis, they may not be appropriate for judging the suitability.
(7) The modulus of deformation of the refuse increases with confining pressure. This increase is more dramatic for the wet refuse which was tested under undrained conditions.
(8) The modulus of deformation of the refuse is much lower than the modulus of deformation of intact coal for the confining pressures used in this investigation.
(9) It was determined that the refuse does have a moderate amount of plasticity, low permeability, and low total stress angle of friction; therefore, it is felt that the coal refuse tested is inadequate as a backfill material.

It is anticipated that there may be drainage problems when using this material as stowing material in wet mines, and that it will not readily dissipate pore pressure. Excessive pore pressures caused by increasing stresses can lead to fill liquefaction if the amount of confinement provided to the fill is not adequate. A possible solution to this problem would be to remove some portion of the fines (minus 0.075 mm material), which might also help improve placement performance.

In summary, it is felt 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 as a stowing material.

Acknowledgements

The cooperation and assistance of the Paramont Coal Corporation, the United Coal Company and the Westmoreland Coal Company in obtaining coal refuse samples and the editing of this manuscript by M.K. Radcliffe, is gratefully acknowledged. The information contained here represents part of C.H. Bowman's Masters Thesis. This research was partially funded by the Generic Mineral Technology Center, Mine System Design and Ground Control.

References

AASHO (1970) *Standard Specifications for Highway Materials and Methods of Sampling and Testing*, Part 2, American Association of State Highway Officials, Washington, DC.

ASTM D4644-87 (1987) Slake durability of shales and similar weak rocks, *Annual Book of ASTM Standards*, Vol. 4.08, Am. Soc. for Testing and Materials, Philadelphia, pp. 848–50.

Bishop, A.W. (1971) Stress strain behaviour of soil, *Proceedings of the Roscoe Memorial Symposium*, Whitefriars, London, pp. 3–58.

Bishop, C.S. and Simon, N.R. (1976) Selected soil mechanics properties of Kentucky coal preparation plant refuse, *Proceedings of the Second Kentucky Coal Refuse Disposal and Utilization Seminar*, Pineville, Kentucky, pp. 61–67.

Bland, A.E., Robl, T.L. and Rose, J.G. (1976) Kentucky coal preparation plant refuse characterization and uses, *Proceedings of the Second Kentucky Coal Refuse Disposal and Utilization Seminar*, Pineville, Kentucky, pp. 21–35.

Blight, G.E. (1979) Properties of pumped tailings fill, *Journal of the South African Institute of Mining and Metallurgy*, October, Vol. 79, 446–53.

Bowles, J.E. (1979) *Physical and Geotechnical Properties of Soils*, McGraw-Hill, New York, 478 pp.

Brown, A., Berryhill, H.L., Taylor, D.A. and Trumbull, J. (1952) *Coal Resources of Virginia*, US Geological Survey Circular No. 171, Washington D.C., 57 pp.

Busch, R.A., Backer, R.R. and Atkins, L.A. (1974) *Physical Property Data on Coal Waste Embankment Materials*, Bureau of Mines Report of Investigations 7964, 142 pp.

Charles, J.A. (1984) Settlement of fill, in *Ground Movements and Their Effects on Structures*, Attewell, P.B. and Taylor R.K. (eds), Chapman and Hall, New York, pp. 26–45.

Coates, D.F. (1981) Caving, subsidence and ground control, in *Rock Mechanics Principles*, Minister of Supply and Services Canada, Quebec, Chapter 5, pp. 5.1–5.42.

D'Appolonia, E. (1975) *Engineering and Design Manual for Coal Refuse Facilities*, Prepared for US Department of the Interior, MESA, Washington, DC.

Drnevich, V.P., Williams, G.P. and Ebelhar, R.J. (1976) Soil mechanics tests on coal mine spoils, *Proceedings of the Second Kentucky Coal Refuse Disposal and Utilization Seminar*, Rose, J.G. (ed.), University of Kentucky, Lexington, KY, Pineville, Kentucky, pp. 47–59.

Dunn, I.S., Anderson, L.R. and Kiefer, F.W. (1980) *Fundamentals of Geotechnical Analysis*, John Wiley and Sons, New York, 414 pp.

Franklin, J.A. and Dusseault, M.B. (1989) *Rock Engineering*, McGraw-Hill, New York, 600 pp.

Gamble, J.C. (1971) Durability–Plasticity: Classification of Shales and Other Argillaceous Rocks, Thesis, University of Illinois, Urbana, Illinois.

Head, K.H. (1982) *Manual of Soil Laboratory Testing*, Vols 1–3, Pentech, London, 1238 pp.

Huddle, J.W., Lyons, E.J., Smith, H.L. and Ferm, J.C. (1963) *Coal Reserves of Eastern Kentucky*, US Geological Survey Bulletin 1120, 247 pp.

Hunt, R.E. (1986) *Geotechnical Engineering Analysis and Evaluation*, McGraw-Hill, New York, 729 pp.

Ko, H.Y. and Gerstle, K.H. (1974) *Constitutive Relations of Coal,* Annual report submitted to USBM under Grant G0110894, 220 pp.

McNay, L.M. and Corson, D.R. (1975) *Hydraulic Sandfill in Deep Metal Mines*, USBM, IC 8663, 40 pp.

Miller, M.S. (1973) Stratigraphy and Depositional Systems of Carboniferous Rocks in Southwest Virginia, Thesis, West Virginia University, Morgantown, West Virginia.

Mitchell, R.J. (1983) *Earth Structures Engineering*, Allen and Unwin, Winchester, Massachusetts, 265 pp.

Morgenstern, N.R. and Eigenbrod, K.D. (1974) Classification of Argillaceous soils and rocks, *Journal of Geotechnical Engineering Division*, **100**, 1137–57.

Moulton, L.K., Anderson, D.A., Seals, R.K. and Hussain, S.M. (1974) Coal mine refuse: an engineering material, *Proceedings of the First Symposium on Mine and Preparation Plant Refuse Disposal*, National Cool Association Louisville, Kentucky, October, pp. 1–25.

Munjeri, D. (1987) Prevention of subsidence using stowing methods, *Colliery Guardman,* **235**, 245–8.

Murphy, D.J. (1987) Stress, degradation, and shear strength of granular material, *Geotechnical Modeling and Applications*, Sayed, S.M. (ed.), Gulf, Houston, pp. 181–211.

Nandy, S.K. and Szwilski, A.B. (1987) Disposal and utilization of mineral wastes as a mine backfill, *Underground Mining Methods and Technology*, Szwilski, A.B. and Richards, M.J. (eds), Elsevier, Amsterdam, pp. 241–52.

National Academy of Sciences, (1975) *Underground Disposal of Coal Mine Wastes*, National Academy of Engineering, Washington, DC, 172 pp.

Rose, J.G., Bland, A.E. and Robl, T.L. (1984) *Utilization Potential of Kentucky Coal Refuse,* University of Kentucky Institute for Mining and Minerals Research Publications Group, Lexington, Kentucky, 49 pp.

Senyur, G. (1989) The flow characteristics through hydraulic filling materials produced from coal washery rejects, *Proceedings of the 4th International Symposium on Mining with Backfill, Innovation in Mining Backfill Technology*, Montreal, October, A.A. Balkema, Amsterdam, pp. 307–14.

Sinha, K.M. (1989) Hydraulic stowing – a solution for subsidence due to underground mining in the U.S.A., *Rock Mechanics as a Guide for Efficient Utilization of Natural Resources*: *Proceedings of the 30th U.S. Symposium*, Balkema, Rotterdam, pp. 827–34.

Sleeman, W. (1987) Colliery spoil in urban development, *Proceedings of the Second International Conference on the Reclamation, Treatment and Utilization of Coal Mining Wastes*, Nottingham, England, Elsevier, Amsterdam, pp. 163–77.

Sowers, G.F. (1979) *Introductory Soil Mechanics and Foundations*, Macmillan, New York, 621 pp.

Stewart B.R. (1990) Physical and Chemical Properties of Coarse Coal Refuse From Southwest Virginia, Thesis, Virginia Polytechnic Institute and State University, Blacksburg, Virginia.

Thomas, E.G. (1976) Thick seam underground coal mining by a slice and fill method, *Proceedings of the Symposium on Thick Seam Mining by Underground Methods*, Rockhampton, Queensland, Australasian Institute of Mining and Metallurgy, September, pp. 75–81.

Vesic, A.S. and Clough, G.W. (1968) Behavior of Granular Materials Under High Stresses, *Journal of the Soil Mechanics and Foundations Division, ASCE,* **94**, 661–88.

Vick, S.G. (1983) *Planning, Design, and Analysis of Tailings Dams*, John Wiley and Sons, New

York, 369 pp.

Vorobjev, B.M. and Deshmukh, R.T. (1966) *Advanced Coal Mining* Asia Publishing House, London, 909 pp.

Wahler and Associates (1973) *Analysis of Coal Refuse Dam Failure, Middle Fork, Buffalo Creek, Saunders, WV*, National Technical Information Service, Springfield, VA, Appendix B, 29 pp.

Wayment, W.R. (1978) Backfilling with tailings – a new approach, *Mining With Backfill, 12th Canadian Rock Mechanics Symposium*, Sudbury, Canadian Institute of Mining, Special Vol. 19, pp. 111–16.

Wray, W.K. (1986) *Measuring Engineering Properties of Soil*, Prentice-Hall, New Jersey, 276 pp.