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COAL WASTE DEWATERING SYSTEMS

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16. Abstracts

Samples of fine coal refuse were obtained from 11 coal preparation plants throughout the United States. These samples were characterized by ash analysis, particle size, and mineral composition. From the characterization, geographical locations, and the dewatering equipment being used by the plants, five sites were selected for obtaining bulk fine refuse samples.

The selected coal preparation plants were using for final dewatering thickeners followed by: a vacuum disc filter, a filter press, a belt press filter, a centrifuge, and a settling pond. None were using thermal drying.

The moisture content of the final cake varied from approximately 23% to 50% depending upon the feed material, flocculant addition, and the type of equipment. The filter press gave the lowest moisture content followed by a belt press and then either a centrifuge or vacuum filter. The centrifuge products were all dried to an approximate 5% moisture in a thermal dryer.

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FOREWORD

Colorado School of Mines Research Institute (CSMRI), Golden, Colorado, prepared this report under USBM Contract J0205012. The contract was initiated under the U.S. Bureau of Mines coal refuse program.

It was administered under the technical direction of Mr. Robert W. McKibbin as Technical Project Officer. Mr. David J. Askin was the contract administrator for the U.S. Bureau of Mines. The report is a summary of the complete work carried out during the period of December 11, 1979, to February 6, 1981.

Patent Statement

This report does not contain patentable information.

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INTRODUCTION

Coal mine operators are facing a growing volume of coal fines refuse produced in cleaning operations. Disposal of this refuse is difficult and expensive.

Impoundment has been the most common disposal method. Recent evaluations have indicated these structures can be unsafe and can be critical during periods of heavy rains. As a result, state and federal regulations now govern the construction and abandonment of such structures to the extent that the average cost of impoundments is now $2.50 per ton of waste.

Where long-term impoundment is not possible, the practice has been to use a series of holding ponds for the refuse. After the solids have settled, the supernate is pumped off, and the settled solids are dug from the pond. These are trucked to a disposal site and mixed with coarse refuse. The handling of these fines is extremely difficult and leads to long-term problems. Inefficient loading equipment is used, because the terrain will not support other equipment. Trucking is difficult due to the thixotropic nature of the fines refuse. Spillage from the trucks makes maintenance of haul roads expensive. Difficulties are experienced when the critical limits for moisture of the coarse and fines mixture are exceeded. Often the material cannot be compacted; sometimes hillside disposal sites move in mass.

Because of these problems and strict regulations, operators are seeking technology to close their preparation water circuits in a proper and reliable manner that would eliminate the need for impoundment.

The U.S. Bureau of Mines issued a contract to Colorado School of Mines Research Institute (CSMRI) to conduct a coal refuse dewatering investigation.

The objective of this project was to determine the extent to which fine coal waste can be dewatered and the system requirements for integration into preparation plant facilities.

The broad scope of the program was to visit up to nine coal preparation plants and to collect fine refuse pulp samples. These preliminary samples would be characterized to provide a basis for selecting locations for collecting bulk samples. Sufficient fine refuse pulp was to be collected from five plants for use in testing five dewatering techniques. Studies would be carried out at the U.S. Bureau of Mines Spokane Mining Research Center. Dewatering techniques to be investigated were vacuum filtration, pressure filtration, belt press filtration, centrifugation, and thermal drying.

Nine preparation plants were visited and fine refuse pulp samples were obtained along with general plant operating information. The samples were subjected to a laboratory characterization study. Subsequently, samples from two more plants were shipped to CSMRI for evaluation.

Five plants were then agreed upon as source sites for the bulk samples. One plant was in the Northern Appalachian Region, two in the Southern Appalachian Region, and two in the Western Region. Selections were made on the basis of relative degrees of difficulty and the type of dewatering equipment being used by the plant. Locations were selected where a filter press, a belt press, a vacuum filter, a centrifuge, and a settling pond were being used as a final stage of dewatering. All of the plants first treated their fine refuse slurry streams in a thickener and the thickener overflow was used as plant recycle water.

Bulk samples were collected from the five locations by pumping thickener underflow slurry into 55-gallon drums. The drums of slurry were transported in insulated trucks to the Spokane Mining Research Center.

Equipment necessary to conduct the dewatering studies was shipped to Spokane and was assembled as required for the various tests. Part of the equipment was furnished by CSMRI, and other equipment was rented. Some samples were also shipped to equipment manufacturers for their laboratory studies. The bulk tests were conducted at Spokane with laboratory characterization tests largely being conducted in Golden.

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SUMMARY AND CONCLUSIONS

The following summary and conclusions are based on the samples studied and on the results reported herein.

Summary

Preliminary fine coal refuse samples were obtained from 11 preparation plants for characterization studies. Five bulk samples were obtained for characterization and dewatering studies.

All of the preparation plants were using thickeners as their first stage of dewatering. The feed to the thickeners ranged from 2.3% to 18% solids, the underflow from 12% to 52% solids, and the overflow from 0.02% to 1.3% solids.

The refuse samples had a wide range of ash content from 25.7% to 72.4%. The bulk samples had the following analysis.

Preparation Plant	Refuse Ash %	Refuse Total S %	Btu
B	63.4	0.73	4,759
K	49.2	0.63	6,680
L	48.1	3.10	7,118
C	34.0	1.14	8,754
J	25.7	0.72	11,284

Preparation Plant B refuse contained an estimated 20.2 weight % montmorillonite clay and 5.0 weight % kaolinite clay. The other four bulk samples contained kaolinite and illite clays with total clay contents as follows:

Preparation Plant	Total Clay Minerals wt %
B	25.2
K	21.3
L	15.2
C	3.1
J	1.7

The principal mineral composition of the refuse was coal, quartz, calcite, dolomite, clays, feldspar, and pyrite. Coal, quartz, and clay were present in all samples with the other minerals being present in some samples.

All of the fine refuse samples were finer than 1/4-inch size, and most were a nominal 28 mesh top size. There was a wide spread in the amount of fines, i.e., from 26.4% to 70.5% minus 325 mesh and from 2.3% to 44.8% minus 2 micrometers. Bulk samples used for dewatering studies had the following particle size distribution.

Preparation Plant	Cumulative Passing Size 8M wt %	28M wt %	100M wt %	325M wt %	2 µm wt %
K	100.0	89.6	75.5	62.9	29.5
B	100.0	98.8	87.4	50.5	28.0
L	97.3[1]	86.5	66.9	52.4	23.0
C	100.0	100.0	76.5	38.0	4.7
J	100.0	97.3	70.1	52.3	2.3

[1] 2.7% +8M.

Samples of thickener underflow pulp were subjected to laboratory vacuum filtration tests as part of the characterization program and to assess the effect of pulp aging. The data were not totally comparable because of differences in the amount of feed solids to the two tests. The second tests were conducted approximately 5 months after the first tests. Of the nine samples tested, only one sample showed a decrease in filtration rate and the others were either the same or greater. Any improved dewatering that took place was due to increased flocculation or coagulation on standing. This could have occurred as all samples had been flocculated and/or coagulated in the plant thickeners. There were no substantial data to show that the refuse samples deteriorated or became more difficult to filter with time.

A series of centrifuge tests were conducted on each of the bulk samples using a continuous solid bowl centrifuge with the following results.

Preparation Plant	Cake Moisture %	Approximate Capacity Full-Scale Centrifuge tph
B	42.5	15
J	31.6	25
K	40.5	25
C	32.7	30
L	31.4	30-40

Tests were conducted using two different belt presses, and these results are summarized as follows.

Preparation Plant	Cake Moisture, % (Andritz)	(Parkson)	Capacity tph/meter Belt Width (Andritz)	(Parkson)
B	37	40	6.0	1.8
C	30	30	5.4	2.9
J	29	30	6.1	3.5
K	30	35	7.3	2.5
L	29	35	8.9	3.0

The two belt presses were not directly comparable machines in regard to capacity. The Arus-Andritz design capacity was based on a much larger and more costly machine that is longer and taller than the Parkson machine.

A drum vacuum filter was used to develop dewatering data as follows:

Preparation Plant	Moisture %	Cake Form Rate dry lb/ft^2-hr
J	28.9	18
C	32.0	64
B	47.4	73
L	26.3	95
K	37.0	102

A laboratory filter press operating at 225 psig was used to develop the following pressure filter data.

Preparation Plant	Moisture %	Chamber Fill Rate dry lb/ft^2-hr
B	62.0	3.82
L	24.5	6.61
C	26.7	7.49
K	24.0	9.75
J	23.0	37.77

The centrifuge dewatered coal refuse was thermally dried using an indirect-type transport dryer. All of the wet cake refuse products were effectively dried to 1.3% to 7.2% moisture without any operating problems. The transporter was heated using oil at 375° to 395°F.

Based on characterization data using ash, total clay, montmorillonite clay, minus 325 mesh, minus 2 micrometers, and laboratory filter rates, the following order of decreasing dewatering difficulty was projected: Preparation Plants B, K, L, J, and C. However, based on the pilot-plant data, Plant B refuse was the most difficult to dewater followed by C, and the remaining three, J, K, and L, were essentially the same. Flocculation was a factor not measured in the characterization study. The degree of flocculation and the type of floc formed are important factors in the dewatering rate and to some extent in final cake moisture.

Conclusions

Plant recycle water of less than 1% solids should be readily obtainable by proper operation of the thickener and by returning filtrate or centrate to the thickeners.

The thickener underflow product from most plants was variable as to pulp density and mineral composition resulting in variable feed to dewatering equipment. To design an effective refuse dewatering system for a particular preparation plant will require careful analysis over a period of time of the material to be treated.

Development of the optimum flocculation system for a particular application is the most important factor in maximizing dewatering equipment performance. Optimization must include selection of the best reagents and best reagent flocculant-slurry mixing technique for each dewatering machine.

All types of dewatering devices tested, i.e., the centrifuge, the belt press, the vacuum filter, and pressure filters, have potential application for all five coal slurry refuses. The total plant operations of coarse and fine refuse along with economics must be assessed to arrive at the best plant for each site.

RECOMMENDATIONS

The following recommendations are based on the five coal refuse dewatering studies covered by this report.

1. A detailed flocculation-coagulation investigation of fine coal refuse should be undertaken. Such a program would include characterization of the refuse as to coal-ash analysis, mineralogical analysis, and particle size analysis. The flocculation-coagulation study would have to include investigation of most reagent types, reagent addition, and mixing techniques. A flocculated pulp evaluation method that would be correlatable to dewatering processes should be established.

2. A standard laboratory refuse characterization procedure should be developed to include a standard coal-ash analysis, mineralogical analysis, particle size analysis, a flocculation procedure, a vacuum filter test, a pressure filter test, and centrifuge tests.

3. On-site field testing should be conducted on the various types of dewatering equipment to define operational problems and to establish realistic performance and operating data. Protracted data are needed on flocculation techniques, capacity, product moisture, water clarity, filter media blinding, labor costs, and maintenance costs.

4. An alternate to Item 3 would be to conduct detailed studies of existing operating plants using fine coal refuse dewatering equipment.

DISCUSSION

The coal waste dewatering program was designed to select five fine coal refuse samples for dewatering studies. These five samples were to be from different coal regions in the country and to have different physical characteristics.

A preliminary field survey was made, and small refuse samples were obtained from nine coal preparation plants. These samples were characterized by CSMRI, and the results were reviewed with the U.S. Bureau of Mines Technical Project Officer. Two of the samples met the desired program criteria for the dewatering studies. Three additional plants were contacted, and two of the plants sent small refuse samples to CSMRI for characterization studies. From the 12 plants contacted, five were selected from which to obtain bulk samples for dewatering studies. All of these preparation plants have thickeners to settle their fine coal refuse but use different processes for final dewatering. The plants have been identified by letter designation (A-L), and the five selected plants for dewatering studies were described as follows:

Preparation Plant	Coal Region	Fine Refuse Final Dewatering Process
B	Western	Belt Press
C	Western	Vacuum Filter
J	Southern Appalachian	Centrifuge
K	Southern Appalachian	Pressure Filter
L	Northern Appalachian	Pond

The original nine preliminary samples collected by CSMRI and the two preliminary samples sent to CSMRI were subjected to a number of characterization tests. These preliminary samples were from Preparation Plants A-K. Samples of thickener feed, thickener overflow, and thickener underflow were obtained from Plants A-I, and only thickener underflow, from Plants J and K. The percent solids, pH, and particle size analyses were made on the thickener feed, underflow, and overflow samples from plants' A-I samples. The thickener underflow samples from the 11 plants, A-K, were subjected to float-sink studies at 1.6 sp gr, x-ray diffraction (XRD), clay separation, XRD analysis of the clay fraction, and laboratory vacuum form filtration tests. These data were used to characterize or to evaluate the various fine refuse samples. General plant operating information was obtained regarding preparation-plant cleaning equipment and flocculants or coagulants used to thicken the fine refuse.

The five bulk samples of fine coal refuse were obtained from thickener underflow at each preparation plant. These samples were collected in 55-gal drums and were shipped by insulated truck to the U.S. Bureau of Mines, Spokane Mining Research Center. At the same time, small 5-gal samples of pulp were collected for characterization studies at Golden. These studies were similar to those performed on the preliminary samples. The specific characterization data determined were: percent solids, pH, particle size analysis, sulfur and Btu content, ash on size fractions, float-sink studies at 1.6 sp gr, XRD on plus 2 sp gr solids, % -2 μm clay, XRD on -2 μm clay, flocculant scoping, and laboratory vacuum filtration tests. These studies were made to characterize the feed to the dewatering studies and to determine the variability of the plant fine refuse products.

Each of the five bulk samples was subjected to the following dewatering studies: pressure filter, vacuum filter, two belt press tests, centrifuge, and thermal drying of the centrifuge product. These dewatering studies were made to determine the final water content of the various products, to obtain dewatering rate data, and to present information to predict the dewatering characteristics of various types of fine coal refuse.

Coal Preparation Plants and Processes

Twelve coal preparation plants were visited during this program to obtain preliminary and/or bulk fine coal refuse samples and to obtain general process information. These plants provided an overview of coal preparation plants and their fine refuse dewatering systems. The obtained general plant information is summarized in Table 1, and the details are given in Exhibit 1.

One half of the plants sampled are using heavy media vessels to treat the coarsest coal, Deister concentrating tables to process the middle size coal, and froth flotation to clean the fine coal. Three plants use only Baum jigs, and the other three plants use a combination of several cleaning devices. The type of coal-cleaning equipment being used did not control the quality of the fine refuse. In each equipment grouping in Table 1, the ash content of the fine refuse ranged from low to high.

Those plants producing coal for the steam coal market have fine refuses with an ash content ranging from 34.1% to 72.4%. The plants producing metallurgical (Met) grade coal have lower ash contents in the refuse, 27.5% to 43.7%, than those producing steam coal. Presumably, the metallurgical coal plant has tighter specifications resulting in less ash or more coal in the fine refuse. Those plants with low ash content refuses may at some future date improve their coal recovery, which could change the overall character of the fine refuses. Such a change could have an adverse effect on the fine refuse dewatering operation; however, it should decrease the amount of refuse to be discarded.

Five plants use settling ponds for the final dewatering, three plants use vacuum disc filters, two use filter presses; one, a centrifuge; and one uses a belt press. Preparation Plants B and K, which have some of the highest ash content refuse, use pressure-type dewatering equipment, which may indicate that dewatering high ash content refuse is more difficult.

Three of the five plants selected for bulk dewatering tests have the highest ash content refuse of the 12 plants, and the other two have medium to low ash content refuse. Thus, the majority of the samples used in the dewatering study were expected to represent the more difficult refuse dewaterng problems. The other two samples were to represent a more average dewatering problem. Plants were selected that use a belt press, a filter press, a vacuum filter, a centrifuge, and a settling pond for the final dewatering step. All program dewatering schemes were represented by the bulk samples except for thermal drying. Thermal drying of refuse is not being used at any of the plants contacted.

Preparation Plant Thickener Performance

All of the visited coal preparation plants from which samples were obtained use thickeners as the first step in fine coal refuse dewatering. Samples of thickener feed, overflow, and underflow were obtained from the first nine plants visited. The data on these samples are presented in Exhibits 1 and 2. The pulp

TABLE 1. - Preparation plants

Preparation Plant	Coal Region	Market for Clean Coal	Coal Preparation Cleaning Device(s)	Ash Content of Thickener Underflow %	Fine Refuse Dewatering Device
B[1]	Western	Steam	Baum Jig	72.4	Belt Press
F	Northern Appalachian	Steam	Baum Jig	41.8	Vacuum Filter
C[1]	Western	Steam	Baum Jig	37.5	Vacuum Filter
K[1]	Southern Appalachian	Steam	Heavy-Media Vessel, Table, Flotation	57.9	Filter Press
I	Alabama	Steam	Heavy-Media Vessel, Table, Flotation	46.6	Pond
H	Southern Appalachian	Met/Steam	Heavy-Media Vessel, Table, Flotation	38.0	Pond
J[1]	Southern Appalachian	Steam	Heavy-Media Vessel, Table, Flotation	35.2	Centrifuge
E	Northern Appalachian	Steam	Heavy-Media Vessel, Table, Flotation	34.1	Filter Press
A	Western	Met	Heavy-Media Vessel, Table, Flotation	27.5	Vacuum Filter
L[1]	Northern Appalachian	Steam	Baum Jig, Table	62.1	Pond
D	Northern Appalachian	Met	Chance Cone, Table, Flotation	43.7	Pond
G	Southern Appalachian	Met	Heavy-Media Vessel, Heavy-Media Cyclone, Flotation	34.3	Pond

[1] Bulk samples obtained from these plants for dewatering studies.

percentage of solids was used to calculate a solids distribution for each thickener. This is presented in Table 2.

Table 2 is presented in the order of increasing solids in the overflow. Neither the solids in nor the clarity of the overflow had any relationship to the percentage solids in the feed or in the underflow.

A very wide variation from 2.3% to 18% solids content existed in the feed, which indicates differences in the amounts of water used in the preparation plants.

There was also a very wide spread in the percent solids in the thickener underflow, 12.5% to 52% solids. This factor is important when considering the next stage of dewatering following a thickener. Dewatering equipment is size and capacity dependent on quantities of both water and solids.

Plant B with the lowest thickener underflow solids content was the most difficult material to dewater. The solids were the only residue containing montmorillonite clay; it had the highest ash content at 72.4% and was the finest material at 70.5% 325 mesh x 0 and 44.8% 2 μm x 0.

Plant A with the highest percent solids in the thickener underflow contained the coarsest solids at 2.8% 2 μm x 0 and the lowest ash content at 27.5%.

The amount of feed solids recovered (efficiency) by the thickeners covered a wide range from 73.8% to 99.8%. The solids in the thickener overflow are expected to be the unflocculated fine particles. Presumably, they would be the most difficult material to filter in the next stage of dewatering. An improvement in the thickener operation in those plants having a high percentage of the total solids in the overflow could result in a marked change in subsequent dewatering steps.

One of the study requirements was to generate recovered water that was less than 1% dry solids and that was directly reusable in the preparation plant process. The thickener overflow from all but Plant H met the criteria of less than 1% solids, and presumably the water was reusable. Certainly, most of the plants, those with less than 99% solids recovery, should examine the operation of their thickeners to improve solids recovery.

Solids content of the effluent from the filters and centrifuge ranged from 0.09% to 8%. Some of these waters would be suitable for direct recycle to the preparation plant. However, normal practice is to return filter and centrifuge water streams to the thickener; then the thickener acts as a backup or safety in case of some mechanical failure of the filters or the centrfuge. In all cases, the greatest quantity of recycle water will be thickener overflow. Therefore, if the thickener is operated properly to produce acceptable recycle water, the quality of filter or centrifuge water would have only a minor effect on the total water for recycle.

Thickener underflow bulk samples were obtained from four plants 1 to 3 months after the preliminary samples wre obtained. Table 3 presents comparative thickener underflow pulp data on the two sets of samples.

TABLE 2. - Preparation plant thickener performance

Preparation Plant	% Solids: Thickener Feed	% Solids: Thickener Underflow	% Solids: Thickener Overflow	Weight % Distribution: Thickener Feed	Weight % Distribution: Thickener Underflow	Weight % Distribution: Thickener Overflow	Efficiency
B	4.5	12.5	0.02	100.0	99.7	0.3	99.7
A	6.5	52.0	0.07	100.0	99.1	0.9	99.1
I	18.0	25.0	0.14	100.0	99.8	0.2	99.8
G	4.0	31.5	0.18	100.0	96.0	4.0	96.0
D	4.2	35.3	0.20	100.0	95.8	4.2	95.8
E	2.3	14.8	0.22	100.0	91.8	8.2	91.8
F	5.2	31.0	0.28	100.0	95.5	4.5	95.5
C	2.5	37.1	0.69	100.0	73.8	26.2	73.8
H	8.7	33.3	1.3	100.0	88.5	11.5	88.5

TABLE 3. - Comparative thickener underflow

Preparation Plant	Preliminary Thickener Underflow Pulp pH	Preliminary Thickener Underflow Pulp Solids %	Bulk Sample Underflow Pulp pH	Bulk Sample Underflow Pulp Solids %
C	7.8	37.1	7.9	19.1
K	--	26.7	6.3	40.8
J	--	25.6	8.2	30.7
B	9.0	12.5	10.1	24.0

The data for Plants B, C, and K samples illustrate that major changes in plant operating conditions take place. Such changes would certainly affect subsequent dewatering equipment requirements, such as filters or centrifuges. To obtain good realistic plant filter or centrifuge equipment design data would require prolonged on-site testing.

Preparation Plant Conditioning Agents

All preparation plants covered by this investigation were thickening their fine coal refuse streams and were adding reagent to the thickener feed pulp. These reagents were flocculants or coagulants used to improve the settling characteristics of the solids. All of the plants used two or three conditioning agents, which are presented in Table 4.

Nineteen different reagents were used in the 12 plants. The best pulp thickening was obtained in Preparation Plant A using Percol 158 and acid. The Plant A refuse contained the least amount of ash (27.5%) and the least amount of 2 µm x 0 size material (2.8%). The high settled pulp density must be related to the ash content and the particle size in addition to the flocculant used.

Examining Table 4 shows that the same reagents will not provide the same results with different refuses.

TABLE 4. - Thickener pulp conditioning

Preparation Plant	Conditioning Agents	Thickener Underflow Solids %	Thickener Underflow pH
A	Percol 158, Acid	52.0	7.8
L	Nalco 8852 and 8873	49.7	8.3
C	Nalco 8852 and 8873	37.1	7.8
G	Nalco 8852 and 8873	31.5	7.9
K	Dowell M191 and M88	40.8	6.3
F	Dowell M191 and M88	31.0	7.6
D	American Cyanamid 550, Betz 3310, Al_2SO_4	35.3	7.7

TABLE 4 -- continued

Preparation Plant	Conditioning Agents	Thickener Underflow Solids %	pH
H	American Cyanamid 1202 and 315	33.8	8.0
E	American Cyanamid 1202 and 330	14.8	7.4
B	American Cyanamid 1202 and Lime	12.5	9.0
J	American Cyanamid 214 and 208 or Percol 351 and American Cyanamid 355	30.7	8.2
I	Calgon 503 and 570	25.0	8.2

The dewatering studies conducted on the five bulk samples used eight different reagents: lime (CaO); Nalco 8873; Celanese 347; Percol 725, 727, and 730; Superfloc 208; and Polyhall 990.

Only three of these reagents were the same ones being used by the preparation plants that used 19 different reagents. Plant J provided the flocculant information that it generally uses American Cyanamid 214 to the thickener feed and American Cyanamid 208 to the centrifuge feed. On the day that the bulk sample was obtained, Allied Colloids 351 was being used in place of American Cyanamid 214. Also, at times, American Cyanamid 355 (cationic) is substituted for American Cyanamid 208 (anionic). This information infers that changes occur in refuse character. Conditioning reagents are very important in fine coal refuse dewatering. One area needing greater investigation is the conditioning or flocculation of the coal refuse.

Ash Sulfur Btu Analysis

As a part of refuse characterization, the ash, sulfur, and Btu analyses of selected products were determined. The ash content of the 11 preliminary thickener underflow sample float-sink products was determined. A more detailed analysis was made on the bulk thickener underflow samples used for dewatering tests. Specifically, the bulk samples were analyzed for sulfur, Btu, ash of float-sink products, and ash of screen products. These data are presented in Exhibit 3.

The total ash content and the ash content of the 1.60 sp gr float-sink are presented in Table 5.

The total ash content of the sampled refuse materials covered a wide range from 25.7% to 72.4% ash. Likewise, the 1.60 float product covered a wide range from 9.7% to 63.3% of the refuse weight and had ash contents from 9.4% to 24.4%. Some of these refuses should be candidates for further cleaning to recover more coal. Cleaning the present refuse would increase the ash content of the final refuse. One of the refuse characteristics to be examined in dewatering studies was ash. Any plant with a low-ash fine refuse product should consider additional coal cleaning in conjunction with coal refuse dewatering studies.

The bulk samples were separated by float-sink at 1.60 sp gr and by the ash, Btu, and sulfur determined on each product. These data are summarized in Table 6.

TABLE 5. - Thickener underflow ash content

Preparation Plant	Coal Region	Preliminary Sample: Total Ash %	Preliminary Sample: Float Weight %	Preliminary Sample: Float Ash %	Preliminary Sample: Sink Ash %	Bulk Sample: Total Ash %	Bulk Sample: Float Weight %	Bulk Sample: Float Ash %	Bulk Sample: Sink Ash %
B	Western	72.4	9.7	20.8	77.9	63.4	9.8	17.7	68.4
L	Northern Appalachian	NA	NA	NA	NA	48.1	29.0	13.9	62.1
K	Southern Appalachian	57.9	13.4	17.3	64.2	49.2	19.6	10.6	58.6
C	Western	37.5	44.8	24.4	48.1	34.0	63.3	18.1	61.4
J	Southern Appalachian	35.2	39.3	12.1	50.1	25.7	59.7	15.9	40.2
I	Alabama	46.6	28.4	16.2	58.6	--	--	--	--
D	Northern Appalachian	43.7	15.4	15.0	48.9	--	--	--	--
F	Northern Appalachian	41.8	23.8	11.5	51.2	--	--	--	--
H	Southern Appalachian	38.0	34.9	9.4	53.4	--	--	--	--
G	Southern Appalachian	34.3	27.9	14.5	41.9	--	--	--	--
E	Northern Appalachian	34.1	21.9	20.7	37.9	--	--	--	--
D	Northern Appalachian	27.5	62.3	11.8	53.5	--	--	--	--

TABLE 6. - Bulk samples ash, sulfur Btu analysis

Preparation Plant	Total Refuse Ash %	S %	Btu	1.60 sp gr Float Weight %	Ash %	S %	Btu
B	63.4	0.73	4,759	9.8	17.7	0.82	10,028
K	49.2	0.63	6,680	19.6	10.6	0.87	12,708
L	48.1	3.10	7,118	29.0	13.9	2.94	12,716
C	34.0	1.14	8,754	63.3	18.1	0.70	11,248
J	25.7	0.72	11,284	59.7	15.9	0.68	12,982

There were great differences in the quality of the refuse as to ash, sulfur, and Btu content.

The C and J plant samples had relatively large amounts of low sulfur coal in the refuse; however, the ash content of the 1.60 sp gr float product was fairly high. If coals C and J were cleaned at 1.60 sp gr, the sink or refuse products would contain 61.4% and 40.2% ash. Thus, the cleaned refuses would have ash contents more similar to the other three refuse products, and their dewatering characteristics would be expected to change.

The bulk refuse samples were screened, and the ash content of each size was determined as shown in Table 7.

TABLE 7. - Bulk refuse samples ash content of screen fractions

Preparation Plant	8 x 28 Weight %	8 x 28 Ash %	28 x 100 Weight %	28 x 100 Ash %	100 x 325 Weight %	100 x 325 Ash %	325 x 0 Weight %	325 x 0 Ash %
B	1.1	23.8	11.4	52.4	36.9	63.4	50.5	75.2
K	10.4	9.2	14.1	25.3	12.6	24.1	62.9	65.2
L[1]	10.8	10.7	19.6	20.3	14.5	40.8	52.4	69.4
C	--	--	23.5	12.1	38.5	11.9	38.0	63.9
J	2.7	14.3	27.2	14.1	17.8	15.8	52.3	36.8

[1] Plus 8 mesh, 2.7 wt %, 7.7% ash.

Based on the particle size analysis in Table 7, Sample C may be a candidate for cleaning, as only 38% of the material is minus 325-mesh size and is a high ash product. All other samples contained more than 50% minus 325-mesh material.

Sample J was an anomalous material, as the minus 325-mesh fraction was greater than 50% of the weight; however, this fraction had a relatively low ash content of 36.8%. The other four minus 325-mesh products had ash contents of approximately 64% to 75%.

The ash content of the refuse has been considered a factor in establishing dewatering characteristics. Table 8 presents an ordering of the samples based on ash content.

TABLE 8. - Dewatering order based on ash

Preparation Plant	Ash %	Projected Dewatering Difficulty
J	25.7	Low
C	34.0	
L	48.1	Medium
K	49.2	
B	63.4	High

The actual equipment performance test results were not directly related to the Table 8 projection of dewatering difficulty. The quantity of ash was undoubtedly a contributing factor to dewatering. However, knowing only the ash content of a refuse sample would not be a reliable basis for directing the development of a dewatering system.

Clay Analysis

The quantity of clay and the types of clay present in the ash were presumed to be important factors in characterizing the refuse. Several different approaches were taken to assess the clay mineralization. Separations were made to quantify the amount of clay size material, 2 μm or less particles as shown in Exhibits 5 and 8. These clay size particles were subjected to an XRD analysis to determine the minerals present, and this information is presented in Exhibit 6. In Exhibit 7, data are presented on the determination of clay minerals by specific gravity, by XRD, and by settling on the bulk samples.

The clay minerals found were kaolinite, illite, and montmorillonite. The thickener underflow refuse from Preparation Plant B was the only one to contain a significant amount of montmorillonite, and it contained little or no illite. All other samples contained kaolinite and illite.

Table 9 contains calculated quantities of clay minerals in the bulk refuse samples. Each refuse was sampled and separated at 2.0 sp gr. The plus 2.0 sp gr particles were then separated at 2 μm by a settling rate procedure based on Stokes law to determine clay size (-2 μm) particles. The amounts of clay minerals were estimated from XRD analysis of the minus 2-μm clay size particles.

Also, the amount of clay in the refuse could be estimated through XRD analysis of the plus 2.0 sp gr material. The results were similar to those presented in Table. 9.

TABLE 9. - Clay content of bulk refuse samples

Preparation Plant	Total Clay Size Weight %	Total Clay Minerals Weight %	Kaolinite Weight %[1]	Illite Weight %[1]	Montmorillonite Weight %[1]	Other[2] Weight %[1]
B	28.0	25.2	5.0	--	20.2	2.8
K	29.5	21.3	13.0	8.3	--	8.2

TABLE 9 -- continued

Preparation Plant	Total Clay Size Weight %	Total Clay Minerals Weight %	Kaolinite Weight %[1]	Illite Weight %[1]	Montmorillonite Weight %[1]	Other[2] Weight %[1]
L	23.0	15.2	6.2	9.0	--	7.8
C	4.7	3.1	2.2	0.9	--	1.6
J	2.3	1.7	1.1	0.6	--	0.6

[1] Estimated quantities of specific minerals by XRD.
[2] Chlorite, quartz, feldspar.

Based on clay and clay size particles, the refuse from Plant B should have been the most difficult to dewater. This material was the only refuse to contain a significant amount of montmorillonite, which is a swelling-type clay. Swelling clays are generally difficult to flocculate and filter. They tend to produce a thixotropic filter cake.

The other four samples would be expected to have filtering rates or dewatering characteristics in the order presented. That is, K material would be the next most difficult to dewater, and J material, the least difficult.

The amount of clay size material (2 μm) in the preliminary samples was determined. Also, the mineral composition of the 2-μm material was identified without a quantitative estimation. The quantity of clay size minerals in the preliminary and bulk samples is shown in Table 10.

TABLE 10. - Clay size minerals

Preparation Plant	Clay Size Material wt % <2 μm: Preliminary Samples	Clay Size Material wt % <2 μm: Bulk Samples
B	36.20	28.0
K	25.70	29.5
L	--	23.0
E	22.88	--
D	21.51	--
F	18.95	--
C	18.68	4.7
I	16.69	--
G	15.77	--
H	15.70	--
J	6.90	2.3
A	6.45	--

Preparation Plant K samples were similar, while the remaining four bulk samples had a lower quantity of clay minerals than the preliminary samples had. On the preliminary sample, the total amount of ≤2 μm material was approximated based on Stokes law. Some fine coal was in the clay-size material. On the bulk samples, only the clay size material in the ≥2.0 sp gr fraction was determined. Therefore, a part of the difference between the two samples can be attributed to <2.0 sp gr fines and part to variation in the coal refuse.

Mineralogical Analysis

The mineral content of nine preliminary samples and the plus 2.0 sp gr fractions of the five bulk and two preliminary samples were determined by XRD. The total coal refuse sample was used for the original samples, and in most cases, the major constituent was amorphous coal. In the XRD studies, coal was not of interest; therefore, in subsequent XRD studies, most of the coal was rejected by separating the refuse at 2.0 sp gr. The details of these studies are in Exhibit 7.

All preliminary samples contained amorphous coal >10%, quartz >10%, and from trace to moderate amounts of crystalline minerals. The most signigicant variations in the mineral composition of the samples are shown in Table 11.

The sample from Plant B was the only one identified as containing montmorillonite clay, and it was present in a moderate amount (10% to 30%). This sample had the smallest amount of quartz (moderate/minor) and the largest amounts of feldspar and unidentified crystalline material (minor, 3% to 10%). The unidentified minerals may have included a zeolite mineral heulandite and cristobolite (SiO_2).

The sample from Plant A had a different composition of largely quartz, calcite, and dolomite with only traces of clay minerals, kaolinite, and illite.

The Preparation Plant K sample was largely quartz, kaolinite, and illite with no calcite or dolomite.

The remaining eight samples had a similar mineral makeup with slight variations in the amounts of quartz, calcite, dolomite, kaolinite, and illite.

The eight samples classified as having similar mineralogy could have similar dewatering characteristics. The other three samples -- A, B, and K -- would be expected to have different dewatering characteristics. The difference in mineral compositions may require different conditioning or flocculation to obtain good dewatering equipment performance.

An XRD analysis of the plus 2.0 sp gr material from the bulk samples showed a large variety of minerals to be present. The minerals were identified as quartz, kaolinite, illite, montmorillonite, feldspar, magnetite, rutile, pyrite, calcite, aragonite, dolomite, siderite, gypsum, and amorphous (coal). The principal minerals were coal, quartz, kaolinite, and feldspar in all samples. Montmorillonite was a major mineral in the sample from Plant B, and illite was present in the samples from Plants C, J, K, and L. The Plant J sample contained an estimated 11% magnetite in the plus 2.0 sp gr material. The other minerals were present in amounts ranging from zero to 5% of the plus 2.0 sp gr material and should have little or no effect on dewatering.

The coal and clay minerals were expected to have the greatest effects on refuse dewatering. No effort was made to characterize the coal other than to quantify based on coal analysis. Flocculating clay minerals can be a problem.

TABLE 11. - XRD analysis of preliminary samples

Preparation Plant	Phase and Estimated Amounts						
	Calcite	Dolomite	Pyrite	Kaolinite	Illite	Montmorillonite	Feldspar
A	Moderate	Moderate/Minor	--	Trace	Trace	--	Trace
B	--	--	--	Minor	Minor	Moderate	Minor
C	Trace	Trace	--	Minor/Trace	Trace	--	--
D	Minor/Trace	--	Trace	Minor	Trace	--	--
E	Minor/Trace	--	Minor	Minor	Trace	--	--
F	Minor/Trace	Trace	Minor	Minor	Minor/Trace	--	--
G	Minor	--	--	Minor	Minor	--	Trace
H	Minor/Trace	Trace	--	Minor	Minor	--	--
I	Minor/Trace	Minor/Trace	Trace	Minor/Trace	Trace	--	--
J	Minor	--	Minor/Trace	Minor	Minor	--	--
K	--	--	Trace?	Moderate	Moderate	--	Trace?

Note: Major >30%, Moderate 10% to 30%, Minor 3% to 10%, Trace <3%.

Kaolinite and illite are fine-grained, nonswelling clay minerals, whereas, montmorillonite is a swelling clay. Montmorillonite can be a difficult mineral to filter, as it can exhibit gel-like characteristics and be thixotropic.

Particle Size Analysis

The particle size of the solids in the refuse dewatering thickener feed and underflow streams from the original nine preliminary samples was determined. Samples were processed by wet screening on 28 mesh, by drying the plus 28-mesh material, and by dry screening using a Ro-Tap. The size distribution of the minus 28-mesh material was determined using a soils hydrometer method. Details of the particle size analysis are presented in Exhibit 2, and the results are summarized in Table 12. A description is given of the coarsest size particles, particles passing 325 mesh, and the quantity of the finest size particles measured. The table is arranged in descending order from the largest amount of 2-µm material in the thickener underflow.

TABLE 12. - Particle size distribution

	Thickener Feed		Thickener Underflow		
Preparation Plant	-28M Weight %	-2 µm Weight %	-28M Weight %	-325M Weight %	-2 µm Weight %
B	92.6	36.5	97.3	70.5	44.8
D	98.0	14.1	96.0	56.7	18.9
E	100.0[1]	6.9[1]	98.2	50.9	17.3
C	96.0	12.0	96.0	47.8	16.6
F	87.5	10.6	94.0	46.3	16.6
H	97.0	10.3	97.5	43.2	13.0
G	NA	NA	94.0	38.8	12.1
I	81.0	1.0	87.0	26.4	7.9
A	99.4	2.8	92.5	32.6	2.8

[1] Accuracy is questionable as too small a sample was used.

The nominal passing size of the feed was 28 mesh (595 µm). All of the samples were 100% passing 1/4-in. size. The sample from Plant I was the coarsest material, and it contained 5.5% of plus 8 mesh.

The thickener feed and underflow particle size analyses were similar but not exactly the same. Differences would be accounted for as being different samples, precision of analysis, and some material was eliminated from the feed into the overflow product.

The projected dewatering of preliminary samples from Plants A-I would be in the order presented in Table 12 based on 2-µm material with only A and I being reversed if based on 325-mesh material.

The nine samples covered a wide range of material relative to fine particle size.

Thickener underflow bulk refuse samples were subjected to wet and dry screen analysis and to clay particle size analysis by sedimentation. These data are presented in Exhibits 2 and 8 of the Appendix and in Table 13 in the order of decreasing amounts of 2-μm material.

TABLE 13. - Particle size analysis of bulk samples

Preparation Plant	Cumulative Passing Size 8M %	28M %	100M %	325M %	2 μm %
K	100.0	89.6	75.5	62.9	29.5
B	100.0	98.8	87.4	50.5	28.0
L	97.3[1]	86.5	66.9	52.4	23.0
C	100.0	100.0	76.5	38.0	4.7
J	100.0	97.3	70.1	52.3	2.3

[1] 2.7% +8M.

The order would be different if a passing size of 100 mesh were used and again different if a passing size of 325 mesh were used. Simply stated, the refuse samples did not have similar particle size distribution patterns.

Preparation Plant B material was the most difficult to dewater. However, based on 2-μm-size particles, it would be the second most difficult and based on 325-mesh particles, it would be the fourth most difficult to dewater. In generalizing terms, B would dewater the same as J and L based on 325-mesh size. Other publications (1-3) on coal refuse dewatering have stated that the quantity of minus 325-mesh material is the main controlling factor in dewatering. From the information developed in this program, the characterization factors controlling dewatering are much more involved than particle size passing 325 mesh.

Two different samples were obtained from Plants B and C, preliminary and bulk. Their analyses are presented in Tables 12 and 13, and the results were different as shown in Table 14.

TABLE 14. - Particle size comparison

Preparation	Sample	Weight % Passing 325M	2 μm
B	Preliminary	70.5	44.8
B	Bulk	50.5	28.0
C	Preliminary	47.8	16.6
C	Bulk	38.0	4.7

The results in Table 14 show a great change in thickener underflow particle size from preliminary to bulk samples. Such changes would be expected to change the dewatering problems.

A precise determination of the particle size of coal refuse presents some unusual problems. The problems are related to surface characteristics of coal and ash minerals as to wetability, dispersion, and specific gravity, especially in the micron size particles of clay.

CSMRI used a soils sedimentation procedure to determine the clay size (2-μm) material. Several different techniques were used to obtain the results presented. The final approach was to first make a separation at 2.0 sp gr. This removed most of the light gravity coal from the sample. Thus, the quantity of 2-μm material could be low if there was any 2-μm coal present. The plus 2.0 sp gr material was dispersed in a blender using sodium hexametaphosphate. The sample was then analyzed using a standard soils sedimentation procedure.

The 2-μm analysis on the preliminary samples was made on the total refuse sample. The analysis has a discrepancy because of a wide range of particle specific gravity. An average specific gravity is used in the calculations. This same problem of specific gravity applies to most micron analysis methods.

Some samples required reanalyzing because good dispersion had evidently not been obtained. The error became apparent from the XRD analysis to determine clay minerals that are 2-μm particles or smaller.

Another problem with most of the samples was that they had already been flocculated. Whether or not all of the flocs, especially in the micron size, have been destroyed cannot be ascertained.

Samples were analyzed with instruments by two outside laboratories. The results are not reported here, because they were greatly different. In discussing the anomaly with these operators, the results became questionable, because they were analyzed on a routine basis, because the dispersing reagent and technique used were questionable, and because the instruments are not accurate below 1 μm, which could be the size of much of the clay.

Size analysis of fine coal refuse is not a simple matter, especially in the micron size range. Developing a reliable particle size analysis method for fine coal refuse would be an interesting study.

Filter Tests of Thickener Underflow Slurries

Preliminary Samples 1 Through 11

A bench-scale vacuum filtration setup was used to gather dewatering data on the preliminary coal refuse slurries from Preparation Plants A through I, CSMRI Samples 1 through 9. The setup consisted of a vacuum pump, filtrate flask, pour-on-type test leaf, vacuum gage, and a quick-release valve between the vacuum source and the test leaf. Each slurry was poured on the test leaf, vacuum was applied, and a stopwatch was started. Filtration continued until water was no longer visible on the top of the formed cake. Wet and dry cake weights were determined along with filtrate volume and filtration time. A second set of filtration tests was performed by a different investigator, approximately 5 months following the first set. These test data are included in Table 15 along with the first set. All details and results are available in Exhibit 10.

TABLE 15. - Results of filtration tests on preliminary samples

CSMRI Sample No.	Preparation Plant	Cake Dry Solids Series 1 g	Cake Dry Solids Series 2 g	Cake Solids Series 1 %	Cake Solids Series 2 %	Filter Capacity[1] (Cake) Series 1 lb/ft²/hr	Filter Capacity[1] (Cake) Series 2 lb/ft²/hr
1	A	103.3	73.6	90.6	74.0	71	146
2	B	[2]	7.3	--	54.9	[2]	148
3	C	68.4	58.4	68.1	62.9	12	23
4	D	[3]	58.7	--	63.5	[3]	3
5	E	27.0	14.5	64.8	63.0	6	13
6	F	54.1	49.1	68.1	68.1	4	4
7	G	59.8	21.1	71.3	66.4	15	15
8	H	59.8	54.7	70.6	68.7	5	9
9	I	47.2	20.8	76.4	69.1	10	9

[1] Filter capacity calculated using the following formula:

$$\frac{\text{Dry Cake Wt, g}}{454\ \text{g/lb}} \times \frac{1}{\text{Filter Area, } 0.05\ \text{ft}^2} \times \frac{1}{\text{Filtration Time, min}} \times \frac{60\ \text{min}}{\text{hr}} = \frac{\text{Dry lb}}{\text{ft}^2\text{-hr}}$$

[2] Test sample deemed unfilterable.
[3] Test aborted due to extremely slow filtration rate.

As can be seen from these data, the tests performed 5 months later resulted in filtration rates greater than or equal to the previous series.

Series 1 and 2 tests were conducted by two different investigators, and the amount of solids were not the same in both series. Overall, the data show that there was no deterioration of the material. Samples 7 and 9 may have deteriorated, as the solids in the second series were only one third to one half of the first series. Some samples showed a significant increase in filtration rate. Possibly, this increase in rate was due to flocculation and/or coagulation on standing. All of the pulps had been flocculated or coagulated in the plant thickeners.

Bulk Samples 13 Through 17

Bench-scale filtration tests were also carried out on portions of the bulk samples from Preparation Plants B, C, J, K, and L. The equipment employed was identical to that described for the two previous test series, and two series were conducted on these five samples as well. The two test series on the bulk samples were carried out 5 months apart, just as in the previous series, but with the exception that the first series employed flocculant addition. All details and results are available in Exhibit 11, and selected data for both series of filtration tests are presented in Table 16.

The use of flocculants as conditioning agents caused a dramatic increase in the rate of filtration, i.e., from a 6-fold increase for Sample K to a 44-fold increase for Sample B. Unfortunately, a comparison to evaluate the 5 months of aging between the first and second series is not possible due to the use of flocculants in the first series. The retained cake moisture for flocculated samples of C, J, and K was less than for those samples left unflocculated. The other two

TABLE 16. - Results of filtration tests on bulk samples

Sample No.	Preparation Plant	Dry Cake Solids Series 1 g	Dry Cake Solids Series 2 g	Cake Solids Series 1 %	Cake Solids Series 2 %	Flocculant Series 1 Brand	Flocculant Addition lb/ton	Filter Capacity[1] (Cake) Series 1 lb/ft^2-hr	Filter Capacity[1] (Cake) Series 2 lb/ft^2-hr
13	B	28	40	50.0	29.0	Nalco 8852	6.3	88	2
14	C	38	23	54.3	65.9	Celanese 347 Celanese 990	0.1	100	10
15	J	34	48	61.8	67.7	American Cyanamid 208	0.2	120	18
16	K	47	72	63.5	66.5	Dowell M88	0.9	18	3
17	L	58	101	70.7	62.6	Celanese 347	3.4	31	4

[1] Filter capacity calculated using the following formula:

$$\frac{\text{Dry Cake Wt, g}}{\text{454 g/lb}} \times \frac{1}{\text{Filter Area, 0.05 ft}^2} \times \frac{1}{\text{Filtration Time, min}} \times \frac{\text{60 min}}{\text{hr}} = \frac{\text{Dry lb}}{\text{ft}^2\text{-hr}}$$

samples, B and L, exhibited the opposite behavior: less cake moisture retained for the flocculated slurries. In general, cake moisture is reduced with the addition of polymers, but that excess addition will cause an increase in cake moisture. Only through an extensive flocculant study can the best flocculant and optimum addition be determined for any coal refuse sample.

Characterization Summary

One of the project goals was to characterize the various coal refuse samples in an effort to predict their dewatering characteristics. Specific data that might be related to dewatering performance were ash content, clay analysis, mineralogical analysis, particle size analysis, and laboratory filtration. Table 17 presents a relative prediction of dewatering based on the probable effect of the various factors. Assumptions were made that the degree of difficulty in dewatering would increase with increases in the following.

1. Ash.
2. Clay.
3. Montmorillonite clay.
4. Minus 325-mesh particles.
5. Minus 2-μm particles.
6. Laboratory filtration rate.

TABLE 17. - Projected dewatering performance

	Degree of Difficulty to Dewater				
	Preparation Plants				
Characteristic	B	C	J	K	L
Ash	High	Low	Low	Medium	Medium
Clay	High	Low	Low	High	Medium-High
Montmorillonite Clay	High	Low	Low	Low	Low
-325M	Medium	Low	Medium	High	Medium
-2 μm	High	Low	Low	High	Medium-High
Filtration Rate	Low	Medium	High	Low	Medium

Based on the characterization results, the order of decreasing dewatering difficulty in the plants should be B, K, L, J, and C.

As previously mentioned, a very important dewatering factor not directly covered by the characterization studies is flocculation. Dewatering results certainly change with effectiveness of flocculation. However, some of the presented characterization factors, such as particle size and type of minerals, directly affect flocculation.

Flocculant Scoping

Portions of the coal refuse slurries from Preparation Plants B, C, J, K, and L (Samples 13 through 17) were subjected to flocculant scoping in Golden. The flocculant scoping tests were conducted using readily available reagents to obtain some preliminary information regarding the degree of difficulty in flocculating the various samples. A variety of flocculants were employed at concentrations ranging from 0.25 to 5 gpl. Pulp concentration was also varied; most of the time dilutions were made to aid visual evaluation of floc formation. The observations were then recorded as to floc formation, water release, and supernatant clarity. Additional flocculant scoping tests were conducted prior to each pilot-plant dewaterng test. All details of procedures and results are provided in Exhibit 9. Selected data for the five series are presented in Tables 18 through 22.

TABLE 18. - Flocculant scoping for Preparation Plant B

Pulp Solids %	Flocculant Brand	Flocculant Volume ml	Flocculant Strength gpl	Observations
12.0	Nalco 8852	10.0	1.0	Medium flocs, slow water release.
12.0	Celanese 347	10.0	1.0	Medium flocs, medium water release.
24.0	CaO to pH = 11.0	--	--	Gelatinous, globular floc formation.
24.0	Nalco 8873	20.0	1.0	Gelatinous, globular floc formation.
24.0	Nalco 8852	50.0	1.0	Some flocculation, slow water release.
24.0	Celanese 990	5.0	1.0	Gelatinous, globular floc formation.
24.0	CaO to pH = 11.0 followed by American Cyanamid 1202	30.0	0.5	No visible flocculation.
24.0	CaO to pH = 11.0 followed by Celanese 990	2.0	1.0	Gelatinous, globular flocs.

The use of a single flocculating agent was unsuccessful for this coal refuse slurry. The preparation plant itself uses lime addition to pH 11 or 12 followed by an anionic flocculant, specifically American Cyanamid 1202. This combination of treatment agents was scoped without any positive result. Celanese 990, another anionic flocculant, formed gelatinous globular flocs either by itself or in combination with lime.

In the pilot-plant dewatering tests, the following reagents were used: lime, Hercules 847 and 1018, Allied Colloids 730, and Celanese 990. The Celanese 990 was used in the scoping and did not appear to be favorable, however, it was effectively used in the pilot-plant tests. American Cyanamid 1202 and lime were used by the operating plant.

TABLE 19. - Flocculant scoping for Preparation Plant C

Solids %	Flocculant Brand	Flocculant Volume ml	Flocculant Strength gpl	Observations
19.1	Celanese 361	2.5	1.0	Small, very fine flocs formed. Later, very good clarity.
19.1	Celanese 347	2.0	1.0	Small, very fine flocs formed. Later, very good clarity.
19.1	Nalco 8852	0.5	1.0	Small, very fine flocs formed. Superior clarity.
19.1	Dowell M88	0.25	1.0	Large flocs, quick water release, cloudy supernate.

Preparation Plant C uses Nalco 8852 in its filtering circuit and a combination of Nalco 8852 and 8873 in thickening. These scoping data indicated that Nalco 8852 was superior to the other reagents tested. The pilot-plant dewatering studies used Hercules 847 and 849, American Cyanamid 208, and Celanese 990, none of which were used in the scoping study.

TABLE 20. - Flocculant scoping for Preparation Plant J

Pulp Solids %	Flocculant: Brand	Volume ml	Strength gpl	Observations
30.7	American Cyanamid 208	5.0	0.25	Small flocs, fair water release, dirty supernate.
30.7	Celanese 361	35.0	0.25	Tiny flocs, minimal water release.
30.7	Celanese 361	35.0	1.0	Small flocs, slow water release.
30.7	Celanese 990	3.0	0.25	Medium flocs, slow water release, cloudy supernate.
30.7	Celanese 347	15.0	1.0	No visible flocculation.

Preparation Plant J normally adds American Cyanamid 214 to its thickener feed; however, it has also made use of Allied Colloids 351 and American Cyanamid 355 and 208. None of the flocculant concentrations employed in this scoping series resulted in acceptable flocculation. This was due to the presence of a white-colored supernate present in all tests completed. Study would be necessary to find a system that would result in acceptable underflow percent solids and supernate clarity. Hercules 1018, American Cyanamid 208, and Celanese 990 were used in the pilot-plant studies. The 208 and 990 reagents were part of the scoping study and 208 is used in the plant.

TABLE 21. - Flocculant scoping for Preparation Plant K

Solids %	Flocculant: Brand	Volume ml	Strength gpl	Observations
20.4	Celanese 361	3.0	1.0	Medium flocs formed, good water release, clean supernate.
20.4	Celanese 990	0.6	1.0	Medium flocs formed, good water release, dirty supernate.
20.4	Nalco 8852	3.0	1.0	Medium flocs formed, good water release, clean supernate.
20.4	Dowell 191	5.0	1.0	Medium flocs formed, slow water release, dirty supernate.
20.4	Dowell 88	3.0	1.0	Large flocs formed, good water release, dirty supernate.
20.4	American Cyanamid 1202	1.0	1.0	Medium flocs formed, medium water release, cloudy supernate.

Preparation Plant K employs Dowell M191 and M88 in its thickening and dewatering equipment. These data indicate that Celanese 361 resulted in a clearer supernate than Dowell M88 at the same concentration and that Nalco 8852, also at the same concentration, performed as well as the Celanese 361.

TABLE 22. - Flocculant scoping for Preparation Plant L

Solids %	Flocculant Brand	Volume ml	Strength gpl	Observations
24.9	Dowell M88	0.4	1.0	Small flocs formed, good water release, cloudy supernate.
24.9	Nalco 8873	0.4	1.0	Medium flocs formed, good water release, very cloudy supernate.
24.9	Celanese 361	5.0	1.0	Small flocs formed, medium water release, clear supernate.
24.9	Nalco 8852	1.5	5.0	Small flocs formed, slow water release, cloudy supernate.
24.9	Celanese 347	5.0	1.0	Small flocs formed, slow water release, very cloudy supernate.
24.9	Dowell M191	2.0	5.0	Small flocs formed, slow water release, cloudy supernate.

Pilot-plant dewatering tests were conducted using Hercules 847, 849, and 1018; American Cyanamid 208; and Celanese 990. Only Celanese 990 had been tested in the scoping tests.

Preparation Plant L uses Nalco 8852 and 8873 in its thickening equipment. These data indicate that the Nalco reagents did not perform as well as Dowell M88 at similar or lower concentrations.

The pilot-plant dewatering tests on Plant L refuse used Celanese 990; Allied Colloids 725, 727, and 757, and lime.

A number of reagents are used by the operating plants, in the scoping tests, and in the actual dewatering tests. In most cases, different reagents are used at the operating plant, in scoping tests, and in pilot-plant tests.

Reagents used in these pilot-plant tests were the result of scoping tests conducted by equipment manufacturers and of additional laboratory tests conducted in Spokane. Every effort was made to use an acceptable reagent for each application.

Obtaining the best results is not only a matter of flocculant choice or dosage. Flocculant concentration will have an effect on proper reagent adsorption, and the effect of mixing must also be considered for optimum reaction between the solid particles in the slurry and the reagent employed. There is a distinct possibility, then, of anomalous test results when optimum flocculation requirements have not been met.

Centrifuge

A Bird Machinery Company Model LB-1400 solid bowl continuous centrifuge was used to dewater the five bulk samples of coal refuse. The refuse, which had been stored in drums, was reslurried and transferred to an agitated feed tank.

The refuse slurry was fed to the centrifuge at a controlled rate through the use of a variable speed Moyno feed pump.

There were 27 runs made on five types of coal refuse over a 2-week period. The effects of the various slurry types, polymer dosage and introduction point, feed rate, and differential speed were investigated.

Several high molecular-weight polymers of various cationic and anionic charges were bench tested. In all cases, anionic polymers flocced the material well, with one sample requiring preconditioning with lime. A liquid anionic polymer, Nalco 8873, was used throughout.

A large batch of slurry was prepared for each preparation plant coal refuse sample. The pulp along with flocculant was fed to the centrifuge. The centrifuge was operated for a period of 2 to 3 hr on each refuse sample. During this period, the operating conditions were changed from two to eight times, and data were collected. The details of the tests are presented in Exhibit 16 of the Appendix.

Preparation Plant B Coal Refuse

Sample 13 required lime as a preconditioning agent prior to flocculating with an anionic polymer to achieve a strong flocced particle that would release water readily. An amount of lime equivalent to 30 lb CaO/ton dry solids was slurried at a 15% consistency in a 20-gal barrel, then was added to the agitated feed tank. The viscosity of the feed slurry increased dramatically when the lime was added; however, it decreased with time during the run.

Selected results from Exhibit 16 are presented in Table 23.

TABLE 23. - Centrifuge dewatering results for Preparation Plant B refuse

Run No.	Nalco 8873 Flocculant lb/ton	Feed Solids %	Feed Dry Solids lb/hr	Cake Solids %	Solids Recovery %	Solids Recovery Dry Solids lb/hr
18	1.75	22.7	628	57.5	99.1	622
19	3.37	22.7	653	54.3	99.8	652
17	1.03	22.7	683	52.2	99.7	681
16	0.95	22.7	685	51.0	99.8	684
20	0.52	22.7	2,015	54.9	94.8	1,910

Better than 99% solids recovery was obtained with cake solids of 57.5% using 1.75 lb liquid polymer per ton dry feed solids. A 44-in.-diam by 132-in.-length 250 hp production-size unit would be able to handle approximately 15 tph while obtaining similar results.

Preparation Plant C Coal Refuse

Sample 14, coal refuse from Preparation Plant C, was much easier to handle with the centrifuge than the refuse from Preparation Plant B. The results are tabulated in Table 24.

TABLE 24. - Centrifuge dewatering results for Plant C refuse

Run No.	Nalco 8873 Flocculant lb/ton	Feed Solids %	Feed Dry Solids lb/hr	Cake Solids %	Solids Recovery %	Solids Recovery Dry Solids lb/hr
14	0.79	17.1	442	67.2	99.9	442
11	0.78	18.0	1,219	67.8	99.3	1,210
10	1.18	18.0	1,308	68.5	99.9	1,307
13	0.26	18.0	1,328	69.3	99.3	1,319
12	0.42	18.0	1,437	68.0	99.6	1,431
15	0.41	17.1	2,212	67.3	100.0	2,212

Cake solids ranged from 67% to 69% with solids recoveries in most cases better than 99.5%. The required polymer dosage is 0.2 to 0.4 lb liquid polymer per ton dry feed solids. A production-size unit obtaining similar results would be able to handle approximately 30 tph of this material.

Preparation Plant J Coal Refuse

The coal refuse identified as CSMRI Sample 15 from Plant J was reslurried at approximately 29% solids. Nine tests were conducted on the pulp using various additions of flocculants and pulp feed rate. The test results are summarized in Table 25.

TABLE 25. - Centrifuge dewatering results for Preparation Plant J coal refuse

Run No.	Nalco 8873 Flocculant lb/ton	Feed Solids %	Feed Dry Solids lb/hr	Cake Solids %	Solids Recovery %	Solids Recovery Dry Solids lb/hr
1	1.01	29.8	841	67.6	99.4	836
2	0.67	29.8	895	66.2	99.3	889
5	1.66	28.4	935	67.1	98.9	925
4	0.43	28.4	1,068	68.0	98.1	1,048
3	0.05	28.4	1,068	70.0	98.4	1,051
6	0.66	28.4	1,068	66.6	99.6	1,064
7	0.64	29.5	2,197	68.9	99.7	2,190
8	0.59	29.5	2,374	68.4	99.8	2,369
9	2.07	29.5	2,321	65.6	98.5	2,286

Cake solids of 68% to 69% were obtained at better than 99.5% solids recovery using 0.6 lb liquid polymer per ton dry feed solids. A production-size unit would be able to accomplish this at about 25 tph throughput.

The results were inconclusive, because at the lowest level of flocculant addition, 0.05 lb/ton, the cake had the highest percent solids with only a slightly lower recovery of solids.

Variation in feed rate from 841 to 2,374 lb/hr of dry solids had no effect on the product cake's moisture content or on cake recovery. A prolonged program would be required to optimize the results.

The preparation plant that generated this sample also makes use of a solid bowl centrifuge for dewatering and produces product cake solids of 68.2% using 0.5 lb of flocculant per ton. The plant results compare favorably to the pilot-plant results for the nine test conditions, as the average cake produced was 67.6% solids using 0.86 lb of flocculant per ton.

Preparation Plant K Coal Refuse

No problems were experienced in the pilot-plant run of this coal refuse slurry. The results are presented in Table 26.

TABLE 26. - Centrifuge dewatering results for Preparation Plant K refuse

Run No.	Nalco 8873 Flocculant lb/ton	Feed Solids %	Feed Dry Solids lb/hr	Cake Solids %	Solids Recovery %	Solids Recovery Dry Solids lb/hr
24	0.93	37.4	975	59.5	99.9	974
22	0.52	37.4	1,073	60.4	99.9	1,072
21	0.80	37.4	1,073	60.8	100.0	1,073
22	1.12	37.4	1,162	59.6	99.9	1,161
25	0.62	37.4	2,329	59.5	99.9	2,327

A solids recovery level of 99.9% was consistently obtained with cake solids of 60%. The polymer dosage required was 0.5 lb liquid polymer per ton of dry feed solids. A production-size unit would be able to repeat this at up to 25 tph throughput.

Preparation Plant L Coal Refuse

The coal refuse thickener underflow pulp from Preparation Plant L contained 50.2% solids. This refuse contained 52.4% minus 325-mesh material and 8% of the plus 2 sp gr material as illite clays.

Only two tests were conducted on this sample, because difficulty was experienced in maintaining consistent feed and the test centrifuge did not have sufficient torgue to handle the high-solids feed. Nonetheless, the two tests indicated reasonably acceptable levels of cake solids production. These results are presented in Table 27.

TABLE 27. - Centrifuge dewatering results for Preparation Plant L coal refuse

Run No.	Nalco 8873 Flocculant lb/ton	Feed Solids %	Feed Dry Solids lb/hr	Cake Solids %	Solids Recovery %	Solids Recovery Dry Solids lb/hr
26	0.65	50.2	1,574	68.6	99.7	1,569
27	0.52	50.2	3,708	74.9	86.8	3,219

The feed slurry was extremely thick at a concentration of 50% solids. Cakes of 70% solids would be obtained with 99% recovery at a throughput of 30 to 40 tph in a large production-size unit. Polymer consumption would be about 0.6 lb liquid polymer per ton dry feed solids. Some oil was observed in the centrate.

The results at the highest product rates of acceptable quality are summarized in Table 28.

TABLE 28. - Summary of centrifuge dewatering tests

CSMRI Sample No.	Preparation Plant	Nalco 8873 Flocculant lb/ton	Feed Solids %	Feed Dry Solids lb/hr	Product Cake Solids %	Solids Recovery %	Solids Recovery Dry Solids lb/hr
13	B	1.75	22.7	628	57.5	99.1	622
14	C	0.41	17.1	2,212	67.3	100.0	2,212
15	J	0.59	29.5	2,374	68.4	99.8	2,369
16	K	0.62	37.4	2,329	59.5	99.9	2,327
17	L	0.65	50.2	1,574	68.6	99.7	1,569

Average cake solids of 64.3% were produced with average flocculant addition of 0.80 lb/ton of solids. This result compares reasonably well with the unit operating at one of the preparation plants sampled, which used 0.5 lb of flocculant per ton of solids and produced a cake of 68.2% solids.

Centrifuge units are compact, easy to operate, and have reasonably low maintenance requirements. However, their one operating drawback is that they can act like classifiers with large amounts of fine particles reporting to the centrate. This behavior will obviously cause problems for all dewatering devices in the plant as the recirculated water takes on a higher percentage of fine particles.

Belt Presses

The performance of two belt presses on coal refuse thickener underflow from Preparation Plants B, C, J, K, and L (CSMRI Samples 13 through 17) was investigated. The Arus-Andritz mobile laboratory unit detailed in Exhibit 14 was run at the Spokane Mining Research Center by Mr. Gary Andrews with the assistance of Mr. P. S. Jacobsen, CSMRI. The other available belt press data detailed in Exhibit 15 were the result of investigations by Parkson Corporation (Fort Lauderdale, Florida) on 5-gal portions of each of the five refuse samples shipped to its facilities. No representative of CSMRI was present during any of this test work, and, as a result, no information regarding the procedure was available to be included here. A synopsis of the Arus-Andritz procedure follows.

The Arus-Andritz press laboratory unit simulated the three dewatering zones of the Standard Andritz Sludge Dewatering Machine (SDM). Initially, a bench-scale polymer addition investigation took place to determine the best available flocculation system. The flocculated sample was then tested in three steps to simulate the progressive dewatering zones of a SDM. First, the free water was allowed to drain through a sample of belt mesh that simulated the gravity zone of the commercial-scale unit. Secondly, another belt sample was placed over the sludge solids, followed by a tray, and hand pressure applied to force out water as in the wedge zone. Finally, the mesh containing the cake was then passed around the S-roll of the laboratory press to simulate the S7 configuration. For the laboratory evaluation, controlled pressures and times were utilized to simulate the pressure application period in gravity and wedge sections and on each S-roll of the high-pressure section. Times were based on a belt speed of 25 ft/min, and the maximum applied roll pressure was 12 psi.

Table 29 presents the Arus-Andritz and Parkson data. According to data presented in Table 29, the Arus-Andritz unit had almost three times the capacity of the Parkson unit, although the Parkson unit required twice the flocculant to produce a cake of essentially the same moisture content. The Parkson unit captured more solids, a trend consistent throughout the test results.

The belt press results are those of the equipment manufacturers and their interpretations of the test data. Parkson markets a low-pressure and a high-pressure machine. The moisture content of the final cake product from the Arus-Andritz machine and the Parkson high-pressure machine was essentially the same. Generally, the belt press produced the second driest cake next to the pressure filter.

There were several significant differences between the two tests and their results. Both organizations arrived at what they considered to be the best flocculation scheme for the preliminary tests. They used different reagents in significantly different amounts. Parkson used more flocculant than Arus-Andritz on three samples, and on the other two, a lesser amount than Arus-Andritz.

The Arus-Andritz results have a solids capture about 2% lower than the Parkson data. Recovery of less solids was probably due to using a more porous medium.

The greatest difference was in the capacity rating of tph/meter of belt width. Arus-Andritz showed a capacity rating between two and three times that of Parkson. These are machine design capacities and are based on each company's machine and its interpretation of test data. The machines are not comparable in size and cost. The Parkson capacity rating was for a relatively short, low (one-level), and low-cost machine. The Arus-Andritz machine is longer, two level, and more costly. There is no method developed to directly compare belt presses on a unit area capacity as can be done for vacuum filters.

Unfortunately, at the time the preparation plant employing belt presses was visited, the units were in the process of being brought on stream. As a result, no comparison of operational data to pilot-plant data is available for the belt press section.

The belt press or double belt filter has been used successfully in the coal, paper, chemical, and municipal waste industries. There are numerous machine configurations that all operate in a similar manner. The material to be dewatered is continuously fed to the unit where it is placed between two belts of permeable material. The double belt travels through a series of rolls where increased pressure squeezes the moisture from the cake. The final product is reportedly dry enough to form a stable mixture when blended with coarse refuse. The tabulated results in Table 29 indicated a range from 50% to 79% cake solids with an average of 65% for both sets of results. The one reported drawback to these units is that they require careful feed pretreatment for proper and optimum operation.

Drum Vacuum Filter

All five coal refuse samples were fed to an Eimco 12-in.-wide by 18-in.-diam drum filter to determine their operating characteristics when subjected to

TABLE 29. - Selected belt press data and results

Belt Press	Preconditioning: Polymer Name	Polymer lb/ton	CaO lb/ton	Cake Solids %	Solids Capture %	Throughput tph/meter of belt width
Sample 13 -- Preparation Plant B						
Refuse Feed Test Conditions: ~24% solids, pH = 10, Specific Gravity = 1.14						
Arus-Andritz	Hercules 1018	0.5-0.75	30	63	96+	6.0
Parkson MPL[1]	Allied Colloids 730	2.5	30	50	98	1.8
Parkson MP[1]	Allied Colloids 730	2.5	30	60	98	1.8
Sample 14 -- Preparation Plant C						
Refuse Feed Test Conditions: ~19% solids, pH = 8, Specific Gravity = 1.07						
Arus-Andritz	Hercules 847	0.25-0.45	--	70	96+	5.4
Parkson MPL[1]	American Cyanamid 208	1.2	--	60	98	2.9
Parkson MP[1]	American Cyanamid 208	1.2	--	70	98	2.9
Sample 15 -- Preparation Plant J						
Refuse Feed Test Conditions: ~30% solids, pH = 8, Specific Gravity = 1.12						
Arus-Adritz	Hercules 1018	0.4-0.65	--	71	96+	6.1
Parkson MPL[1]	American Cyanamid 208	0.3	--	65	98	3.5
Parkson MP[1]	American Cyanamid 208	0.3	--	70	98	3.5
Sample 16 -- Preparation Plant K						
Refuse Feed Test Conditions: ~40% solids, pH = 6.5, Specific Gravity = 1.23						
Arus-Andritz	Hercules 847	0.5-0.75	--	70	96+	7.3
Parkson MPL[1]	American Cyanamid 208	0.9	--	58	98	2.5
Parkson MP[1]	American Cyanamid 208	0.9	--	65	98	2.5
Sample 17 -- Preparation Plant L						
Refuse Feed Test Conditions: ~49% solids, pH = 8.4, Specific Gravity = 1.27						
Arus-Andritz	Allied Colloids 757	1.0-1.3	--	71	96+	8.9
Parkson MPL[1]	Allied Colloids 725	0.5	--	68	99	3.0
Parkson MP[1]	Allied Colloids 727	0.5	--	79	99	3.0

[1] MPL -- low-pressure unit, MP -- high-pressure unit.

vacuum filtration. Each of the samples was subjected to flocculant scoping prior to pilot testing, and the best available flocculant system was used. Standard 55-gal drums of bulk samples were flocculated and fed to the filter using a Sand Piper pump. The drum filter was a self-contained unit with its own source of vacuum and a blower for cake product discharge. The total surface area of the drum was 4 ft^2, and the time required for one complete drum revolution was 1.75 min. A multiplication factor of four times was used in the cake form rate calculations. The results for the drum vacuum filtration series are presented in Table 30. Details are available in Exhibit 12.

The best result, in terms of form rate, was with the K sample, which achieved 102 lb of dry solids/ft^2-hr. This sample was run with the smallest amount of flocculant, 0.10 lb/ton, but also had the largest amount of solids contained in the filtrate[1].

The filtrate samples ranged from 0.01% to 8% solids. The variation was attributed to variation of flocculation and/or a too-coarse filter media. The filter media was National Filter Media No. S/15405899, which is in the screen category. This filter media was selected, because it is used in Plant C. Plant C is using a vacuum disc filter producing a cake of 60.8% solids using 0.49 lb of Nalco 8852/8873 per ton of solids. Data developed on a drum vacuum filter at the Spokane Mining Research Center on Plant C refuse resulted in a 68% solids cake using 0.30 lb of Polyhall 990 flocculant per ton of dry solids treated. The filtrate contained 2.10% solids, and the cake form rate was 64 lb (dry)/ft^2-hr. No comparable plant rate and filtrate data were available.

Proper flocculation of the minus 2-µm particles was probably the most important factor affecting effluent clarity and filter capacity. Plants B and K were similar in fine particle content at 28% and 29.5% minus 2 µm. The sample from Plant B was flocculated with 0.61 lb/ton Polyhall 990 and 102 lb/ton lime, resulting in a relatively clean filtrate of 1.30% solids and a 73-lb/ft^2-hr form rate. The Plant K sample used only 0.1 lb/ton Polyhall 990, giving a filtrate of 8% solids and a 102-lb/ft^2-hr form rate.

Another example of the effect of flocculation was the testing of Plants J, K, and L samples. The tests on these three samples used essentially the same amounts of flocculant. The amount of minus 2-µm material in samples from Plants J, K, and L was 2.3%, 29.5%, and 23%, respectively. The solids in the filtrate were in the same relationship as the minus 2-µm material, i.e., J, 0.01%; K, 8%; and L, 7.4%.

The cake form rate on the sample from Plant J was exceptionally low. It was assumed that there may have been overflocculation as evidenced by the fact that tests on samples from Plant J, K, and L used the same amount of flocculant, but the sample from Plant J contained considerably less -2 µm material. An alternate explanation for the low form rate would be poor flocculation because of the high coal content of the refuse. The Plant J sample had the lowest ash and highest coal content of the five bulk samples.

A common application of vacuum filtration for fine coal refuse dewatering has been in areas that lack land for impoundment. Selection of the proper

[1] This is quite common that near-optimum flocculation of a pulp will leave a much more porous network of pulp from which the water can be removed at a higher rate; however, this occurs at the loss of filtrate clarity.

TABLE 30. - Results of dry vacuum filtration tests

CSMRI Sample No.	Preparation Plant	Feed Solids %	Average Vacuum psig	Operating Time hr	Wet Cake Recovered lb	Cake Solids[1] %	Filtrate Rate gph	Filtrate Solids %	Slurry Conditioning: Flocculant lb/hr	Slurry Conditioning: Flocculant Type[3]	Slurry Conditioning: CaO lb/ton	Cake Form Rate[2] dry lb/ft^2-hr
13	B	24.0	10.5	2.0	341	52.6	24.9	1.30	0.61	Polyhall 990	102	73
14	C	19.1	10.0	2.5	242	68.0	45.6	2.10	0.30	Polyhall 990	--	64
15	J	30.7	12.5	4.0	93	71.1	5.1	0.01	0.11	Polyhall 990	--	18
16	K	40.8	13.0	2.5	462	63.0	12.5	8.00	0.10	Polyhall 990	--	102
17	L	49.7	12.0	3.0	295	73.7	7.8	7.40	0.11	Polyhall 990	13	95

[1] Average moisture for the five best cake samples taken during operation.

[2] Determined by using the flollowing calculation.

$$\frac{\text{Wet Cake Recovered, lb} \times \text{Cake Solids, \%} \times 4 \text{ (Drum Filter Form Factor)}}{4 \text{ ft}^2 \text{ (total drum filter area)} \times \text{Operating Time, hr}}$$

[3] Celanese Polyhall 990.

dewatering device must consider a number of operating parameters: (1) solids tonnage-average and range, (2) feed solids concentration, (3) ash content of solids, (4) clay content of solids, (5) particle size distribution, particularly the subsieve range, (6) flocculation characteristics of the slurry, and (7) handling and final disposal system for the dewatered refuse. These items hold true for all given modes of coal refuse dewatering, not just vacuum filtration. As far as vacuum filtration is concerned, a number of parameters should be investigated and correctly designed for each individual coal refuse slurry. Among these items are (1) type of unit, e.g., drum, disc, roller discharge drum, or horizontal belt; (2) filter media; (3) optimum flocculant system (including flocculant type(s), concentration at addition time, method of addition); and (4) instrumentation/operator experience. The last item is actually two items; however, no amount of instrumentation is going to substitute for intelligent decisions regarding the operation of the unit. Too often, an operator will set the variable speed drive on maximum to avoid overflowing the feed well, which causes the filter to run with a partly full tank. The result is low vacuum, poor cake discharge, filter media blinding, greater recycle of filtrate solids, and excessive wear on moving parts.

Vacuum filtration is frequently the lowest capital and operating cost system with a sacrifice in cake moisture content compared to a belt press or a pressure filter. The feasibility of using a vacuum filter will depend on the character of the mixed filter cake and coarse refuse.

Pressure Filter

A laboratory-scale pressure filter was obtained from Passavant Corporation for dewatering studies on the five bulk samples of coal refuse thickener underflow. Refuse from Preparation Plants B, C, J, K, and L (CSMRI Samples 13 through 17) were tested on the Passavant Model 600 Pilot Pressure Filter System by CSMRI personnel at the Spokane Mining Research Center. Each test batch was conducted with 4,600 ml of flocculated refuse underflow, which was placed in a feed tank. Appropriate high-pressure hoses connected a nitrogen cylinder to the feed tank, which was, in turn, connected to the 6-in.-ID filter unit. The effective cross-sectional filtration area was 0.33 ft^2, and cake thicknesses of 1.57 in. were formed.

The test cycle started with application of 225 psig N_2 to the feed tank. A valve allowing pressurized pulp to enter the filter unit was opened, and timed samples of filtrate were then taken and recorded. The sampling continued until the flow rate flattened out, that is, when the current flow rate was essentially the same as the previous flow rate. The test unit was then disassembled, and the cake was collected. Wet and dry cake weights were determined, and all data were sent to Passavant Corporation for equipment sizing.

All test data and details are included in Exhibit 13. Selected test data are presented in Table 31 with the exception that no Passavant-supplied sizing information on Preparation Plant B (Sample 13) was made available. The test on Plant B refuse did not result in formation of a solid cake; the material was a thick slurry. Supposedly, the data generated were not suitable for filter sizing due to underconditioning of the slurry at the time that the test was conducted.

TABLE 31. - Results and equipment sizing by Passavant Corporation

Data	Preparation Plant C	J	K	L
CSMRI Sample No.	14	15	16	17
Anticipated Cake Solids, %	73.3	77.0	76.0	75.5
Filter Cake Weight, tpd	3,274	3,117	3,158	3,307
Filter Cake Volume, ft^3/day	66,146	59,369	50,934	55,121
Conditioning Agents				
Lime as 100% $Ca(OH)_2$, tpd	--	--	--	96
Polymer, lb/day:	--	--	--	1,920
Sizing				
Cake Thickness, in.	1.57	1.57	1.57	1.57
Filter Volume per Chamber, ft^3	5.086	5.086	5.086	5.086
Number of Cambers Required	544	204	420	605
Cake Volume, ft^3/day	66,146	59,369	50,934	55,121
Total Dry Solids, tpd	2,400	2,400	2,400	2,400

The anticipated cake solids reported here are those that were developed using the laboratory Model 600 unit. As can be seen, three out of five of the refuse streams could be dewatered to a relatively high cake solids, on the average 75%, without the use of additional flocculating agents. Preparation Plant K that was sampled and that had a filter press in operation produced a cake of 76% solids. All of the refuse streams used as feed had been flocculated in the thickener dewatering step before being treated in the pressure filter. Sample 13 from Preparation Plant B gave a cake of 38% solids using 107 lb of lime per ton of dry refuse solids. Although Passavant did not choose to use these data to develop design information, the possibility exists that, with additional test work and conditioning agents, this sample could give acceptable results as well.

As reported in the January 1980 issue of Coal Age, the Martin County Coal Corporation near Inez, Kentucky, has purchased and installed Passavant Model 20 filter presses for the fine coal refuse dewatering. The plant itself handles 1,200 tph of raw feed with 40% to 45% rejects so that a considerable amount of fine material must be discarded. The pressure filter consists of a stack of 2 x 2 meter plates covered with a tightly woven synthetic monofilament fabric that when clamped together forms a unit containing 150 chambers. There is a 6-in.-diam center line slurry feed hole through the entire stack and smaller holes located at each of the four corners for filtrate removal. Initially, sludge is pumped into the press by a fast-fill centrifugal pump to a pressure of approximately 160 psi. At this point, Passavant 3800 series hydraulically actuated flow-control ram pumps finish the cake buildup and compact the cake to a terminal pressure of 225 psi. The resultant cakes, two 1.5 in. thick each to a plate, weigh approximately 450 lb. At Martin County, each press goes through two cycles per hour with a maximum of 65 cycles in a day having been attained so far. For our four samples, the number of cycles required per day varies from 18 to 57. The Martin County presses produce a filter cake averaging 22% moisture as compared to the laboratory results achieved here that varied from 27% to 23% moisture. No information on filtrate clarity was available, although a "clear" filtrate was reportedly produced at the Martin County facility.

In general then, pressure filters, also called plate and frame presses, have found application in Great Britain where limited land area for refuse disposal and government requirements to operate with closed water circuits have made this method of handling fine size refuse a viable alternative. The equipment produces a cake of approximately 75% solids. This is removed via a moving belt and subsequently is mixed with coarse refuse and stockpiled. Major objections to the presses are that they (1) have an intermittent time cycle, (2) are rather costly to operate, and (3) require a high capital cost.

Thermal Dryer

An indirect-type transport dryer using circulating hot oil was rented from EBSP-Envirotech for pilot-plant drying studies. The unit was set up and was run at the Spokane Mining Research Center by Mr. Norbert Stevens of Envirotech and representatives of CSMRI. Feed for the five test runs consisted of partially dewatered refuse from the centrifuge pilot-plant studies detailed in Exhibit 16.

Solids were fed to the dryer using an inclined belt that had been calibrated by setting the belt speed and weighing the drop-off. The material was leveled by a movable bar, and the feed rate was adjusted for each sample. Product samples were taken on a continuous basis to ensure uniformity. A plastic container was fitted on the dryer discharge nozzle and 10-min timed samples were taken.

Data sheets representing the entire pilot run are available in Exhibit 17. Selected data and results are presented in Table 32.

TABLE 32. - Selected data and results for entire pilot run

	Preparation Plant				
	B	C	J	K	L
CSMRI Sample No.	13	14	15	16	17
Feed Rate, lb (wet)/hr	134	152	150	123	132
Feed Moisture, %	43.8	31.1	30.0	38.0	24.2
Product Rate, lb/hr	84	88	108	54	84
Product Moisture, %	7.2	7.0	4.1	1.3	2.2
Oil Temperature - In/Out, °F	388/374	390/384	398/388	395/384	395/384
Thermal Duty, Btu/hr	62,904	51,962	50,978	53,692	37,730
Transfer Coefficient, But/hr-ft^2-°F	18.39	14.48	13.90	15.11	10.34
Water Evaporated, Btu/lb[1]	1,195	1,264	1,256	1,166	1,254

[1] Thermal Duty ÷ Water Evaporated.

Note: Where thermal duty is the amount of heat transferred per unit time and the transfer coefficient is the heat transferred per unit time, per unit area, and per °F of driving force.

No difficulties were experienced with the pilot operation of the thermal dryer. Average feed cake moisture from the five centrifuge pilot runs was 33.2%. The average cake moisture for the thermal dryer products was 5.1%. This represents an 84.6% reduction of moisture for the two averaged values given. On an individual basis, the dewatered centrifuge product from Preparation Plant K showed the largest percent decrease in moisture at 97%, and Preparation Plant C, the smallest at 77.5%.

With today's ever increasing fuel costs, any thermal process employed will have operating costs increased cocurrently.

Ultimately, drying coal refuse down to 5% moisture may not be necessary or desirable due to dusting problems and to the dust control measures that would be required. Proper dewatering operations and/or partial thermal drying may result in acceptable moisture levels, product stability, and dust control for ultimate refuse disposal.

Summary of Pilot-Plant Studies

The data developed during the pilot-plant runs have been difficult to evaluate due to the number of variables involved. Sample results have been selected from the respective series on each unit operation and are presented in Table 33. For the purposes of direct comparison, the results of the centrifuge and vacuum filter are the most useful for interpretive purposes as they give quantitative information on flocculant consumption, cake moisture, and filtrate clarity.

Cake moisture contents cannot be evaluated in isolation, because there is always an interplay between cake moisture and percent solids in the filtrate. The relationship is well illustrated in the vacuum filter test work in which the lowest cake moisture was achieved for Preparation Plant L, 26%, but the filtrate contained 7.4% solids. An analysis of the significance of cake moisture was also rendered extremely difficult due to the varying behavior of pulp flocculation and dosage.

The lowest overall cake moisture contents were achieved with the use of the filter press. The order of increasing mean cake moisture value is as follows:

1. Filter Press (mean of four results without Preparation Plant B)
2. Belt Press (Andritz and Parkson)
3. Vacuum Filter
4. Solid Bowl Centrifuge

An evaluation of the results on a purely technical basis reveals that the best results, in terms of low cake moisture contents, at filtrate solids levels below 1%, were achieved in the filter press, followed by the belt press (assuming acceptable filtrate solids contents), solid bowl centrifuge, and vacuum filter. The performance of the solid bowl centrifuge was considered to be superior to that of the vacuum filter due to the lower filtrate solids contents achieved with the former unit.

In Table 34, the rate or capacity information for various types of equipment is compared. The presentation is made in increasing order of capacity.

There was no truly consistent order of dewatering as related to feed material. The material from Plant B was probably the most difficult to dewater.

Table 35 presents a ranking by dewatering rate and by final cake percent solids content.

The numerical ranking shows that B was the most difficult material to dewater followed by C and that the remaining three were essentially the same.

The summary of the characterization study suggests that Plant B material would be the most difficult to dewater followed by K, L, J, and C. The rankings by characterization and by dewatering tests are not the same, and the difference is attributed to differences in flocculation. Flocculation should have been a part of characterization, but there was no method to quantify flocculation.

TABLE 33. - Summary of pilot plant studies

Equipment	Preparation Plant	Slurry Specific Gravity g/cm^3	Feed Data Ash %	Feed Data Solids %	Flocculant Consumption lb/ton	Cake Product Solids %	Solids Recovery %	Centrate Solids %	Cake Product Moisture %
Solid Bowl Centrifuge	B	1.13	63.4	22.7	30 - CaO	--	--	--	--
					1.75 - Nalco 8873	57.5	99.1	0.45	42.5
	C	~1.10	34.0	17.1	0.41 - Nalco 8873	67.3	100.0	0.10	32.7
	J	~1.20	25.7	29.5	0.59 - Nalco 8873	68.4	99.8	0.10	31.6
	K	1.22	49.2	37.4	0.62 - Nalco 8873	59.5	99.9	0.12	40.5
	L	~1.20	48.1	50.2	0.65 - Nalco 8873	68.6	99.7	0.50	31.4
Belt Press (Arus-Andritz)	B	1.14	63.4	24.0	30 - CaO				
					0.6 - Hercules 1018	61.2	96+	4	38.8
	C	1.07	34.0	19.1	0.25 - Hercules 847	70.9	96+	4	29.1
	J	1.12	25.7	30.7	0.55 - Hercules 1018	72.7	96+	4	27.3
	K	1.23	49.2	40.8	0.50 - Hercules 847	72.8	96+	4	27.9
	K	1.23	49.2	40.8	0.75 - Hercules 1018	73.7	96+	4	26.3
	L	1.27	48.1	49.7	1.00 - Percol 757	72.0	96+	4	28.0
Belt Press (Parkson)	B	1.131	63.4	21.43	30 - CaO				
					2.5 - Percol 730	60[1]	98	2	40[1]
	C	1.061	34.0	16.21	1.2 - Superfloc 208	70[1]	98	2	30[1]
	J	1.109	25.7	28.76	0.3 - Superfloc 208	70[1]	98	2	30[1]
	K	1.238	49.2	37.29	0.9 - Superfloc 208	65[1]	98	2	35[1]
	L	1.230	48.1	48.00	0.5 - Percol 725	79[1]	98	1	21[1]
Vacuum Filter	B	1.14	63.4	24.0	120 - CaO	--	--	--	--
					0.61 - Polyhall 990	52.6	98.7	1.33	47.4
	C	1.07	34.0	19.1	0.30 - Polyhall 990	68.0	97.9	2.1	32.0
	J	1.12	25.8	30.7	0.11 - Polyhall 990	71.1	100.0	0.01	28.9
	K	1.23	49.2	40.8	0.10 - Polyhall 990	63.0	92.0	8.0	37.0
	L	1.27	48.1	49.7	13.0 - CaO	--	--	--	--
					0.11 - Polyhall 990	73.7	92.6	7.4	26.3
Pressure Filter (No results for Preparation Plant B)	C	1.07	34.0	19.1	--	73.3	99	1	26.7
	J	1.12	25.8	30.7	--	77.0	99	1	23.0
	K	1.23	49.2	40.8	--	76.0	99	1	24.0
	L	1.27	48.1	24.9[2]	81 - CaO	--	--	--	--
					0.8 - Percol 727	75.5	99	1	24.5
Thermal Disc	B	--	63.4	56.2	--	--	--	--	7.2
	C	--	34.0	68.9	--	--	--	--	7.0
	J	--	25.8	70.0	--	--	--	--	4.1
	K	--	49.2	62.0	--	--	--	--	1.3
	L	--	48.1	75.8	--	--	--	--	2.2

[1] MP -- high-pressure unit.
[2] Pulp diluted to obtain proper flocculation.

TABLE 34. - Summary of rate on capacity data

Equipment	Preparation Plant	Feed Solids %	Cake Product Moisture %	Approximate Capacity Full-Scale Centrifuge tph	Relative Ranking
Solid Bowl Centrifuge	B	22.7	42.5	15	Low
	J	29.5	31.6	25	Medium
	K	37.4	40.5	25	Medium
	C	17.1	37.4	30	Medium-High
	L	50.2	31.4	30-40	Medium-High
				Belt Width tph/meter	
Belt Press (Andritz)	C	19.1	29.1	5.4	Low
	B	24.0	38.8	6.0	Medium
	J	30.7	27.3	6.1	Medium
	K	40.8	29.4	7.3	Medium-High
	L	49.7	30.0	8.9	High
Belt Press (Parkson)	B	21.43	40.0	1.8	Low
	K	37.29	35.0	2.5	Medium
	C	16.21	30.0	2.9	Medium
	L	48.00	35.0	3.0	Medium
	J	28.76	30.0	3.5	High
				Cake Form Rate dry lb/ft^2-hr	
Vacuum Filter	J	30.7	28.9	18	Low
	C	19.1	32.0	64	Medium
	B	24.0	48.2	73	Medium
	L	49.7	26.0	95	High
	K	40.8	37.0	102	High
				Chamber Fill Rate dry lb/ft^2-hr	
Pressure Filter	B	24.0	62.0	3.82	Low
	L	24.9	24.5	6.61	Medium
	C	19.1	26.7	7.49	Medium
	K	40.8	24.0	9.75	Medium
	J	30.7	23.0	37.77	High

TABLE 35. - Ranking by dewatering rate and cake percent solids

Preparation Plant	B	C	J	K	L
Centrifuge, Rate	Low	Medium-High	Medium	Medium	Medium-High
% Solids	Low	Medium	Medium	High	Medium
Belt Press, Rate	Medium	Low	Medium	Medium-High	High
(Andritz) % Solids	Low	High	High	High	High
Belt Press, Rate	Low	Medium	High	Medium	Medium
(Parkson) % Solids	Low	High	High	Medium	Medium
Vacuum Filter, Rate	Medium	Medium	Low	High	High
% Solids	Low	Medium-High	High	Medium	High
Pressure Filter, Rate	Low	Medium	High	Medium	Medium
% Solids	Low	High	High	High	High
Numerical Ranking[1]					
Total	12	18	25	24.5	25.5
Average	1.2	1.8	2.5	2.45	2.55

[1] Low = 1, Medium = 2, Medium-High = 2.5, High = 3.

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2. Noone, W. H. Eliminating Stream Pollution from a Coal Preparation Plant. Mining Congress Journal, v. 49, No. 8., 1963, pp. 26-30.

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APPENDIX

EXHIBIT 1

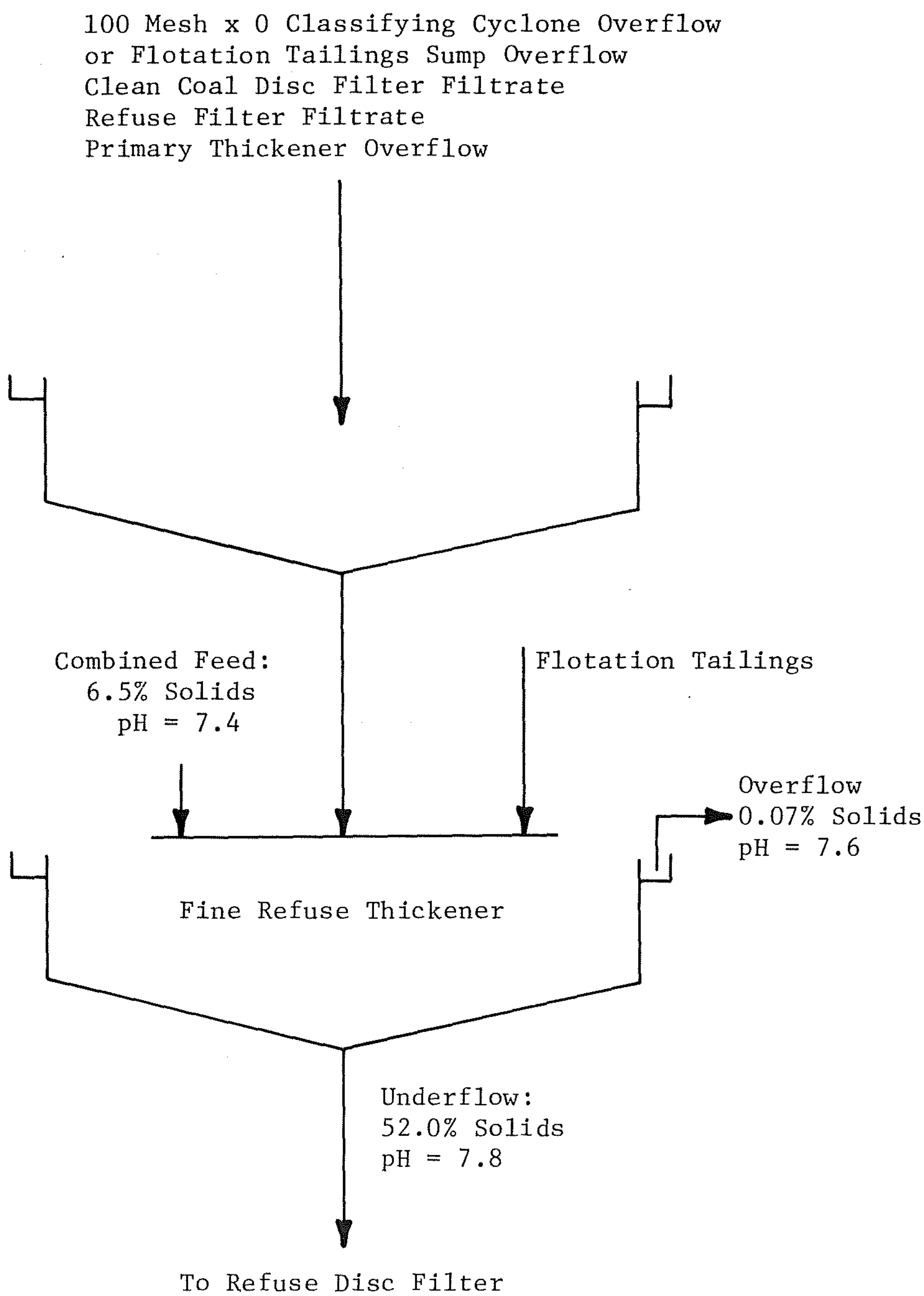
SAMPLE AND PREPARATION PLANT DESCRIPTION

CSMRI Sample 1

Sample Origin:	Preparation Plant A.
Date of Sampling:	December 18, 1979.
Samples Taken and Amount:	Refuse thickener feed - 5 gal. Refuse thickener underflow - 5 gal. Refuse thickener overflow - 5 gal.
Sample Description:	Refuse thickener feed - •100 mesh x 0 classifying cyclone overflow or flotation tailings sump overflow. •Clean coal disc filter filtrate. •Refuse filter filtrate. •Primary thickener overflow. Refuse thickener underflow - •Flocculated and thickened feed. Refuse thickener overflow - •Clarified water for plant reuse.
Preparation Plant Description:	Product is for the metallurgical market. Cleaning equipment - •Heavy media bath for coarse size coal. •Deister concentrating table for intermediate size coal. •Froth flotation for fine size coal. Thickening and dewatering equipment - •Static thickener. •Refuse disc vacuum filter. •Flocculant: Percol 158 and acid.

EXHIBIT 1

CSMRI Sample 1



Preparation Plant A: Origin of thickener underflows. Solids concentration and pH values of feed, overflow, and underflow of Fine Refuse Thickener.

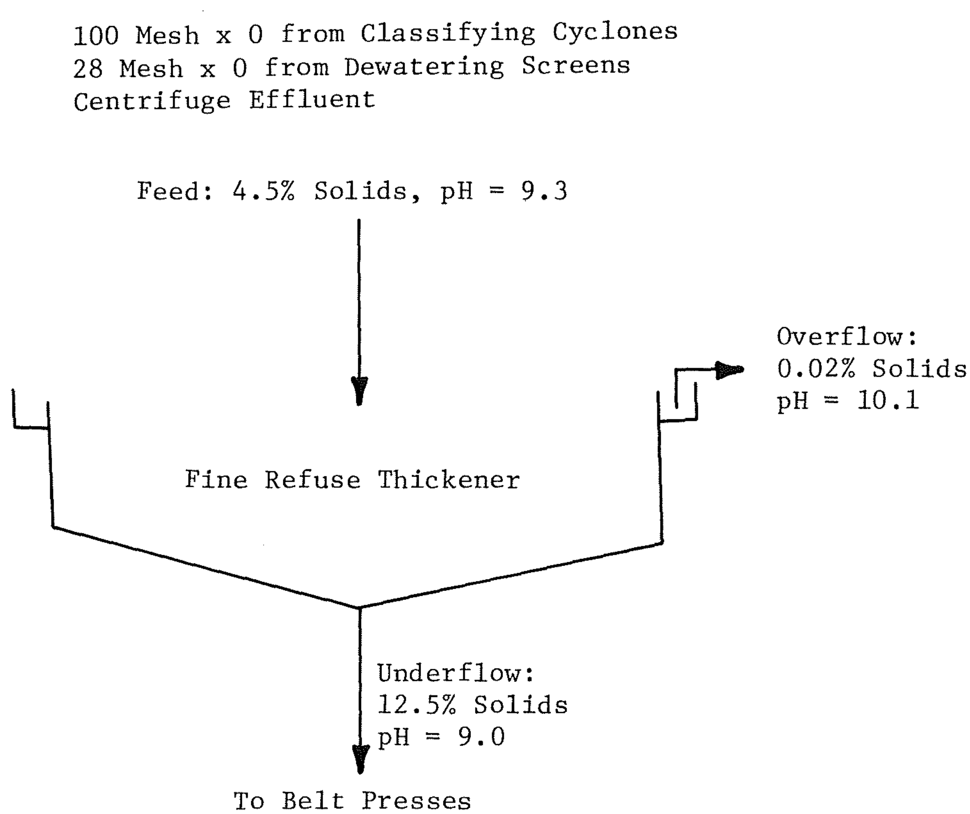
EXHIBIT 1

CSMRI Sample 2

Sample Origin:	Preparation Plant B.
Date of Sampling:	December 20, 1979.
Samples Taken and Amount:	Refuse thickener feed - 5 gal. Refuse thickener underflow - 5 gal. Refuse thickener overflow - 5 gal.
Sample Description:	Refuse thickener feed - •100 mesh x 0 classifying cyclone overflow. •28 mesh x 0 dewatering screens underflow. •Centrifuge effluent. Refuse thickener underflow - •Flocculated and thickened feed. Refuse thickener overflow - •Clarified water for plant reuse.
Preparation Plant Description:	Product is for electric power generation. Cleaning equipment - •Baum jig treating full size range. Thickening and dewatering equipment - •Static thickener. •Belt press, Andritz. •Flocculant: American Cyanamid 1202 and lime.

EXHIBIT 1

CSMRI Sample 2



Preparation Plant B: Origin of thickener underflows. Solids concentration and pH values of feed, overflow, and underflow of fine refuse thickener.

EXHIBIT 1

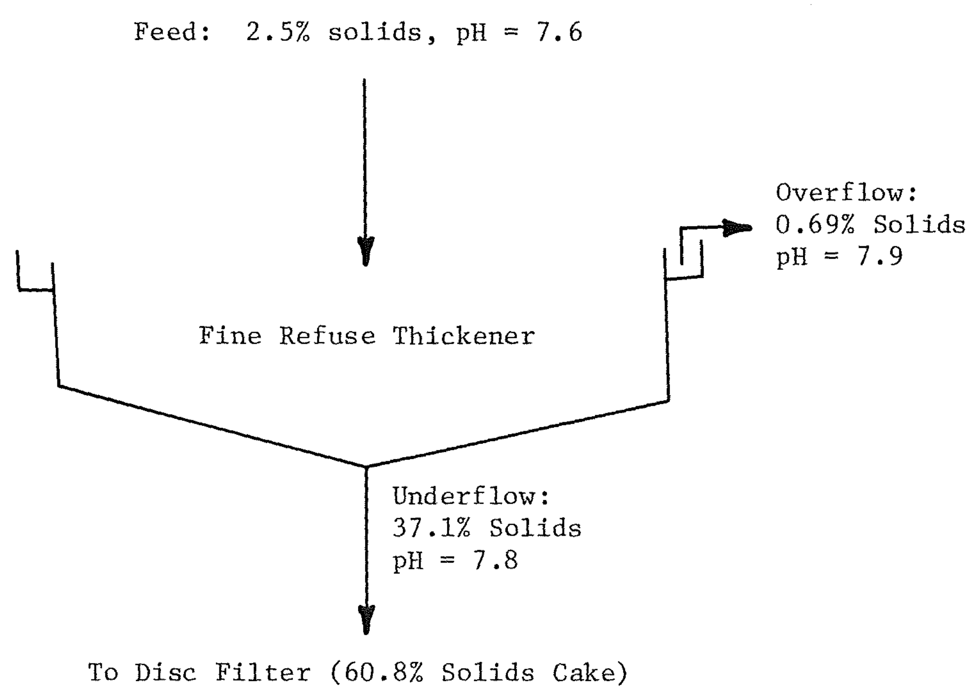
CSMRI Sample 3

Sample Origin:	Preparation Plant C.
Date of Sampling:	Approximately January 9, 1980.
Samples Taken and Amount:	Refuse thickener feed - 10 gal. Refuse thickener underflow - 10 gal. Refuse thickener overflow - 5 gal. Refuse disc vacuum filter cake - ~10 lb.
Sample Description:	Refuse thickener feed - •28 mesh x 0 refuse screen underflow. •Water head tank by-pass. •Screen bowl centrifuge effluent. •Disc filter filtrate. Refuse thickener underflow - •Flocculated and thickened feed. Refuse thickener overflow - •Clarified water for plant reuse.
Preparation Plant Description:	Product is for electric power generation. Cleaning equipment - •Baum jig treating full size range. Thickening and dewatering equipment - •Static thickener. •Refuse disc vacuum filter. •Flocculant: Nalco 8852 and 8873.

EXHIBIT 1

CSMRI Sample 3

28 Mesh x 0 from Refuse Dewatering Screens after Jig
Water Head Tank By-Pass
Screen Bow Effluent
Disc Filter



Preparation Plant C: Origin of thickener underflow. Solids concentration and pH value of feed, overflow, and underflow of fine refuse thickener; solids concentration of refuse cake feed to disc filter.

EXHIBIT 1

CSMRI Sample 4

Sample Origin: Preparation Plant D.

Date of Sampling: January 15, 1980.

Samples Taken and Amount:
Refuse thickener feed - 5 gal.
Refuse thickener underflow - 5 gal.
Refuse thickener overflow - 5 gal.

Sample Description:
Refuse thickener feed -
- 28 mesh x 0 desilter overflow.
- Dirty water head tank.
- 100 mesh x 0 classifying cyclones overflow.

Refuse thickener underflow -
- Flocculated and thickened feed.

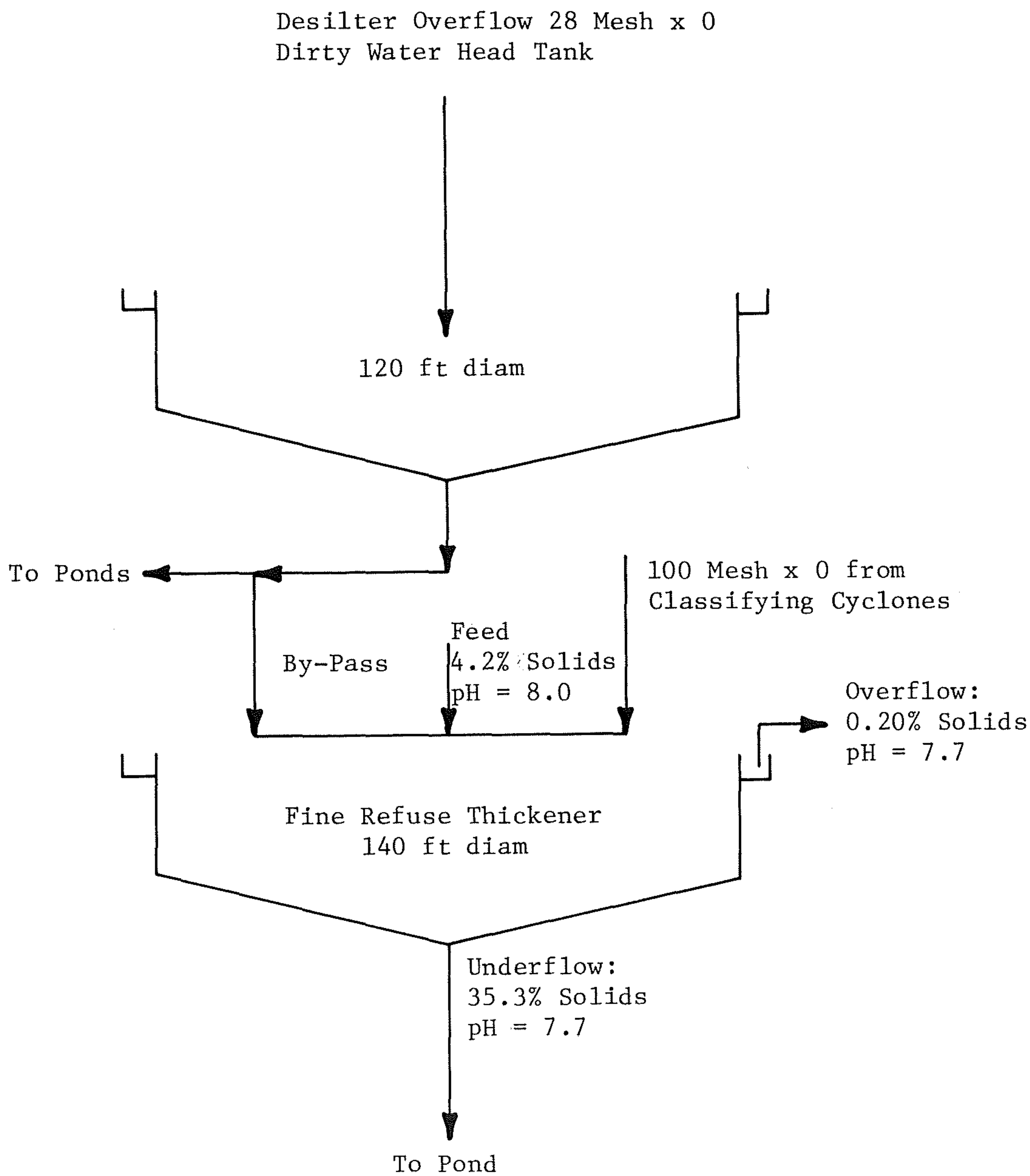
Refuse thickener overflow -
- Clarified water for plant reuse.

Preparation Plant Description:
Product is for the metallurgical market.
Cleaning equipment -
- Chance cone sand separator for coarse size coal.
- Deister concentrating table for intermediate size coal.
- Froth flotation for fine size coal.

Thickening and dewatering equipment -
- Static thickener.
- Pumping to pond rather than dewatering of fine refuse.
- Flocculant: American Cyanamid 550, Betz 3310, and aluminum sulfate.

EXHIBIT 1

CSMRI Sample 4



Preparation Plant D: Origin of thickener underflow. Solids concentration and pH values of feed, overflow, and underflow of fine refuse thickener.

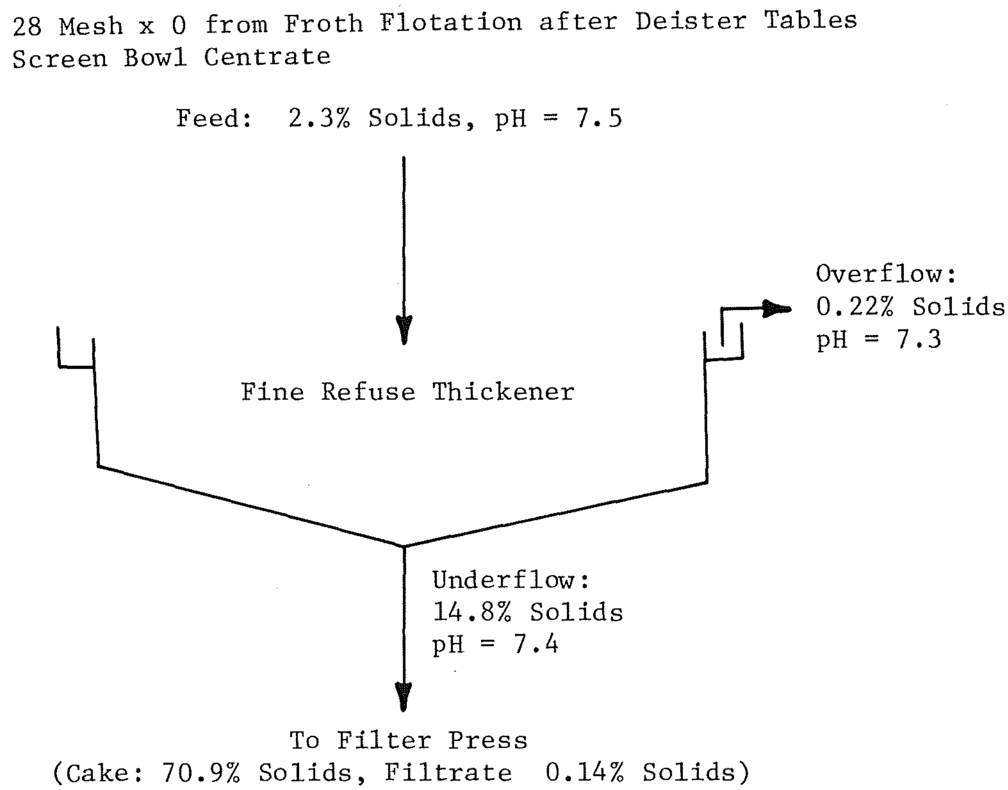
EXHIBIT 1

CSMRI Sample 5

Sample Origin:	Preparation Plant E.
Date of Sampling:	January 15, 1980.
Samples Taken and Amount:	Refuse thickener feed - 10 gal. Refuse thickener underflow - 10 gal. Refuse thickener overflow - 5 gal. Belt press cake - 3 lb. Belt press filtrate - 1 qt.
Sample Description:	Refuse thickener feed - ·28 mesh x 0 from froth flotation. ·Screen bowl centrifuge centrate. Refuse thickener underflow - ·Flocculated and thickened feed. Refuse thickener overflow - ·Clarified water for plant reuse.
Preparation Plant Description:	Product is for electric power generation. Cleaning equipment - ·Heavy media bath for coarse size material. ·Deister concentrating table for intermediate size coal. ·Froth flotation for fine size coal. Thickening and dewatering equipment - ·Static thickener. ·Refuse filter-press installation. ·Flocculant: American Cyanamid 1202 and 330.

EXHIBIT 1

CSMRI Sample 5



Preparation Plant E: Origin of thickener underflow. Solids concentration and pH value of feed, overflow and underflow of fine refuse thickener; and solids concentration of refuse filter press cake and filtrate.

EXHIBIT 1

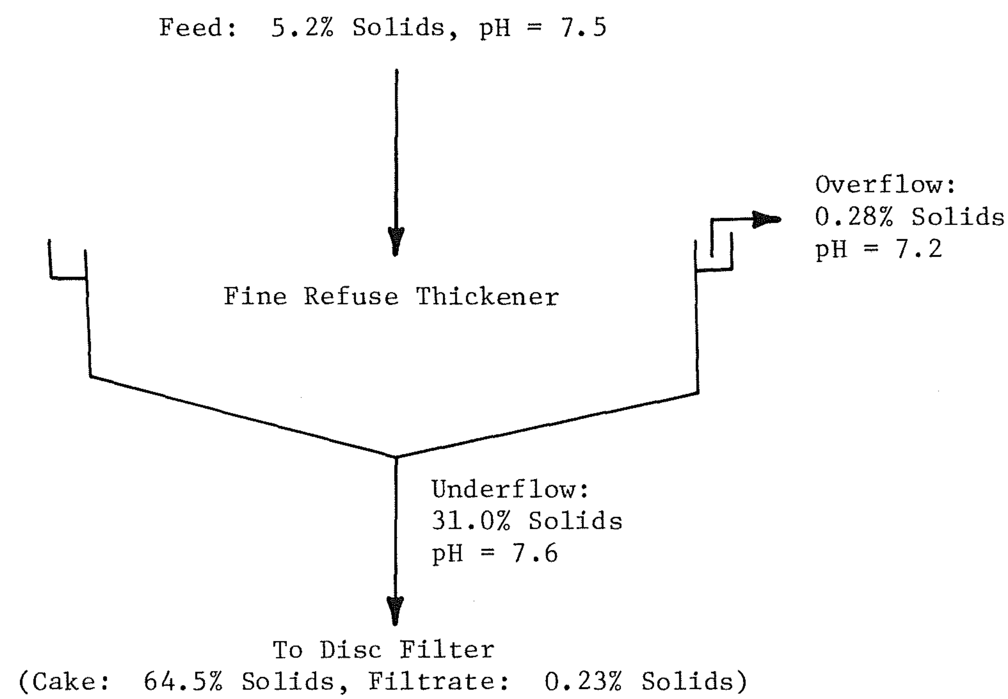
CSMRI Sample 6

Sample Origin:	Preparation Plant F.
Date of Sampling:	January 18, 1980.
Samples Taken and Amount:	Refuse thickener feed - 10 gal. Refuse thickener underflow - 10 gal. Refuse thickener overflow - 5 gal. Refuse disc vacuum filter cake - 3 lb. Refuse disc vacuum filter filtrate - 1 qt.
Sample Description:	Refuse thickener feed - •28 mesh x 0 refuse from dewatering screens. •Classifying cyclone overflow. •Disc filter filtrate. Refuse thickener underflow - •Flocculated and thickened feed. Refuse thickener overflow - •Clarified water for plant reuse.
Preparation Plant Description:	Product is for electric power generation. Cleaning equipment - •Baum jig cleaning entire size range. Thickening and dewatering equipment - •Static thickener. •Refuse disc vacuum filter. •Flocculant: Dowell M88 and M191.

EXHIBIT 1

CSMRI Sample 6

28 Mesh x 0 from Refuse Dewatering Screens after Jig
Classifying Cyclone Overflow
Disc Filter Filtrate



Preparation Plant F: Origin of thickener underflow. Solids concentration and pH value of feed, overflow, and underflow of fine refuse thickener; and solids concentration of refuse disc filter cake and filtrate.

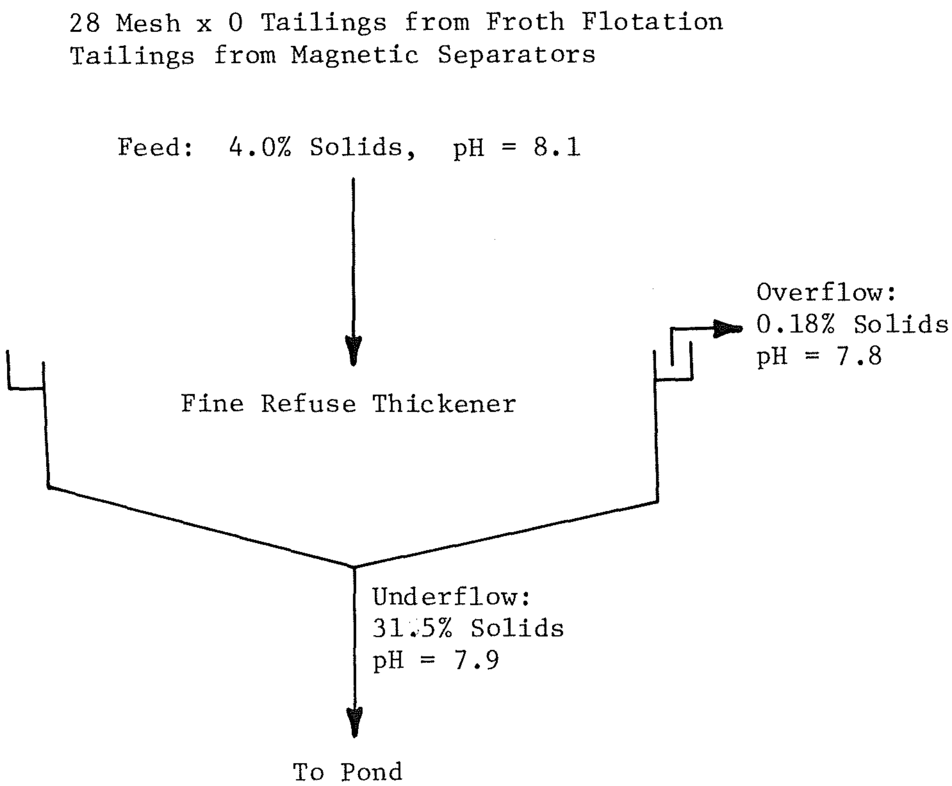
EXHIBIT 1

CSMRI Sample 7

Sample Origin:	Preparation Plant G.
Date of Sampling:	January 21, 1980.
Samples Taken and Amount:	Refuse thickener feed - 10 gal. Refuse thickener underflow - 10 gal. Refuse thickener overflow - 5 gal.
Sample Description:	Refuse thickener feed - •28 mesh x 0 tailings from froth flotation. •Tailings from magnetic separators (treating magnetite from heavy-media cleaning cyclones). Refuse thickener underflow - •Flocculated and thickened feed. Refuse thickener overflow - •Clarified water for plant reuse.
Preparation Plant Description:	Product is for the metallurgical market. Cleaning equipment - •Heavy media vessel treating coarse size material. •Heavy media cyclones treating intermediate size material. •Froth flotation treating fine size material. Thickening and dewatering equipment - •Static thickener. •Thickened underflow pumped to pond disposal. •Flocculant: Nalco 8852 and 8873.

EXHIBIT 1

CSMRI Sample 7



Preparation Plant G: Origin of thickener underflow. Solids concentration and pH value of feed, overflow, and underflow of fine refuse thickener.

EXHIBIT 1

CSMRI Sample 8

Sample Origin:	Preparation Plant H.
Date of Sampling:	January 22, 1980.
Samples Taken and Amount:	Refuse thickener feed - 10 gal. Refuse thickener underflow - 10 gal. Refuse thickener overflow - 5 gal.

Sample Description:

Refuse thickener feed -
- Secondary froth flotation tailings.
    - Feed to secondary froth flotation.
        - 100 mesh x 0 classifying cyclones overflow.
        - 28 mesh x 0 from Deister table, dryers, and sieve bend effluent.
        - 28 mesh x 0 middlings drag tank overflow.
        - Middlings and clean coal disc filter filtrate.
- Refuse drag tank overflow.

Refuse thickener underflow -
- Flocculated and thickened feed.

Refuse thickener overflow -
- Clarified water for plant reuse.

Preparation Plant Description:

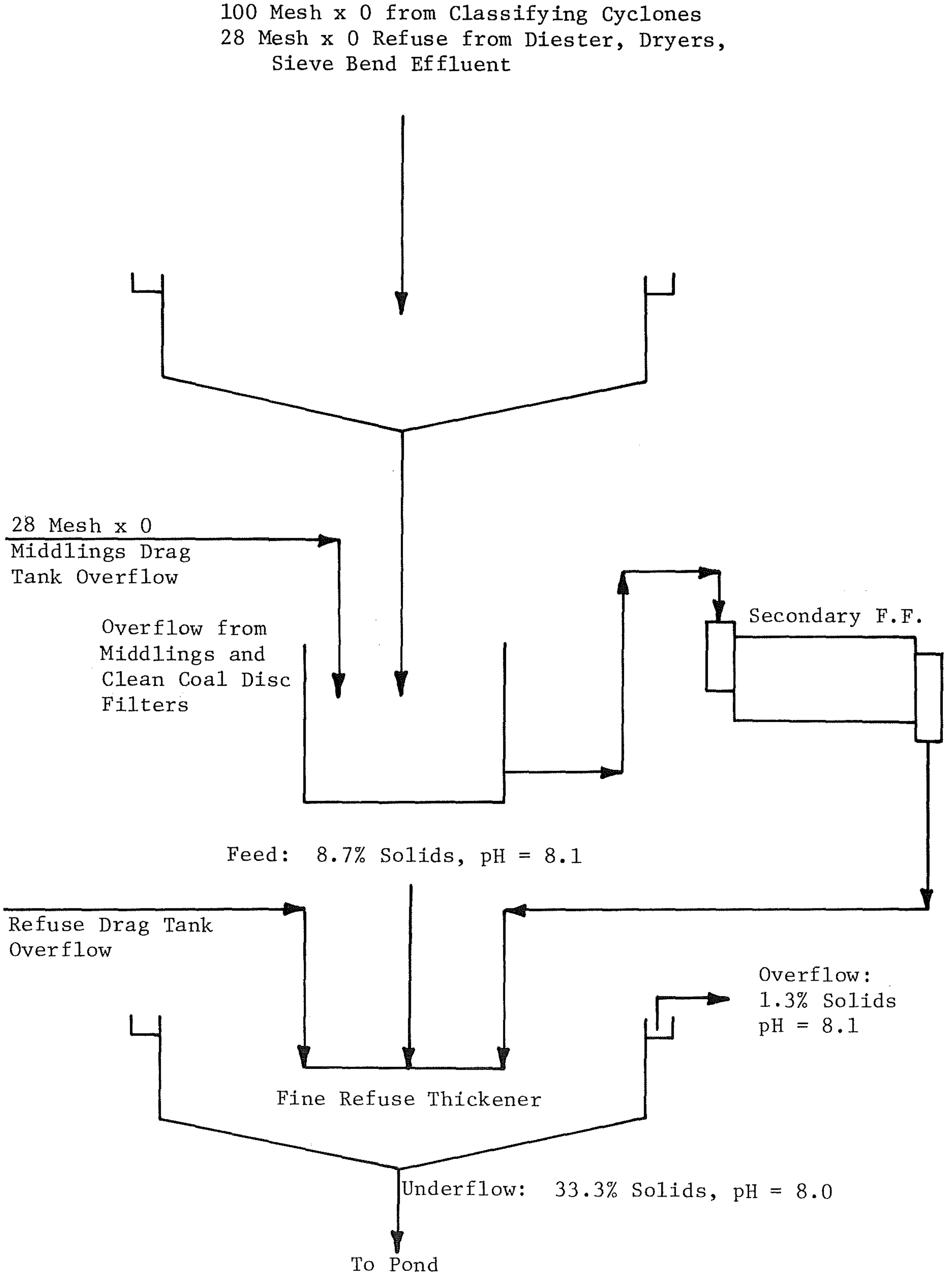
Product is for both the metallurgical and electric power generation market.

Cleaning equipment -
- Heavy media vessel treating coarse size material.
- Deister concentrating table treating intermediate size material.
- Froth flotation treating fine size material.

Thickening and dewatering equipment -
- Static thickener.
- Thickener underflow pumped to pond disposal.
- Flocculant: American Cyanamid 1202 and 315.

EXHIBIT 1

CSMRI Sample 8



Preparation Plant H: Origin of thickener underflows. Solids concentration and pH value of feed, overflow, and underflow of fine refuse thickener.

EXHIBIT 1

CSMRI Sample 9

Sample Origin: Preparation Plant I.

Date of Sampling: January 24, 1980.

Samples Taken and Amount:
Refuse thickener feed - 10 gal.
Refuse thickener underflow - 10 gal.
Refuse thickener overflow - 5 gal.

Sample Description:
Refuse thickener feed -
- 28 mesh x 0 froth flotation tailings.
- 28 mesh x 0 Deister concentrating table refuse from screen undersize.

Refuse thickener underflow -
- Flocculated and thickened feed.

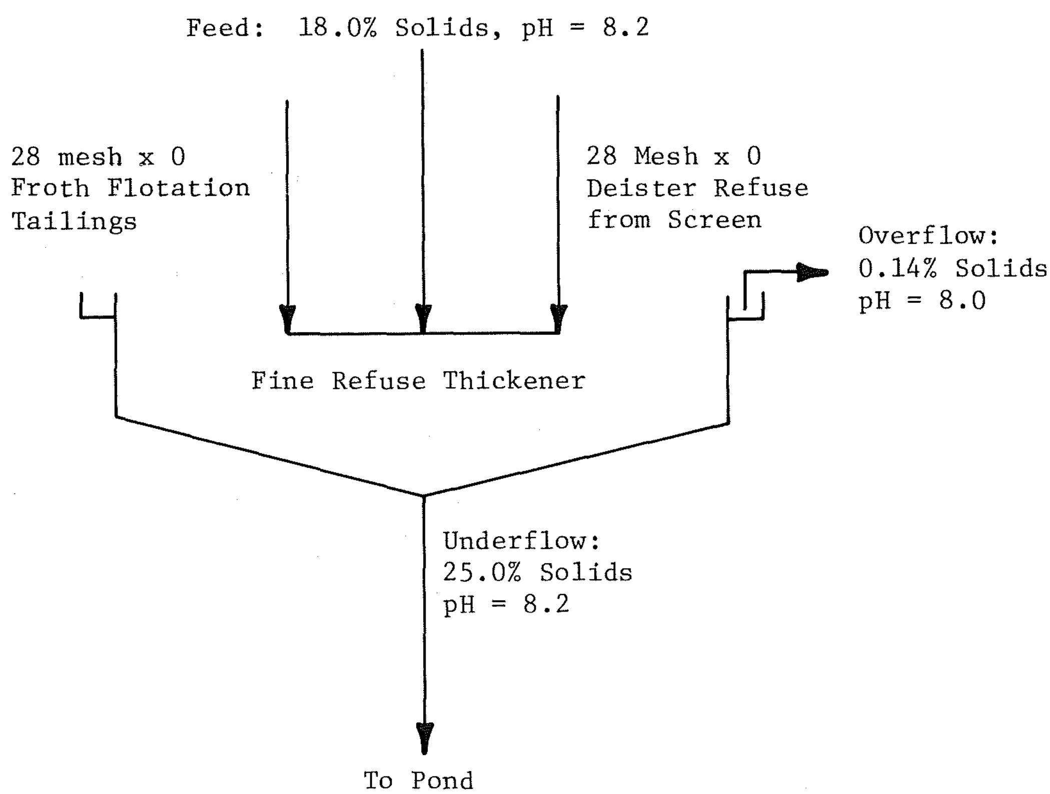
Refuse thickener overflow -
- Clarified water for plant reuse.

Preparation Plant Description:
Product is for electric power generation.
Cleaning equipment -
- Heavy media vessel treating coarse size material.
- Deister concentrating table treating intermediate size material.
- Froth flotation treating fine size material.

Thickening and dewatering equipment -
- Static thickener.
- Thickener underflow pumped to pond disposal.
- Flocculant: Calgon 503 and 570.

EXHIBIT 1

CSMRI Sample 9



Preparation Plant I: Origin of Thickener underflow. Solids concentration and pH value of feed, overflow, and underflow of fine refuse thickener.

EXHIBIT 1

CSMRI Sample 10

Sample Origin: Preparation Plant J.

Date of Sampling: March 3, 1980.

Samples Taken and Amount:
Refuse thickener underflow - 5 gal.
Solid bowl centrifuge cake.

Sample Description:
Refuse thickener underflow -
- Flocculated and thickened 28 mesh x 0 froth flotation tailings.

Solid bowl centrifuge concentrate.

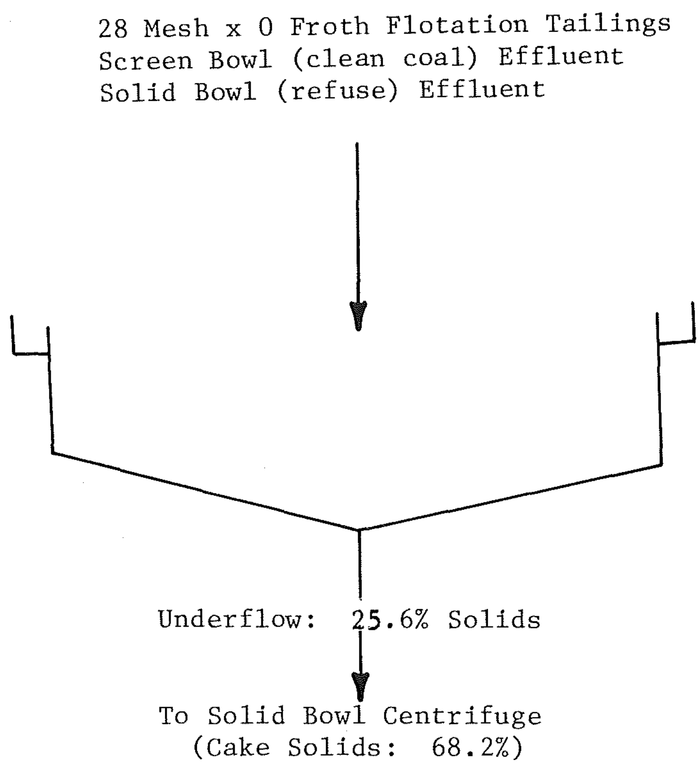
Preparation Plant Description:
Product is for electrical power generation.
Cleaning equipment -
- Heavy-media vessel treating coarse-size material.
- Deister concentrating table treating intermediate size material (¼ in. by 28 mesh).
- Froth flotation treating fine-size material.

Thickening and dewatering equipment -
- Static thickener (100 ft diam).
- Bird solid bowl centrifuge.
- Flocculants: American Cyanamid 214 and 208, Allied Colloids 351.

Note: Samples taken by the coal company.

EXHIBIT 1

Origin of Thickener Feed Solids
CSMRI Sample 10



Preparation Plant J: Origin of thickener underflow. Solids concentration and pH of underflow of fine refuse thickener.

EXHIBIT 1

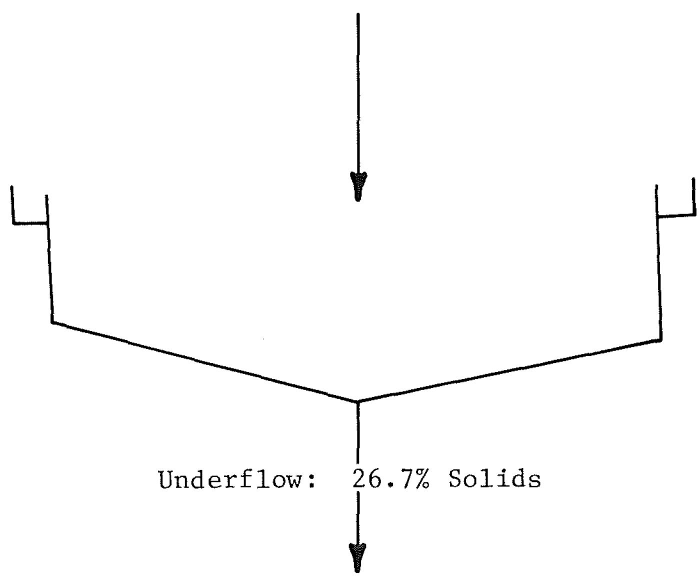
CSMRI Sample 11

Sample Origin:	Preparation Plant K.
Date of Sampling:	March 3, 1980.
Samples Taken and Amount:	Refuse thickener underflow - 5 gal. Filter press cake.
Sample Description:	Refuse thickener underflow - •Flocculated and thickened 28 mesh x 0 froth flotation tailing. Filter press cake.
Preparation Plant Description:	Product is for the steam market. Cleaning equipment - •Heavy-media vessel treating coarse-size material. •Deister concentrating table treating intermediate size material (¼ in. x 0). •Froth flotation treating fine-size coal. Thickening and dewatering equipment - •Static thickener. •Passavant filter press. •Flocculant: Dowell M191 and M88.
Note:	Samples taken by coal company.

EXHIBIT 1

Origin of Thickener Feed Solids
CSMRI Sample 11

Refuse Centrifugal Dryer Effluent
Deister Concentrating Table Refuse Spiral Classifier Effluent
Froth Flotation Tailings



To Filter Press
(Cake Solids: 76.0%)

Preparation Plant K: Origin of thickener underflow. Solids concentration and pH of underflow of fine refuse thickener.

EXHIBIT 1

CSMRI Sample 12

Preparation Plant L: No preliminary sample was received from Preparation Plant L.

EXHIBIT 1

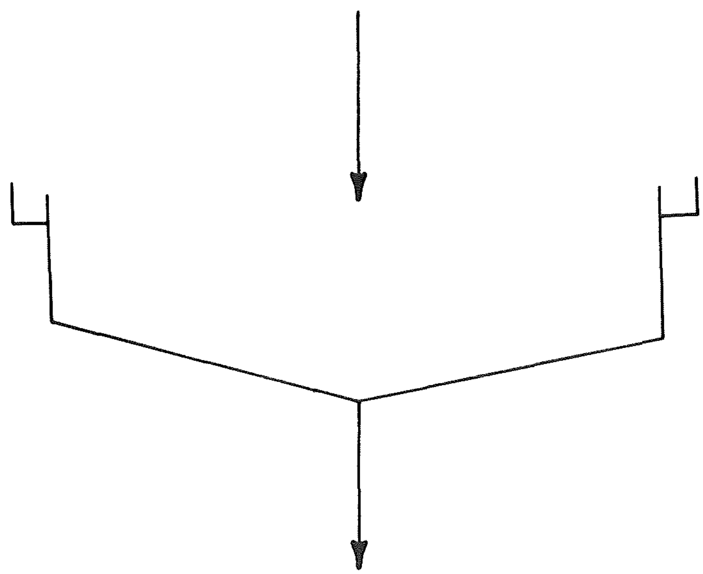
CSMRI Sample 13

Sample Origin:	Preparation Plant B.
Date of Sampling:	April 15, 1980.
Samples Taken and Amount:	Refuse thickener underflow - 65 55-gal drums.
Sample Description:	Refuse thickener feed - •100 mesh x 0 classifying cyclone overflow. •28 mesh x 0 dewatering screens underflow. •Centrifuge effluent. Refuse thickener underflow - •Flocculated and thickened feed.
Preparation Plant Description:	Product is for electric power generation. Cleaning equipment - •Baum jig treating full size range. Thickening and dewatering equipment - •Static thickener. •Belt press, Andritz. •Flocculant: American Cyanamid 1202 and lime.

EXHIBIT 1

CSMRI Sample 13

100 Mesh x 0 from Classifying Cyclones
28 Mesh x 0 from Dewatering Screens
Centrifuge Effluent



To Belt Presses

Thickener Underflow: 24.0% Solids, pH = 10.1
Pulp Density = 1.140 g/ml

EXHIBIT 1

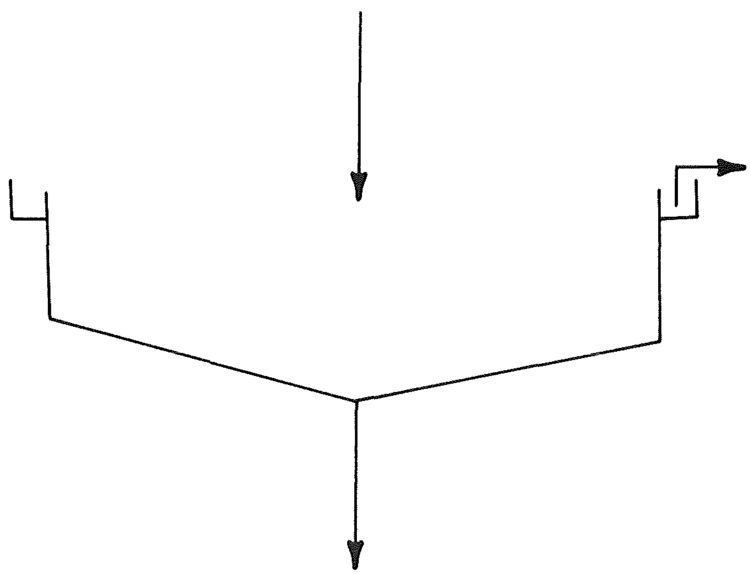
CSMRI Sample 14

Sample Origin:	Preparation Plant C.
Date of Sampling:	March 28, 1980.
Samples Taken and Amount:	Refuse thickener underflow - 60 55-gal drums.
Sample Description:	Refuse thickener feed - ·28 mesh x 0 refuse screen underflow. ·Water head tank bypass. ·Screen bowl centrifuge effluent. ·Disc filter filtrate. Refuse thickener underflow - ·Flocculated and thickened feed.
Preparation Plant Description:	Product is for electrical power generation. Cleaning equipment - ·Baum jig treating full size range. Thickening and dewatering equipment - ·Static thickener. ·Refuse disc vacuum filter. ·Flocculant: Nalco 8852 and 8873.

EXHIBIT 1

CSMRI Sample 14

28 Mesh x 0 from Refuse Dewatering Screens after Jig
Water Head Tank By-Pass
Screen Bowl Effluent
Disc Filter



To Disc Filter
Thickener Underflow: 19.1% Solids, pH 7.9
Pulp Density = 1.07 g/ml

EXHIBIT 1

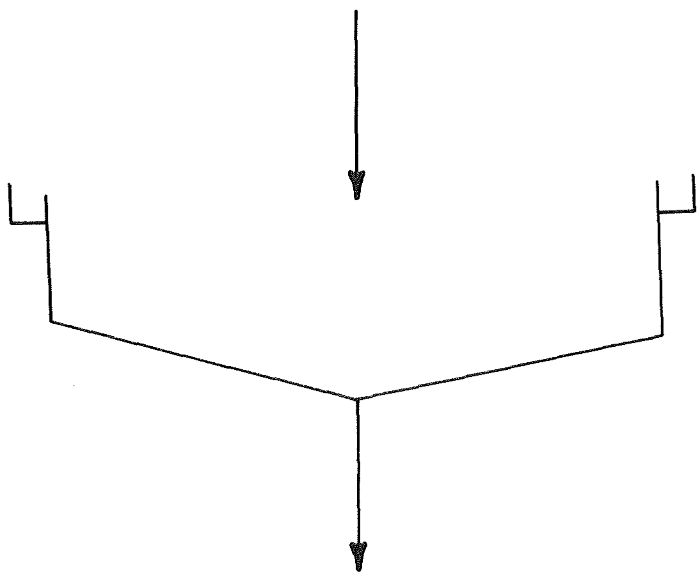
CSMRI Sample 15

Sample Origin:	Preparation Plant J.
Date of Sampling:	March 27, 1980.
Samples Taken and Amount:	Refuse thickener underflow - 55 55-gal drums.
Sample Description:	Refuse thickener underflow - •Flocculated and thickened 28 mesh x 0 froth flotation tailings.
Preparation Plant Description:	Product is for electrical power generation. Cleaning equipment - •Heavy-media vessel treating coarse-size material. •Deister concentrating table treating intermediate-size material (¼ in. by 28 mesh). •Froth flotation treating fine-size material. Thickening and dewatering equipment - •Static thickener (100 ft diam). •Bird solid bowl centrifuge. •Flocculant: American Cyanamid 214 and 208, Allied Colloids 351.
Note:	American Cyanamid 214 to thickener feed. American Cyanamid 208 to solid bowl centrifuge feed. Allied Colloids 351 to thickener feed (used the day of sampling rather than American Cyanamid 214). American Cyanamid 355 (cationic) is used also at times as a substitute for American Cyanamid 208 (anionic).

EXHIBIT 1

Origin of Thickener Feed Solids
CSMRI Sample 15

28 Mesh x 0 Froth Flotation Tailings
Screen Bowl (clean coal) Effluent
Solid Bowl (refuse) Effluent



To Solid Bowl Centrifuge
Thickener Underflow: 30.7% Solids, pH = 8.2,
Pulp Density = 1.115 g/ml

EXHIBIT 1

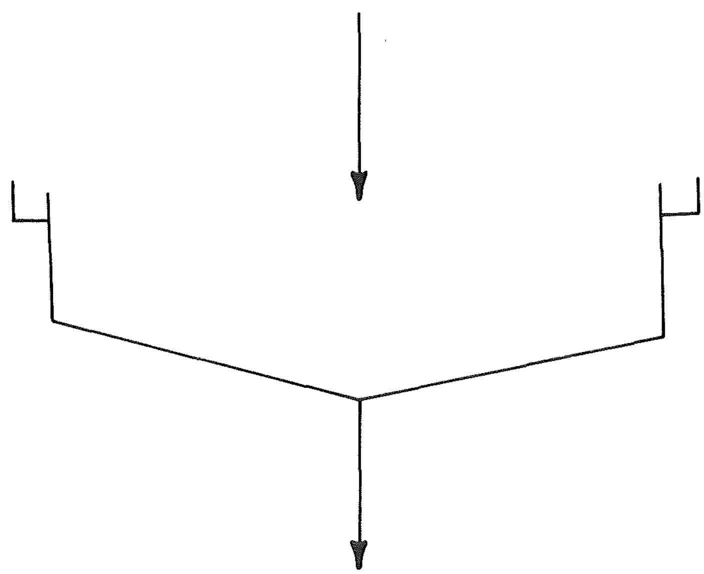
CSMRI Sample 16

Sample Origin:	Preparation Plant K.
Date of Sampling:	March 25, 1980.
Samples Taken and Amount:	Refuse thickener underflow - 45 55-gal drums.
Sample Description:	Refuse thickener underflow - •Flocculated and thickened 28 mesh x 0 froth flotation tailing.
Preparation Plant Description:	Product is for the steam market. Cleaning equipment - •Heavy-media vessel treating coarse-size material. •Deister concentrating table treating intermediate-size material (¼ in. by 0). •Froth flotation treating fine-size coal (inoperative at time of sampling). Thickening and dewatering equipment - •Static thickener. •Passavant filter press. •Flocculant: Dowell M191 and M88.

EXHIBIT 1

Origin of Thickener Feed Solids
CSMRI Sample 16

Refuse Centrifugal Dryer Effluent
Deister Concentrating Table Refuse Spiral Classifier Effluent
Froth Flotation Tailings



To Filter Press
Thickener Underflow: 40.8% Solids, pH = 6.3,
Pulp Density = 1.227 g/ml

EXHIBIT 1

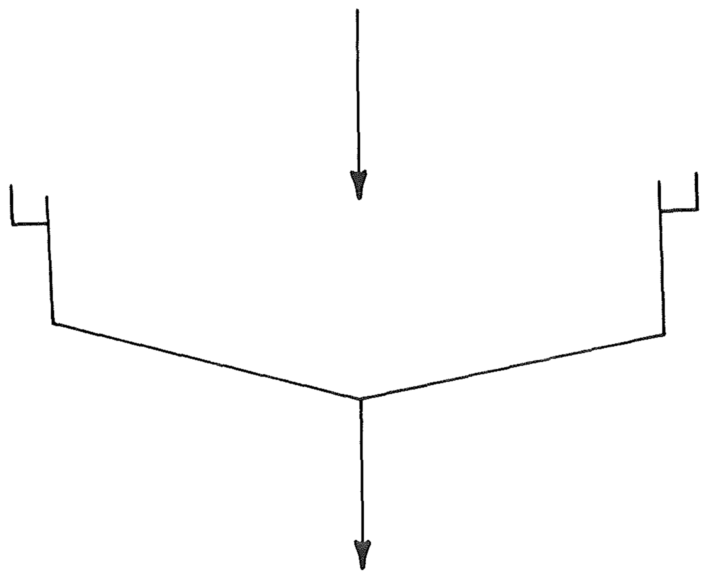
CSMRI Sample 17

Sample Origin:	Preparation Plant L.
Date of Sampling:	March 28, 1979.
Samples Taken and Amount:	Refuse thickener underflow - 69 55-gal drums.
Sample Description:	Refuse thickener underflow - •Flocculated and thickened 28 mesh x 0 raw coal feed. Diester concentrate table was not operating at the time of sampling.
Preparation Plant Description:	Product is for the steam market. Cleaning equipment - •Baum jig treating full size range. •Deister concentrating table retreating ¼ in. x 150 mesh clean coal from Baum jig. Thickening and dewatering equipment - •Static thickener (180 ft diam). •Underflow pumped to slurry pond. •Flocculant: Nalco 8852 and 8873.

EXHIBIT 1

Origin of Thickener Feed Solids
CSMRI Sample 17

Spiral Classifier Effluent (coarse refuse is the feed solids)
Cyclone Classifier Effluent (Deister concentrating table
Refuse is the Feed Solids
Scrubber Effluent (thermal dryer stack-gas solids are the
feed solids)



To Pond
Thickener Underflow: 49.7% Solids, pH = 8.4,
Pulp Density = 1.270 g/ml

EXHIBIT 2

PARTICLE SIZE DISTRIBUTION

Screen Size Analyses 1 through 9

Samples: Fine refuse thickener feeds from Plant A through I inclusive; CSMRI Samples 1 through 9.

Procedure: The samples were wet screened on 28 mesh. Each respective oversize product was dry screened on appropriate screens using the Ro-Tap method. Each undersize was tested using hydrometer method per Methods of Soil Analysis, Part 1, American Society of Agronomy, Inc. (1965), Section 43-3, Method for Separation of Clay, Silt, and Sand Fractions.

Results: The following tabulation contains the data for the resultant particle size distributions for the nine preliminary samples.

Particle Size Distributions for Preliminary Thickener Feed Samples

Feed Sample	Sample Weight g	Cumulative Weight % Passing, Size ¼ in.	8M	28M	100M	325M	22 μm	11μm	2 μm
1	~135	100.0	99.9	99.4	65.0	23.9	18.4	12.8	2.8
2	~689	100.0	97.2	92.6	67.1	56.9	51.1	44.9	36.5
3	~105	--	100.0	96.0	71.5	44.0	35.2	27.4	12.0
4	~113	--	100.0	98.0	78.0	54.5	48.9	42.3	14.1
5[1]	~17	--	--	100.0	70.0	45.0	39.3	26.5	6.9
6	~144	--	100.0	87.5	65.0	41.0	35.7	28.5	10.6
7	~116[2]	--	--	--	--	--	--	--	--
8	~85	--	100.0	97.0	63.0	36.6	30.1	23.2	10.3
9	~457	100.0	94.5	81.0	63.0	7.3	6.2	4.7	1.0

[1]Accuracy of this sample analysis questionable due to small sample weight.

[2]Flocculation occurred during the test run which invalidated the results. Additional material with which to repeat the test was not available.

EXHIBIT 2

Screen Size Analyses 10 through 18

Samples: Fine refuse thickener underflows from Plant A through I inclusive; CSMRI Samples 1 through 9.

Procedure: The samples were wet screened on 28 mesh. In each case, oversize product was dry screened on appropriate screens using the Ro-Tap method. Each undersize was tested using hydrometer method per Methods of Soil Analysis, Part 1, American Society of Agronomy, Inc. (1965), Section 43-3, Method for Separation of Clay, Silt, and Sand Fractions.

Results: The following tabulation contains the data for the resultant particle size distributions for the nine preliminary samples.

Particle Size Distributions for
Preliminary Thickener Underflow Samples

Feed Sample	Sample Weight g	Cumulative Weight % Passing, Size							
		¼ in.	8M	28M	100M	325M	22 µm	11µm	2 µm
1	~340	--	100.0	92.5	69.0	32.6	23.2	15.3	2.8
2	~145	--	100.0	97.3	88.0	70.5	68.1	60.3	44.8
3	~375	--	100.0	96.0	87.5	47.8	40.4	31.2	16.6
4	~388	--	100.0	96.0	75.0	56.7	49.9	41.4	18.9
5	~144	--	100.0	98.2	72.0	50.9	46.1	39.4	17.3
6	~337	--	100.0	94.0	73.0	46.3	41.9	34.9	16.6
7	~571	--	100.0	94.0	77.0	38.8	31.9	25.0	12.1
8	~341	--	100.0	97.5	79.0	43.2	34.6	27.6	13.0
9	~296	100.0	99.8	87.0	60.0	26.4	21.1	17.4	7.9

EXHIBIT 2

Ash Determination of Size Fractions

Sample: CSMRI Samples 13 through 17.

Procedure: A sample of each of the thickener underflows was subjected to particle size distribution via standard wet and dry screening. The dried individual screen fractions were then submitted for ash analysis which was performed according to ASTM[1] procedures.

Results: Tabulated as follows.

CSMRI Sample No.	Preparation Plant	Size Fraction mesh	Direct Weight %	Direct Ash %	Cumulative Weight %	Cumulative Ash %
13	B	8 x 28	1.2	23.8	1.2	23.8
		28 x 100	11.4	52.4	12.6	49.7
		100 x 325	36.9	63.4	49.5	59.9
		-325	50.5	75.2	100.0	67.6
14	C	8 x 28	<0.1	--	--	--
		28 x 100	23.5	12.1	23.5	12.1
		100 x 325	38.5	11.9	62.0	12.0
		-325	38.0	63.9	100.0	31.7
15	J	8 x 28	2.7	14.3	2.7	14.3
		28 x 100	27.2	14.1	29.9	14.1
		100 x 325	17.8	15.8	47.7	14.8
		-325	52.3	36.8	100.0	26.3
16	K	8 x 28	10.4	9.2	10.4	9.2
		28 x 100	14.1	25.3	24.5	18.5
		100 x 325	12.6	24.1	37.1	20.4
		-325	62.9	65.3	100.0	48.6
17	L	+8	2.7	7.7	2.7	7.7
		8 x 28	10.8	10.7	13.5	10.1
		28 x 100	19.6	20.3	33.1	16.1
		100 x 325	14.5	40.8	47.6	23.7
		-325	52.4	69.4	100.0	47.6

[1]ASTM -- D3174-73.

EXHIBIT 3

SPECIFIC GRAVITY (FLOAT-SINK) ANALYSES AND ASH DETERMINATION OF THICKENER UNDERFLOW SOLIDS

Sample: About 250 g of solids from thickener underflow Samples 1 through 11.

Procedure: The samples were subjected to specific gravity analyses at 1.60 specific gravity. Procedures employed are those described in USBM Bulletin 638. Products were analyzed for ash only according to ASTM[1] procedures.

Results:

CSMRI Sample No.	Preparation Plant	Specific Gravity	Direct Weight %	Direct Ash %	Cumulative Weight %	Cumulative Ash %
1	A	Float 1.60	62.3	11.8	62.3	11.3
		Sink 1.60	37.7	53.5	100.0	27.5
2	B	Float 1.60	9.7	20.8	9.7	20.8
		Sink 1.60	90.3	77.9	100.0	72.4
3	C	Float 1.60	44.8	24.4	44.8	24.4
		Sink 1.60	55.2	48.1	100.0	37.5
4	D	Float 1.60	15.4	15.0	15.4	15.0
		Sink 1.60	84.6	48.9	100.0	43.7
5	E	Float 1.60	21.9	20.7	21.9	20.7
		Sink 1.60	78.1	37.9	100.0	34.1
6	F	Float 1.60	23.8	11.5	23.8	11.5
		Sink 1.60	76.2	51.2	100.0	41.8
7	G	Float 1.60	27.9	14.5	27.9	14.5
		Sink 1.60	72.1	41.9	100.0	34.3
8	H	Float 1.60	34.9	9.4	34.9	9.4
		Sink 1.60	65.1	53.4	100.0	38.0
9	I	Float 1.60	28.4	16.2	28.4	16.2
		Sink 1.60	71.6	58.6	100.0	46.6
10	J	Float 1.60	39.3	12.1	39.3	12.1
		Sink 1.60	60.7	50.1	100.0	35.2
11	K	Float 1.60	13.4	17.3	13.4	13.4
		Sink 1.60	86.6	64.2	100.0	57.9

[1]ASTM -- D3174-73.

EXHIBIT 3

Bulk Samples

Sample: About 250 g of solids from thickener underflow Samples 13 through 17. (Sample taken at the same time as bulk sample.)

Procedure: The samples were subjected to specific gravity analyses at 1.60 sp gr. Procedures employed are those described in USBM Bulletin 638. Products were analyzed for ash, total sulfur, and Btu according to ASTM[1] procedures.

Results:

			Direct				Cumulative			
CSMRI Sample No.	Preparation Plant	Specific Gravity	Weight %	Ash %	Total S %	Btu	Weight %	Ash %	Total S %	Btu
13	B	Float 1.60	9.8	17.7	0.82	10,028	9.8	17.7	0.82	10,028
		Sink 1.60	90.2	68.4	0.72	3,362*	100.0	63.4	0.73	4,759
14	C	Float 1.60	63.3	18.1	0.70	11,248	63.3	18.1	0.70	11,248
		Sink 1.60	36.7	61.4	1.90	4,451	100.0	34.0	1.14	8,754
15	J	Float 1.60	59.7	15.9	0.68	12,982	59.7	15.9	0.68	12.982
		Sink 1.60	40.3	40.2	0.77	8,768	100.0	25.7	0.72	11,284
16	K	Float 1.60	19.6	10.6	0.87	12,708	19.6	10.6	0.87	12,708
		Sink 1.60	80.4	58.6	0.57	5,211	100.0	49.2	0.63	6,680
17	L	Float 1.60	29.0	13.9	2.94	12,716	29.0	13.9	2.94	12,716
		Sink 1.60	71.0	62.1	3.16	4,831	100.0	48.1	3.10	7,118

*Predicted using least squares regression analysis of C, J, K, and L data.

[1]ASTM -- D3174-73.

EXHIBIT 4

X-RAY DIFFRACTION ANALYSIS OF COAL REFUSE

Samples: Approximately 500 ml of fine refuse thickener underflow from Samples 1 through 11 inclusive.

Procedure: Each slurry sample was oven-dried and a split of the dried solids for each was ground to a fine powder (approximately 270 mesh) and run on the x-ray diffractometer over the range $2\theta = 3^\circ$ to 61° (CuK radiation). This was done to determine the minerals present without regard to particle size. A bulk x-ray diffraction (XRD) scan allows determination of the major minerals present. It generally does not allow detection of minerals present in amounts less than a few percent. It does not respond to coal, except to indicate that a certain amount of amorphous material is present.

Results: The interpreted results of the bulk XRD scans, with semiquantitative estimates of the amounts of the minerals identified based on peak intensities, are given in tabular form on the following page.

EXHIBIT 4

X-Ray Diffraction Analyses of Dried Solids from Fine Refuse Thickener Underflows

Phase and Estimated Amounts

Sample	Quartz	Calcite	Dolomite	Pyrite	Kaolinite	Illite	Montmorillonite	Feldspar	Amorphous	Total Unidentified Crystalline
1	Moderate	Moderate	Moderate/Minor	--	Trace	Trace	--	Trace	Major	Trace
2	Moderate/Minor	--	--	--	Minor	Minor	Moderate	Minor	Moderate	Minor[1]/Trace
3	Moderate/Major	Trace	Trace	--	Minor/Trace	Trace	--	--	Moderate	Trace
4	Moderate	Minor/Trace	--	Trace	Minor	Trace	--	--	Major/Moderate	Trace
5	Moderate	Minor/Trace	--	Minor	Minor	Trace	--	--	Major	Trace
6	Moderate	Minor/Trace	Trace	Minor	Minor	Minor/Trace	--	--	Major	Trace
7	Moderate	Minor	--	--	Minor	Minor	--	Trace	Major	Trace
8	Moderate	Minor/Trace	Trace	--	Minor	Minor	--	--	Major	Trace[2]
9	Moderate	Minor/Trace	Minor/Trace	Trace	Minor/Trace	Trace	--	--	Major	Trace
10[3]	Moderate	Minor	--	Minor/Trace	Minor	Minor	--	--	Moderate	Minor/Trace
11[3]	Moderate	--	--	Trace?	Moderate	Moderate	--	Trace?	Moderate	Minor/Trace

Note: Major, >30%; Moderate, 10%-30%; Minor, 3%-10%; Trace, <3%.

[1]May include a zeolite mineral (heulandite?) and cristobalite (SiO_2).
[2]May include siderite ($FeCO_3$).
[3]Done at a later date on +2.0 sp gr material to eliminate dilution by amorphous (coal) material.

EXHIBIT 5

QUANTITATIVE CLAY SEPARATIONS

Samples: Dried solids from fine refuse thickener underflow slurries from Samples 1 through 11 inclusive.

Procedure: An approximation method based upon Stokes Law was used to determine the weight percentage of clay size material, that is <2 μm.

Results: The yields of clay-sized material via the method of approximation for each sample were as follows:

CSMRI Sample No.	Clay Sized Material Weight % <2 μm
1	6.45
2	36.20
3	18.68
4	21.51
5	22.88
6	18.95
7	15.77
8	15.70
9	16.69
10	6.90
11	25.70

EXHIBIT 5

Approximate Quantitative Clay Separation Procedure

1. Disaggregate sample -- split until a 20 to 30 g sample size is achieved.

2. Weigh sample -- mark a 1,000 ml graduated cylinder in cm from the top: tare a 50 ml beaker.

3. Disperse sample in the 1,000 ml graduated cylinder half filled with distilled water. Add 10 ml of 10% Calgon solution and continue filling with distilled water to 1,000 ml.

4. Agitate sample with the sediment plunger until a good dispersion is achieved. Withdraw plunger being careful to wash off any clinging sediment. Begin timing immediately.

5. Allow suspension to settle for a convenient Stokes Law time for the <2 μm fraction (see chart).

6. After settling time has elapsed, withdraw a 20 ml aliquot from the specified settling depth. Transfer to the preweighed 50 ml beaker. Wash pipette into beaker to insure all clay has been transferred.

7. Evaporate to dryness in an oven at low temperature. Allow beaker to cool before weighing.

8. Record weight of clay fraction and multiply by 50.

9. To find weight percent clay, divide the recorded weight of clay by the total weight of the sample and multiply by 100.

EXHIBIT 6

X-RAY DIFFRACTION OF CLAY SAMPLES

Samples: Individual portions of clay-sized material from Samples 1 through 11 as obtained in the procedures detailed in Exhibit 5.

Procedure: A portion of the clay-sized (<2 μm) material from each sample was deposited as an oriented x-ray mount on a millipore filter and scanned on the diffractometer over a range $2\theta = 3^\circ$ to 31° (CuK radiation). The mounts were then treated with ethylene glycol and scanned again, this time over a range $2\theta = 3^\circ$ to 31° to detect the presence of any swelling (montmorillonite-type) clays.

Results: The interpreted results of the clay XRD scans are given in tabular form below. Estimates are again semiquantitative. The clay-size fractions in general still contain a great deal of coal and are very black. The B sample is the only one where swelling clay was detected, and the montmorillonite content is substantial. For Samples 10 (Plant J) and Sample 11 (Plant K); treatment of both oriented clay mounts with ethylene glycol gave no indication of detectible swelling clay.

Clay Diffraction Analysis
(<2 μm material)

PHASE AND ESTIMATED AMOUNT

CSMRI Sample No.	Kaolinite	Illite	Montmorillonite	Quartz	Amorphous	Total Unidentified (crystalline)
1	Trace	Trace	--	Trace	Major	Minor-Trace[1]
2	Moderate-Minor	Trace?	Major	Trace?	Moderate-Major	Trace
3	Moderate-Minor	Trace	--	Minor	Major	Trace
4	Minor	Trace	--	Trace	Major	Trace
5	Minor	Trace	--	Trace	Major	Trace
6	Moderate-Minor	Trace	--	Trace	Major	--
7	Minor	Minor	--	Trace	Major	Trace
8	Minor	Minor	--	Trace	Major	Trace[2]
9	Minor	Trace	--	Trace	Major	Trace
10	Moderate	Moderate	--	Minor	Moderate	--
11	Major	Moderate	--	Minor	Minor	--

Note: Major -- >30%, Moderate -- 10%-30%, Minor 3%-10%, Trace -- <3%.

[1]May include a zeolite (erionite?) or hydrobiotite.
[2]May include chlorite.

EXHIBIT 7

X-RAY DIFFRACTION ANALYSIS ON +2.0 SPECIFIC GRAVITY

Samples: +2.0 specific gravity portions of dried Samples 13 through 17.

Procedure: A portion of each sample was ground and prepared as a diffractometer mount. The samples were scanned from 3°-61° of 2θ with CuKα radiation. Interpretation of the resulting diffraction patterns showed the following phases with amounts given as estimated weight percentages:

Phase	Composition	Sample No. 13 +2.0	14 +2.0	15 +2.0	16 +2.0	17 +2.0
Quartz	SiO_2	13	62	46	39	48
Kaolinite	$Al_2Si_2O_5(OH)_4$	2	9	7	11	10
Illite-Mica	K Al silicate	--	3	7	9	8
Feldspar	K NaCaAl silicate	30	2	4	6	2
Magnetite	Fe_3O_4	--	--	11	4	--
Rutile	TiO_2	--	4	1	4	1
Pyrite	FeS_2	--	1	2	1	5
Calcite	$CaCO_3$	2	2	1	--	2
Aragonite	$CaCO_3$	--	--	2	--	1
Dolomite	$CaMg(CO_3)_2$	--	--	3	1	--
Siderite	Fe CO_3	--	--	1	1	--
Gypsum	$CaSO_4 \cdot 2H_2O$	--	3	--	--	--
Amorphous	?	21	11	14	22	21
Unidentified	?	--	3	1	2	2
Montmorillonite-Chlorite	--	32	--	--	--	--

Amounts are based on visual estimates and measurements of diffraction peak heights and are only semiquantitative. Precisions vary with amount and are estimated as follows:

Est. wt %	1	2	4	7	10	16	23	30	40	50	67	85
Est. σ	1	1.5	2	2.5	3	4	5	6	7	8	10	12

The unidentified category accounts for that part of the diffraction pattern not accounted for by the identified phases.

A portion of each (~10 g) was subjected to a settling rate procedure based on Stokes Law. The objectives of this procedure were to provide an approximation to the total weight of clay size (<2 μm) material present in the sample and to provide a separated fraction of the clay size material. The first objective was met with the following results.

EXHIBIT 7

	Sample No. 13 +2.0	14 +2.0	15 +2.0	16 +2.0	17 +2.0
Total Sample, mg	10.5694	10.2556	10.2201	10.9664	11.0106
Clay Size Fraction, mg	4.302	2.215	0.245	1.570	1.140
Weight % Clay Size	40.7	21.6	2.4	14.3	10.3

Part of the clay size fraction was prepared as an oriented clay mount and its diffractometer pattern was recorded (scan from 3° to 31° of 2θ with CuKα radiation). The clay mount was treated with ethylene glycol to expand the structures of any montmorillonitic clays that might be present. The clay mount was then rescanned from 3° to 31°. The two diffraction patterns from each clay size fraction were interpreted and showed the following minerals with amounts given as estimated weight percentages:

Mineral	Sample No. 13 +2.0	14 +2.0	15 +2.0	16 +2.0	17 +2.0
Kaolinite	18	46	46	44	27
Illite-Mica	--	20	25	28	39
Chlorite	3	4	6	3	5
Montmorillonite	72	--	2	--	--
Quartz	2	27	17	20	24
Feldspar	5	3	4	5	5

Amounts are based on peak areas within the diffraction range and should be considered to be only semiquantitative. It is possible that the clay fractions may contain some amorphous material, but the limited scan range makes it difficult to determine that aspect.

EXHIBIT 8

CLAY CONTENT OF THICKENER UNDERFLOW SOLIDS FROM BULK SAMPLES

Sample: Thickener underflow samples taken at the time the bulk samples were taken.

Procedure: Dried solids from thickener underflow sample was subjected to specific gravity analysis at 2.0 sp gr. Procedures employed as described in USBM Bulletin 638. Clay content of sink 2.0 sp gr material determined by subjecting solids to settling rate procedures based on Stokes' Law. The final figure for percentage of clay in total solids of thickener underflow sample determined by weighted average calculation.

Results:

CSMRI Sample No.	Preparation Plant	Sink 2.0 Specific Gravity Material wt %	Clay Size Material (<2 μm) in Sink 2.0 Fraction
13	B	68.9	40.7
14	C	13.6	34.6
15	J	6.6	35.6
16	K	67.4	43.7
17	L	50.8	45.2

EXHIBIT 9

FLOCCULANT SCOPING

Purpose: To determine the best flocculant system on a preliminary basis using the reagents available.

Samples: CSMRI Samples 13 through 17.

Procedure: Portions of coal refuse thickener underflow samples were either diluted or left undiluted and mixed by hand with a glass rod with various flocculant systems. The flocculants were added at various concentrations, but none higher than 5 gpl. Observations were then made and recorded as to flocculant formation, water release, and supernatant clarity. Results are provided in the following tables.

Results:

Flocculant Scoping Results for Sample 13
Preparation Plant B

Constant Conditions: Pulp diluted 1 to 1 with water, 50 ml of diluted pulp used in each case. Original pulp % solids = 24.0.

Flocculant Used	Volume ml	Concentration gpl	Observations
Nalco TX-14	5.5	1.0	No visible flocculation.
Nalco TX-14	25.0	0.25	No visible flocculation.
Nalco 8873	8.0	1.0	No visible flocculation.
Nalco 8873	30.0	0.25	No visible flocculation.
Celanese 361	10.0	1.0	Slight flocculation, very dirty supernate.
Celanese 361	25.0	0.25	Medium floccculations formed, slow water release.
Celanese 990	9.0	1.0	No visible flocculation.
Celanese 990	20.0	0.25	Little flocculation.
Nalco 8852	10.0	1.0	Medium flocculation, slow water release.
Celanese 347	10.0	1.0	Medium flocculation, medium water release.
Nalco 8852	26.0	0.25	Medium flocculation, cloudy supernate, medium water release.
Celanese 347	15.0	0.25	No visible flocculation.
Dowell 191	25.0	0.25	No visible flocculation.
Dowell M88	25.0	0.25	No visible flocculation.
American Cyanamid 1202	35.0	0.25	No visible flocculation.
Celanese 990 with	10.0	0.25	No visible flocculation.
Nalco 8852	10.0	0.25	No visible flocculation.
Nalco 8852 with	10.0	0.25	No visible flocculation.
Nalco 8873	10.0	0.25	

EXHIBIT 9

Constant
Conditions: Flocculant added to undiluted refuse pulp.

Pulp Volume ml	Flocculant Used	Volume ml	Concentration gpl	Observations
100	Nalco 8852	50	0.25	No visible flocculation.
100	Nalco 8852	10	1.0	No visible flocculation.
200	CaO - 3.0 g	--	--	No visible flocculation.
50	CaO to pH 11.0 with	--	--	No visible flocculation.
	American Cyanamid 1202	30	0.5	
50	CaO to pH 11.0 with	--	--	Gelatinous globular flocs.
	Celanese 990	2	1.0	
50	Nalco 8873	20	1.0	Gelatinous globular flocs.
50	Nalco 8852	50	1.0	Some flocculation, slow water release.
50	Celanese 990	5	1.0	Gelatinous globular flocs.
50	Celanese 347	10	0.25	Some flocculation, dirty supernate, slow water release.

Flocculant Scoping Results for Sample 14
Preparation Plant C

Constant
Conditions: 50 ml of undiluted refuse slurry, flocculant concentration = 1 gpl. Original pulp % solids = 19.1.

Flocculant Used	Volume ml	Observations
Nalco TX 14	1.0	Good water release, cloudy supernate, floating scum.
Nalco 8873	1.0	Good water release, cloudy supernate, floating scum.
Celanese 361	2.5	Small, very fine floc formed. Later very good clarity.
Celanese 347	2.0	Small, very fine floc formed. Later very good clarity.
Nalco 8852	0.5	Small, very fine floc formed. Superior clarity.
Celanese 990	0.5	Large flocs formed, very dirty supernate.
Dowell M191	2.0	Small flocculant formation, slow water release.
Dowell M88	0.25	Large flocs, quick water release, cloudy supernate.
American Cyanamid 1202	0.40	Medium flocs, good water release, trash on top.
Celanese 347 with	1.0	Medium flocs, quick water release, cloudy supernate.
Celanese 990	0.4	

EXHIBIT 9

Flocculant Scoping Results for Sample 15
Preparation Plant J

Constant Conditions: 50 ml of undiluted refuse slurry. Original pulp % solids = 30.7.

Flocculant Used	Volume ml	Concentration gpl	Observations
American Cyanamid 208	1.0	1.0	No visible flocculation.
American Cyanamid 208	5.0	0.25	Small flocs, fair water release, dirty supernate.
Celanese 361	35.0	0.25	Tiny flocs, minimal water release.
Celanese 361	35.0	1.0	Small flocs, slow water release.
Celanese 990	3.0	0.25	Medium flocs, slow water release, cloudy supernate.
Celanese 347	15.0	1.0	No visible flocculation.
American Cyanamid 208	5.0	0.25	No visible flocculation.
with Celanese 361	10.0	0.25	
Celanese 361	100.0	0.25	No visible flocculation.
Dowell M88	15.0	1.0	No visible flocculation.
Nalco 8852	15.0	1.0	No visible flocculation.

Flocculant Scoping Results for Sample 16
Preparation Plant K

Constant Conditions: Pulp diluted 1 to 1 with water, flocculants added as 1 gpl. Original pulp % solids = 40.8.

Flocculant Used	Volume ml	Observations
American Cyanamid 315	5.0	Small flocs formed, small water release, clean supernate.
American Cyanamid 214	0.6	Small flocs formed, small water release, dirty supernate.
American Cyanamid 208	0.7	Medium flocs formed, slow water release, dirty supernate.
Nalco TX14	0.5	Medium flocs formed, slow water release, dirty supernate.
Nalco 8873	0.8	Small flocs formed, very slow water release, dirty supernate.
Celanese 361	3.0	Medium flocs formed, good water release, clean supernate.
Celanese 990	0.6	Medium flocs formed, good water release, dirty supernate.
Nalco 8852	3.0	Medium flocs formed, good water release, clean supernate.

EXHIBIT 9

Preparation Plant K -- Continued

Flocculant Used	Volume ml	Observations
Celanese 347	3.0	Small flocs formed, medium water release, cloudy supernate.
Dowell 191	5.0	Medium flocs formed, slow water release, dirty supernate.
Dowell 88	3.0	Large flocs formed, good water release, dirty supernate.
American Cyanamid 1202	1.0	Medium flocs formed, medium water release, cloudy supernate.

Flocculant Scoping Results for Sample 17
Preparation Plant L

Constant Conditions: Pulp diluted 1 to 1 with water. Original pulp % solids = 49.7.

Flocculant Used	Volume ml	Concentration gpl	Observations
American Cyanamid 315	4.0	1.0	Small flocs formed, slow water release, cloudy supernate.
American Cyanamid 214	0.4	1.0	Medium flocs formed, good water release, murky supernate.
Dowell M88	0.4	1.0	Small flocs formed, good water release, murky supernate.
Nalco TX-14	0.3	5.0	Medium flocs formed, good water release, very cloudy supernate.
Nalco 8873	0.4	1.0	Medium flocs formed, good water release, very cloudy supernate.
Celanese 361	5.0	1.0	Small flocs formed, medium water release, clear supernate.
Celanese 990	0.1	1.0	Small flocs formed, fast water release, very dirty supernate.
Nalco 8852	1.5	5.0	Small flocs formed, slow water release, cloudy supernate.
Celanese 347	5.0	1.0	Small flocs formed, slow water release, very cloudy supernate.
Dowell M191	2.0	5.0	Small flocs formed, slow water release, cloudy supernate.
American Cyanamid 1202	5.0	1.0	Not well flocculated.
American Cyanamid 208	0.3	1.0	Medium flocs formed, good water release, very cloudy supernate.

EXHIBIT 10

FILTER TESTS OF THICKENER UNDERFLOW SLURRY -- PRELIMINARY SAMPLES

Sample: CSMRI Samples 1 through 9.

Procedure: A bench-scale filtration set-up was used involving a vacuum pump, filtrate flask, pour-on-type test leaf, vacuum gage, and a quick-release valve arrangement in the vacuum line. Approximately 150 ml of slurry was added to the test leaf and the vacuum turned on. Filtration was continued until there was no visible water on top of the formed cake. Time to this end point was noted.

Results:

CSMRI Sample No.	Preparation Plant	Filtration Time min[1]	Cake: Dry Solids g	Cake: Solids %	Filtrate Volume ml	Filter Capacity[4] (Cake) $lb/ft^2/hr$
1	A	3.8	101.3	90.6	83	71
2	B	--[2]	--	--	--	--
3	C	15.0	68.4	68.1	84	12
4	D	45.0[3]	--	--	55	--
5	E	11.5	27.0	64.8	141	6
6	F	37.0	54.1	68.1	95	4
7	G	10.5	59.8	71.3	106	15
8	H	33.0	59.8	70.6	95	5
9	I	12.3	47.2	76.4	127	10

[1]Time to no visible water on top of the formed cake.
[2]Unfiltereable.
[3]Filtration rate so low that test was aborted.
[4]Filter capacity calculated using the following formula:

$$\frac{\text{Dry Cake wt, g}}{\text{454 g/lb}} \times \frac{1}{\text{Filter Area, 0.05 ft}^2} \times \frac{1}{\text{Filtration Time, min}} \times \frac{\text{60 min}}{\text{hr}} = \frac{\text{Dry lb}}{\text{ft}^2\text{-hr}}$$

EXHIBIT 10

Possible Effects of Aging

Sample: CSMRI Sample 1 through 9 (preliminary samples).

Procedure: Identical to that described previously. However, the test procedure was carried out by a different investigator and the samples had aged approximately 5 mo.

Results:

CSMRI Sample No.	Preparation Plant	Filtration Time min[1]	Cake: Dry Solids g	Cake: Solids %	Filtrate Volume ml	Filter Capacity[2] (Cake) lb/ft^2/hr
1	A	1.33	73.6	74.0	74	146
2	B	0.13	7.3	54.9	150	148
3	C	6.82	58.4	62.9	86	23
4	D	60.00	58.7	63.5	75	3
5	E	2.97	14.5	63.0	146	13
6	F	30.05	49.1	68.1	98	4
7	G	3.68	21.1	66.4	133	15
8	H	16.12	54.7	68.7	102	9
9	I	5.88	20.8	69.1	134	9

[1]Time to no visible water on top of formed cake.
[2]Filter capacity calculated using the following formula:

$$\frac{\text{Dry Cake wt, g}}{\text{454 g/lb}} \times \frac{1}{\text{Filter Area, 0.05 ft}^2} \times \frac{1}{\text{Filtration Time, min}} \times \frac{\text{60 min}}{\text{hr}} = \frac{\text{Dry lb}}{\text{ft}^2\text{-hr}}$$

EXHIBIT 11

FILTER TESTS OF THICKENER UNDERFLOW SLURRY -- BULK SAMPLES

Sample: CSMRI Samples 13 through 17.

Procedure: Identical to that described in Exhibit 10 -- using the same equipment. Tests performed during the first week of May, 1980 in conjunction with flocculant scoping work detailed in Exhibit 9.

Results: Tabulated results contain the best filtration rate achieved for flocculant systems investigated.

CSMRI Sample No.	Preparation Plant	Filtration Time min	Cake: Dry Solids g	Cake: Solids %	Filtrate Volume ml	Flocculant Type	Addition lb/ton	Filter Capacity (Cake) lb/ft²/hr
13	B	1.5	28.0	50.0	150	Nalco 8852	6.3	88
14	C	1.0	38.0	54.3	152	Celanese 347 Celanese 990	0.1 0.2	100
15	J	0.75	34.0	61.8	94	American Cyanamid 208	0.2	120
16	K	7.0	47.0	63.5	92	Dowell M88	0.9	18
17	L	19.0	58.0	70.7	121	Celanese 347	3.4	31

[1]Filter capacity calculated using the following formula:

$$\frac{\text{Dry Cake wt, g}}{\text{454 g/lb}} \times \frac{1}{\text{Filter Area, 0.05 ft}^2} \times \frac{1}{\text{Filtration Time, min}} \times \frac{\text{60 min}}{\text{hr}} = \frac{\text{Dry lb}}{\text{ft}^2\text{-hr}}$$

EXHIBIT 11

Sample: CSMRI Samples 13 through 17 (bulk samples).

Procedure: Identical to that previously described with the exception that: (1) no flocculant addition was made to any of the tests; (2) a different investigator performed the actual test work; and (3) the tests were performed approximately 5 months after those described just previously.

Results:

CSMRI Sample No.	Preparation Plant	Filtration Time min	Cake Dry Solids g	Cake Solids %	Filtrate Volume ml	Filter Capacity[1] (Cake) $lb/ft^2/hr$
13	B	65.0	40.0	29.0	26	2
14	C	6.4	23.2	65.9	122	10
15	J	6.9	47.8	67.7	107	18
16	K	60.0	72.3	66.5	72	3
17	L	60.0	100.5	62.6	30	4

[1]Filter capacity calculated using the following formula:

$$\frac{\text{Dry Cake wt, g}}{\text{454 g/lb}} \times \frac{1}{\text{Filter Area, 0.05 ft}^2} \times \frac{1}{\text{Filtration Time, min}} \times \frac{\text{60 min}}{\text{hr}} = \frac{\text{Dry lb}}{\text{ft}^2\text{-hr}}$$

EXHIBIT 12

DRUM VACUUM FILTER

Purpos: To dewater coal refuse slurries on a continuous basis for at least 1 hr.

Samples: CSMRI Sample 13 through 17. Preparation Plants B, C, J, K, and L.

Procedure: A vacuum drum filter was set up at the Spokane Mining Research Center to dewater five coal refuse samples. The drum measured 12-in. wide x 18-in.-diam and one complete revolution took 1.75 min. The unit was self-contained with its own source of vacuum for filtration, blower for product discharge, and continuous rake unit for pulp agitation. Bulk samples in 55-gal drums were agitated to suspend all particles and dip samples were taken for flocculant scoping. A number of variously charged polyacrylamides were tested and a decision was reached on type and concentration based on visual results. Sufficient flocculant was added to the bulk sample, and agitation was provided to aid the process. A 1-in.-diam Sand Piper pump was used to transfer the flocculated material to the tub of the drum filter. The filter was operated for a minimum of 1 hr on each of the five refuse slurries. Samples of the feed as well as periodic samples of cake discharge were taken and the percent solids was determined. Filtration rate was determined using the following formula:

$$\frac{\text{Dry Cake wt, g}}{1} \times \frac{\text{lb}}{\text{454 g}} \times \frac{1}{\text{Time Period, min}} \times \frac{\text{60 min}}{\text{hr}} \times \frac{\text{4 (drum filter form fraction)}}{\text{Filter Area} = 4.00\ \text{ft}^2} =$$

Cake Form Rate

Operating parameters were noted and are contained in the following pages.

EXHIBIT 12

Drum Filter Test 1 -- Sample 13
Preparation Plant B

Clock Time	Cake Sample No.	Cake Weight Wet g	Cake Weight Dry g	Cake Solids %	Cake Form Rate lb/ft^2-hr
9:11					
9:30	1	2,005.9	969.4	48.3	128.1
9:45	2	1,718.5	840.6	48.9	111.1
10:15	3	3,181.8	1,750.2	55.0	231.3
10:30	4	3,056.8	1,643.0	53.7	217.1
10:45	5	2,344.2	1,225.5	52.3	162.0
11:15	6	1,850.1	941.7	50.9	124.5
11:30	7	1,476.0	745.2	50.5	98.5
11:45	8	1,375.4	699.0	50.8	92.4
12:00	9	1,411.2	719.3	51.0	95.1

Total Wet Cake Product Recovered: 349 lb

Total Filtrate Recovered: 415 lb

Filtrate Clarity: 1.33% solids

Average Vacuum Reading: 10.5 psig

Cake Sampling Duration: 1 min

Feed Data: 30.5 liters of 1 gpl Polyhall 990 floccu- equal to 0.61 lb/ton
5,128 g of hydrated CaO equal to 102 lb/ ton
97.5 gal of coal refuse slurry at 24.0% solids

Drum Specifications: 12-in. wide by 18-in. diam
Time for 1 drum revolution was 1.75 min

Filter Media Specifications: National Filter Media, Salt Lake City, Utah
Media No. S/15405899
50 x 50 count 6.7 oz/yd^2
8 mil yarn, air flow >900 cfm
(screen category equivalent to a 50M Tyler screen size).

Overall Cake Form Rate =

$$\left(349\ \text{lb} + \frac{18{,}419.9^{2}\ \text{g}}{454}\right) \times 0.526^{1} \times \frac{1}{2.8\ \text{hr}} \times \frac{4\ (\text{form fraction})}{4\ \text{ft}^2} = 73.2\ \text{lb/ft}^2\text{-hr}$$

[1]Average five highest cake solids percent.
[2]Total of 1 min sample wet cake weight.

EXHIBIT 12

Drum Filter Test 2 -- Sample 14
Preparation Plant C

Clock Time	Cake Sample No.	Cake Weight Wet g	Cake Weight Dry g	Cake Solids %	Cake Form Rate lb/ft^2-hr
3:15					
3:30	1	1,095.4	728.8	66.5	96.3
3:45	2	1,122.0	747.3	66.6	98.8
4:00	3	625.2	396.9	63.5	52.5
4:15	4	954.2	630.2	66.0	83.3
4:45	5	1,506.7	1,032.1	68.5	136.4
5:15	6	1,605.5	1,102.0	68.6	145.6
5:45	7	1,508.6	1,039.5	68.9	137.4
6:05	8	1,201.7	809.1	67.3	106.9

Total Wet Cake Product Recovered:	242 lb
Total Filtrate Recovered:	952 lb
Filtrate Clarity:	2.1% solids
Average Vacuum Reading:	10.0 psig
Cake Sampling Duration:	1 min
Feed Data:	16 liters of 0.25 gpl Polyhall 990 equal to 0.30 lb/ton 137 gal of coal refuse slurry at 19.1% solids
Drum Specifications:	12-in. wide by 18-in. diam Time for 1 drum revolution was 1.75 min
Filter Media Specifications:	National Filter Media, Salt Lake City, Utah Media No. S/15405899 50 x 50 count 6.7 oz/yd^2 8 mil yarn, air flow >900 cfm (screen category equivalent to a 50M Tyler screen size).

Overall Cake Form Rate =

$$\left(242 \text{ lb} + \frac{9{,}619.3^{2} \text{ g}}{454}\right) \times 0.68^{1} \times \frac{1}{2.8 \text{ hr}} \times \frac{4 \text{ (form fraction)}}{4 \text{ ft}^2} = 63.9 \text{ lb/ft}^2\text{-hr}$$

[1]Are five highest cake solids percent.
[2]Total of 1 min sample dry cake weight.

EXHIBIT 12

Drum Filter Test 3 -- Sample 15
Preparation Plant J

Clock Time	Cake Sample No.	Cake Weight Wet g	Cake Weight Dry g	Cake Solids %	Cake Form Rate lb/ft^2-hr
12:30					
12:40	1	377.4	269.8	71.5	35.7
12:50	2	422.3	298.2	70.6	39.4
1:00	3	302.0	216.8	71.8	28.7
1:10	4	404.8	288.9	71.4	38.2
1:20	5	360.6	254.0	70.4	33.6
1:30	6	368.3	256.4	69.6	33.9
1:40	7	380.0	259.0	68.2	34.2
2:40	8	287.6	191.5	66.6	25.3
3:40	9	220.5	145.7	66.1	19.3
4:30	10	167.5	111.3	66.4	14.7

Total Wet Cake Product Recovered: 93 lb

Total Filtrate Recovered: 171 lb

Filtrate Clarity: 0.01% solids

Average Vacuum Reading: 12.5 psig

Cake Sampling Duration: 1 min

Feed Data: 6 liters of Polyhall 990 flocculant equal to 0.11 lb/ton
82.5 gal of coal refuse slurry at 30.7% solids

Drum Specifications: 12-in. wide by 18-in. diam
Time for 1 drum revolution was 1.75 min

Filter Media Specifications: National Filter Media, Salt Lake City, Utah
Media No. S/15405899
50 x 50 count 6.7 oz/yd^2
8 mil yarn, air flow >900 cfm
(screen category equivalent to a 50M Tyler screen size).

Overall Cake Form Rate =

$$\left(93 \text{ lb} + \frac{3{,}291.0^{2} \text{ g}}{454}\right) \times 0.711^{1} \times \frac{1}{4.0 \text{ hr}} \times \frac{4 \text{ (form fraction)}}{4 \text{ ft}^2} = 17.8 \text{ lb/ft}^2\text{-hr}$$

[1]Are five highest cake solids percent.
[2]Total of 1 min sample dry cake weight.

EXHIBIT 12

Drum Filter Test 4 -- Sample 16
Preparation Plant K

Clock Time	Cake Sample No.	Cake Weight Wet g	Cake Weight Dry g	Cake Solids %	Cake Form Rate lb/ft^2-hr
10:45					
11:00	1	1,378.1	871.0	63.2	115.1
11:15	2	1,352.2	850.4	62.9	112.4
11:30	3	1,400.4	884.6	63.2	116.9
11:45	4	1,377.8	868.6	63.0	114.8
12:15	5	1,284.9	808.3	62.9	106.8
12:45	6	1,237.0	774.1	62.9	102.3
1:15	7	1,293.3	813.0	62.9	107.4
1:45	8	1,349.9	844.2	62.5	111.6

Total Wet Cake Product Recovered:	462 lb
Total Filtrate Recovered:	260 lb
Filtrate Clarity:	8.0% solids
Average Vacuum Reading:	13.0 psig
Cake Sampling Duration:	1 min
Feed Data:	31 liters of 0.25 gpl Polyhall 990 equal to 0.10 lb/ton 81 gal of coal refuse slurry at 40.8% solids
Drum Specifications:	12-in. wide by 18-in. diam Time for 1 drum revolution was 1.75 min
Filter Media Specifications:	National Filter Media, Salt Lake City, Utah Media No. S/15405899 50 x 50 count 6.7 oz/yd^2 8 mil yarn, air flow >900 cfm (screen category equivalent to a 50M Tyler screen size).

Overall Cake Form Rate =

$$\left(462 \text{ lb} + \frac{10{,}673.6^{2} \text{ g}}{454}\right) \times 0.63^{1} \times \frac{1}{3.0 \text{ hr}} \times \frac{4 \text{ (form fraction)}}{4 \text{ ft}^2} = 102.0 \text{ lb/ft}^2\text{-hr}$$

[1]Are five highest cake solids percent.
[2]Total of 1 min sample dry cake weight.

EXHIBIT 12

Drum Filter Test 5 -- Sample 17
Preparation Plant L

Clock Time	Cake Sample No.	Cake Weight Wet g	Cake Weight Dry g	Cake Solids %	Cake Form Rate lb/ft^2-hr
12:35					
12:50	1	1,789.7	1,319.5	73.7	174.4
1:05	2	1,865.0	1,374.6	73.7	181.7
1:20	3	1,820.5	1,341.7	73.7	177.3
1:35	4	1,827.2	1,345.3	73.6	177.8
2:05	5	1,491.6	1,100.6	73.8	145.5
2:35	6	1,609.3	981.9	61.9	129.8
2:40	7	1,479.9	824.5	55.7	109.0
3:05					

Total Wet Cake Product Recovered: 295 lb

Total Filtrate Recovered: 196 lb

Filtrate Clarity: 7.4% solids

Average Vacuum Reading: 12.0 psig

Cake Sampling Duration: 1 min

Feed Data: 12 liters of Polyhall 990 flocculant equal to 0.11 lb/ton
1,070 g of hydrated CaO equal to 13 lb/ton
71 gal of coal refuse slurry at 49.7% solids

Drum Specifications: 12-in. wide by 18-in. diam
Time for 1 drum revolution was 1.75 min

Filter Media Specifications: National Filter Media, Salt Lake City, Utah
Media No. S/15405899
50 x 50 count 6.7 oz/yd^2
8 mil yarn, air flow >900 cfm
(screen category equivalent to a 50M Tyler screen size).

Overall Cake Form Rate =

$$\left(295 \text{ lb} + \frac{11{,}883.2^{2} \text{ g}}{454}\right) \times 0.737^{1} \times \frac{1}{2.5 \text{ hr}} \times \frac{4 \text{ (form fraction)}}{4 \text{ ft}^2} = 94.7 \text{ lb/ft}^2\text{-hr}$$

[1]Are five highest cake solids percent.
[2]Total of 1 min sample dry cake weight.

EXHIBIT 13

PRESSURE FILTER

Purpose: To determine coal refuse dewatering characteristics using a pressure filter.

Samples: Thickener underflow coal refuse from Preparation Plants B, C, J, K, and L; identified as CSMRI Samples 13 through 17.

Procedure: These tests were conducted using a Passavant Model 600 Pilot Pressure Filter System. The unit was supplied by Passavant Corporation and test work was conducted by CSMRI personnel at the Spokane Mining Research Center. The unit had an I.D. of 6 in. and an internal area of 0.33 ft^2 with average formed cake thickness of 1.57 in. The equipment consisted of a high pressure source (cylinder) of N_2 plus regulator, a pulp holding tank, the small stainless steel plate and frame filter, and connecting high pressure hose from the tank to the filter. Each test batch consisted of 4,600 ml of treated coal refuse thickener underflow which was placed in the holding tank. The filter unit having been assembled was piped to the holding tank which, in turn, was connected to the N_2 source. The test was started with the application of 225 psig N_2 to the pulp holding tank; starting time was also noted. Filtrate samples were taken at specific time intervals which varied with the performance of any given refuse sample. A test was stopped when the flow rate flatened out; that is when the current flow rate for a given time period was almost the same as that determined for the previous time period. The unit was then disassembled and the cake discharged. A wet and dry weight for the product cake was determined and all laboratory data was sent to Passavant Corporation for equipment sizing.

Results: Tabulation of all laboratory data as well as the recommendations by Passavant Corporation follow.

EXHIBIT 13

Laboratory Pressure Filter Test 1

Sample: Thickener underflow pulp from Preparation Plant B, CSMRI Sample 13.

Test Data:

	Feed Pulp Solids, %:	24.0
	Filter Pressure, psig:	225
	Flocculants, lb/ton solids:	CaO = 107

Filtration Data

Direct		Cumulative		
Time min	Volume ml	Time min	Volume ml	Rate ml/min
2	140	2	140	70
4	105	6	245	41
4	75	10	320	32
5	75	15	395	26
5	65	20	460	23
5	60	25	520	21
5	53	30	573	19
5	50	35	623	18

Filter Cake Product:

	Wet Weight, g:	878.6
	Dry Weight, g:	334.2
	Cake Solids, %:	38.0
	Filter Rate[1], lb/ft^2-hr:	3.82

Note: The cake was sloppy and the filter chamber was not completely filled. The data was determined by Passavant as unsuitable for sizing determinations.

[1] $\frac{\text{Dry Cake Weight, g} \times 60}{454 \times 0.33 \times \text{Time, min}} = lb/ft^2\text{-hr}$

Test filter area, 0.33 ft^2

EXHIBIT 13

Laboratory Pressure Filter Test 2

Sample: Thickener underflow pulp from Preparation Plant C, CSMRI Sample 14.

Test Data:

Feed Pulp Solids, %:	19.1
Filter Pressure, psig:	225
Flocculants, lb/ton solids:	None

Filtration Data

Direct		Cumulative		
Time min	Volume ml	Time min	Volume ml	Rate ml/min
1	410	1	410	410
1	310	2	720	360
1	280	3	1,000	333
1	250	4	1,250	313
1	235	5	1,485	297
1	200	6	1,685	281
1	190	7	1,875	268
1	180	8	2,055	257
1	160	9	2,215	246
2	290	11	2,505	228
2	255	13	2,530	195
2	225	15	2,555	170
2	200	17	2,575	151
3	270	20	2,845	142
5	370	25	3,215	129
5	285	30	3,500	117
5	210	35	3,710	106
5	200	40	3,910	98
5	175	45	4,085	91

Filter Cake Product:

Wet Weight, g:	1,148.6
Dry Weight, g:	841.7
Cake Solids, %:	73.3
Filter Rate, lb/ft^2-hr:	7.49

Note: These data were used by Passavant Corporation to size filter equipment as given in the tables following Test 6.

EXHIBIT 13

Laboratory Pressure Filter Test 3

Sample: Thickener underflow pulp from Preparation Plant J, CSMRI Sample 15.

Test Data:

Test Data:	Feed Pulp Solids, %:	30.7
	Filter Pressure, psig:	225
	Flocculants, lb/ton solids:	None

Filtration Data

Direct		Cumulative		
Time min	Volume ml	Time min	Volume ml	Rate ml/min
1	500	1	500	500
1	325	2	825	413
1	270	3	1,095	365
1	245	4	1,340	335
1	215	5	1,555	311
1	150	6	1,705	284
1	75	7	1,780	254
1	21	8	1,801	225
1	7	9	1,808	201
1	5	10	1,813	181

Filter Cake Product:

Wet Weight, g:	1,225.2
Dry Weight, g:	943.0
Cake Solids, %:	77.0
Filter Rate, lb/ft^2-hr:	37.77

Note: These data were used by Passavant Corporation to size filter equipment as given in the tables following Test 6.

EXHIBIT 13

Laboratory Pressure Filter Test 4

Sample: Thickener underflow pulp from Preparation Plant K, CSMRI Sample 16.

Test Data:

Feed Pulp Solids, %:	40.8
Filter Pressure, psig:	225
Flocculants, lb/ton solids:	None

Filtration Data

Direct		Cumulative		
Time min	Volume ml	Time min	Volume ml	Rate ml/min
2	230	2	230	115
3	175	5	405	81
5	215	10	620	62
5	175	15	795	53
5	150	20	945	47
5	135	25	1,080	43
5	120	30	1,200	40
5	105	35	1,305	37
5	85	40	1,390	35
5	55	45	1,445	32

Filter Cake Product:

Wet Weight, g:	1,441.8
Dry Weight, g:	1,096.1
Cake Solids, %:	76.0
Filter Rate, lb/ft^2-hr:	9.75

Note: These data were used by Passavant Corporation to size filter equipment as given in the tables following Test 6.

EXHIBIT 13

Laboratory Pressure Filter Test 5

Sample: Thickener underflow pulp from Preparation Plant L, CSMRI Sample 17.

Test Data:
- Feed Pulp Solids, %: 24.9[1]
- Filter Pressure, psig: 225
- Flocculants, lb/ton solids: CaO = 81, Percol 727 = 0.8

Filtration Data

Direct		Cumulative		
Time min	Volume ml	Time min	Volume ml	Rate ml/min
2	490	2	490	245
3	350	5	840	168
5	450	10	1,290	129
5	365	15	1,655	110
5	310	20	1,965	98
5	255	25	2,220	89
5	205	30	2,425	81
5	160	35	2,585	74
5	125	40	2,710	68
5	100	45	2,810	62
5	150	50	2,960	59
5	140	55	3,100	56
5	135	60	3,235	54
5	150	65	3,385	52

Filter Cake Product:
- Wet Weight, g: 1,422.5
- Dry Weight, g: 1,073.5
- Cake Solids, %: 75.5
- Filter Rate, lb/ft^2-hr: 6.61

Note: These data were used by Passavant Corporation to size filter equipment as given in the tables following Test 6.

[1]Pulp diluted to obtain proper flocculation.

EXHIBIT 13

Laboratory Pressure Filter Test 6

Sample: Thickener underflow pulp from Preparation Plant L, CSMRI Sample 17.

Test Data:
- Feed Pulp Solids, %: 24.9[1]
- Filter Pressure, psig: 225
- Flocculants, lb/ton solids: Percol 722 = 0.4, Percol 727 = 0.2

Filtration Data

Direct		Cumulative		
Time min	Volume ml	Time min	Volume ml	Rate ml/min
2	130	2	130	65
3	105	5	245	47
5	140	10	385	39
5	100	15	485	32
5	85	20	570	29
5	85	25	655	26
5	75	30	730	24
5	70	35	800	23
5	65	40	865	22
5	65	45	920	20
5	60	50	980	20
5	60	55	1,040	19
5	50	60	1,090	18

Filter Cake Product:
- Wet Weight, g: 763.8
- Dry Weight, g: 454.5
- Cake Solids, %: 59.5
- Filter Rate, lb/ft^2-hr: 3.03

[1]Pulp diluted to obtain proper flocculation.

EXHIBIT 13

Equipment Sizing Performed by
Passavant Corporation

Based on treatment of 100 tph of coal refuse thickener underflow solids.

Preparation Plant:	C	J	K	L
CSMRI Sample No.:	14	15	16	17
Slurry Sample Solids, %:	19.1	30.7	40.8	24.9
Slurry, sp gr:	1.070	1.115	1.227	1.270

Refuse Solids Dewatered Daily

Refuse Production, day/wk:	7	7	7	7
Refuse Dewatering, day/wk:	7	7	7	7
Refuse Dewatered, ton/hr:	100	100	100	100

Refuse Conditioning (for Sample 17 only)

Lime as 100% CaO: 4%; Polymer: 0.04%

Total Refuse Solids (for Sample 17 only)

Refuse Dewatered, ton/hr:				100
Lime as 100% $Ca(OH)_2$, lb/day:				192,000
Polymer, lb/day:				1,920
Total Solids, lb/day:				4,993,920

Filter Cake Calculations

Total Solids, lb/day:	4,800,000	4,800,000	4,800,000	4,993,920
Anticipated Cake Solids, %:	73.3	77.0	76.0	75.5
Total Filter Cake Weight, lb/day:	6,548,431	6,233,766	6,315,790	6,614,464
Filter Cake Density, lb/ft^3:	99	105	124	120
Filter Cake Volume, ft^3/day:	66,146	59,369	50,934	55,121

Operating Data

Operating Time, hr/day:	24	24	24	24
Filtration Time, min:	45	10	45	65
Turnaround Time, min:	15	15	15	15
Total Filter Cycle, min:	60	25	60	80
Total Filter Cycles:	24	57	24	18

Filter Design

Filter Cake Volume, ft^3/day:	66,146	59,369	50,934	55,121
Total Filter Cycles:	24	57	24	18
Filter Cake, ft^3/cycle:	2,756	1,042	2,122	3,063
Design Capacity of Each Filter, %:	25	50	33	20
Required Volume per filter, ft^3:	689	521	707	613

EXHIBIT 13

Equipment Sizing -- Continued

Preparation Plant	C	J	K	L
Filter Cake Thickness, in.:	1.57	1.57	1.57	1.57
Filter Volume per Chamber, ft^3:	5,086	5,086	5,086	5,086
Filter Chambers Required:	136	102	140	121
Number of Filters Required:	4	2	3	5
Filter Model:	20	20	20	20
Filter Max. Chambers Capacity:	150	150	150	150
Refuse Volume Per Day				
Refuse, gpd:	2,816,164	1,681,365	1,149,112	948,671
Lime, gpd:	--	--	--	306,954
Polymer, gpd:	--	--	--	115,108
Total, gpd:	2,816,164	1,681,365	1,149,112	1,370,733

EXHIBIT 14

BELT PRESS

Purpose:

To determine coal refuse dewatering characteristics using a belt press.

Sample:

Thickener underflow coal refuse from Preparation Plants, B, C, J, K, and L identified as CSMRI Samples 13 through 17.

Procedure:

The Arus-Andritz mobile laboratory belt press was used for all tests performed. Testing was conducted at the Spokane Mining Research Center by Mr. Gary Andrews of Arus-Andritz with Mr. P.S. Jacobsen of CSMRI assisting.

The Arus-Andritz laboratory press simulates the three dewatering zones of the Standard Andritz Sludge Dewatering Machine (SCM). This allows accurate bench-scale testing of a sludge for amenability to SDM dewatering. Initially, beaker testing of various polymers is conducted to determine the charge characteristics of the sludge particles.

The flocculated sample is then tested in a series of steps, which simulate the progressing dewatering zones of the SDM. The gravity zone is simulated by pouring the flocculated slurry onto a sample of belt mesh (which is supported by the perforated metal rectangular tray) and allowing the free water to drain. An upper belt sample is placed over the sludge solids, followed by the upper tray, and hand pressure is applied, forcing out capillary water as in the wedge zone. The cake containing mesh is then passed around the S-roll of the lab press, between the supportive mesh and the roll, seven times to simulate the S7 configuration. The support belt tension is increased with each subsequent S-roll. This increases the area pressure applied to the cake with each roll. On the production SDM the belt tension remains constant while the roll diameter decreases as the cake progresses through the high pressure zone. Hence the same pressure effects are obtained on the lab unit as on the full scale SDM.

This dewatering method is used to evaluate SDM efficiency based on derived cake solids, drainage rate, solids capture; polymer consumption, etc. Varying one of these parameters while holding all other constant allows the effects of each to be observed.

This procedure yields accurate determinations for all essential parameters required for efficient operation of the production SDM.

For the laboratory evaluation, controlled pressures and times were utilized in order to accurately simulate the pressure application period in gravity and wedge sections and on each S-roll of the high pressure section. The times were based on a belt speed of 25 ft/min. Maximum applied pressure (sic: Pressure on seventh S-roll) was 12 psi.

EXHIBIT 14

The following two tables contain information as regards sample idenitfication and the polymers investigated during the test program.

Sample Identification and Information

Preparation Plant:	B	C	J	K	L
CSMRI Sample No.:	13	14	15	16	17
Total Solids, % DS[1]:	24.0	19.1	30.7	40.8	49.7
pH at Ambient:	10.52	8.25	8.10	6.58	8.44
Specific Gravity, g/cm^3:	1.140	1.070	1.115	1.227	1.270

[1]Dry solids.

Listing of Polymers Evaluated During Testing

Designation	Molecular Weight	Charge/Density
Hercules 847	V. High	Anionic/Med. High
Hercules 849	V. High	Cationic/High
Hercules 1018	V. High	Anionic/Low
Percol 725	V. High	Anionic/Low
Percol 757	V. High	Cationic/Med. High
Percol 788N	V. High	Cationic/V. High
Nalco 7763	High	Anionic/High
Cyanamid 1839A	High	Anionic/High

Results:

All results are contained in the following tables.

EXHIBIT 14

Summary of Individual Tests

Arus-Andritz Test No.	Preparation Plant	CSMRI Sample No.	Polymer	Polymer Consumption lb/ton Dry Solids	SDM Module	Belt Mesh	Belt Speed ft/min	Cake Thickness in.	Solids Capture %	Cake Moisture %	Average % Solids in Filtrate
1	K	16	1018	0.90	S7	9,912	25	1/4	96+	22.4	--
2	K	16	1018	0.75	S7	9,912	25	1/4	96+	26.3	--
3	K	16	1018	0.65	S7	9,912	25	1/4	96+	29.2	--
4	K	16	1018	0.75	S7	9,912	25	1/4	96+	30.7	0.09
5	K	16	849	--	--	--	--	--	--	--	--
6	K	16	847	0.50	S7	9,912	25	1/4	96+	27.9	--
7	K	16	847	0.55	S7	9,912	25	1/4	96+	30.1	--
8	K	16	847	0.40	S7	9,912	25	1/4	96+	30.2	--
9	B	13	Lime/847	30/0.70	S7	9,912	25	1/8	95+	38.8	--
10	B	13	Lime/847	30/0.70	S7	9,912	25	1/8	95+	39.7	--
11	B	13	Lime/1018	30/0.50	S7	9,912	25	1/4	96+	38.8	0.02
12	B	13	Lime/1018	30/0.60	S7	9,912	25	1/4	96+	38.8	--
13	L	17	849	2.50	S7	9,912	25	1/4	95+	28.7	--
14	J	15	1018	0.55	S7	9,912	25	1/4	96+	27.3	--
15	J	15	1018	0.40	S7	9,912	25	1/8	95+	28.9	0.07
16	J	15	1018	0.55	S7	9,912	25	1/4	96+	27.9	--
17	C	14	847	0.45	S7	9,912	25	1/8	96+	29.6	--
18	C	14	847	0.45	S7	9,912	25	1/4	96+	29.4	--
19	C	14	847/849	0.25/0.15	S7	9,912	25	1/8	96+	29.4	0.07
20	C	14	847	0.25	S7	9,912	25	1/4	96+	29.1	--
21	C	14	847	0.45	S7	9,912	25	1/4	96+	30.4	--
22	C	14	847	0.25	S7	9,912	25	1/4	96+	30.7	--
23	L	17	757	1.30	S7	9,912	25	1/4	96+	28.0	
24	L	17	757	1.00	S7	9,912	25	1/4	96+	28.0	0.17
25	L	17	757	1.25	S7	9,912	25	1/4	96+	29.2	--
26	L	17	757	1.30	S7	9,912	25	1/4	96+	28.7	--
27	L	17	757	1.00	S7	9,912	25	1/4	96+	28.7	--
28	L	17	757	1.30	S7	9,912	25	1/4	96+	29.0	--

EXHIBIT 14

Summary of Test Results and Recommendations by Arus-Andritz

CSMRI Sample No.	Prepration Plant	Additions: Polymer	Additions: Polymer lb/ton	Additions: CaO lb/ton	SDM Module	Belt Mesh	Cake Solids %	Cake Thickness in.	Solids Capture %	Throughput tph/meter of Belt Width
13	B	Hercules 1018	0.5-0.70	30	S7[1]	9,912	60-65	0.25	96+	6.0
14	C	Hercules 847	0.25-0.45	--	S7	9,912	69-71	0.25	96+	5.4
15	J	Hercules 1018	0.4-0.65	--	S7	9,912	70-72	0.25	96+	6.1
16	K	Hercules 847	0.5-0.75	--	S7	9,912	68-72	0.25	96+	7.3
17	L	Percol 757	1.0-1.3	--	S	9,912	70-72	0.25	96+	8.9

[1]The SDM 3500 S7 has a belt width of 3.5 meter.

EXHIBIT 15

BELT PRESS NO. 2

Purpose: To determine coal refuse dewatering characteristics using a belt press.

Samples: Thickener underflow coal refuse from Preparation Plants B, C, J, K, and L identified as CSMRI Samples 13 through 17.

Procedure: A single 5-gal pail of each bulk sample was sent to Parkson Corporation (5601 N.E. 14th Ave., P.O. Box 24407, Fort Lauderdale, Florida, 33307) for test work. Investigations were carried out using the Parkson MPL 0.25 m wide press unit and standard Parkson test procedures. No representative of CSMRI was present during any of the proceedings.

Results: Provided in the following table.

EXHIBIT 15

Results of Belt Press No. 2 Investigation

CSMRI Sample No.	13	14	15	16	17
Preparation Plant	B	C	J	K	L
Pulp Solids; Weight, %	21.43	16.21	28.76	37.29	48.00
pH	9.4	7.2	6.8	5.3	7.7
Pretreatment Recommendation:					
CaO, lb/ton	30	--	--	--	--
Flocculant	Allied Colloids 730	American Cyanamid 208	American Cyanamid 208	American Cyanamid 208	Allied Colloids 725
Dosage, lb/ton	2.5	1.2	0.3	0.9	0.5
Unit Capacity; gpm/meter of width	30	67	44	21	20
lb/hr-meter of width	3,640	5,800	7,000	4,850	5,900
Belt Product Solids, %[1]	50	60	65	58	68
Belt Product Solids, %[2]	60	70	70	65	79
Solids Recovery, %	98	98	98	98	99
Solids, sp gr	1.131	1.061	1.109	1.238	1.230

[1]MPL unit low pressure.
[2]MP unit high pressure.

EXHIBIT 16

CENTRIFUGE

Purpose: To determine coal refuse dewatering characteristics using a solid bowl continuous centrifuge.

Samples: Thickener underflow coal refuse from Preparation Plants B, C, J, K, and L identified as CSMRI Samples 13 through 17.

Procedure: The tests were conducted using a Bird Machine Comapny Model LB-1400 solid bowl continuous centrifuge. Thickener underflow material was stored in 55-gal drums until added to the 350-gal Fiberglas feed tank, agitated with a two blade mixer. These samples had been settling in the 55-gal drums for periods up to 3 months. Mixing a batch of feed material would take several hours due to the difficulty in resuspending the drums and adding them to the tank. Once mixed, this coal refuse was pumped with a Moyno 1L6 positive displacement pump.

Polymer was mixed in a 20-gal barrel and metered with a Randolph pump. Polymer injection was at the feed pump inlet except for Test 6 where it was internal to the feed pipe. Cake was dumped into a cut down 55-gal drum and saved for dryer studies.

Centrate discharged by gravity. A run of 200 ft through 1½ in. flexible hose with a head of only 4 ft sometimes imposed a restriction to centrate flow.

Flow measurements were determined by measuring tank drop for feed rate and weighing polymer barrel before and after a run for polymer rate. Runs were from 2 to 3 hr in duration.

No dilution water was added.

Results: Test details as well as results are presented in the following tables.

EXHIBIT 16

Test No.	Feed: Solids %	Feed: gpm[1]	Feed: Dry Solids lb/hr	Feed: Temperature	Feed: sp gr	Cake Solids %	Centrate Solids %	Solids Recovery %	Flocculant, Nalco 8873: Solution %	Flocculant, Nalco 8873: gpm[1]	Flocculant, Nalco 8873: 100% Liquid lb/hr	Flocculant, Nalco 8873: lb/ton	Centrifuge Conditions: Bowl Speed rpm	Centrifuge Conditions: Gravity	Centrifuge Conditions: Slip/Gear Ratio	Centrifuge Conditions: Pool Depth
Sample: CSMRI 15, Preparation Plant J																
1	29.8	4.70	841	Ambient	~1.2	67.6	0.319	99.4	0.5	0.17	0.43	1.01	--	--	--	--
2	29.8	5.48	895	Ambient	~1.2	66.2	0.360	97.3	0.5	0.13	0.33	0.67	2,350	1,100	10.4	X-2
3	28.4	6.26	1,068	Ambient	~1.2	70.0	0.776	98.4	0.5	0.01	0.03	0.05	2,350	1,100	15.4	X-2
4	28.4	6.26	1,068	Ambient	~1.2	68.0	0.929	98.1	0.5	0.09	0.23	0.43	2,350	1,100	19.9	X-2
5	28.4	5.48	935	Ambient	~1.2	67.1	0.554	98.9	0.5	0.31	0.78	1.66	2,350	1,100	11.0	X-2
6	28.4	6.26	1,068	Ambient	~1.2	66.6	0.190	99.6	0.5	0.14	0.35	0.66	2,350	1,100	11.0	X-2
7	29.5	12.4	2,197	Ambient	~1.2	68.9	0.180	99.7	0.5	0.28	0.70	0.64	2,350	1,100	13.6	X-2
8	29.5	13.4	2,374	Ambient	~1.2	68.4	0.100	99.8	0.5	0.28	0.70	0.59	2,350	1,100	20.0	X-2
9	29.5	13.1	2,321	Ambient	~1.2	65.6	0.800	98.5	0.5	0.96	2.40	2.08	2,350	1,100	20.0	X-2
Sample: CSMRI 14, Preparation Plant C																
10	18.0	13.2	1,308	Ambient	~1.1	68.5	0.020	99.9	0.5	0.31	0.78	1.18	2,350	1,100	13.0	X-2
11	18.0	12.3	1,219	Ambient	~1.1	67.8	0.170	99.3	0.5	0.19	0.48	0.78	2,350	1,100	13.0	X-2
12	18.0	14.5	1,437	Ambient	~1.1	68.0	0.090	99.6	0.5	0.12	0.30	0.42	2,350	1,100	13.0	X-2
13	18.0	13.4	1,328	Ambient	~1.1	69.3	0.170	99.3	0.5	0.07	0.18	0.26	2,350	1,100	13.5	X-2
14	17.1	4.7	442	Ambient	~1.1	67.2	0.020	99.9	0.5	0.07	0.18	0.79	2,350	1,100	11.1	X-2
15	17.1	23.5	2,212	Ambient	~1.1	67.3	0.010	100.0	0.5	0.18	0.45	0.41	2,350	1,100	22.1	X-2
Sample: CSMRI 13, Preparation Plant B[2]																
16	22.7	5.34	685	Ambient	1.13	51.0	0.070	99.8	0.5	0.13	0.33	0.95	2,350	1,100	11.1	X-2
17	22.7	5.32	683	Ambient	1.13	52.2	0.140	99.7	0.5	0.14	0.35	1.03	2,350	1,100	15.0	X-2
18	22.7	4.89	628	Ambient	1.13	57.5	0.450	99.1	0.5	0.22	0.55	1.75	2,350	1,100	15.0	X-2
19	22.7	5.09	653	Ambient	1.13	54.3	0.080	99.8	0.5	0.44	1.10	3.37	2,350	1,100	15.0	X-2
20	22.7	15.7	2,015	Ambient	1.13	54.9	1.930	94.8	0.5	0.21	0.53	0.52	2,350	1,100	15.0	X-2
Sample: CSMRI 16, Preparation Plant K																
21	37.4	4.70	1,073	Ambient	1.22	60.8	0.030	100.0	0.5	0.17	0.43	0.80	2,350	1,100	15.0	X-2
22	37.4	4.70	1,073	Ambient	1.22	60.4	0.100	99.9	0.5	0.11	0.28	0.52	2.350	1,100	15.0	X-2
23	37.4	5.09	1,162	Ambient	1.22	59.6	0.070	99.9	0.5	0.26	0.65	1.12	2,350	1,100	11.1	X-2
24	37.4	4.27	975	Ambient	1.22	59.5	0.070	99.9	0.5	0.18	0.45	0.93	2,350	1,100	11.1	X-2
25	37.4	10.2	2,329	Ambient	1.22	59.5	0.120	99.9	0.5	0.29[3]	0.73[3]	0.62[3]	2,350	1,100	11.1	X-2
Sample: CSMRI 17, Preparation Plant L																
26	50.2	5.22	1,574	Ambient	~1.2	68.6	0.50	99.7	0.5	0.17	0.42	0.65	2,350	1,100	11.1	X-2
27	50.2	12.3	3,708	Ambient	~1.2	74.9	15.8	86.8	0.5	0.32	0.80	0.52	2,350	1,100	11.1	X-2

[1]Gallon per minute.
[2]Added 30 lb/ton lime.
[3]Flocculant may have run out before sample was taken.

EXHIBIT 17

THERMAL DRYER

Purpose: To determine the dewatering performance of a thermal dryer on partially dewatered coal refuse sludges.

Sample: Partially dewatered samples of coal refuse from Plant B, C, J, K, and L; CSMRI Sample 13 through 17.

Procedure: The Envirotech thermal dryer was run with the partially dewatered filter cakes from the Bird solid bowl centrifuge pilot plant detailed in Exhibit 16. Five runs were made through the processor: one run with cake products from each of the five samples available.

Solids were fed to the dryer using an inclined belt calibrated by setting a belt speed and weighing the drop off. The material was leveled by an adjustible bar. The feed rate was adjusted for each sample. Product samples were taken on a continuous basis to insure uniformity. A plastic container was fitted on the dryer discharge nozzle and a time sample taken over a 10 min period.

Results: The following tables contain test details, results, and calculated data provided by Envirotech representatives.

Test Data for Thermal Dryer

CSMRI Sample No.	Preparation Plant	Clock Time	Sample Feed = F Product = P	Cake Weight Wet g	Cake Weight Dry g	Contained Water %
16	K	08:00	F	748.0	463.2	38.0
16	K	08:24	P	670.5	661.7	1.3
16	K	08:37	P	537.7	531.7	1.1
16	K	09:03	F	1,372.0	862.6	37.1
15	J	10:00	F	767.5	537.7	30.0
15	J	11:00	F	2,309.4	1,608.5	30.4
15	J	11:28	P	740.2	709.6	4.1
15	J	11:45	F	2,000.0	1,369.8	31.5
13	B	14:40	F	2,089.1	1,173.3	43.8
13	B	14:50	P	678.9	629.7	7.2
14	C	08:10	F	1,149.6	791.8	31.1
14	C	08:20	P	954.5	880.4	7.0
17	L	09:25	F	1,013.9	768.8	24.2
17	L	09:45	P	608.1	571.0	6.0
17	L	10:05	P	663.7	648.7	2.2

EXHIBIT 17

Thermal Drying Tests -- Coal Refuse

Run No.	Drum No.	Time	Feed Rate Wet lb/hr	H_2O %	Product Rate lb/hr	H_2O %	Oil Temperature °F In	Oil Temperature °F Out	Oil Flow gpm
1	K-21/22/23	0800	120	38.00	75.5	1.31	395	386	16.3
		0815	123	--	88.0	--	395	386	16.3
		0830	123	--	70.0	--	396	384	16.3
		0907	123	37.12	54.0	1.11	396	384	16.3
		0917	126	--	56.0	--	396	382	16.3
		0927	124	--	54.0	--	393	382	16.3
		0947	--	--	54.0	--	396	382	16.3
		1007	--	--	54.0	--	392	382	16.3
2	J-5-6	1100	107	30.36	--	--	--	--	--
		1130	150	29.96	108.0	4.13	398	388	16.3
		1145	150	31.50	108.0	4.30	388	377	16.3
		1155	150	--	108.0	--	388	377	16.3
		1205	150	--	112.0	--	390	374	16.3
3	B16-17	1330	132	--	84.0	--	388	372	16.3
		1435	134	43.8	84.0	7.24	388	374	16.3
		1445	132	--	84.0	--	389	375	16.3
		1505	132	--	82.0	--	391	376	16.3
4	C-11-12-13	0815	152	31.10	88.0	7.00	390	384	16.3
		0836	152	--	90.0	--	396	384	16.3
		0846	152	--	101.0	--	395	385	16.3
5	L-26-28	0937	132	--	78.0	--	392	384	16.3
		0947	132	24.18	84.0	--	395	384	16.3
		0957	132	--	84.0	2.00	395	384	16.3

EXHIBIT 17

Thermal Drying Tests -- Coal Refuse
Consolidation of Dryer Heat Transfer Rate

Time	Sample No.	Thermal Duty Btu/hr	Driving Force, °F Sensible	Latent	Weighted	Transfer Coefficient Btu/hr/ft^2/°F	Remark	Moisture Inlet	Discharge
0800	K	53,692	283.6	175	186	15.11	--	38.00	1.31
1130	J	50,978	241.0	178	192	13.90	--	29.96	4.13
1435	B	62,904	220.0	168	179	18.39	--	43.80	7.24
0845	C	51,962	230.0	175	188	14.48	Dusty-material. Balance so indicates.	31.10	7.00
0947	L	37,730	237.0	175	191	10.34	Dusty	24.18	2.20