RESPONSE TO PARAMETER VARIATION OF A ONE-INCH DIAMETER HYDROCYCLONE FOR PYRITIC SULFUR REMOVAL

by

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Dissertation submitted to the Graduate Faculty of the

Virginia Polytechnic Institute and State University

in partial fulfillment of the requirements for the degree of

DOCTOR OF PHILOSOPHY

in

Environmental Sciences and Engineering

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October, 1975

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ACKNOWLEDGMENTS

Grateful acknowledgments is hereby made to the Division of Minerals Engineering, Virginia Polytechnic Institute and State University for the financial support of this project.

Also, appreciation is expressed to the members of the author's committee: Dr. William E. Foreman, Chairman of the committee, and Associate Professor, Minerals Engineering; Dr. J. Richard Lucas, Head, Division of Minerals Engineering; Dr. Paul King, Chairman, Environmental Sciences and Engineering; Dr. Thomas Stephens, Department of Civil Engineering; Dr. Charles Sears, Department of Geology, and Dr. William Payne, Department of Civil Engineering.

The author would like to recognize the invaluable contributions of Mr. Wayne Slusser, Instrument Maker, Division of Minerals Engineering, who provided technical support.

Finally, the author owes a special debt to her advisor, Dr. William E. Foreman, for his guidance, suggestions, and constructive criticism during the course of this investigation.

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I. INTRODUCTION

In recent years there has been a growing concern about the quality of the environment. Realization that technological growth has resulted in degradation of air, water, and land quality has caused pressure to be brought to bear on Congress and industry to correct the situation. Congress responded by creating the Environmental Protection Agency and by enacting laws designed to restore and protect the environment. Industry has started development of the technology required to comply with these new laws.

One of the targeted pollutants under these new laws is oxides of sulfur. Fossil-fuel generating stations account for approximately 60 per cent of the yearly emissions of oxides of sulfur in the United States. Figure 1 shows the major sources of sulfur oxides in the United States.(1)

Fossil-fuel electric plants have taken a two-fold approach in attempting to meet the sulfur emissions requirements. First, they used low sulfur fuels, and many plants have been converted from coal to oil or natural gas. In late 1973 the availability of these fuels was restricted, due to political factors, and the cost of these fuels increased dramatically. Other steam generation stations switched to low-sulfur coals. These coals are in limited supply in the eastern half of the nation, while the low sulfur western coals are economically unattractive because of their low BTU content and the high transportation costs charged to bring these coals to eastern coal-burning steam generation stations.

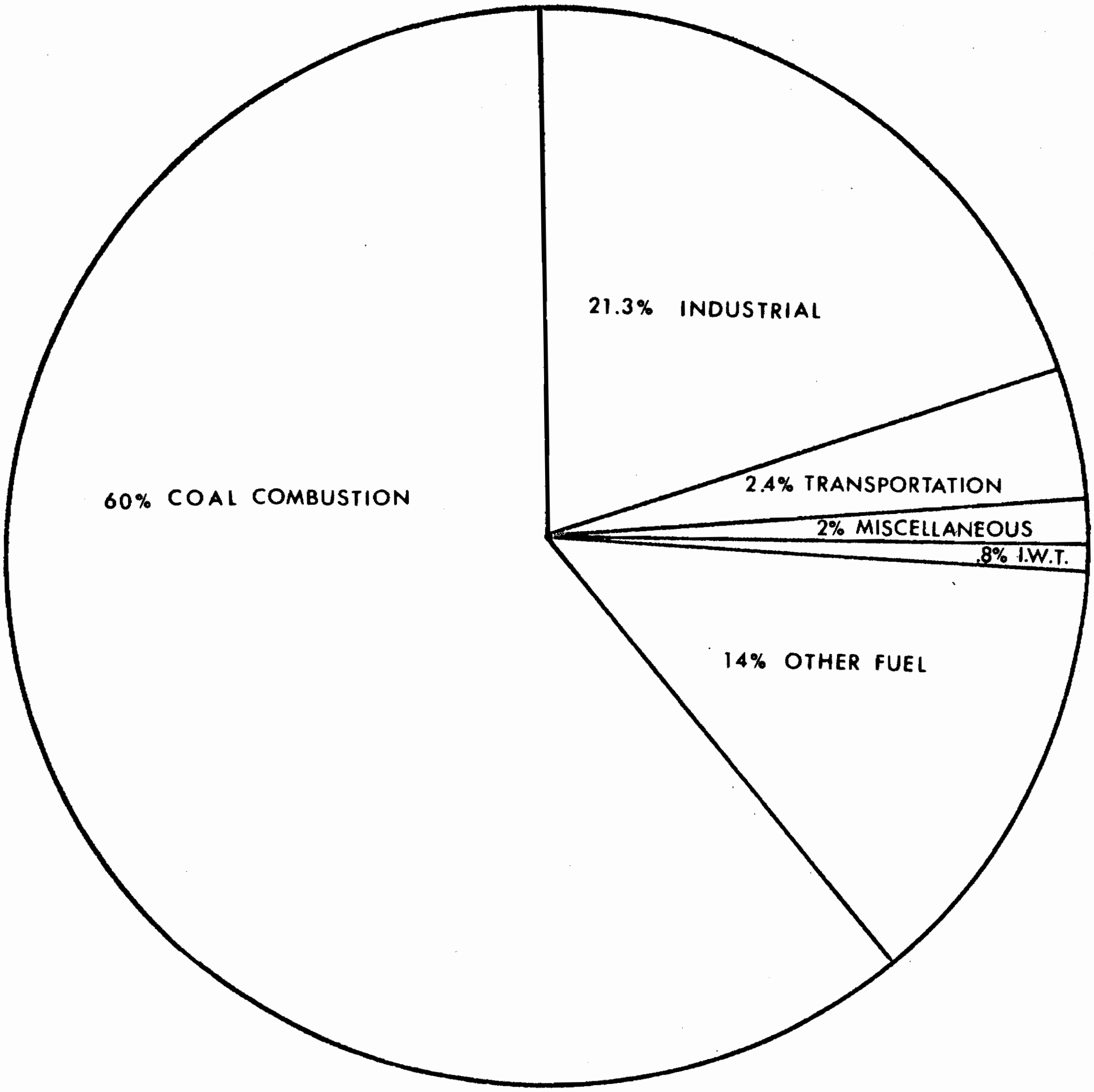


FIGURE 1 - SOURCES OF OXIDES OF SULFUR

The second approach was to add air pollution control equipment. Hot stack gases were routed through these devices in order to remove the pollutants. High capital costs and maintenance costs are associated with these devices with long delivery times. Many utility companies feel that these devices are not technically feasible.

Coal is one of the indigenous fuels which will enable us to meet our expanding energy requirements. These energy demands must be met without impairing air quality; therefore, it is necessary to develop an economic method of removing sulfur from coal.

Three forms of sulfur are found in coal: sulfate sulfur, organic sulfur, and pyritic sulfur. In an analysis the total sulfur, sulfate sulfur and pyritic sulfur are determined directly. The organic sulfur is determined by the difference between the total sulfur and the sulfate and pyritic constituents.(2)

Sulfate sulfur usually occurs in conjunction with calcium or iron. As sulfate sulfur accounts for only a small fraction of a per cent of the total sulfur, it is of minor importance.(2)

Sulfur also occurs in coal as an organic combination. Given and Wyss stated that organic sulfur is present as: mercaptan or thiol, sulfide or thio-ether, disulfide, or aromatic systems containing the thiophene ring.(3) Organic sulfur is chemically bound to the coal. Large amounts of energy must be supplied in order to break these bonds. Therefore, the organic sulfur content represents the theoretical lowest limit to which a coal can be cleaned by physical methods. Table 1 lists 17 American coals together with the total sulfur content and the

TABLE 1 - FORMS OF SULFUR IN VARIOUS COALS

Coal Seam	Total Sulfur %	Pyritic Sulfur %	Organic Sulfur %	Pyritic Sulfur as % of Total Sulfur
Pittsburgh	1.13	0.35	0.78	31.0
Upper Freeport	3.56	2.82	0.74	79.9
Thick Freeport	0.92	0.46	0.45	51.1
B	0.78	0.19	0.57	26.9
C Prime	2.00	1.43	0.54	73.0
Miller	1.25	0.56	0.65	48.0
No. 6	2.52	1.50	1.02	59.5
No. 9	3.28	1.05	2.23	32.0
No. 12	1.48	0.70	0.78	41.3
Freeburn	0.46	0.13	0.33	29.3
Elkhorn	0.68	0.13	0.51	25.0
Pocahontas No. 3	0.55	0.08	0.46	17.4
Eagle	2.48	1.47	1.01	59.3
Pratt	1.72	0.97	0.72	59.1
Lower Mercer	3.92	2.13	1.79	54.3
Deep River	2.32	1.52	0.80	65.5
8 - A	2.51	1.61	0.86	65.7

Leonard J., and Mitchell D., Coal Preparation, A.I.M.E., Baltimore, (1968), I-47.

contribution of the pyritic and organic sulfur to this total.

The pyritic sulfur refers either to pyrite or marcasite. These two minerals have the same chemical formula, FeS_2, but have different crystalline forms. Pyrite is cubic while marcasite is orthorhombic. In American coals the pyrite structure predominates with the size of the pyritic particles ranging from nodules several feet in diameter to finely disseminated particles.[(2)] Once liberation has been achieved, a physical cleaning method may be used to separate the pyritic sulfur from the coal. Many coals liberate at approximately 65 mesh, but at present no cleaning device exists to remove pyritic sulfur from 65 x 0 mesh coal.

A review of the literature indicates that a small diameter hydrocyclone may be suited for separating pyritic sulfur from coal. Figure 2 shows the results of experimental work performed by Visman.[(4)] As the diameter of the hydrocyclone is decreased, the probable error is decreased. Figure 3 shows the results obtained for a 2.0" diameter hydrocyclone by LePage.[(5)] Extrapolation of the data indicates that a hydrocyclone of 1.0" in diameter would be suited for cleaning coal ranging in size from minus 28 mesh down to plus 10 microns.

A 1.0" diameter hydrocyclone was constructed from 303 stainless steel, and a number of interchangable parts were fabricated for the hydrocyclone. These parts allowed for variation of the inlet orifice diameter, the apex discharge diameter, the cone angle, the vortex finder diameter, and the vortex finder position. In addition, the feed pressure could be varied. Tests of the performance of the hydrocyclone with various combinations of parts were performed.

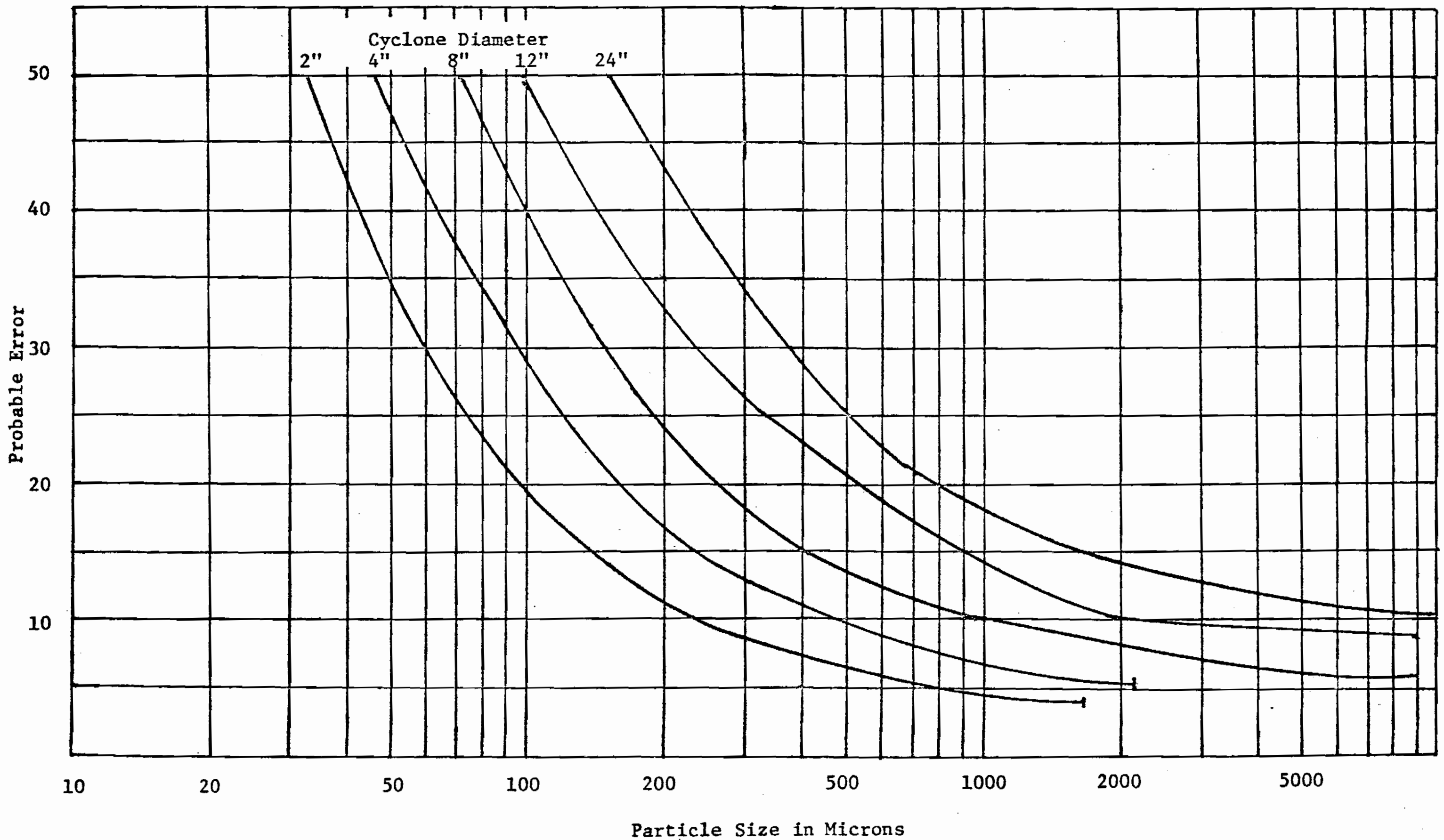


FIGURE 2 - PROBABLE ERRORS OF DIFFERENT DIAMETER HYDROCYCLONES

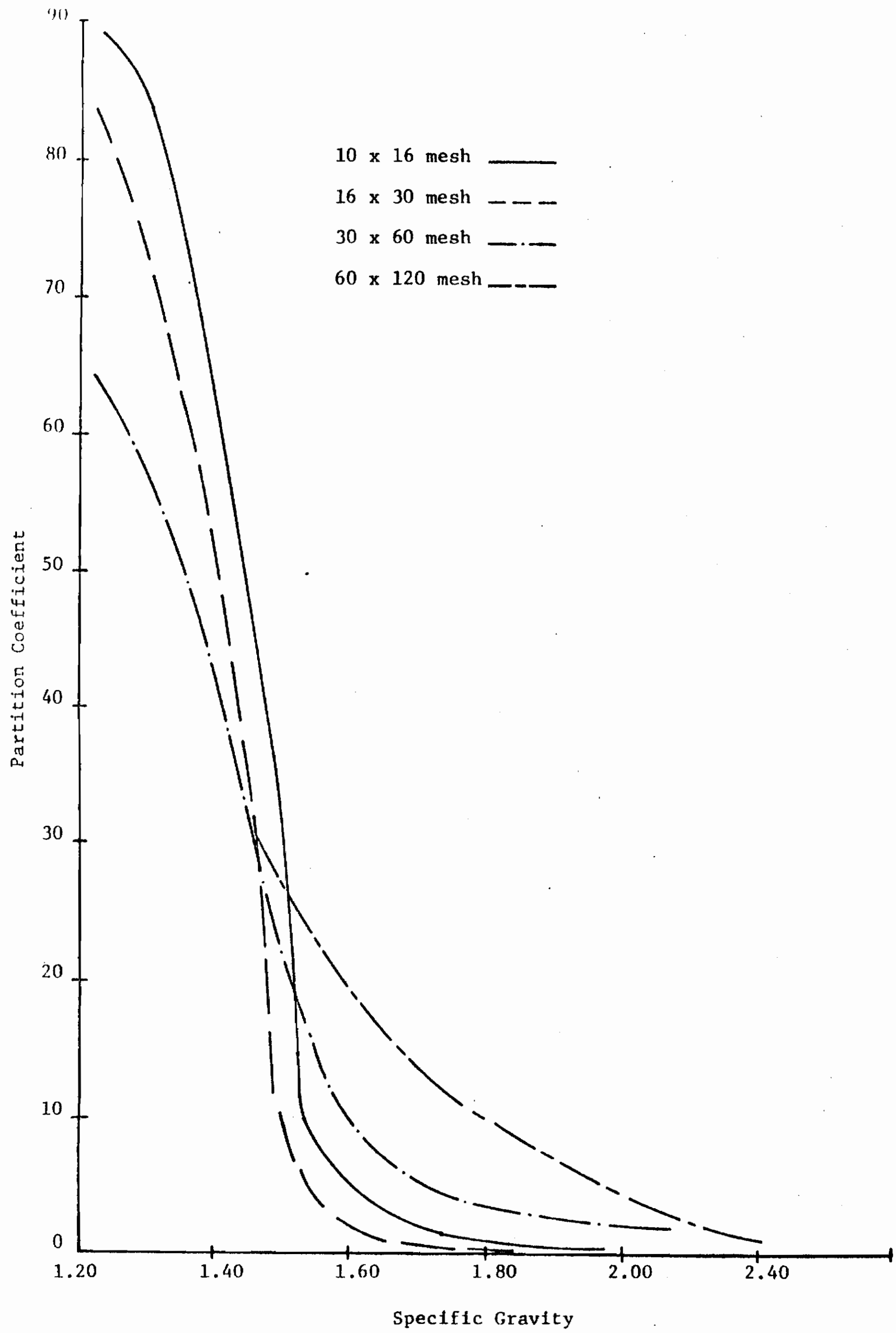


FIGURE 3 - PARTITION CURVES FOR 2" HYDROCYCLONE

The results of these tests were plotted as pyritic sulfur rejection versus the geometry of the hydrocyclone. The combination which yielded the greatest pyritic sulfur rejection value was considered the optimum configuration.

II. LITERATURE REVIEW

Definition of hydrocyclone

For the purpose of this investigation, a small diameter hydrocyclone is defined to be a steel cylinder 1.0" in diameter with a tangential feed inlet. The vortex finder extends through one closed end of the unit and an apex serves as the discharge.(6)

Factors which influence particle movement in a hydrocyclone

There are a number of factors which influence the movement of particles in a hydrocyclone. Dreissen and Fontein in their work on cyclones have determined the following factors to be of importance:(7)

1. Shape of the particles
2. Solids concentration
3. Feed pressure
4. Back pressure
5. Cyclone diameter
6. Diameter of the feed opening
7. Overflow opening
8. Apex opening
9. Length of the vortex finder
10. Cone angle
11. Specific gravity
12. Average grain size of the particles
13. Solids concentration of the inlet feed and apex discharge

14. Viscosity of the feed suspension and the liquid

Visman has stated that the diameter of the cyclone is a significant factor in determining the operating characteristics of the device. For design considerations, certain operating characteristics of the cyclone may be expressed as functions of the cyclone diameter. They are as follows, where D is the diameter of the cyclone.[8]

Maximum particle size of coal processed.........D/10

Recommended maximum particle size (allowing for 15% oversize).....................D/10

Gravity cutpoint control is good over the specific gravity range of 1 to 2.5 for particles of greater diameter than..............D/300

The specific gravity cutpoint increases and the cutpoint range decreases as particle diameter decreases below..........................D/300

Theories of hydrocyclone operation

Investigators have tried to determine the mechanism of gravity separation which occurs inside of the hydrocyclone. The precise mechanism of separation in a hydrocyclone is not known at this time. A review of the literature reveals three major concepts. In general, it is held that inside of the hydrocyclone a dense media is formed. Particles of intermediate and high specific gravity migrate to, and circulate in the conical section. These particles form a dense medium through which the lighter particles cannot penetrate. These lighter particles are entrained in the ascending vortex and report to the overflow. The refuse, composed of heavier material, is able to penetrate the dense medium and report to the underflow discharge.[9]

Fontein and Dijksman have noted that the dense medium cannot account for the separation because, in the absence of high gravity material, the low gravity fraction of the feed still reports to the overflow.[10] They believe that the particles in a hydrocyclone are separated during their residence in the central region. While in the ascending vortex, the particles are subjected to centrifugal forces. During the initial period of acceleration, the influence of particle size on settling velocity is minimal. The equation for the centrifugal force acceleration on a particle may be expressed as follows:

$$a = \frac{(M_p - M_f)}{M_p} \cdot \frac{V^2}{R}$$

where:

a = acceleration of particle

M_p = mass of the particle

M_f = mass of fluid displaced by particle

V = tangential velocity

R = radius of cyclone

For particles in the same plane, this equation may be simplified and rewritten as:

$$a = C(1 - \frac{1}{SG})$$

where:

C = constant

SG = specific gravity[9]

It may be seen that a dense particle will have a greater acceleration than a less dense particle. For example, consider a coal particle with a specific gravity of 1.4 and a shale particle, of equal size, with a specific gravity of 2.6. The acceleration of the coal particle is 0.286C, while the acceleration of the shale particle is 0.615C. Therefore, the dense particle is more likely to migrate out of the ascending vortex and report to the apex discharge.

A basic flaw in this theory is that it is only applicable to the larger-sized particles. For very fine particles, the drag of the fluid cannot be neglected. For the larger particles the time required for fluid drag to influence particle movement is more than the time required for the movement of these particles to their proper product stream. The very small particles, however, tend to report to the clean-coal overflow.[10]

G. Tarjan presents a differing viewpoint. He proposes that a plane exists within the hydrocyclone in which equal size and specific gravity particles are in equilibrium with respect to centrifugal mass and centripetal drag forces.[11] Figure 4 shows the location of these planes for a hypothetical case. The higher density particles accumulate along the inner wall of the hydrocyclone. Tarjan postulates that this amassment of particles functions as a dense medium. It is this mechanism which is responsible for the gravity separation in a hydrocyclone. It is important to note the line $v_a = 0$, on Figure 4, which represents the points where the axial velocity equals zero. According to Tarjan this is the boundary between those particles which will report to the overflow and those which will report to the underflow discharge. The solids in equilibrium at this point give the limit

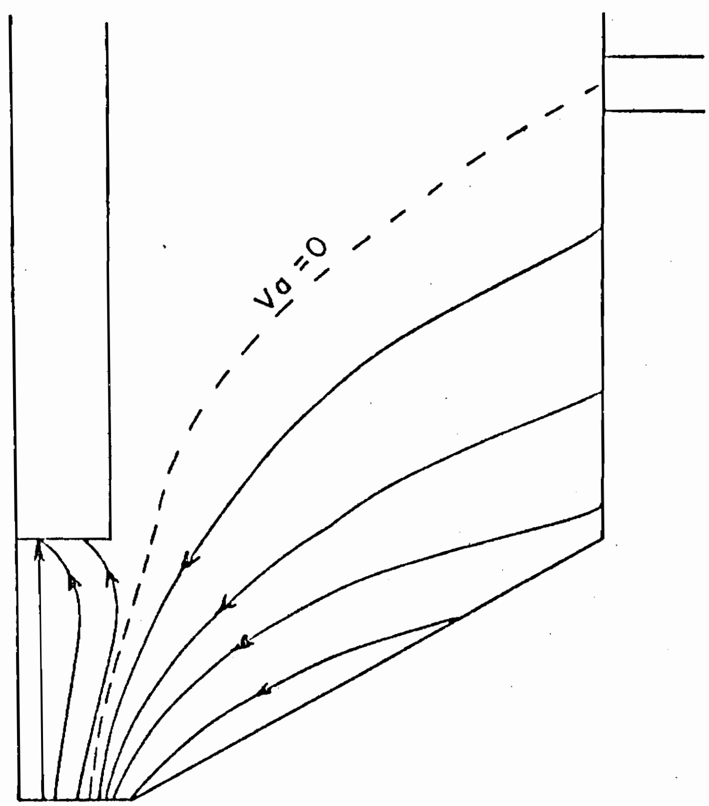


FIGURE 4 - CROSS SECTION OF HYDROCYCLONE SHOWING TARJAN'S EQUILIBRIUM PLANES

grain size of the separation. The specific gravity of the medium formed is the specific gravity of separation. Tarjan's theory does not explain separations at densities higher than 1.6. Such separations have been found to exist by other investigators.[12]

Visman feels that the mechanism of separation in a hydrocyclone is similar to that of the rheolauver. He assumes that the particles stratify into a bed according to specific gravity along the inner wall of the cone. This bed forms a barrier to the passage of large particles of low density which cannot penetrate this bed, and these large particles of low density then report to the overflow. The fine particles of higher density are also unable to penetrate the bed and report to the overflow. The large particles of high specific gravity report to the apex discharge.[13]

Two types of hydrocyclones

Hydrocyclones may be divided into two catagories: those which classify particles on the basis of size and those which separate particles on the basis of specific gravity. The main difference between the two types of hydrocyclones is the geometry of the cone. Classifying hydrocyclones have very steep cone angles. The angle of the cone is greater than 60 degrees. Hydrocyclones which perform a separation on the basis of specific gravity are characterized by cones with angles of around 40 degrees.[13]

Findings of Visman and LePage

J. Visman has worked extensively on the application of hydrocyclones to the beneficiation of coal. Figure 2 shows probable errors for the various size fractions of coal processed through different size hydrocyclones. Probable error is defined as one half of the specific gravity interval spanned by the distribution curve in passing from the 25 per cent to the 75 per cent recovery ordinates.[14] A good, or sharp, separation has a low probable error. Figure 2 shows that as the diameter of the hydrocyclone is decreased, smaller particles of coal may be efficiently cleaned. Extrapolation of this data indicates that a 1.0" diameter hydrocyclone will effectively clean minus 65 mesh coal.

LePage has also used hydrocyclones to process run-of-mine fine coal. Figure 3 shows partition curves for various size fractions of coal plotted against specific gravity. The results are for a 2.0" diameter hydrocyclone.[5] A partition coefficient is the weight of a specific gravity fraction in a product stream divided by the total weight in that specific gravity fraction in the feed. Figure 3 is a plot of the overflow or clean coal partition coefficient. This figure shows that a 2.0" diameter hydrocyclone can effectively clean 10 x 120 mesh coal.

The above data indicate that a 1.0" diameter hydrocyclone may be suited for cleaning minus 65 mesh coal. Pyritic sulfur will be liberated in minus 65 mesh coal. The specific gravity of pyrite ranges from 4.95 to 5.17.[15] Therefore, the pyritic sulfur should report with the ash to the refuse product stream.

III. PURPOSE OF THE INVESTIGATION

Coal fired electric generating stations are a major source of oxides of sulfur. State regulatory agencies have adopted strict emission standards which require the burning of low sulfur coal in order to achieve compliance. Since few coal seams meet the required sulfur standards, it is necessary to remove the excess sulfur during the beneficiation process. Of the three forms of sulfur found in coal, only pyrite may be removed by a physical cleaning method. No cleaning device exists to separate pyritic sulfur from fine coal; accordingly it is necessary to develope an economical and reliable device to remove pyritic sulfur from 65 x 0 mesh coal.

Partition curve data for larger hydrocyclones indicated that a 1.0" hydrocyclone may be suitable for removing pyrite from 65 x 0 mesh coal. Therefore, a 1.0" diameter stainless steel hydrocyclone with interchangable parts was constructed and various combinations of the components were tested. The configuration which yielded the maximum pyritic sulfur rejection was considered the optimum.

The large number of data points required for an optimization study required that a new technique be developed to replace the time-consuming conventional float-sink method of determining partition coefficients. A magnetically tagged tracer material was introduced into the system. A data point on the partition curve could be obtained in a few minutes by comparing the amount of tracer material in the overflow product to the amount present in the underflow.

If successful, this project would have tremendous impact in the electric utility and mining industries. Coal companies with reserves in coal seams which do not presently meet the sulfur requirements would be able to market these coals at an attractive price to power generating stations.

The magnetic tracer method of obtaining partition curve data will have great impact on the mining equipment industry. Designing and testing of new cleaning equipment will be facilitated by use of this technique. Preparation plant operators will be able to rapidly optimize plant performance to meet variations in the feed to the plant.

If the preliminary results of the investigation are encouraging, the next phase would involve a pilot plant installation of a hydrocyclone system. Should the pilot plant prove successful, full scale industrial trials would follow.

With further research and development, it would be possible to develop the magnetic tracer technique into a continuous monitoring device. By combining a process control computer with the magnetic sensing device, it would be possible to automate a coal preparation plant. This would allow the plant to be rapidly adjusted to changes in the composition of the feed and result in greater efficiency of plant operation.

IV. THE HYDROCYCLONE

Background

The hydrocyclone was developed by Fontein, and United States Patent, number 2,573,192, describes his invention. The Dutch State Mines, an independently operated corporation owned by the Netherlands, was an early leader in developing and distributing hydrocyclone systems. Significant work has been done in the field by the Australian Coal Research Laboratory, Ltd. Recently, Cyclones, Ltd. has entered the field and is marketing small diameter hydrocyclones. However, no application has yet been made of the hydrocyclone to the problem of removal of pyritic sulfur from coal.

Materials of construction

The best material from which to construct a hydrocyclone is 303 stainless steel. 303 stainless steel has excellent resistance to corrosion. This corrosion resistance is necessary because the intake water to a cleaning plant may be high in acidity and have a low pH. In addition, the pyritic sulfur may react in the plant circuits to produce acid water. Stainless steel also possesses excellent wear characteristics, which is necessary because the fine coal - water slurry is highly abrasive.

An alternate material of construction would be a corrosion resistant plastic with high abrasion resistance. The major advantage of plastic is the ease of fabrication, which will significantly lower the unit cost for individual hydrocyclones.

The need for a hydrocyclone

Current preparation plant practice is to treat fine coal in froth flotation cells. Problems associated with such cells fall into three main catagories: operation, economics, and size. For efficient operation the feed to the flotation cell must be uniform. The cell is conditioned to handle one type of coal, and this means that the plant can handle only one type of coal. If the coal is oxidized, flotation ceases, and the water to the cell must be free of organic material that would interfere with the flotation process. The costs associated with froth flotation cells are high, and the capital investment required is high. Maintenance costs are high, as periodic replacement of the skimmers, troughs, and other parts is required. The cost of reagents must also be considered. Finally, if the cell fails, the whole plant operation must be stopped. This plant stoppage will result in a large amount of lost revenue. Finally, froth flotation cells require much space. A smaller system would result in a smaller, less expensive preparation plant.

Hydrocyclones do not have the problems discussed above. First, the mechanism of the hydrocyclone is gravity separation, and the gravity of separation of the hydrocyclone can be rapidly changed by adjusting the vortex finder depth. This means one preparation plant can handle a blend of coals, or different coals at different times. Because the mechanism of separation is gravity, oxidized coal presents no problems. The initial cost of the hydrocyclone system is less than for a froth flotation cell system. Because there are no moving parts, maintenance costs are less. Reagent costs for the hydrocyclone are zero as none are required. If one hydrocyclone goes down, plant operation continues because other hydrocyclones are on line. A new hydrocyclone could be installed within minutes. Because of their small size and weight, it is possible to stack hydrocyclones. This results in less space being required for an equivalent hydrocyclone system than for a froth flotation system, and in a smaller, less expensive preparation plant.

Disadvantages of the hydrocyclone

The hydrocyclone has some disadvantages associated with it. The major disadvantage is that as the amount of near gravity material (defined as plus or minus 10 per cent specific gravity at the gravity of separation) increases to a quantity above 10 per cnet, the sharpness of the separation decreases rapidly. Occasionally a hydrocyclone will clog, but this could be remedied by vibrating the unit or applying abck pressure. Finally, pumping costs are higher for hydrocyclones than for conventional froth flotation cells.

V. EXPERIMENTAL MATERIAL AND EQUIPMENT

Synthetic material

It was decided to use synthetic material in place of actual coal. The synthetic material was harder than coal and would not degrade into finer particles when it was used repeatedly for testing purposes. The synthetic material was constructed so as to duplicate coal with respect to specific gravity, size, and particle shape. It was compounded as follows:

Casting resin, specific gravity 1.24, was impregnated with barium sulfate which has a specific gravity of 4.7. Various ingots of material of specific gravities 1.2 - 1.3, 1.3 - 1.4, 1.4 - 1.5, 1.5 - 1.6, 1.6 - 1.7, 1.7 - 1.8, 1.8 - 1.9, were compounded. The formulas used were:

$$CZ = W$$

$$1.24x + 4.7(Z - x) = W$$

$$R = 1.24x$$

$$BS = 4.7(Z - x)$$

where:

C = desired specific gravity of mixture

Z = volume of mixture in cubic centimeters

W = total weight of mixture in grams

1.24 = specific gravity of casting resin

4.70 = specific gravity of barium sulfate

R = required weight of resin in grams

BS = required weight of $BaSO_4$ in grams.

The resin was measured into an enamelled pan. The barium sulfate (minus 200 mesh) was sifted into the resin. The resin catalyst was then added. The mixture was agitated with a hand-held electric mixer until it began to set up. The agitation was required to prevent the barium sulfate from settling out before the mixture hardened. Approximately one hour elapsed between the addition of the catalyst and the setting up of the mixture.

After the mixture had hardened, it was crushed and ground. The ingots were first broken with a hammer into minus 3 inches top size. These pieces were then crushed in the Deco laboratory jaw crusher. After processing through the jaw crusher, the fragments were further reduced in the American Pulverizer hammer mill. During the crushing process it was necessary to mix dry ice with the fragments to be processed. This was due to the fact that the barium sulfate - casting resin ingots tended to deform rather than crush at room temperature.

The product from the hammer mill was screened in Tyler screens. The minus 65 mesh material was then separated by the float - sink technique at the desired specific gravity range to insure that no material outside of the specific gravity range was present. The finished material was placed in appropriately labeled containers. This was the material used to simulate the solids portion of the feed slurry.

Tagged material

To eliminate the need for float - sink operations to determine each data point on the partition curve, it was decided to use a tracer. The tracer material consisted of a mixture of casting resin and magnetite, Fe_3O_4. The resulting material was identical with the synthetic coal material, except for the fact that it was strongly magnetic. The tagged material was compounded in the following specific gravities: 1.2 - 1.3, 1.3 - 1.4, 1.4 - 1.5, 1.5 - 1.6, 1.6 - 1.7, 1.7 - 1.8, 1.8 - 1.9. After crushing and float - sink separation by the procedure previously described, each specific gravity fraction was screened in Tyler screens into the following size fractions: 65 x 100 mesh, 100 x 150 mesh, 150 x 200 mesh and minus 200 mesh.

The hydrocyclone

The one inch diameter hydrocyclone was constructed of 303 stainless steel. 303 stainless steel was chosen because it is more easily machined than other stainless steel alloys and is resistant to abrasion and corrosion. The cyclone was constructed in such a manner that its configuration was easily variable. The following parts were fabricated:

Vortex finder inner diameters	.375", .250", .125"
Apex discharge diameters	.125", .145", .250", .375"
Inlet diameters	.125", .250", .367"
Cone angles	20°, 40°, 60°

Figures 5 through 10 are shop drawings of the various component parts. Tolerances of $\pm$ .001 inches were held.

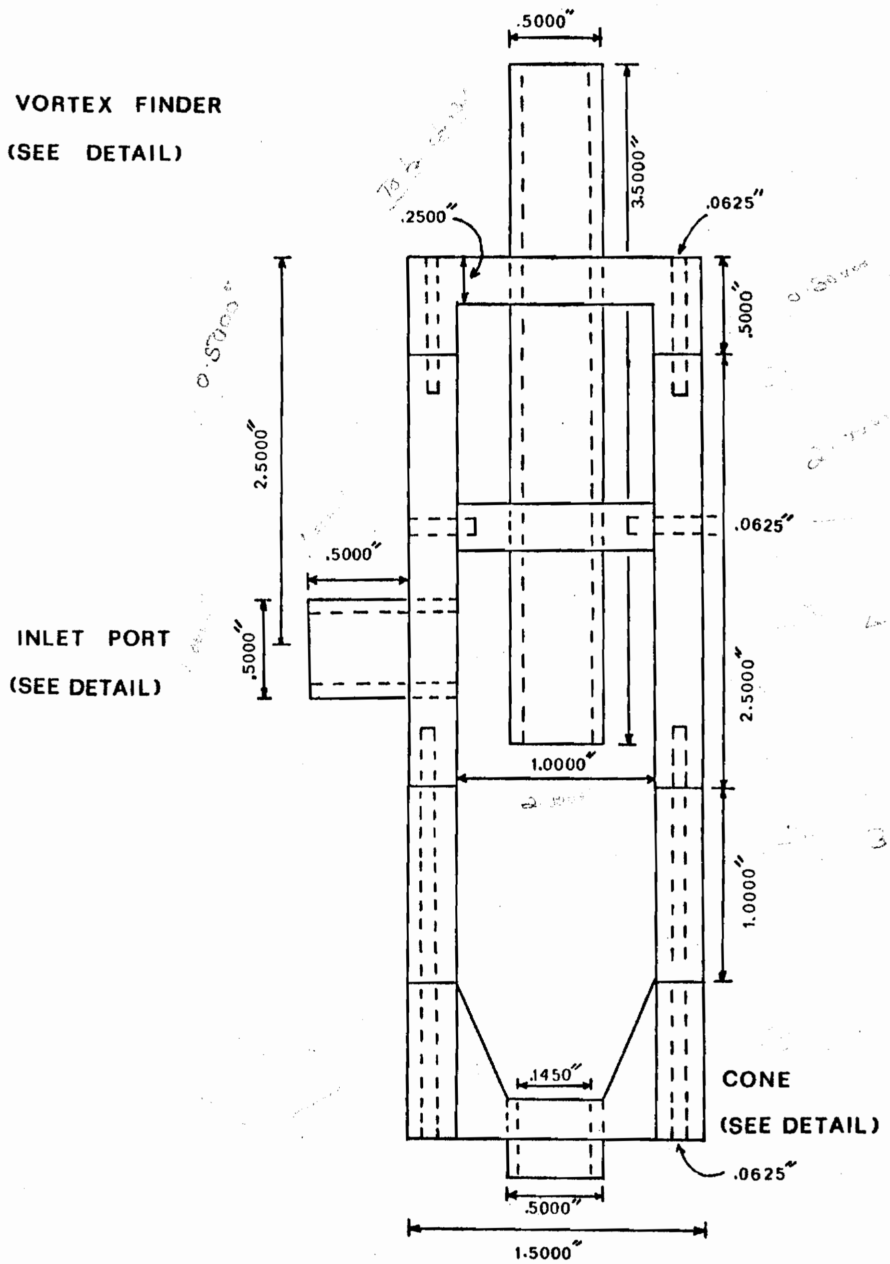


FIGURE 5 FRONT VIEW HYDROCYCLONE

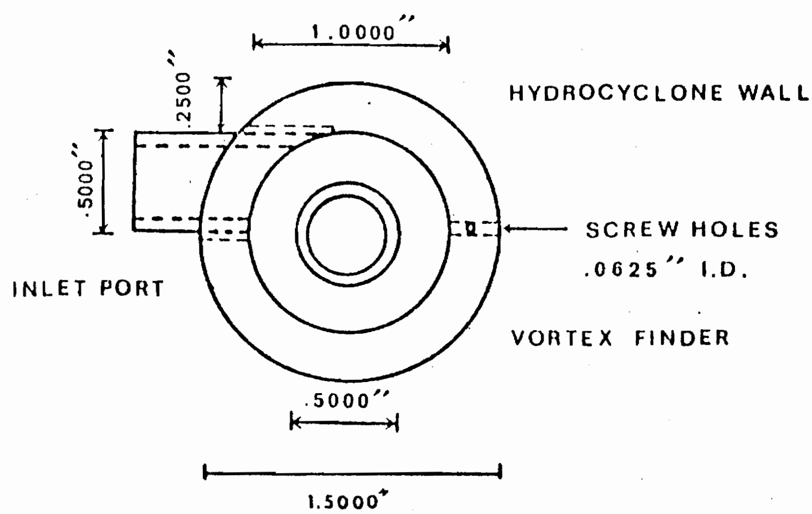


FIGURE 6 - TOP VIEW OF HYDROCYCLONE

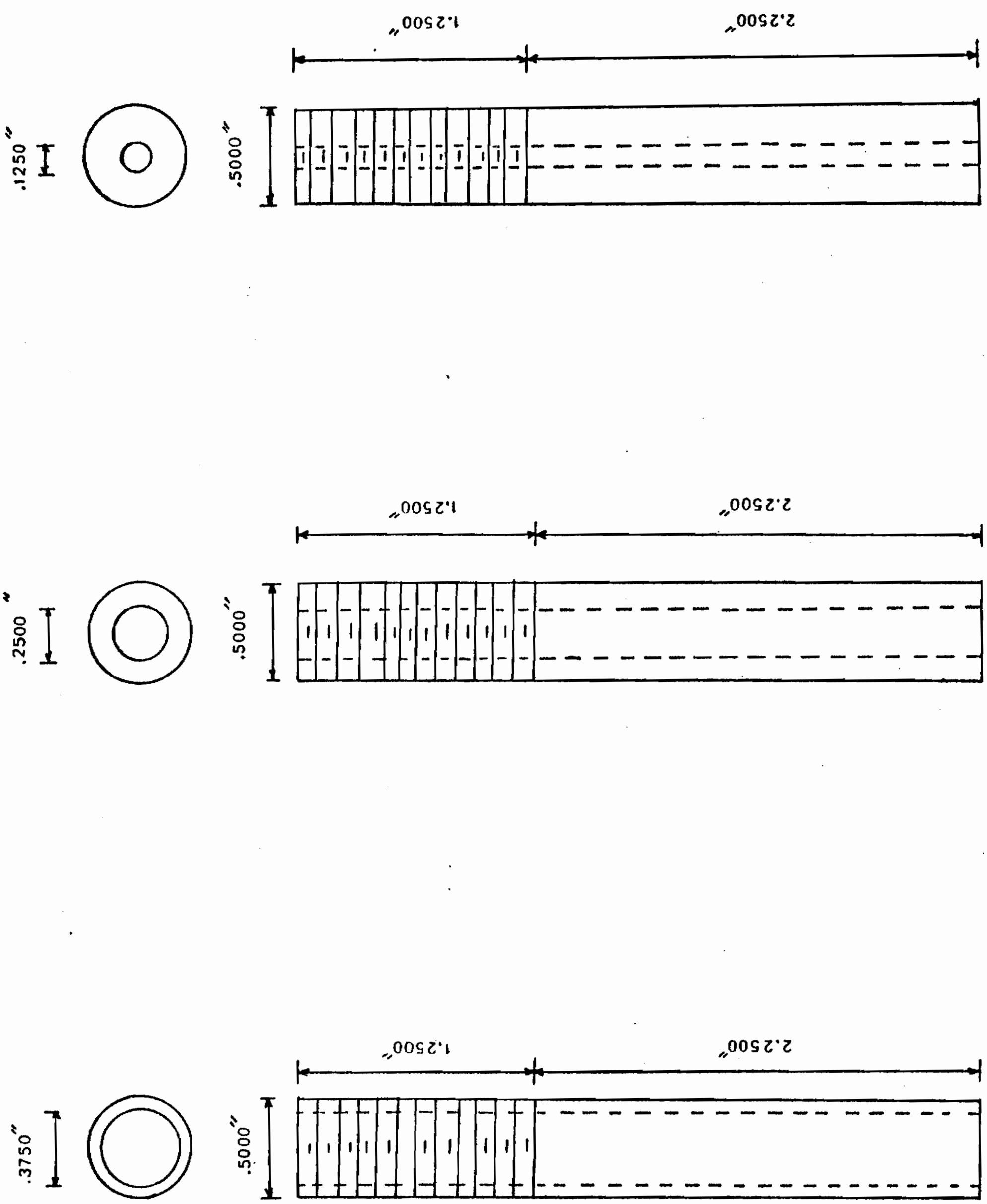


FIGURE 7 — VORTEX FINDER DETAIL

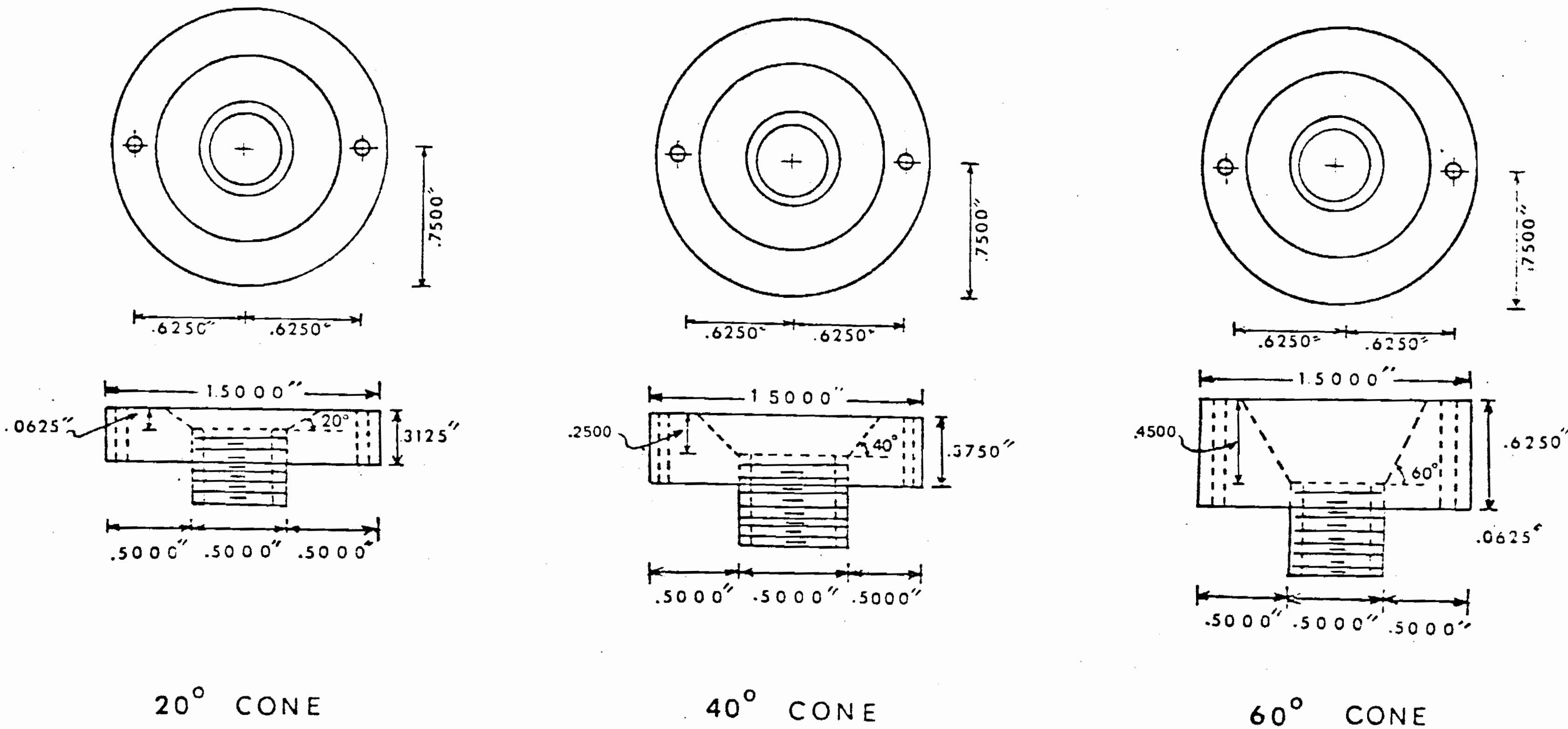
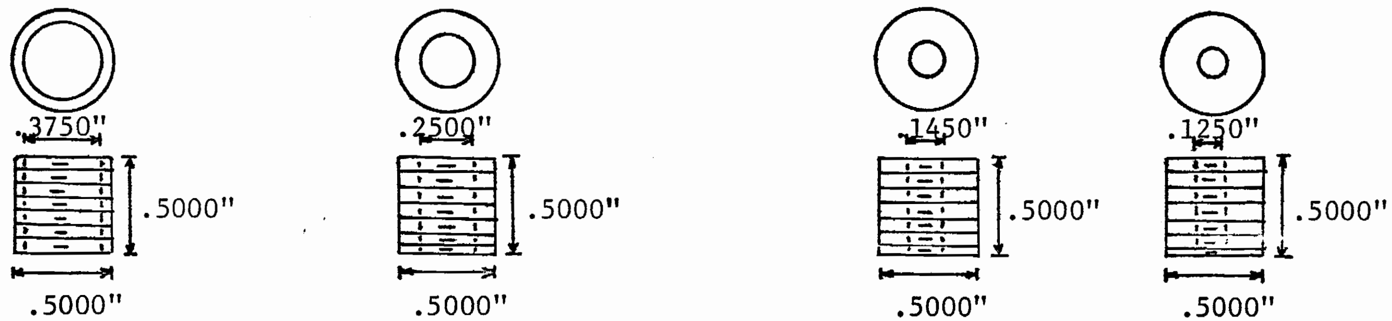
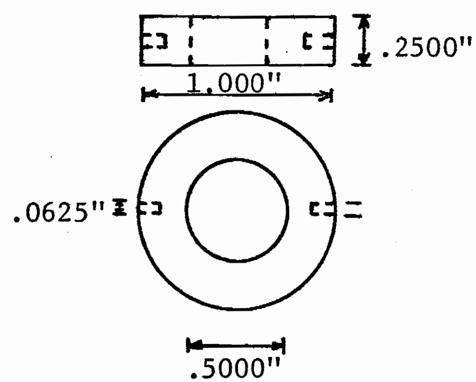


FIGURE 8 – CONE DETAIL



Apex Detail



Partition Detail

FIGURE 9 - APEX AND PARTITION DETAIL

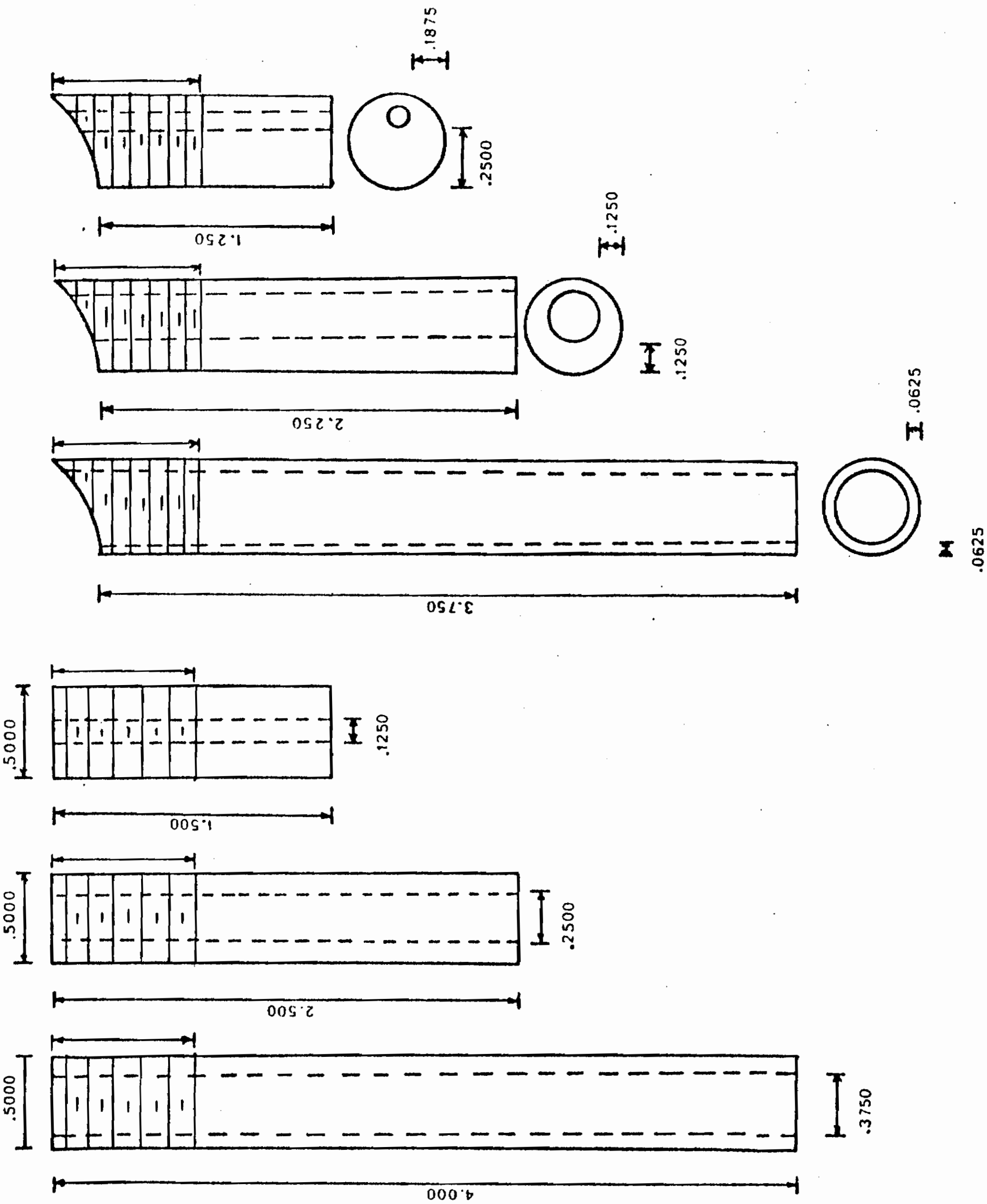


FIGURE 10 – INLET PORT DETAIL

The pump

A Lawrence centrifugal pump was used in this investigation. A centrifugal pump was chosen because of its inherent high degree of reliability and ease of maintenance. Because rust would interfere with the evaluation of the partition coefficients, the pump head, volute, impeller, and housing were cleaned with concentrated sulfuric acid and spray painted with epoxy paint. At the end of this investigation, observation revealed the epoxy coating to be intact.

The motor

The pump was driven by a direct coupled U. S. Vari-Drive Westinghouse Electric Motor. The motor was a 5 horsepower, 220 volt, 3 phase unit. The variable speed aspect aided in maintaining the exact desired pressure.

The gauge

A Duragauge 0 to 30 psig diaphram gauge was chosen. A diaphram gauge was required because the slurry would clog a conventional gauge and render it inoperable.

The pipe

It was decided to use polyvinyl chloride Schedule 40 (ASTM) pipe. The advantages of plastic pipe are its ease of fabrication and its non-rusting properties. The principal disadvantage is its lack of rigidity. This necessitated acaffolding to support the pipe.

The valves

Bronze 1/2" gate valves were used in this experiment. Gate valves were preferred because they will seat even when the fluid is a slurry.

The magnetic separator

A Carpco Magnetic Separator serial number MWL3465 - 115 was used to remove the magnetically tagged material at the end of each run. This machine is a high intensity magnetic separator. A steel cage, which held steel ball bearings, sat between the pole pieces. The cage and balls were painted with epoxy paint to prevent rusting. A fluid stream containing the magnetic particles was directed through the balls. The magnetic particles were retained on the balls, while the rest of the slurry was returned to the reservoir. Figure 11 is a diagram of the system used in this experiment.

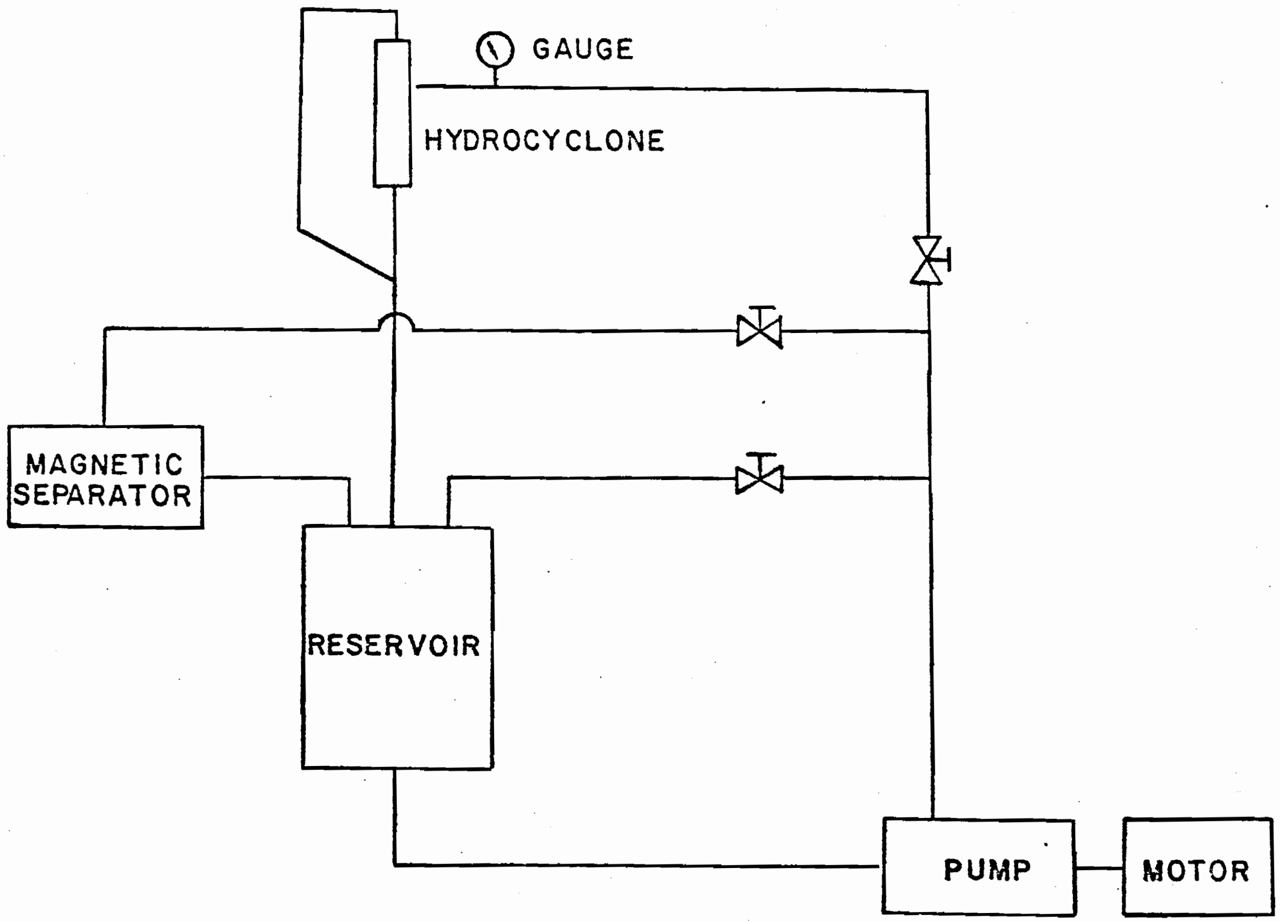


FIGURE 11 — EXPERIMENTAL SETUP

VI. EXPERIMENTAL PROCEDURE

Three coals tested

It was decided to optimize the hydrocyclone for three coals. Washability studies of a "light" coal, a "medium" coal, and a "heavy" coal were found in the literature. A Virginia Pocahontas #3 was selected as the light coal because most of its weight was in the 1.3 or lighter specific gravity fractions. The medium coal chosen was the Hagy seam found in Virginia. Most of its weight lies in the 1.4 - 1.5 specific gravity fraction. A coal from the Cortes area in New Mexico was selected as the heavy coal. Most of its weight is greater than 1.6 in specific gravity. Figure 12 shows the comparison by weight of these three coals. Tables 2,3, and 4 show the washability data for the three coals. By optimizing the hydrocyclone for these three coals, it should be possible to extend the results of this investigation to any type of coal.

Mixing of slurry

After the coal to be investigated was chosen, it was necessary to mix a 10 per cent by weight slurry. It was decided to operate the system with 5400 milliliters of water. Therefore, 600 grams of solids were required. The 600 grams were added on a percentage basis to duplicate the specific gravity distribution of the selected coal. The solids exhibited a tendency to form a stable froth. To eliminate this froth, 3 milliliters of Dowell M - 45 defoamer, and 2 milliliters of Dowell F - 65 wetting agent were added to the system.

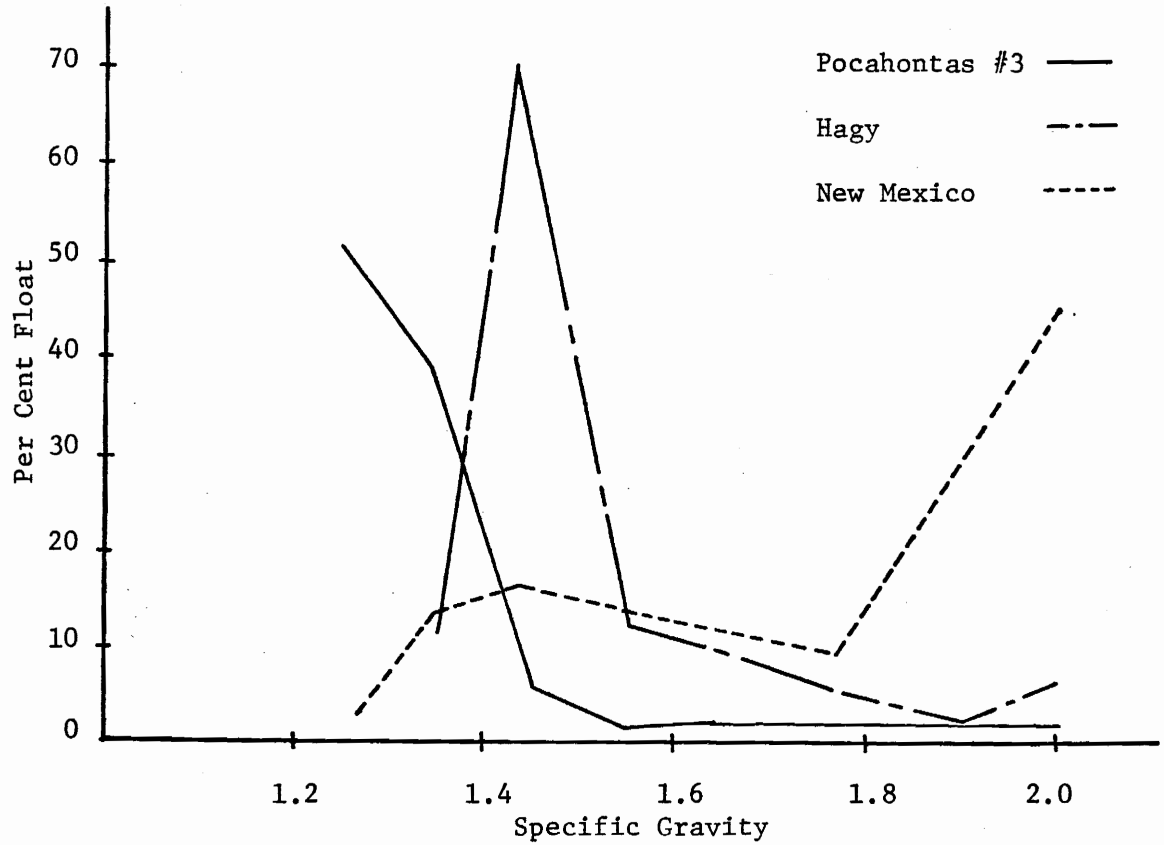


FIGURE 12 - COMPARISON OF WEIGHT DISTRIBUTION OF THE THREE COALS

TABLE 3 - FLOAT SINK ANALYSIS OF POCAHONTAS #3 COAL[16]

Specific Gravity	Weight in grams	Percentage
Float - 1.3	312	52.0
1.3 - 1.4	228	38.0
1.4 - 1.5	30	5.0
1.5 - 1.6	6	1.0
1.6 - 1.7	6	1.0
1.7 - 1.8	6	1.0
1.8 - 1.9	6	1.0
1.9 - Sink	0	0.0

TABLE 4 - FLOAT SINK ANALYSIS OF HAGY SEAM[17]

Specific Gravity	Weight in grams	Percentage
Float - 1.3	0.0	0.0
1.3 - 1.4	12.0	2.0
1.4 - 1.5	378.0	63.0
1.5 - 1.6	72.0	12.0
1.6 - 1.7	54.0	9.0
1.7 - 1.8	24.0	4.0
1.8 - 1.9	6.0	1.0
1.9 - Sink	48.0	9.0

TABLE 5 - FLOAT SINK ANALYSIS OF NEW MEXICO COAL [18]

Specific Gravity	Weight in grams	Percentage
Float - 1.3	7.2	1.2
1.3 - 1.4	78.6	13.1
1.4 - 1.5	94.8	15.8
1.5 - 1.6	81.1	13.6
1.6 - 1.8	70.8	11.8
1.8 - Sink	267.5	44.5

VII. ANALYSIS OF RESULTS

Tagged material method verses float - sink technique

At the beginning of this investigation, it was necessary to prove that the use of tagged material to obtain partition coefficients yielded the same results as the conventional float - sink method. Ten data points were taken on different specific gravity fractions using the magnetic method. These ten data points were repeated using conventional float - sink techniques to obtain the data points. The two samples were tested using the statistical Student's t test. The results indicated that for an alpha of 0.05 the two methods yielded identical results. This proof is shown in Appendix A.

Repeatability of data points using tagged material method

After it was shown that the tagged material method yielded results which compared favorably with those obtained by float - sink analysis, it was necessary to insure that the data points were reproducible. Thirteen paired observations were taken. Statistical manipulation of the data using the Student's t test showed that for an alpha of o.0d the data were reproducible. This proof is shown in Appendix B.

Parameters that were investigated

It was not possible to examine all of the numerous parameters which influence the performance of a hydrocyclone. Therefore, it was necessary to restrict this investigation to those parameters which exercise a major impact on the system. Possible parameters which could have been investigated included:

1. apex diameter to vortex finder diameter ratio
2. slurry density
3. inlet diameter
4. pressure
5. cone angle
6. vortex finder position

It was decided to restrict this investigation to items 3, 4, 5, and 6.

The apex diameter to the vortex finder diameter ratio was felt not to be a critical factor. Weyher has stated that changing this ratio has a most pronounced effect on the separation.[12] The author found this to be correct during the initial familiarization runs. However, there is an extremely limited range in which the cyclone will function. With a very low ratio, no cycloning action took place. When the ratio was high, most of the material reported to the underflow product stream. Therefore, it was decided to run the hydrocyclone at the ratio which yielded the highest recovery of clean coal, while still maintaining the cyclone action. For this investigation this ratio was determined to be 0.38. This compares well with the ratio given by Weyher of 0.35 for a 5.5" diameter hydrocyclone.[19]

It was decided not to investigate the slurry density. Sands and Weyher have intensively investigated the effect of slurry density on the separation obtained with a hydrocyclone.[20] [19] They have shown that as the density of the slurry was increased, the gravity of separation was increased. It was decided to run the experimental system at a 10 per cent slurry ratio by weight. This is the upper limit suggested by Weyher.[19] A higher density of solids would increase the chances of clogging the hydrocyclone.

It was decided to investigate the effect of feed pressure. An increase in feed pressure was accompanied by an increase in throughput capacity. The adjustment of the inlet feed pressure provided a simple method of varying the feed to the hydrocyclone unit. This would allow control of the hydrocyclone to meet changes in rate or feed composition to a coal cleaning plant.[20] The pressures investigated were 6 psig, 10 psig, and 15 psig. This compared well with the limits of 8 to 15 psig established by Weyher for 2" to 4" diameter hydrocyclones.[19]

The cone angle has long been known to exert a great effect on the performance of a hydrocyclone. Visman stated that hydrocyclones with steep cone angles act as classifying hydrocyclones.[13] That is, they separate the particles by size and not by differences in specific gravity. He found that low cone angles caused the hydrocyclone to separate the particles on the basis of specific gravity. Cone angles of 20 degrees, 40 degrees, and 60 degrees were investigated.

The effect of the position of the vortex finder with respect to the bottom of the unit was investigated. Weyher and Lovell found that the yields and ash values were heavily influenced by this parameter.[12] They found that yield and ash increased as the vortex finder approached the bottom of the cone. As the purpose of this investigation was to maximize the pyritic sulfur rejection, it was necessary to investigate this parameter. Vortex finder positions, measured from the top of the outside of the unit, of 2.470 inches, 2.770 inches, 3.070 inches, and 3.370 inches were investigated.

All of the data points taken during the course of this investigation may be found in Appendix C.

Preliminary data points

A number of data points was taken early in the course of this investigation. The purpose of these points was to gain familiarity with the system in general and the hydrocyclone in particular. For example, certain configurations of the hydrocyclone prevented the cycloning action from occuring. (See Figure 13.) In this case, it was found that the ratio of the apex discharge to the vortex finder diameter was too low. Other configurations resulted in no separation. Figure 14 shows that when the apex diameter exceeds the vortex finder diameter, most of the solids report to the underflow.

Pocahontas #3 coal simulation

The first stage of this investigation involved the simulation of the Pocahontas #3 coal. Pocahontas #3 is a coal which has most of its material in the lighter specific gravity fractions. (See Table 2.) In a float - sink analysis performed by Shelton, over 96 per cent of the material was lighter than 1.5 in specific gravity.[16] This coal served as the light endpoint for the purpose of the investigation.

The results of the variation of the vortex finder are shown in Figure 15. The four curves follow a basic trend. The rejection of pyritic sulfur first increases as the vortex finder is moved further away from the cone. This was consistent with the findings of Weyher

DATE: 1/3/75

TIME: 9:00 a.m.

PERSONNEL: L. Amundson, E. Loud

COAL NUMBER: #1 Pocahontas #3

SPECIFIC GRAVITY UNDER INVESTIGATION: 1.2 - 1.3 65 x 100 mesh

BODY LENGTH: 3"

INLET DIAMETER: 0.375"

CONE ANGLE: 40°

VORTEX FINDER: 0.375"

APEX DISCHARGE: 0.125"

PRESSURE: 6 psi

SLURRY DENSITY: 10 %

VORTEX FINDER POSITION: 2.770"

Overflow Volume in ml: 900

Underflow Volume in ml: 0

Time in seconds: 4.0

WT. OF TAGGED MATERIAL IN OVERFLOW: ----

WT. OF TAGGED MATERIAL IN UNDERFLOW: ----

PARTITION COEFFICIENT: ----

With this geometry no cycloning action occured

Figure 13 - Geometry of Hydrocyclone When no Separation Occured

DATE: 1/3/75

TIME: 3:00 p.m.

PERSONNEL: L. Amundson, W. D. Bays

COAL NUMBER: #1 Pocahontas #3

SPECIFIC GRAVITY UNDER INVESTIGATION: 1.3 - 1.4 65 x 100

BODY LENGTH: 3.0"

INLET DIAMETER: 0.375"

CONE ANGLE: 40^{o}

VORTEX FINDER: 0.250"

APEX DISCHARGE: 0.375"

PRESSURE: 6 psi

SLURRY DENSITY: 10 %

VORTEX FINDER POSITION: 2.770"

Overflow Volume in ml 340

Underflow Volume in ml 760

Time in seconds 6.0

WT. OF TAGGED MATERIAL IN OVERFLOW: .1408 g

WT. OF TAGGED MATERIAL IN UNDERFLOW: .4783 g

PARTITION COEFFICIENT: 22.7

Figure 14 - Geometry of Hydrocyclone With Most Solids Reporting to Underflow

BODY LENGTH 3"
PRESSURE 6 psi
DENSITY 10%
CONE ANGLE 40°
INLET DIA. .375"
VORTEX DIA. .250"
APEX DIA. .145"
S.G. 4.7-5.1

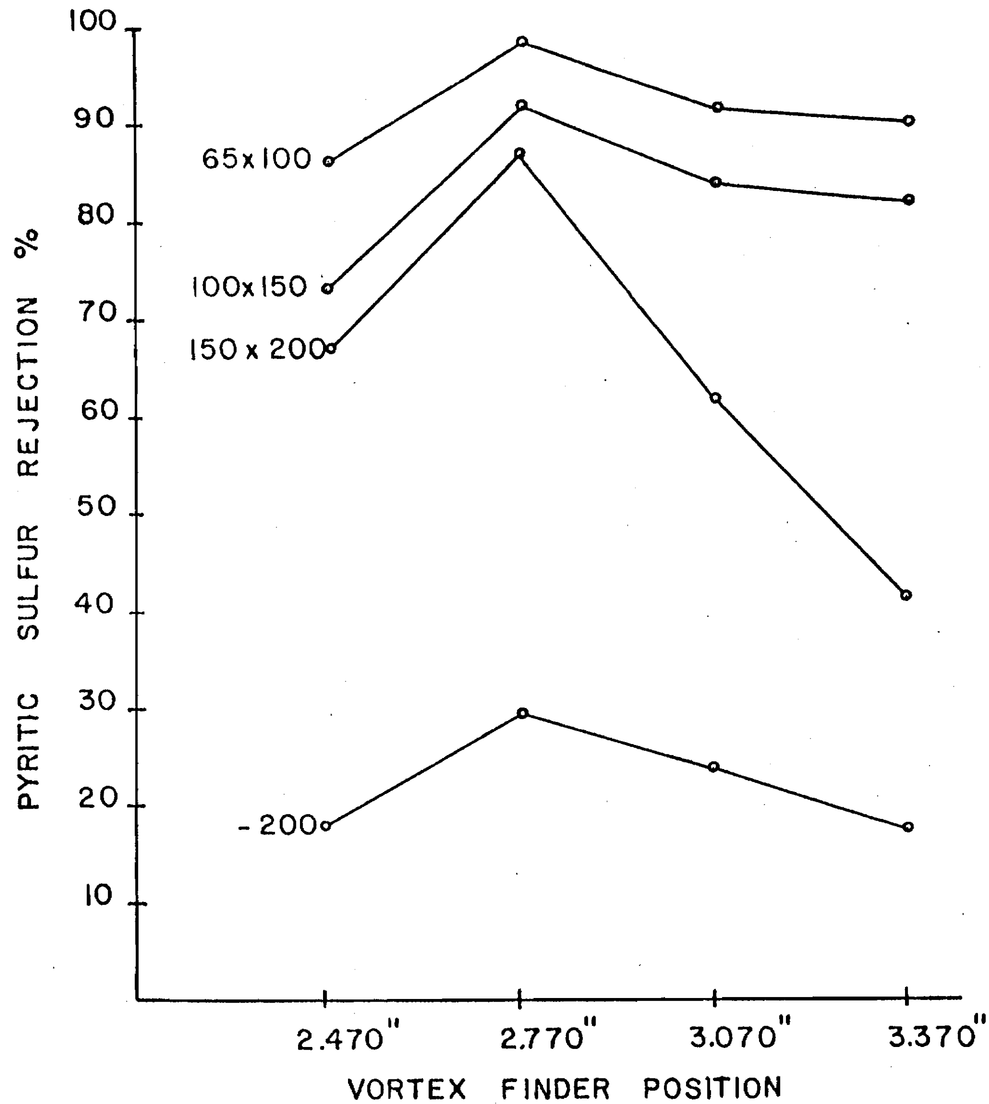


FIGURE 15 - VARIATION OF VORTEX FINDER POSITION POCAHONTAS #3 COAL

and Sands.[19] [20] They noted a decreasing recovery of ash as the vortex finder was raised. As the vortex finder was further raised, the rejection of pyritic sulfur was decreased. The author felt that this was due to short-circuiting of material. The vortex finder has been raised above the point where the particles had reached the wall and had been classified. No confirmation or rejection of the hypothesis was found in the literature.

It is important to note the fact that the pyritic sulfur rejection decreased with decreasing particle size. Good rejection was achieved with the 65 x 100 mesh size and the 100 x 150 mesh size material, yielding 98.3 per cent and 92.2 per cent pyritic sulfur rejection, respectively. The results for the 150 x 200 mesh size fraction were acceptable, resulting in 87.3 per cent pyritic sulfur rejection. The results for the minus 200 mesh size fraction were poor. Only 29.9 per cent of the pyritic sulfur reported to the apex discharge. This indicated that the one inch diameter hydrocyclone was too large to effectively handle material of this size. A smaller diameter hydrocyclone could perform a better separation on this material.

Figure 16 shows the results of the variation of the inlet diameter. The 0.250 inch inlet gave the best results. The pyritic sulfur rejection was 92.6 per cent for the 65 x 100 mesh size fraction, 91.5 per cent for the 100 x 150 mesh size fraction, 70.5 per cent for the 150 x 200 mesh size fraction, and 28.8 per cent for the minus 200 mesh size fraction.

As the inlet diameter was decreased, the pyritic sulfur rejection decreased. This may be due to the fact that the hydrocyclone was

BODY LENGTH	3"
PRESSURE	10 psi
DENSITY	10 %
CONE ANGLE	40°
VORTEX DIA.	.250"
VORTEX POS.	3.370"
APEX DIA.	.145"
S.G.	4.7-5.1

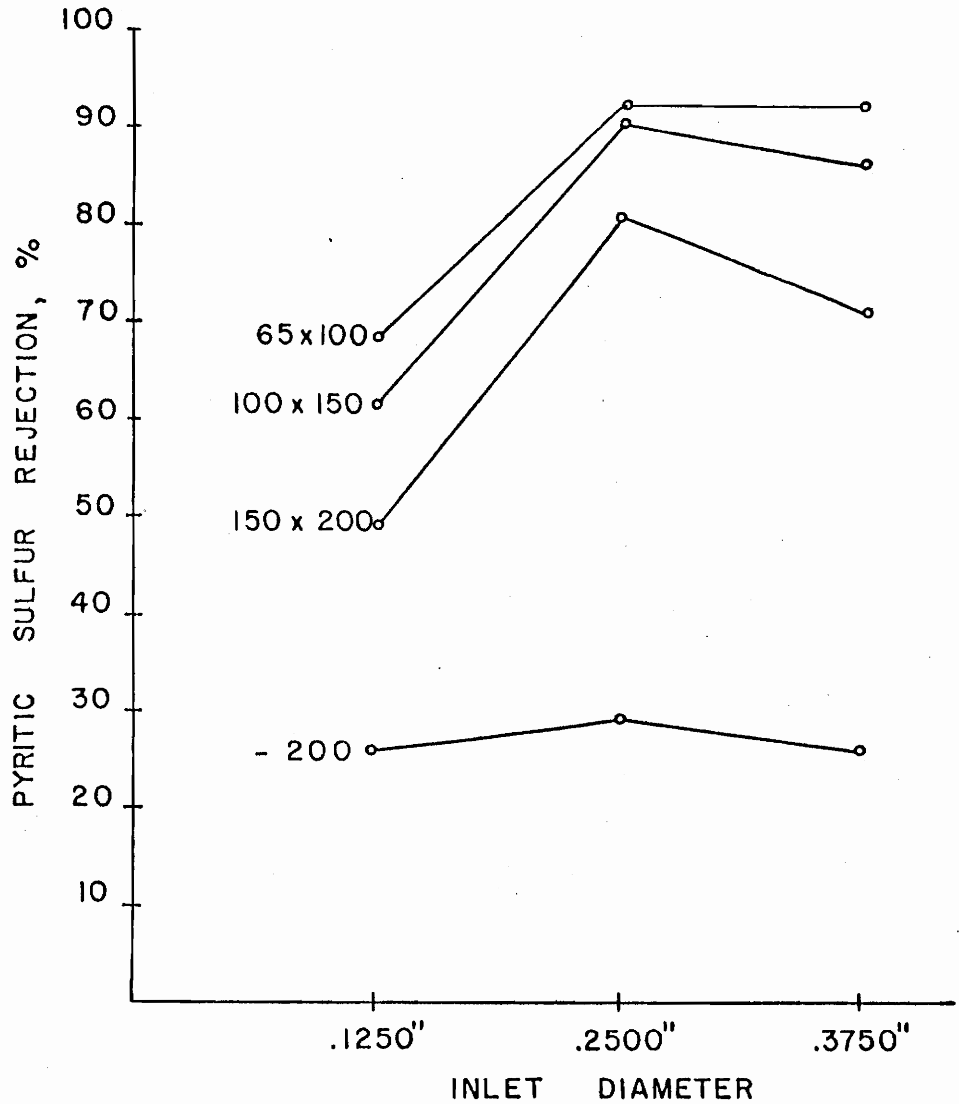


FIGURE 16 - VARIATION OF INLET DIAMETER POCAHONTAS #3 COAL

underloaded. If this was so, the dense bed would not have developed as fully as it did when more material was present. No reference was found to this phenomena in the literature. Throughput capacity was substantially decreased at this smaller inlet diameter.

As the diameter of the inlet was increased to 0.375 inches from 0.250 inches, the rejection of the pyritic sulfur was decreased. This was due to the fact that as the throughput capacity was increased, the dense bed that formed along the inner wall of the hydrocyclone increased in thickness. This in turn resulted in an increase in the gravity of separation. Sands and Weyher have observed this fact.[20] [19] As the gravity of separation increased, the chances of entraining some pyritic particles increased.

These data show that the 1.0" diameter hydrocyclone could effectively remove pyrite from 65 x 150 mesh coal. The results of the tests on the 150 x 200 mesh size fraction were not as good. The 1.0" diameter hydrocyclone was not suited for cleaning the minus 200 mesh size fraction for this type coal.

Variations of the inlet pressure were investigated. The response of the hydrocyclone to inlet pressures of 6 psig, 10 psig, and 15 psig was observed. Figure 17 is a graph of these results. The best results were obtained at a pressure of 10 psig. This compared well with the limits of 8 to 15 psig for a 2.0" to 4.0" diameter hydrocyclone suggested by Weyher.[19] At 6 psig the rejection of the pyrite was decreased. It may have been due to the fact that the inlet pressure was too low to give the particles the acceleration required to classify them according to specific gravity. At 15 psig the rejection of

S.G.	4.7 - 5.1
BODY LENGTH	3"
DENSITY	10%
CONE ANGLE	40°
INLET DIA.	.375"
VORTEX DIA.	.250"
VORTEX POS.	3.370"
APEX DIA.	.145"

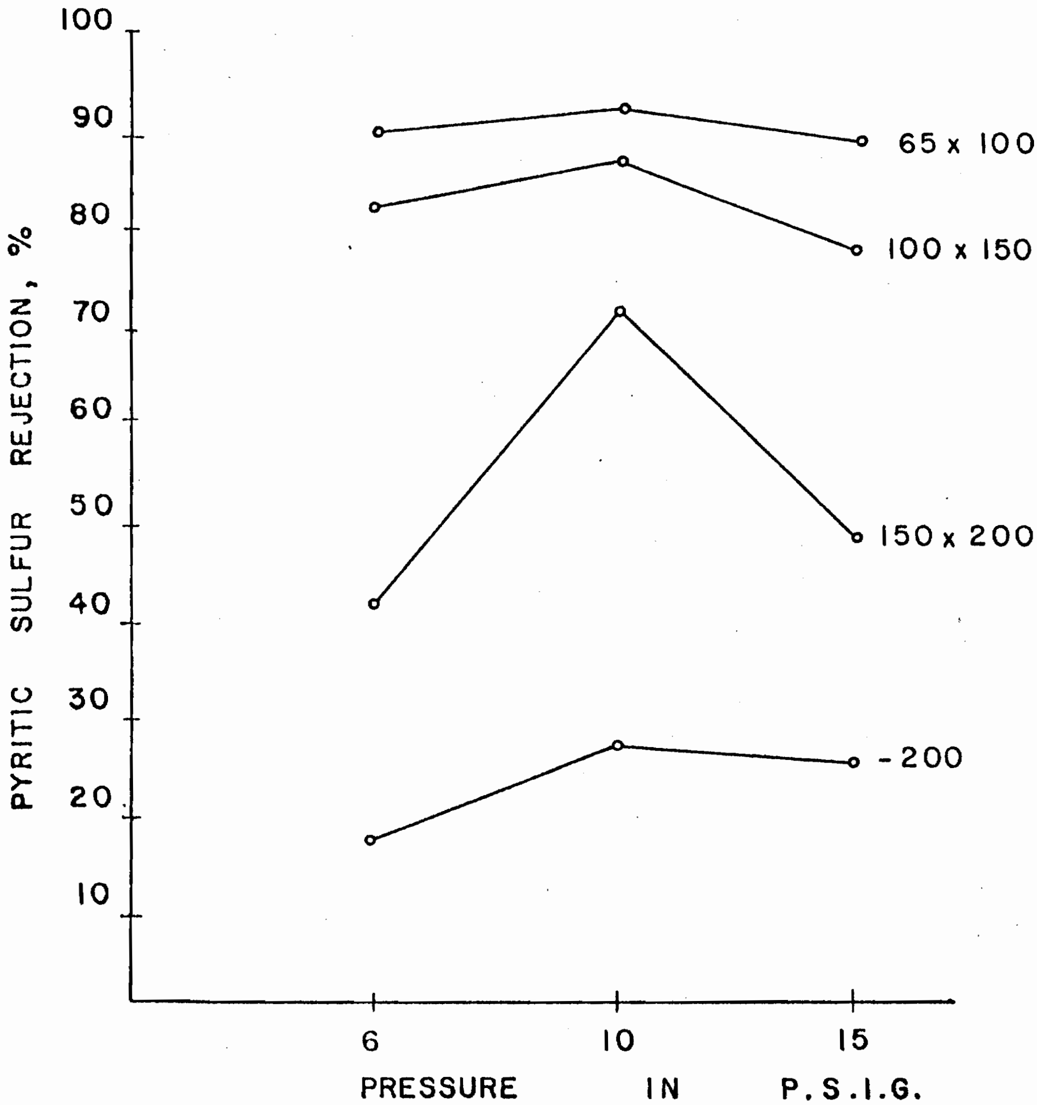


FIGURE 17 - VARIATION OF PRESSURE POCAHONTAS *3 COAL

pyritic sulfur was decreased. This may be explained by the fact that the denser bed formed resisted penetration by the pyritic sulfur particles. This allowed the pyritic sulfur to become entrained in the vortex finder stream.[20]

The pyritic sulfur rejection was 92.5 per cent for the 65 x 100 mesh size fraction, 85.9 per cent for the 100 x 150 mesh size fraction, 70.8 per cent for the 150 x 200 mesh size fraction, and 25.8 per cent for the minus 200 mesh size fraction.

Three cone angles were investigated. The three cones measured 20 degrees, 40 degrees, and 60 degrees. The best results were obtained with the 40 degree cone. According to Visman, the best results were expected with a relatively flat cone.[13] The pyritic sulfur rejection was 92.5 per cent for the 65 x 100 mesh size fraction, 85.9 per cent for the 100 x 150 mesh size fraction, 70.8 per cent for the 150 x 200 mesh size fraction, and 25.8 per cent for the minus 200 mesh size fraction. These results are shown in Figure 18.

The flatter cone, the 20 degree cone, yielded decreased pyritic sulfur rejection. It may be that the thickness of the bed formed with this cone decreased the distance between the bed and the vortex finder current.

The 60 degree cone also yielded decreased pyritic sulfur rejection. This is due to the fact that 60 degrees is starting to approach the steeper cones which are found in classifying hydrocyclones. Visman states that the steep cones form very thin beds with no stratification. Therefore, the smaller particles, regardless of their specific gravity tend to report to the overflow product stream.

BODY LENGTH	3"
PRESSURE	10 psi
DENSITY	10 %
INLET DIA.	.375"
VORTEX DIA.	.250"
VORTEX POS.	3.370"
APEX DIA.	.145"
S.G.	4.7 - 5.0

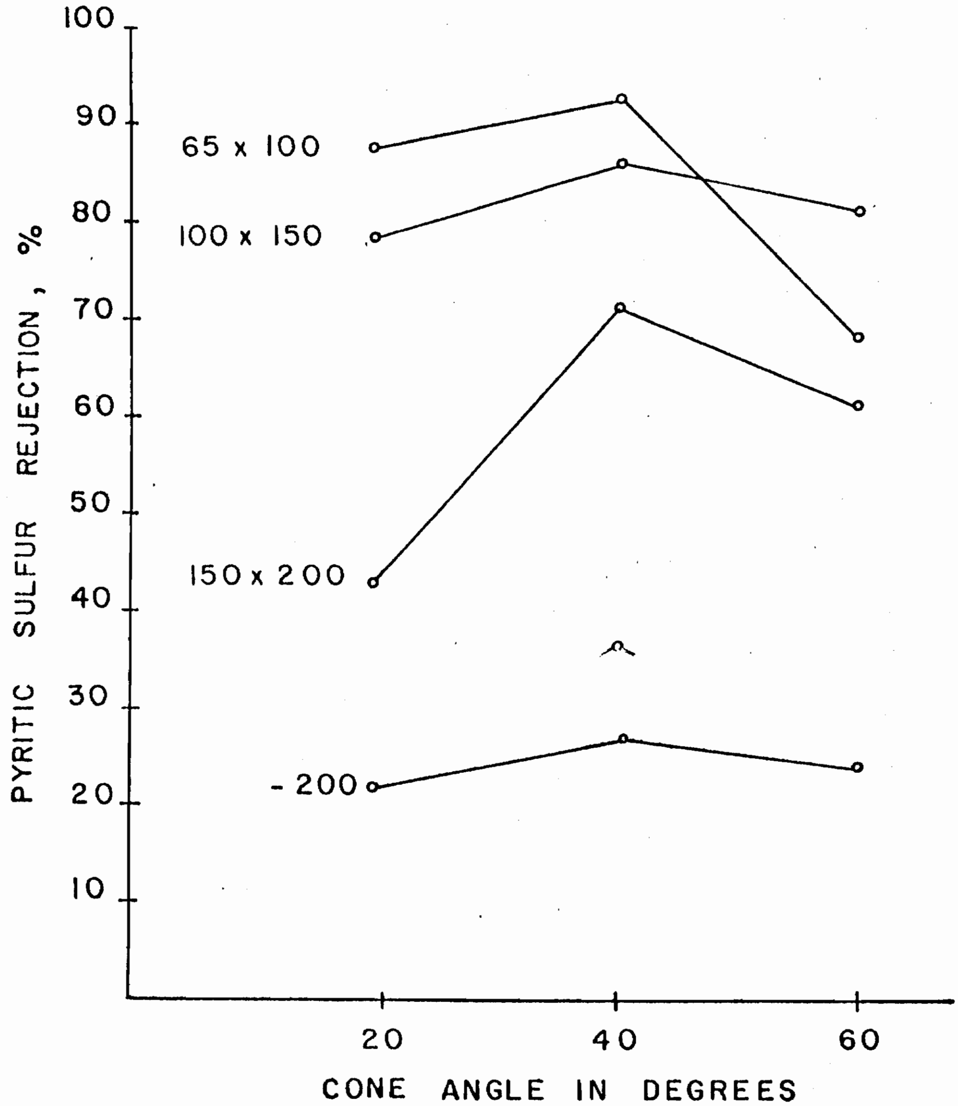


FIGURE 18 - VARIATION OF CONE ANGLE POCAHONTAS *3 COAL

Figure 19 shows the partition recovery curves for the four size fractions. These curves were taken at the configuration which yielded the best pyritic sulfur rejection. This figure shows the trend of the smaller particles to report to the vortex finder product stream.

Hagy coal simulation

The second coal simulated was the Hagy seam. The washability study data performed by the author is shown in Table 3. The Hagy seam has the majority of its weight, 75 per cent, in the 1.4 - 1.6 specific gravity range. This coal served as the medium weight coal.

The results of the variation of the vortex finder position are shown in Figure 20. The curves for the four size fractions showed the same basic trend. When the vortex finder was at its highest point away from the cone, the pyritic sulfur rejection was at its lowest value. Short-circuiting of the material was probably responsible for this. As the vortex finder is moved closer to the cone, the pyritic sulfur rejection was increased. At the 2.770 inch position, the best pyritic sulfur rejection values were obtained. They are: 95.9 per cent for the 65 x 100 mesh size fraction, 80.6 per cent for the 100 x 150 mesh size fraction, 75.0 per cent for the 150 x 200 mesh size fraction, and 57.6 per cent for the minus 200 mesh size fraction. When the vortex finder was moved closer to the cone, to 3.070 inches, the pyritic sulfur rejection decreased. Other researchers have noted an increase in ash recovery as the vortex finder approached the cone. When the vortex finder was moved to its closest point to the cone, even more pyritic sulfur reported to the overflow product.

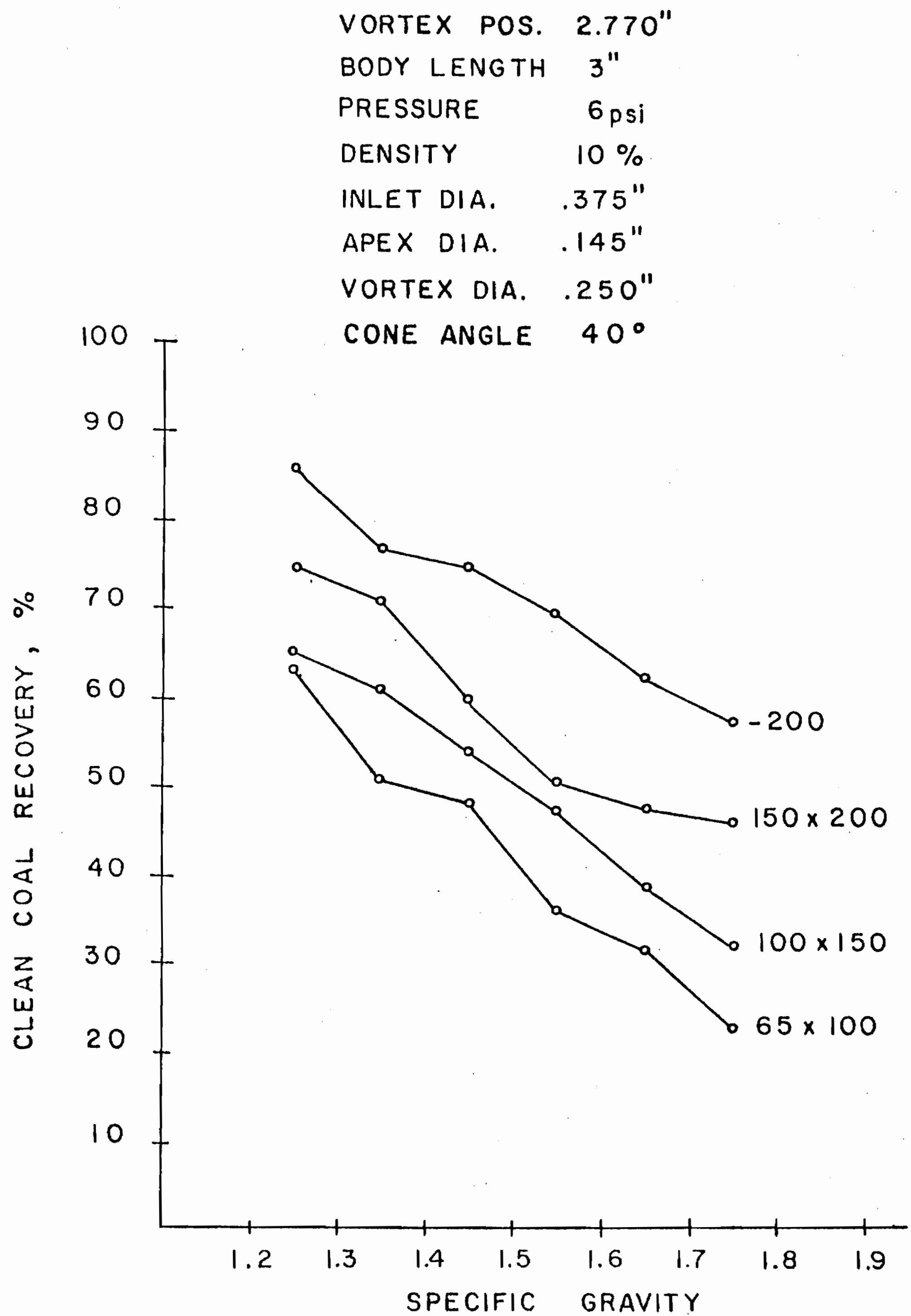


FIGURE 19 - RECOVERY PARTITION CURVES POCAHONTAS #3 COAL

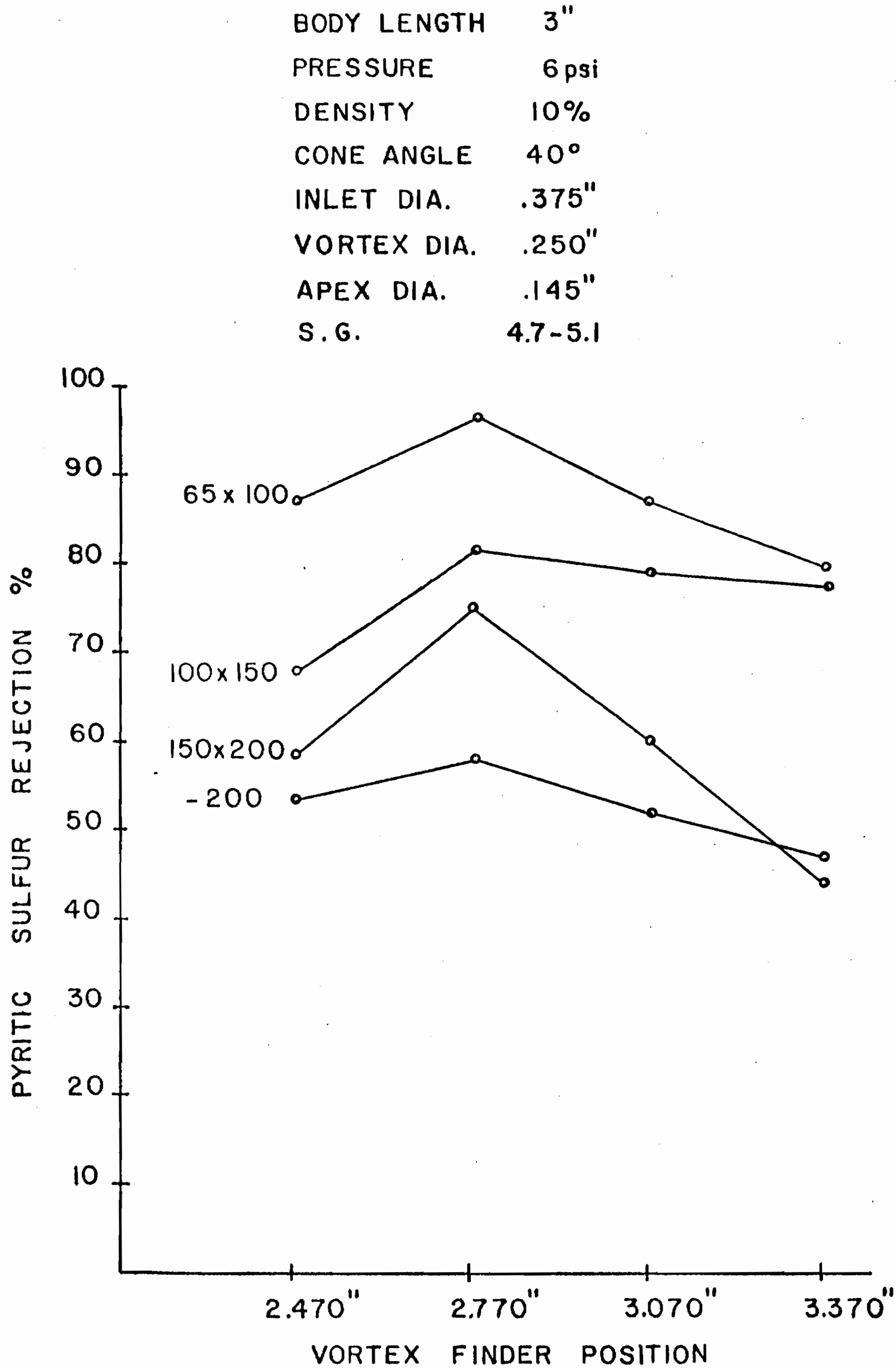


FIGURE 20–VARIATION OF VORTEX FINDER POSITION
HAGY COAL

The results of the variation of the inlet diameter are plotted on Figure 21. For the three finer sizes, the best results were obtained at the 0.250 inch inlet diameter. These results were 93.6 per cent rejection for the 100 x 150 mesh size fraction, 80.8 per cent for the 150 x 200 mesh size fraction, and 73.8 per cent for the minus 200 mesh size fraction. The best results for the 65 x 100 mesh size fraction were obtained at the 0.375 inch inlet diameter. However, it is felt that the 0.250 inch inlet diameter point is in error for the 65 x 100 mesh size fraction. This was reinforced by the fact that the value of the 100 x 150 mesh size fraction point, 93.6 per cent rejection, is approximately equal to the value of the 65 x 100 mesh point, 93.4 per cent rejection. If this premise was correct, then this graph followed the same trend as the Pocahontas coal.

As the diameter of the inlet increased to 0.375 inch, the pyritic sulfur rejection was decreased. Increasing the inlet diameter increased the throughput capacity. This caused the bed which formed along the inner cone wall to increase in thickness. This resulted in an increase in the gravity of separation. Sands has observed this fact.[20] As the gravity of separation increased, more pyritic sulfur reported to the overflow.

Decreasing the inlet diameter to 0.125 inch from 0.250 inch resulted in an increase in the amount of pyritic sulfur which reported to the overflow. This may be due to the fact that the hydrocyclone was underloaded. This prevented the dense bed from completely developing and resulted in more pyritic sulfur reporting to the overflow. No

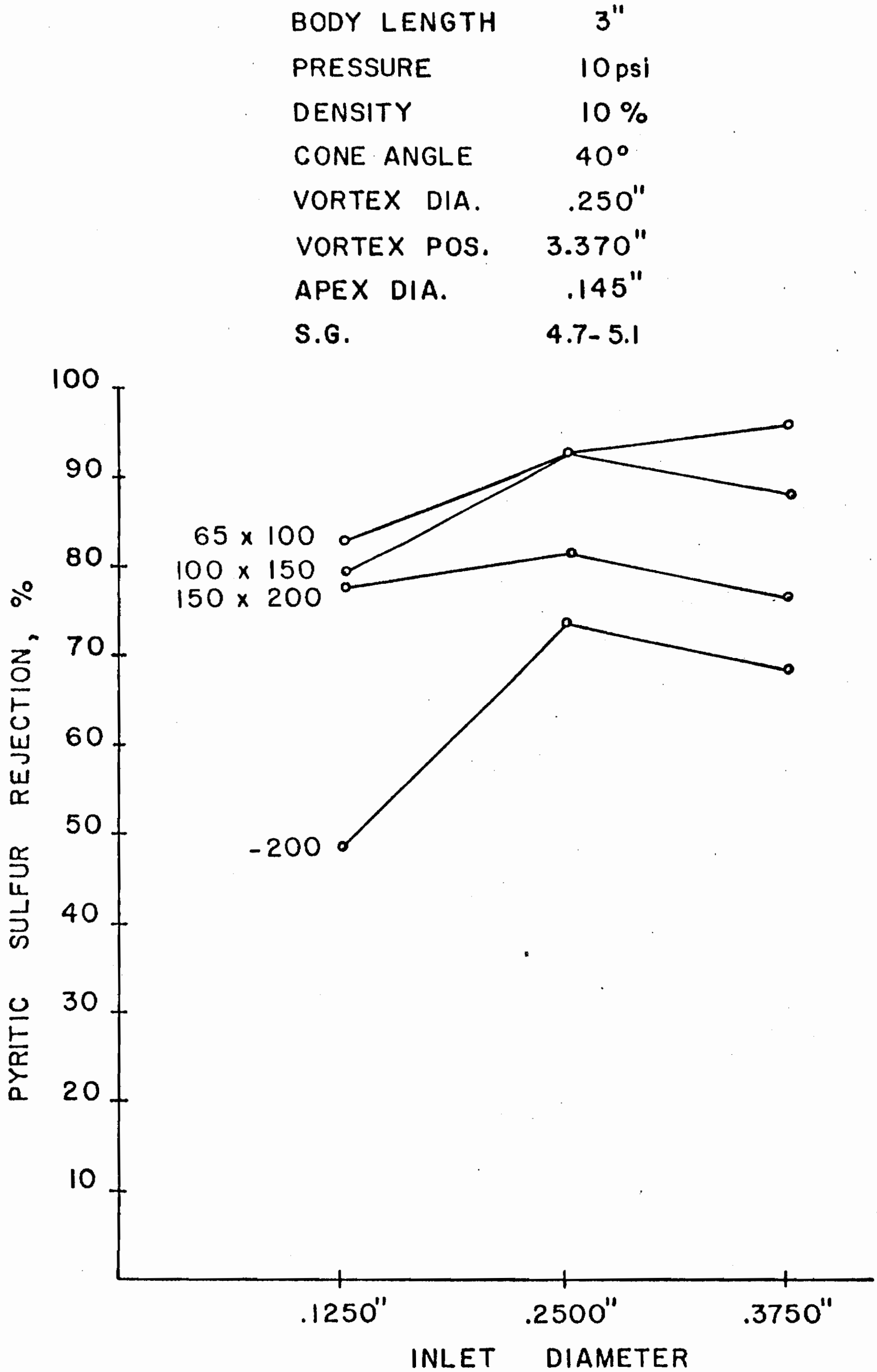


FIGURE 21- VARIATION OF INLET DIAMETER HAGY COAL

confirmation or rejection of this hypothesis was found in the literature. The throughput capacity of the hydrocyclone was substantially reduced at this inlet diameter.

Three inlet pressures were investigated. Figure 22 shows the response of the hydrocyclone at 6 psig, 10 psig, and 15 psig. The best results were obtained at an inlet pressure of 10 psig. The pyritic sulfur rejection was 95.9 per cent for the 65 x 100 mesh size fraction, 86.2 per cent for the 100 x 150 mesh size fraction, 75.7 per cent for the 150 x 200 mesh size fraction, and 67.5 per cent for the minus 200 mesh size fraction.

At a pressure of 6 psig the pyritic sulfur rejection was decreased. The author felt that this was because the particles had not received sufficient acceleration to classify them according to specific gravity.

At a pressure of 15 psig, the capacity of the unit was increased. This resulted in the formation of a denser bed along the inner wall of the cone. This caused an increase in the gravity of separation. With a higher gravity of separation, more pyritic sulfur particles reported to the overflow product stream. This was confirmed by the findings of Sands.[20]

Cone angles of 20 degrees, 40 degrees, and 60 degrees were examined with respect to their influence on pyritic sulfur rejection. It was anticipated that the best results would occur with a relatively flat cone.[13] The 40 degree cone yielded the best pyritic sulfur rejection. The results showed rejections of 96.9 per cent for the 65 x 100 mesh size fraction, 86.2 per cent for the 100 x 150 mesh size fraction, 75.7 per cent for the 150 x 200 mesh size fraction, and 67.5

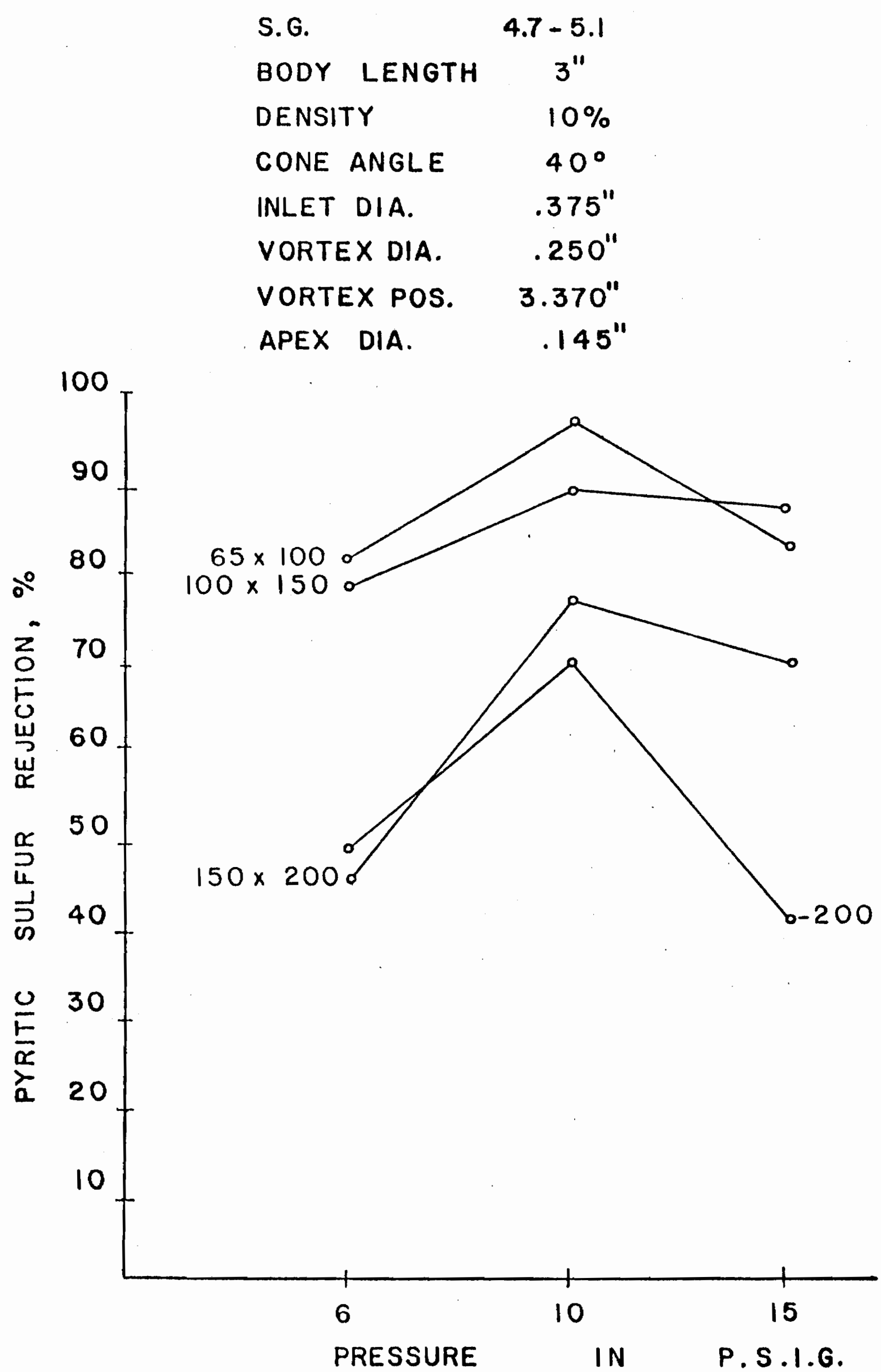


FIGURE 22 - VARIATION OF PRESSURE HAGY COAL

per cent for the minus 200 mesh size fraction. The bed which formed at this cone angle was thin enough so that there was sufficient clearance between it and the vortex finder. It was, at the same time, thick enough so that the hydrocyclone performed a separation by specific gravity and not by size. Figure 23 shows the results of this investigation.

The flatter, 20 degree cone, yielded decreased pyritic sulfur rejection. This was due to the fact that the thickness of the bed decreased the distance between it and the vortex finder. This caused more of the pyritic sulfur particles to report to the overflow.

The 60 degree cone also yielded decreased pyritic sulfur rejection. This cone is steep enough that the hydrocyclone was starting to respond as a classifying hydrocyclone. According to Visman the steeper cones form thin beds with no stratification.(13) Therefore, the smaller particles tended to report to the overflow no matter what their specific gravity.

Figure 24 shows the recovery partition curves of clean coal for the Hagy coal. These points were taken at the configuration which yielded the best pyritic sulfur rejection. The curves show that the finer sized fractions tend to report to the overflow product stream.

New Mexico coal simulation

The third coal investigated was from the Cortes area in New Mexico. The coal seams in this area are currently under investigation and little data is available on them at this time. Washability studies reported by Duerbrock are shown in Table 4.(18) This coal is extremely heavy. Over

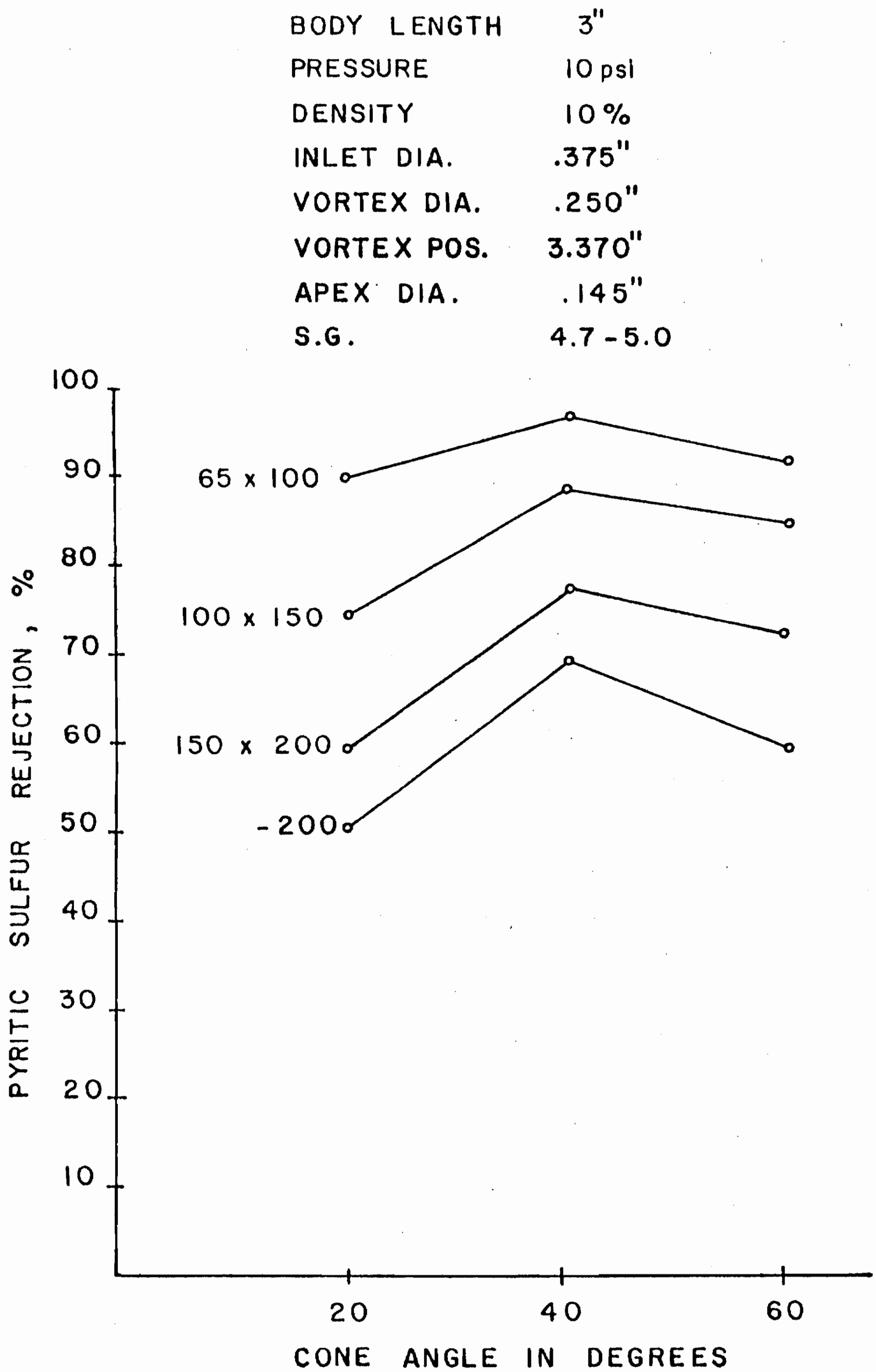


FIGURE 23 - VARIATION OF CONE ANGLE HAGY COAL

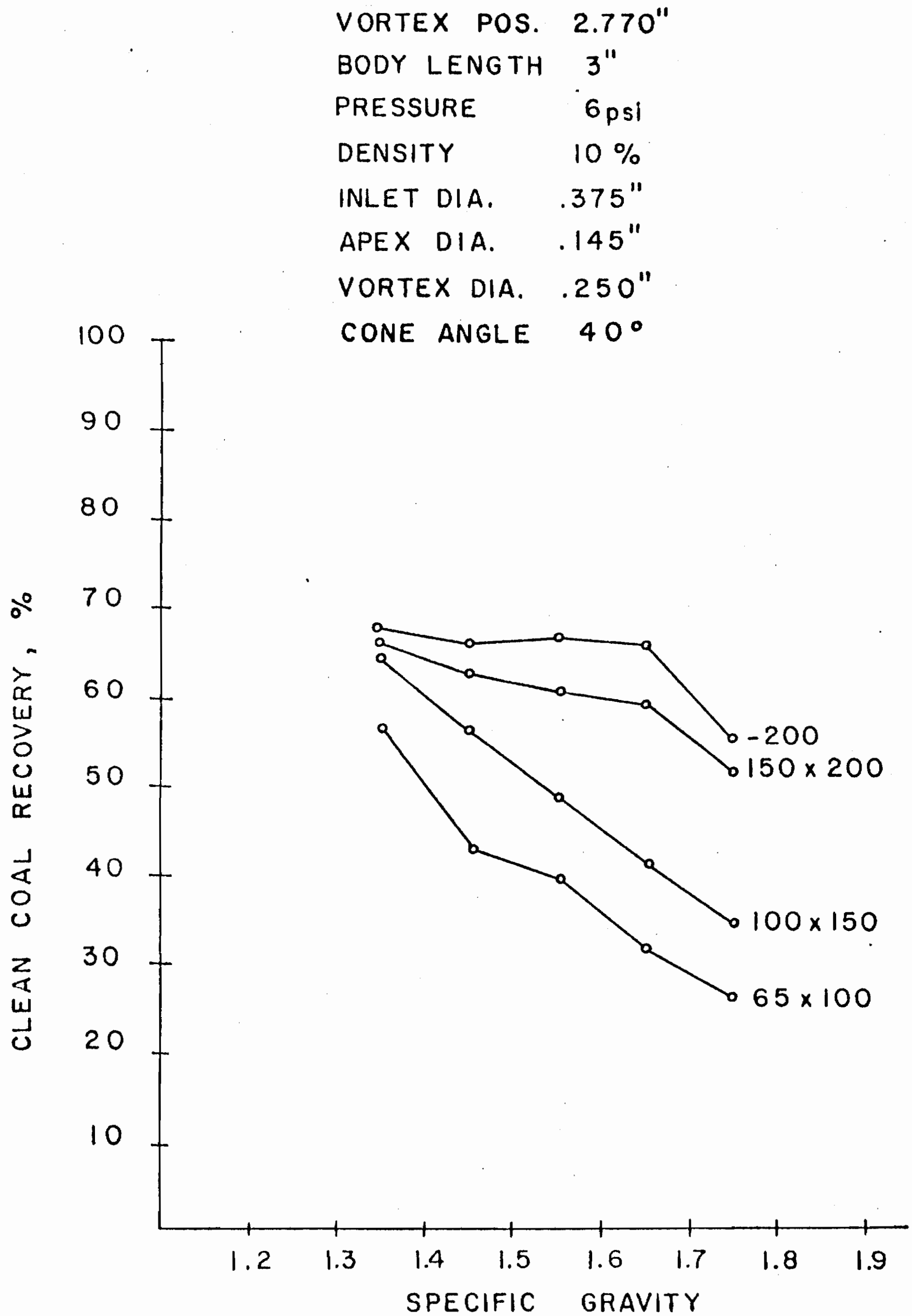


FIGURE 24 - RECOVERY PARTITION CURVE HAGY COAL

69 per cent of its weight has a specific gravity greater than 1.5. This coal will serve as the heavy endpoint in this investigation.

The response of the hydrocyclone with respect to the location of the vortex finder is shown in Figure 25. For the three coarser size fractions, the best results were obtained at the 3.070 inch vortex finder position. The pyritic sulfur rejections were 87.7 per cent for the 65 x 100 mesh size fraction, 83.9 per cent for the 100 x 150 mesh size fraction, and 76.6 per cent for the 150 x 200 mesh size fraction. The vortex finder position is closer to the cone than the optimum position found for the previous, lighter coals. This may be explained by the fact that the New Mexico coal is significantly heavier. A greater residence time was required for this coal to be classified by specific gravity because of the large number of heavy particles. It is felt that they physically interfered with each other while they were being classified according to specific gravity.

As the vortex finder was raised, to the 2.770 inch position, short-circuiting occured. This caused an increase in the amount of pyrite reporting to the overflow. As the vortex finder was further raised, to the 2.470 position, even more pyritic particles were entrained in the upward central current before they had time to reach the outer wall.

When the vortex finder was lowered to the 3.370 inch position, the amount of pyritic sulfur reporting to the overflow increased. This was due to the fact that the vortex finder was now close enough to the bed to be entraining heavy particles off of it.

The minus 200 mesh size fraction did not follow the pattern of the three coarser size fractions. The optimum position for the vortex

BODY LENGTH	3"
PRESSURE	6 psi
DENSITY	10%
CONE ANGLE	40°
INLET DIA.	.375"
VORTEX DIA.	.250"
APEX DIA.	.145"
S.G.	4.7-5.1

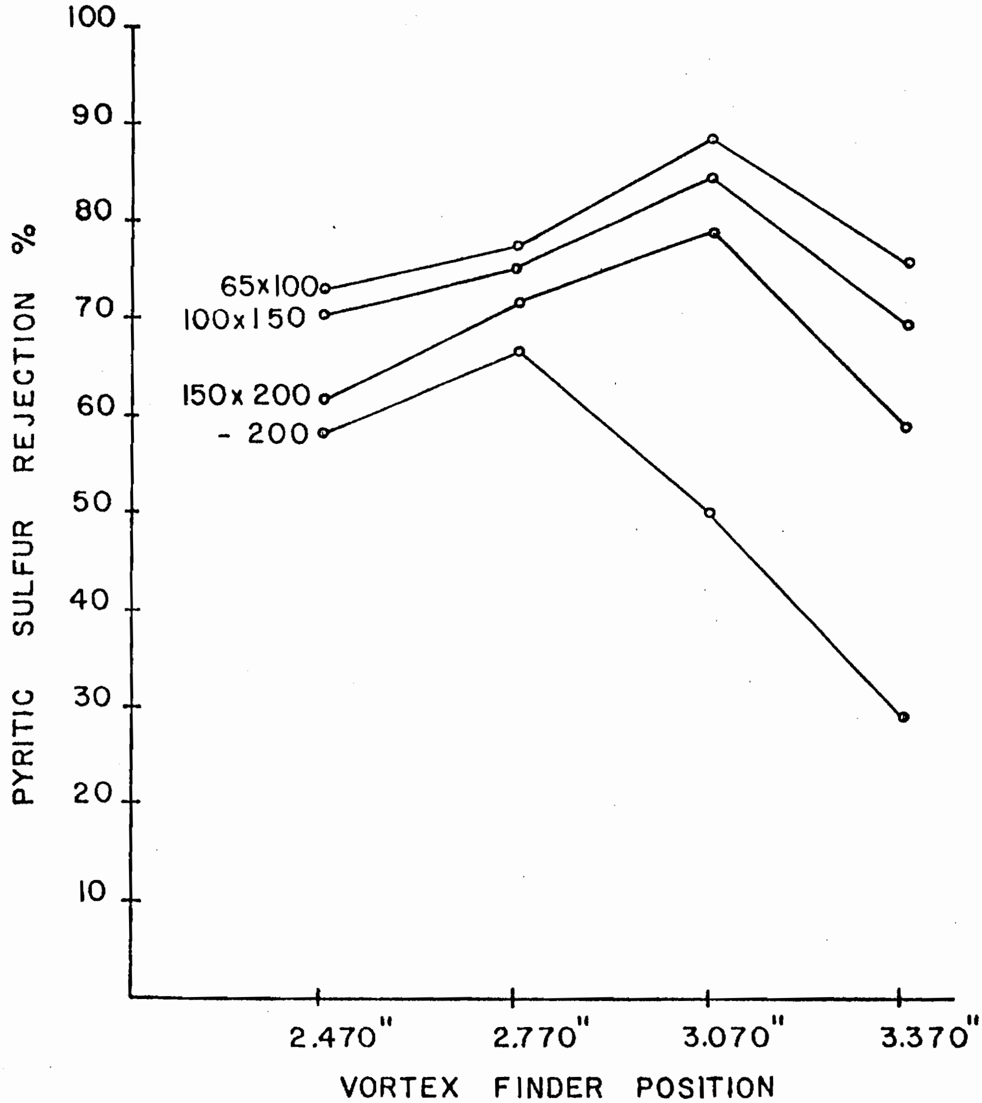


FIGURE 25 - VARIATION OF VORTEX FINDER POSITION NEW MEXICO COAL

finder was 2.770 inch. The pyritic sulfur rejection at this point was 66.0 per cent. It appeared that size was exerting the controlling factor rather than specific gravity. When the vortex finder was closer to the bed, these small pyritic sulfur particles were unable to penetrate the bed before they were entrained in the upward central current. When the vortex finder was raised above the optimum point, the small particles were entrained in the vortex finder current before they had had time to reach the outer wall. Short-circuiting was occurring.

Figure 26 shows the results of the variation of the inlet diameter. The optimum point was the 0.125 inch inlet diameter. The pyritic sulfur rejections were found to be: 94.8 per cent for the 65 x 100 mesh size fraction, 92.9 per cent for the 100 x 150 mesh size fraction, 94.4 per cent for the 150 x 200 mesh size fraction, and 59.4 per cent for the minus 200 mesh size fraction. At the 0.125 inch inlet diameter the throughput volume of the hydrocyclone was restricted. This allowed the particles enough time to be classified before they reached the vortex finder.

The pyritic sulfur rejection was decreased as the diameter was increased. Since the New Mexico coal has a high particle density in the heavier specific gravity fractions, it was possible that these particles interfered with each other at the higher flow rate. Therefore, the percentage of pyrite reporting to the overflow increased. In addition, as the throughput volume was increased, the gravity of separation was increased. This also caused more of the heavier particles to report to the overflow.[19]

BODY LENGTH	3"
PRESSURE	10 psi
DENSITY	10 %
CONE ANGLE	40°
VORTEX DIA.	.250"
VORTEX POS.	3.370"
APEX DIA.	.145"
S.G.	4.7-5.1

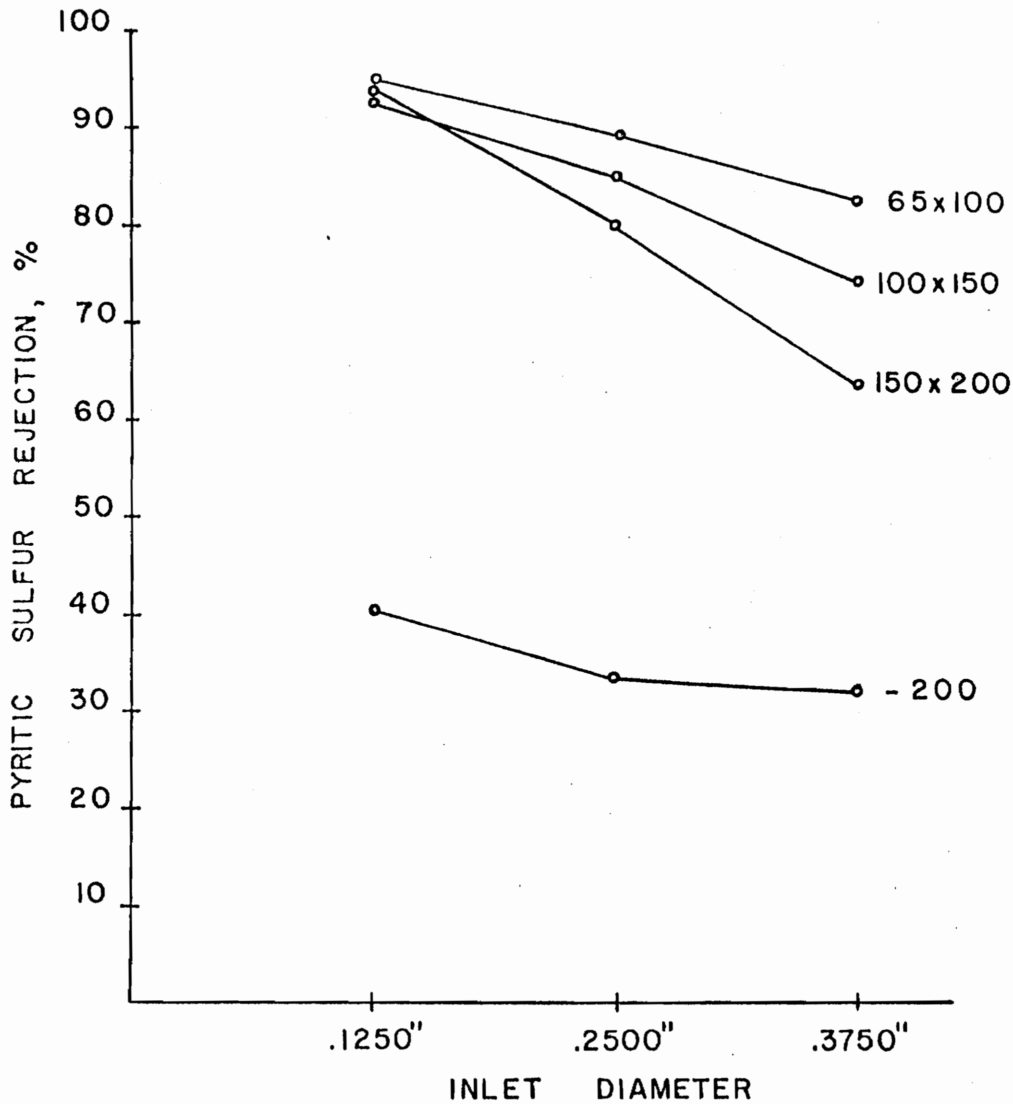


FIGURE 26 - VARIATION OF INLET DIAMETER NEW MEXICO COAL

The results of the variation of the inlet pressure are shown in Figure 27. The best results were obtained at the 15 psig point. The pyritic sulfur rejections were: 87.5 per cent for the 65 x 100 mesh size fraction, 75.9 per cent for the 100 x 150 mesh size fraction, 69.2 per cent for the 150 x 200 mesh size fraction, and 51.9 per cent for the minus 200 mesh size fraction. It appeared that because most of the particles are greater than 1.6 in specific gravity, greater pressure was required to overcome the interference between the particles and thereby effect a good separation by specific gravity.

As the inlet pressure was reduced, first to 10 psig and then to 6 psig, the rejection of pyritic sulfur was decreased. It was felt that the acceleration given the particles was not enough to overcome the interference between them, and that good segregation by specific gravity was not achieved.

The response of the hydrocyclone to variations in the cone angle is shown in Figure 28. The best results were obtained with the 40 degree cone. The following pyritic sulfur rejections were obtained: 81.9 per cent for the 65 x 100 mesh size fraction, 73.9 per cent for the 100 x 150 mesh size fraction, 62.7 per cent for the 150 x 200 mesh size fraction, and 31.5 per cent for the minus 200 mesh size fraction. The bed formed with this cone angle was thin enough that there was sufficient clearance between it and the vortex finder, and thick enough so that the hydrocyclone separated on the basis of specific gravity.[13]

The 20 degree cone caused more pyritic sulfur to report to the overflow. This was due to the fact that the bed which formed was so thick that the distance between it and the vortex finder was signifi-

S.G.	4.7 - 5.1
BODY LENGTH	3"
DENSITY	10%
CONE ANGLE	40°
INLET DIA.	.375"
VORTEX DIA.	.250"
VORTEX POS.	3.370"
APEX DIA.	.145"

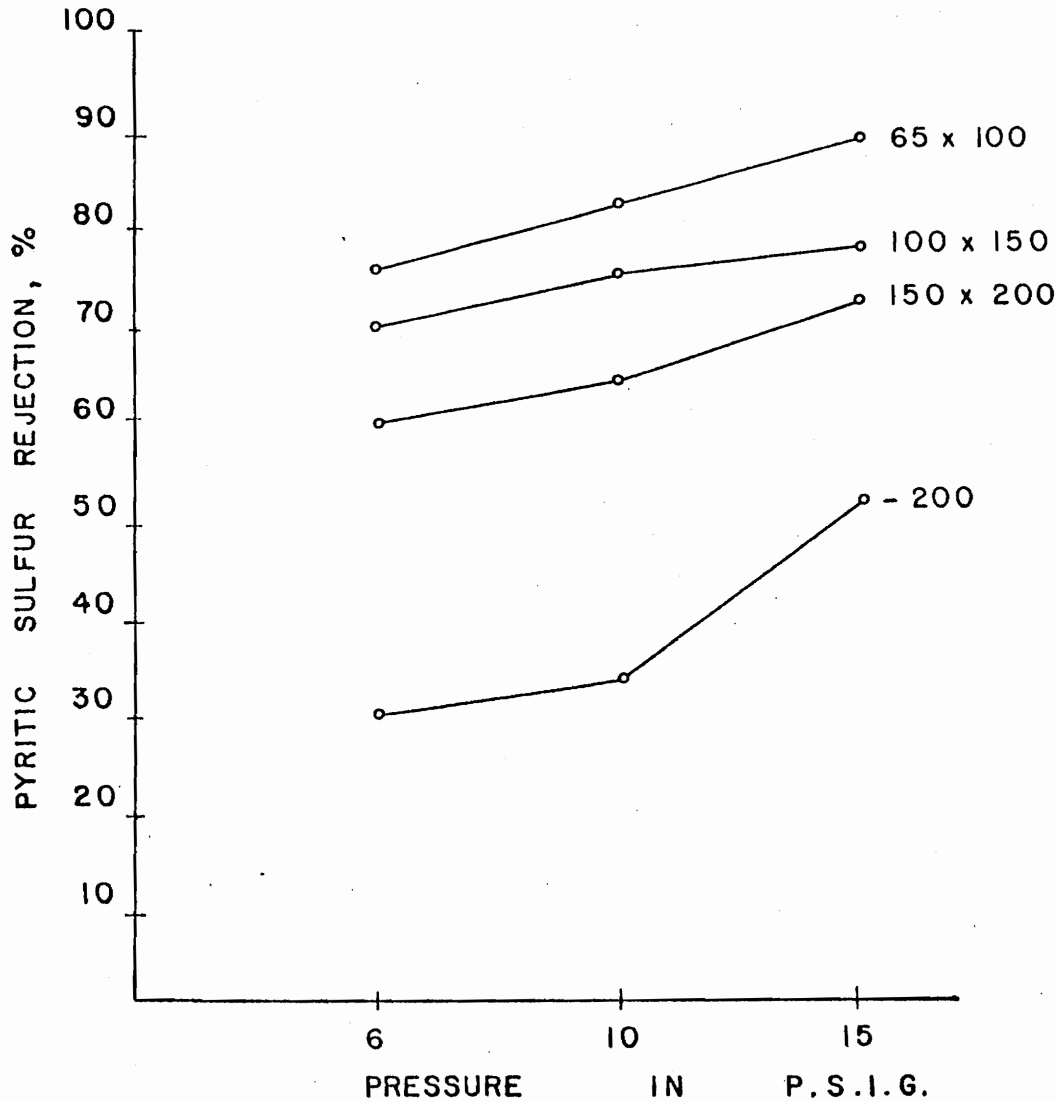


FIGURE 27 - VARIATION OF PRESSURE NEW MEXICO COAL

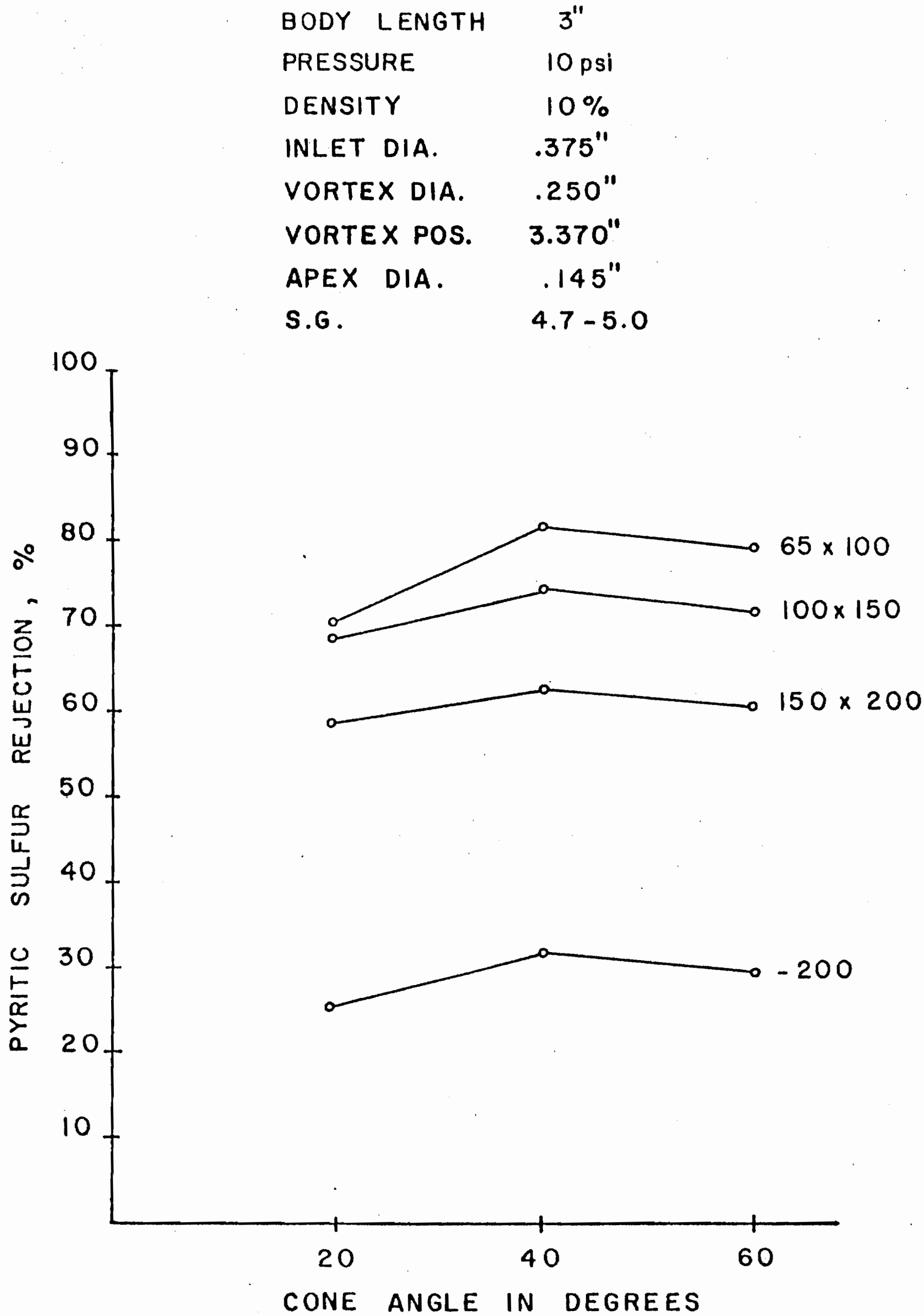


FIGURE 28 - VARIATION OF CONE ANGLE NEW MEXICO COAL

cantly decreased.

The 60 degree cone also yielded decreased pyritic sulfur rejection. This cone was steep enough that the hydrocyclone was starting to act as a classifying hydrocyclone.

The clean coal recovery curves are shown in Figure 29. They reflect the previously established trend that the finer particles tended to report to the overflow product stream.

Comparison of the Pocahontas #3 coal to the Hagy coal

The results of the tests on the Pocahontas #3 coal and the Hagy coal compared quite closely for the three coarser size fractions. Each coal followed the same trends with respect to the variations in vortex finder position, inlet diameter, inlet pressure, and cone angle.

However, for the minus 200 mesh size fraction there was considerable difference between the two coals. The rejection coefficients were much lower for the Pocahontas #3 coal than for the Hagy coal. This was explained by the fact that the Pocahontas #3 coal has a small quantity of near gravity material in the gravity of separation specific gravity fraction. The bed which developed was less dense for the Pocahontas #3 coal than for the Hagy coal. Therefore, with the Pocahontas #3 coal these very fine pyritic sulfur particles had a greater tendency to report to the overflow product stream.

It was expected that the slightly heavier Hagy coal would have a higher gravity of separation than the Pocahontas #3 coal. However, no conclusive results were obtained. Some points on the recovery partition curve were higher than for the Pocahontas #3 coal and some

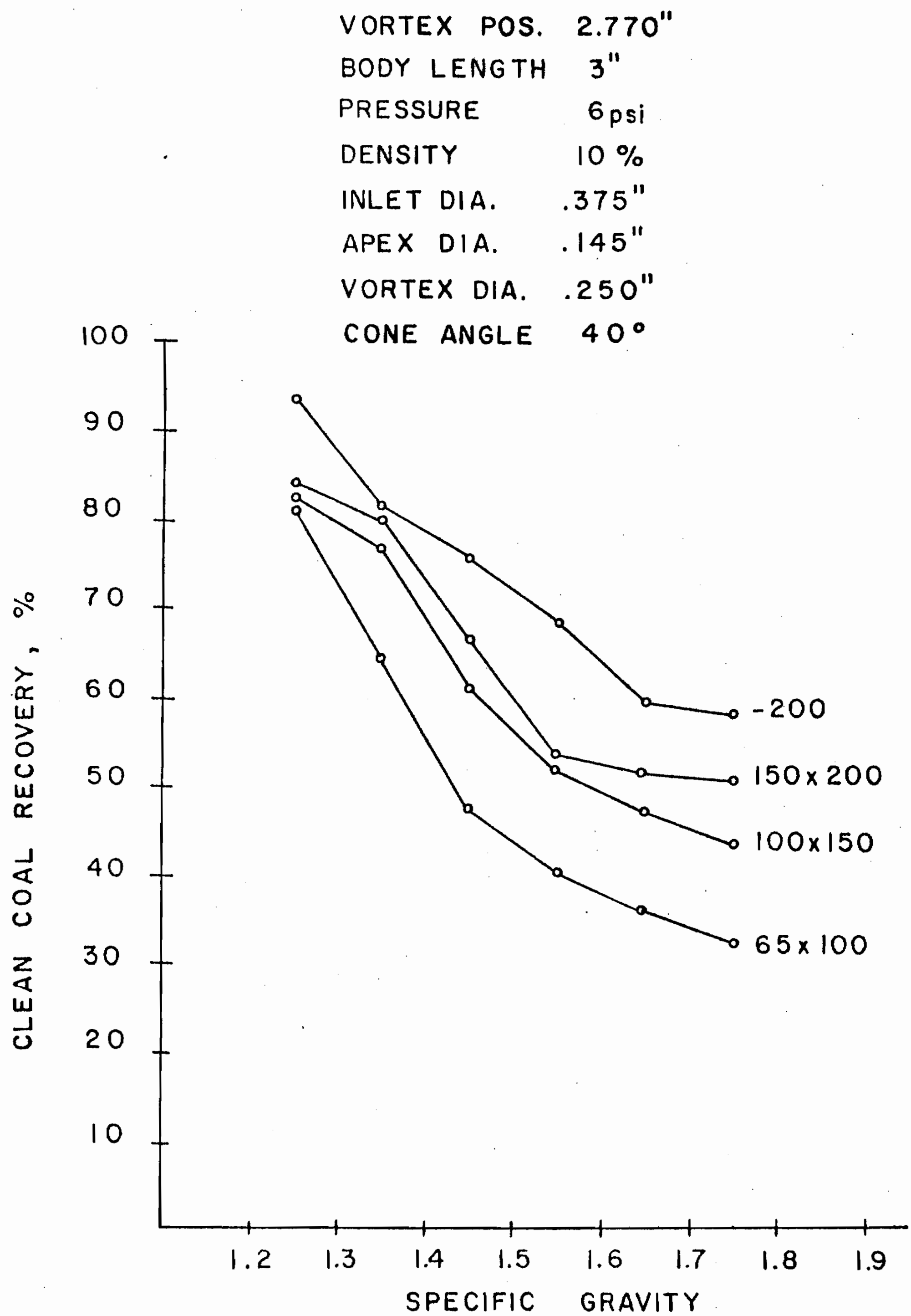


FIGURE 29 RECOVERY PARTITION CURVES NEW MEXICO COAL

were lower. Since the response of these coals was so similar, they will be treated together when compared to the New Mexico coal.

Comparison of the Hagy coal and the Pocahontas #3 coal to the New Mexico coal

Significant differences were observed between the results obtained for the New Mexico coal and the results from the tests on the Pocahontas #3 coal and the Hagy coal.

The first difference may be noted on the recovery partition curves for each coal. The New Mexico coal showed much higher recoveries. This may be attributed to the fact that the much heavier New Mexico coal gave a higher gravity of separation due to the heavier bed formed.

All three coals yielded the best pyritic sulfur rejection at the 40 degree cone angle. However, for the New Mexico coal, pyritic sulfur rejection was lower than for the two other coals. This was explained by the fact that the denser bed formed by the heavier New Mexico coal increased the gravity of separation. Therefore, more pyritic sulfur reported to the overflow.

A comparison of the results of the variation of the inlet pressure for the three coals showed a marked difference. Best results were obtained at 10 psig for the two lighter coals. For the New Mexico coal best results were obtained at 15 psig. Again, it appeared that the great amount of heavier particles in the New Mexico coal required more acceleration to properly classify according to specific gravity. It would be interesting to see if the sulfur rejection decreased at higher pressures. This would be expected as a thicker bed would be formed due to the increase in throughput capacity.[20]

Differences were noted between the heavy coal and the two lighter coals with respect to the inlet diameter variations. For the lighter coals, the best results were obtained at the 0.250 inch diameter inlet. For the heavy coal, best results were obtained at the 0.125 inch diameter inlet. The heavier coal evidently required a lower throughput for the particles to separate efficiently by specific gravity.

A comparison between the graphs of the two lighter coals and the heavy coal for the vortex finder positions showed differences. The lighter coals have the best sulfur rejection at the 2.770 inch vortex finder position. The New Mexico coal yielded the best results at the 3.070 inch vortex finder position. This indicated that a heavy coal required more residence time to classify than a lighter coal.

<u>Addition of 30 per cent limestone to the Pocahontas #3 coal</u>

The results of the comparison of the heavy coal to the two lighter coals showed that the recovery of clean coal was much higher for the heavy coal. It was decided to investigate the performance of the hydrocyclone on the Pocahontas #3 coal when heavy material was added. 30 per cent by weight of limestone was added to the reservoir. The 180 grams was sized to follow the size distribution of the synthetic material already in the system. The results of these tests are shown in Figures 30, 31, 32, and 33. In each case the recovery of clean coal was increased.

This finding may have application in an industrial installation of a hydrocyclone system. If the reject material were recycled into the

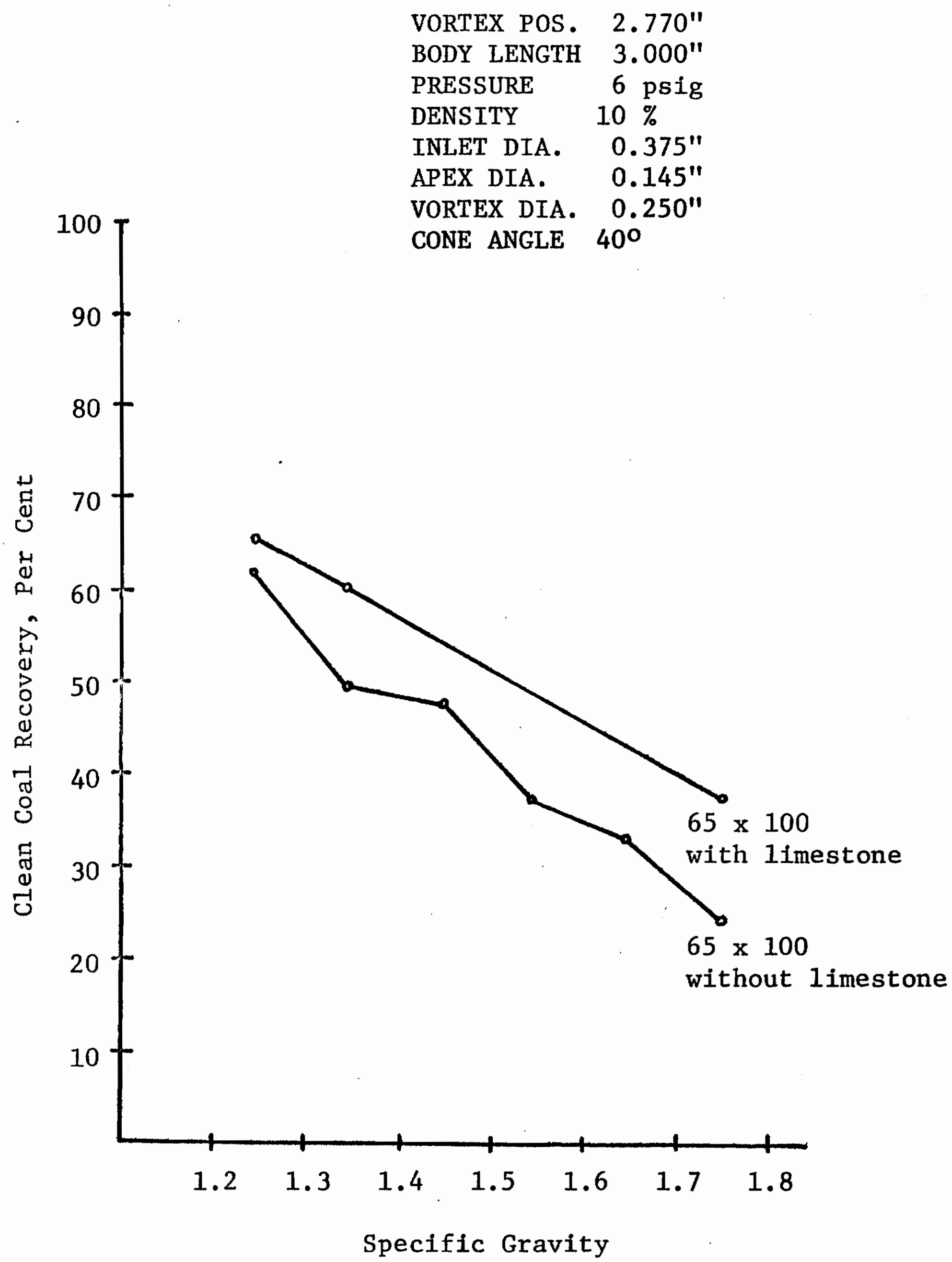


FIGURE 30 - COMPARISON OF RECOVERY PARTITION CURVES, 65 x 100 MESH, POCAHONTAS #3 COAL

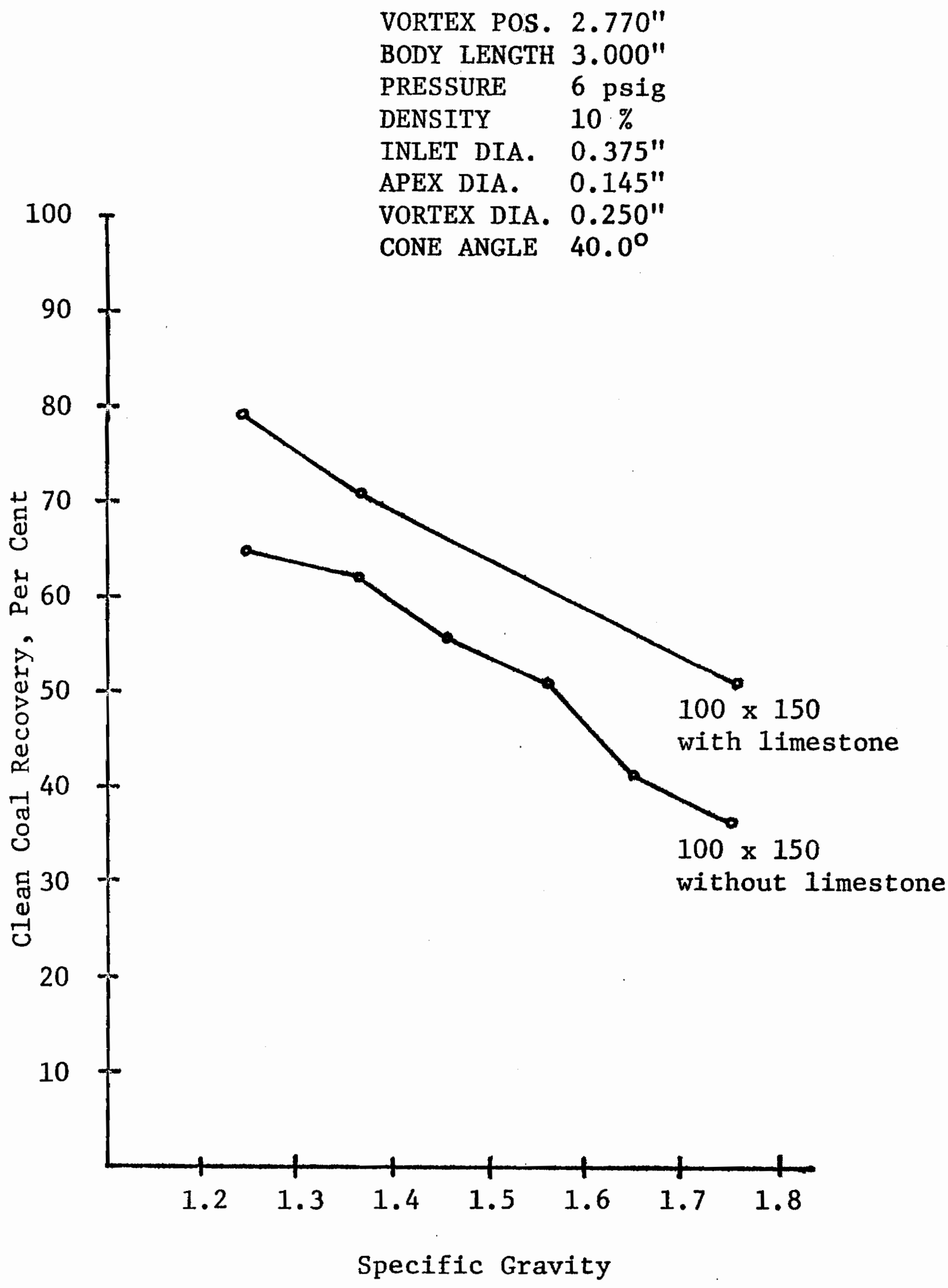


FIGURE 31 - COMPARISON OF RECOVERY PARTITION CURVES, 100 x 150 MESH, POCAHONTAS #3 COAL

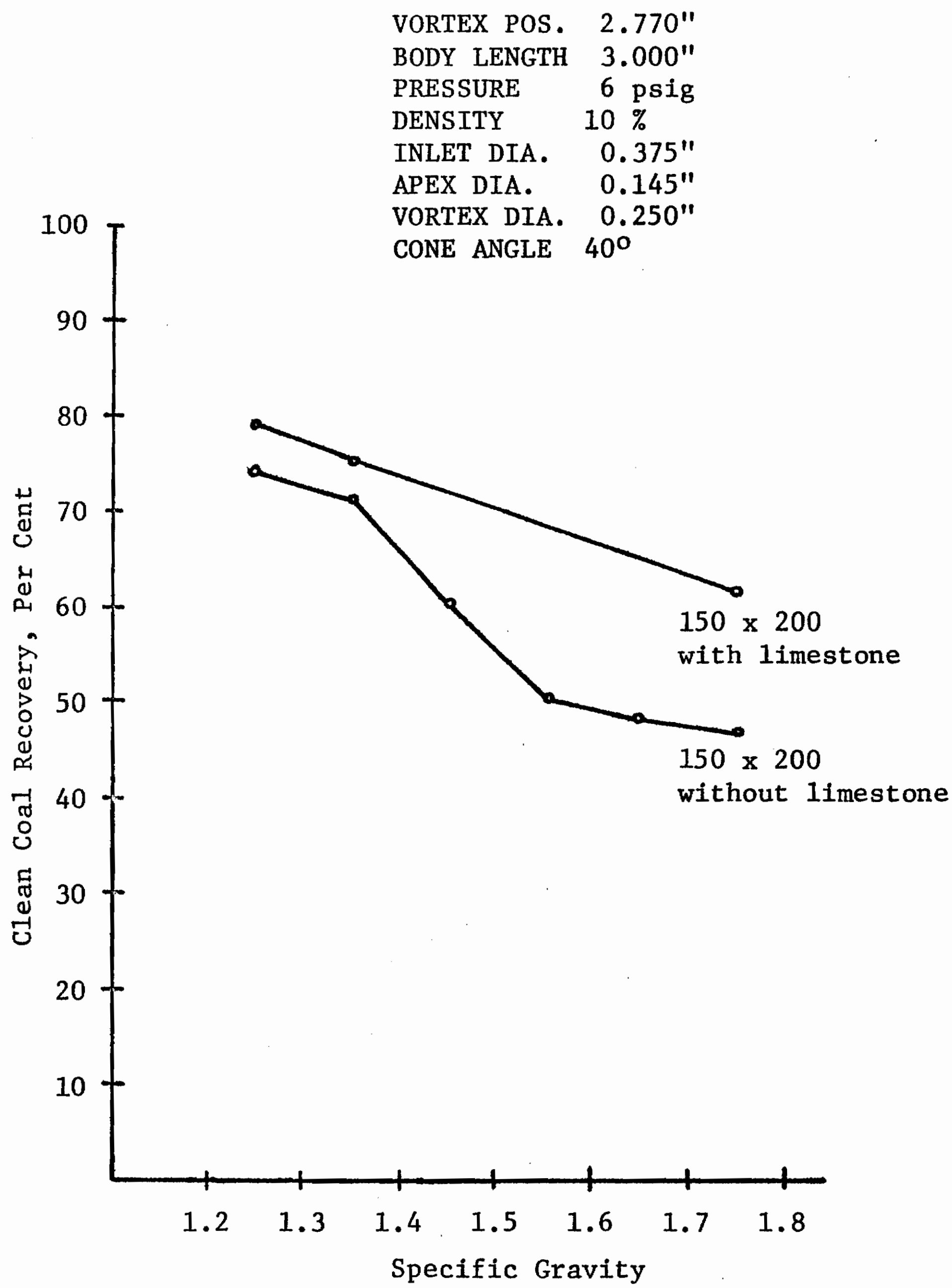


FIGURE 32 - COMPARISON OF RECOVERY PARTITION CURVES, 150 x 200 MESH, POCAHONTAS #3 COAL

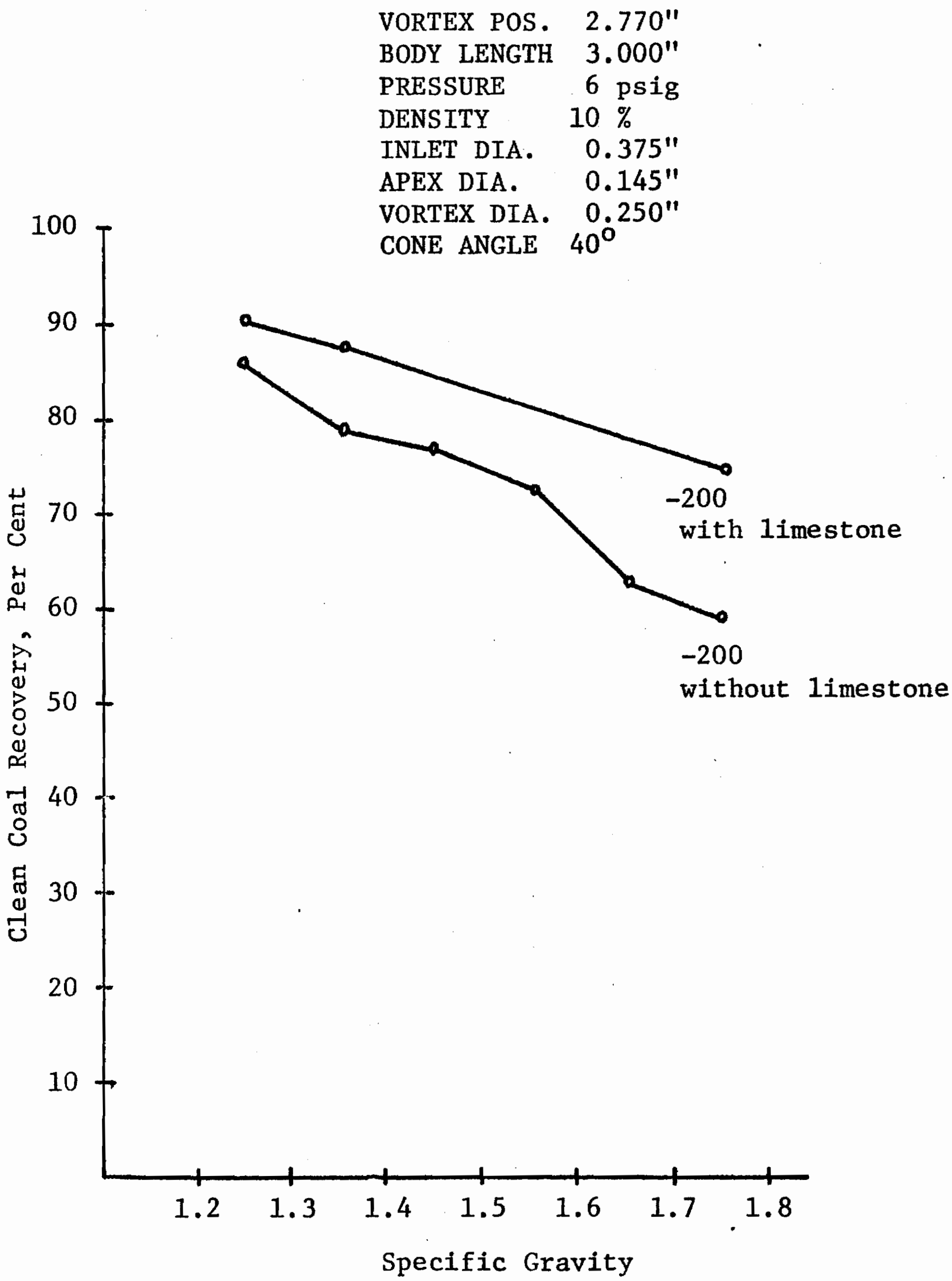


FIGURE 33 - COMPARISON OF RECOVERY PARTITION CURVES, MINUS 200 MESH, POCAHONTAS #3 COAL

feed to the unit, it might be possible to raise the clean coal recovery without adversely affecting the pyritic sulfur rejection.

Comparison of simulated results to actual results on Hagy coal

The calculated pyritic sulfur rejection for the Hagy coal was 82.2. per cent. The results of laboratory analysis, using the method described in Unived States Bureau of Mines Bulletin 638, on actual Hagy coal showed a pyritic sulfur rejection of 66.2 per cent.[21] These data and calculations may be found in Appendix D.

The difference between the two values was thought to be due to the fact that the Hagy coal did not completely liberate at minus 65 mesh. An examination under the microscope confirmed this fact. It would be necessary to further reduce this coal in order to achieve the predicted pyritic sulfur rejection.

Results of test on Pittsburgh coal

It was decided to repeat the test on actual coal using a coal which liberated at 65 mesh. The Pittsburgh coal was chosen. The results of this test showed that 78.9 per cent of the pyritic sulfur was removed. These data may be found in Appendix D.

VIII. ECONOMIC ANALYSIS

This analysis will provide an estimate of the cost of a hydrocyclone system to process 100 tons per hour of coal.[22]

Capital Costs

Thickener (installed)	$300,000.00
Vacuum Filter (installed)	250,000.00
Floor space 200 $ft.^2$ @ $50.00/$ft.^2$	10,000.00
Settling Pond, pipe, pumps	8,000.00
Pumps for Hydrocyclone 14.67 HP @ $150.00/installed HP	2,200.00
Total Capital Costs	$570,200.00

Analysis

Item	Requirement	Unit Cost	Processing Cost $/ton
Direct Production Cost			
Hydrocyclones	6,800/year	$10.00/unit	0.091
Power	86,250 KW/year	.02/KW	0.002
Water	1.7×10^9 gal/year	.03/1000 gal	0.065
Operating Labor	1 person/shift	$6.25/man/hr	0.049
Maintenance	5 % working capital		0.036
Supervision	1 person/week	$7.50/man/hr	0.019
Laboratory			0.050
		Sub-Total	0.312

Indirect Production Cost		
Depreciation	10 % of capital	0.072
Plant Overhead	80 % of labor	0.054
Taxes and Insurance	2 % yr. of capital	0.014
Interest on working capital @	9 % year	0.007
	Sub-Total	0.147
Net Production Cost in $/ton		$0.459

IX. RECOMMENDATIONS FOR FUTURE WORK

Bench scale studies illustrated that a substantial reduction in pyritic sulfur can be realized by processing coal through a 1.0" diameter hydrocyclone. According to preliminary estimates, the hydrocyclone process is economically attractive. It is obvious that removing the pyritic sulfur before burning will eliminate the need for expensive air pollution equipment to remove the sulfur oxides from the hot stack gases.

In order to investigate this method in more detail, a pilot scale study is recommended. It is suggested that a system which can process 10 tons per hour be installed at the Virginia Polytechnic Institute and State University power plant. A system of this size would provide needed operating data on abrasion, corrosion, and plugging characteristics of the hydrocyclones. The advantages of the close proximity of the Virginia Polytechnic Institute and State University power plant to the technical support facilities of the Division of Minerals Engineering cannot be discounted.

The magnetic material tracer method has substantially reduced the amount of time required to obtain partition curves. Further research is needed to refine this batch method into a continuous sensing system. Such a system could be installed in an operating preparation plant to provide a means of constantly monitoring plant performance. This would enable the cleaning plant to respond to changes in the feed and result in more efficient plant operation.

X. SUMMARY AND CONCLUSIONS

An investigation of the literature showed that as the diameter of a hydrocyclone was decreased, smaller particles could be effectively separated on the basis of specific gravity. Consequently, a laboratory scale study was conducted to determine the possibility of removing pyritic sulfur from fine coal using a 1.0" diameter hydrocyclone.

A 1.0" diameter hydrocyclone was constructed from stainless steel. The hydrocyclone was fabricated with easily interchangable parts that permitted rapid variations in the inlet diameter, apex discharge diameter, vortex finder diameter, vortex finder position, and cone angle. The feed pressure to the hydrocyclone could also be varied. A recirculating system was used.

Synthetic material was used in place of actual coal during this investigation. The synthetic material, made from casting resin and barium sulfate, duplicated the coal with respect to particle size, shape, and specific gravity distribution. The synthetic coal had the advantage of not degrading during the course of the experiment.

Three coals were simulated for the purposes of this experiment. Pocahontas #3 coal served as the "light" coal, the Hagy seam was selected as the "medium" coal, and coal from the Cortes area of New Mexico was chosen as the "heavy" coal. By optimizing the 1.0" diameter hydrocyclone for these three coals, it would be possible to extend the results of the investigation to any coal.

The results of the investigation showed that the 1.0" diameter hydrocyclone was an effective device for removing pyritic sulfur from

65 x 200 mesh coal. The results for the minus 200 mesh size fraction were not good. A smaller diameter hydrocyclone would be necessary in order to effectively process the minus 200 mesh size fraction.

The results of tests on the simulated Hagy coal showed that 82.2 per cent of the pyritic sulfur could be removed using the 1.0" diameter hydrocyclone. A test run on actual Hagy coal yielded a pyritic sulfur rejection value of 66.2 per cent. The difference between the predicted value and the actual value of pyritic sulfur rejection was explained by the fact that the Hagy coal did not liberate until crushed to approximately 150 mesh. Therefore, non-liberated pyrite particles were carried by the coal into the clean coal product stream.

The test was repeated on coal from the Pittsburgh seam. This coal liberated at approximately 65 mesh. The pyritic sulfur rejection was found to be 78.9 per cent.

Magnetically tagged material was used to rapidly obtain the data necessary to calculate partition coefficients. One gram of tagged material of the size range and specific gravity under investigation was introduced into the system. The weight of the tagged material in the overflow was divided by the weight of material in the overflow and underflow to yield the pyritic sulfur rejection coefficient. This method allowed a partition coefficient to be obtained within a few minutes. This method is much faster that the float-sink method.

The effect of introducing refuse material into the slurry was investigated. It was found that as the percentage of refuse in the slurry was increased the recovery of clean coal was increased.

An economic study was performed to determine the feasibility of using a 1.0" diameter hydrocyclone system to remove pyritic sulfur from coal. The hydrocyclone system was sized to process 100 tons of coal per hour. Included in the analysis are direct production costs, such as water and power, and indirect production costs, such as insurance and depreciation. The cost of processing coal through the system was estimated to be $0.459 per ton.

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APPENDIX A

PROOF THAT MAGNETIC MATERIAL PARTITION COEFFICIENTS ARE IDENTICAL WITH FLOAT-SINK METHOD PARTITION COEFFICIENTS AND TEST DATA

Proof that magnetic method equals float-sink method

DATA POINTS

Test	Tagged Material Partition Coefficient	Test	Float-Sink Partition Coefficient	d_i	d_i^2
1	60.0		57.0	3.0	9.0
2	59.0		65.0	-6.0	36.0
3	64.0		82.0	-18.0	324.0
4	75.0		81.0	6.0	36.0
5	55.0		53.0	2.0	4.0
6	60.0		54.7	5.3	28.1
7	54.8		61.0	-6.2	38.4
8	89.7		79.0	10.7	114.5
9	41.0		35.0	6.0	36.0
10	50.8	20	46.7	4.1	16.8
	$\overline{X}_1 = 60.9$		$\overline{X}_2 = 61.4$	$\Sigma d_i = 6.9$	$\Sigma d_i^2 = 642.8$

$\overline{d} = .5$

$t = 10 - 1 = 9$

Hypothesis: $\overline{X}_1 = \overline{X}_2$

Choose $\alpha = .05$

$t_{.975} = 2.262$

Reject if $t < -2.262$ or $t > 2.262$

$$t = \frac{d - 0}{S_d / \sqrt{N}}$$

$$S_d = \frac{642.8 - (6.9)^2/10}{9} = 70.9$$

$$t = \frac{-0.5}{\frac{8.4}{3.2}} = 0.19$$

Therefore: accept hypothesis $\overline{X}_1 = \overline{X}_2$.

TABLE 5 - TAGGED MATERIAL DATA

TEST NUMBER	COAL NUMBER	SPECIFIC GRAVITY UNDER INVESTIGATION	MESH	BODY LENGTH IN INCHES
1	2	1.4 - 1.5	65 x 100	3.0
2	2	1.4 - 1.5	100 x 150	3.0
3	2	1.4 - 1.5	150 x 200	3.0
4	2	1.4 - 1.5	minus 200	3.0
5	2	1.5 - 1.6	65 x 100	3.0
6	2	1.5 - 1.6	100 x 150	3.0
7	2	1.5 - 1.6	150 x 200	3.0
8	2	1.5 - 1.6	minus 200	3.0
9	2	1.6 - 1.7	65 x 100	3.0
10	2	1.6 - 1.7	100 x 150	3.0

TABLE 5 CONTINUED

TEST NUMBER	INLET DIAMETER IN INCHES	CONE ANGLE IN DEGREES	VORTEX FINDER IN INCHES	APEX DISCHARGE IN INCHES
1	0.375	40	0.250	0.125
2	0.375	40	0.250	0.125
3	0.375	40	0.250	0.125
4	0.375	40	0.250	0.125
5	0.375	40	0.250	0.125
6	0.375	40	0.250	0.125
7	0.375	40	0.250	0.125
8	0.375	40	0.250	0.125
9	0.375	40	0.250	0.125
10	0.375	40	0.250	0.125

TABLE 5 CONTINUED

TEST NUMBER	PRESSURE IN PSIG	SLURRY DENSITY IN PER CENT	VORTEX FINDER POSITION IN INCHES	OVERFLOW VOLUME IN MILLILITERS
1	6	10	2.770	750
2	6	10	2.770	750
3	6	10	2.770	730
4	6	10	2.770	760
5	6	10	2.770	800
6	6	10	2.770	710
7	6	10	2.770	730
8	6	10	2.770	770
9	6	10	2.770	750
10	6	10	2.770	720

TABLE 5 CONTINUED

TEST NUMBER	UNDERFLOW VOLUME IN MILLILITERS	TIME IN SECONDS	WEIGHT TAGGED MATERIAL OVERFLOW IN GRAMS
1	100	6.0	.0781
2	100	6.0	.0912
3	105	6.0	.0607
4	90	6.0	.0474
5	105	6.5	.0534
6	80	5.9	.0974
7	105	6.4	.0559
8	105	6.7	.0816
9	110	6.0	.0768
10	100	6.0	.0524

TABLE 5 CONTINUED

TEST NUMBER	WEIGHT TAGGED MATERIAL IN UNDERFLOW IN GRAMS	PARTITION COEFFICIENT PER CENT
1	.0520	60
2	.0630	59
3	.0344	64
4	.0156	75
5	.0429	55
6	.0634	60
7	.0462	54.8
8	.0216	79
9	.1076	41.6
10	.0508	50.8

TABLE 6 - FLOAT - SINK DATA

TEST NUMBER	COAL NUMBER	SPECIFIC GRAVITY UNDER INVESTIGATION	MESH	BODY LENGTH IN INCHES
11	2	1.4 - 1.5	65 x 100	3.0
12	2	1.4 - 1.5	100 x 150	3.0
13	2	1.4 - 1.5	150 x 200	3.0
14	2	1.4 - 1.5	minus 200	3.0
15	2	1.5 - 1.6	65 x 100	3.0
16	2	1.5 - 1.6	100 x 150	3.0
17	2	1.5 - 1.6	150 x 200	3.0
18	2	1.5 - 1.6	minus 200	3.0
19	2	1.6 - 1.7	65 x 100	3.0
20	2	1.6 - 1.7	100 x 150	3.0

TABLE 6 CONTINUED

TEST NUMBER	INLET DIAMETER IN INCHES	CONE ANGLE IN DEGREES	VORTEX FINDER IN INCHES	APEX DISCHARGE IN INCHES
11	0.375	40	0.250	0.125
12	0.375	40	0.250	0.125
13	0.375	40	0.250	0.125
14	0.375	40	0.250	0.125
15	0.375	40	0.250	0.125
16	0.375	40	0.250	0.125
17	0.375	40	0.250	0.125
18	0.375	40	0.250	0.125
19	0.375	40	0.250	0.125
20	0.375	40	0.250	0.125

TABLE 6 CONTINUED

TEST NUMBER	PRESSURE IN PSIG	SLURRY DENSITY IN PER CENT	VORTEX FINDER POSITION IN INCHES	OVERFLOW VOLUME IN MILLILITERS
11	6	10	2.770	750
12	6	10	2.770	755
13	6	10	2.770	730
14	6	10	2.770	760
15	6	10	2.770	800
16	6	10	2.770	710
17	6	10	2.770	730
18	6	10	2.770	770
19	6	10	2.770	750
20	6	10	2.770	720

TABLE 6 CONTINUED

TEST NUMBER	UNDERFLOW VOLUME IN MILLILITERS	TIME IN SECONDS	WEIGHT FLOAT - SUNK MATERIAL IN OVERFLOW IN GRAMS
11	100	6.0	5.1
12	105	6.0	3.5
13	105	6.0	1.9
14	90	6.0	7.1
15	105	6.5	0.36
16	80	5.9	2.5
17	105	6.4	0.78
18	105	6.7	3.1
19	110	6.0	0.20
20	100	5.9	0.44

TABLE 6 CONTINUED

TEST NUMBER	UNDERFLOW VOLUME IN MILLILITERS	TIME IN SECONDS	WEIGHT FLOAT - SINK MATERIAL IN OVERFLOW IN GRAMS
11	100	6.0	5.1
12	105	6.0	3.5
13	105	6.0	1.9
14	90	6.0	7.1
15	105	6.5	0.36
16	80	5.9	2.5
17	105	6.4	0.78
18	105	6.7	3.1
19	110	6.0	0.20
20	100	5.9	0.44

TABLE 6 CONTINUED

TEST NUMBER	WEIGHT FLOAT - SUNK MATERIAL IN UNDERFLOW IN GRAMS	PARTITION COEFFICIENT PER CENT
11	3.9	57
12	1.9	65
13	0.4	82
14	1.7	81
15	0.33	53
16	2.1	54.7
17	0.50	61
18	0.36	89.7
19	0.37	35.6
20	0.52	46.7

APPENDIX B

PROOF THAT MAGNETIC MATERIAL PARTITION COEFFICIENTS ARE REPEATABLE AND TEST DATA

Proof that data is repeatable

Data Points

Run 1	Run 2	d_i	d_i^2
60.6	64.9	-4.3	18.49
62.7	62.7	0.0	0.00
69.6	67.2	2.4	5.76
83.5	81.0	2.5	6.25
68.3	67.5	0.8	0.64
42.5	51.6	-9.1	82.81
49.6	45.4	4.2	17.64
64.1	63.5	0.6	0.36
68.2	65.0	3.2	10.24
61.8	65.4	-3.6	12.96
55.7	60.3	-4.6	21.16
54.0	63.8	-9.8	96.04
70.6	79.0	-8.4	70.56
$\overline{X}_1 = 62.4$	$\overline{X}_2 = 64.4$	$\Sigma d_i = -26.1$	$\Sigma d_i^2 = 342.81$

$\overline{d} = -2.0$

$t = 13 - 1 = 12$

Hypothesis: $\overline{X_1} = \overline{X_2}$

Choose $\alpha = .05$

$t_{.975} = 2.160$

Reject if $t < -2.160$ or $t > 2.160$

$$t = \frac{\overline{d} - 0}{S_d/\sqrt{N}}$$

$$S_d = \frac{342.81 - (26.1)^2/13}{12} = 24.2$$

$$t = \frac{-2.0}{\frac{4.9}{3.6}} = -1.47$$

Therefore: accept hypothesis $\overline{X_1} = \overline{X_2}$.

TABLE 7 - PAIRED DATA POINTS

TEST NUMBER	COAL NUMBER	SPECIFIC GRAVITY UNDER INVESTIGATION	MESH	BODY LENGTH IN INCHES
21	2	1.4 - 1.5	65 x 100	3.0
22	2	1.4 - 1.5	65 x 100	3.0
23	2	1.4 - 1.5	100 x 150	3.0
24	2	1.4 - 1.5	100 x 150	3.0
25	2	1.4 - 1.5	150 x 200	3.0
26	2	1.4 - 1.5	150 x 200	3.0
27	2	1.4 - 1.5	minus 200	3.0
28	2	1.4 - 1.5	minus 200	3.0
29	2	1.5 - 1.6	minus 200	3.0
30	2	1.5 - 1.6	minus 200	3.0
31	2	1.6 - 1.7	65 x 100	3.0
32	2	1.6 - 1.7	65 x 100	3.0
33	2	1.6 - 1.7	100 x 150	3.0
34	2	1.6 - 1.7	100 x 150	3.0
35	2	1.6 - 1.7	150 x 200	3.0
36	2	1.6 - 1.7	150 x 200	3.0
37	2	1.6 - 1.7	minus 200	3.0
38	2	1.6 - 1.7	minus 200	3.0
39	2	1.7 - 1.8	65 x 100	3.0
40	2	1.7 - 1.8	65 x 100	3.0
41	2	1.7 - 1.8	100 x 150	3.0
42	2	1.7 - 1.8	100 x 150	3.0
43	2	1.7 - 1.8	150 x 200	3.0
44	2	1.7 - 1.8	150 x 200	3.0
45	2	1.7 - 1.8	minus 200	3.0
46	2	1.7 - 1.8	minus 200	3.0

TABLE 7 CONTINUED

TEST NUMBER	INLET DIAMETER IN INCHES	CONE ANGLE IN DEGREES	VORTEX FINDER IN INCHES	APEX DISCHARGE IN INCHES
21	0.375	40	0.250	0.125
22	0.375	40	0.250	0.125
23	0.375	40	0.250	0.125
24	0.375	40	0.250	0.125
25	0.375	40	0.250	0.125
26	0.375	40	0.250	0.125
27	0.375	40	0.250	0.125
28	0.375	40	0.250	0.125
29	0.375	40	0.250	0.125
30	0.375	40	0.250	0.125
31	0.375	40	0.250	0.125
32	0.375	40	0.250	0.125
33	0.375	40	0.250	0.125
34	0.375	40	0.250	0.125
35	0.375	40	0.250	0.125
36	0.375	40	0.250	0.125
37	0.375	40	0.250	0.125
38	0.375	40	0.250	0.125
39	0.375	40	0.250	0.125
40	0.375	40	0.250	0.125
41	0.375	40	0.250	0.125
42	0.375	40	0.250	0.125
43	0.375	40	0.250	0.125
44	0.375	40	0.250	0.125
45	0.375	40	0.250	0.125
46	0.375	40	0.250	0.125

TABLE 7 CONTINUED

TEST NUMBER	PRESSURE IN PSIG	SLURRY DENSITY IN PER CENT	VORTEX FINDER POSITION IN INCHES	OVERFLOW VOLUME IN MILLILITERS
21	6	10	2.770	775
22	6	10	2.770	750
23	6	10	2.770	615
24	6	10	2.770	606
25	6	10	2.770	690
26	6	10	2.770	700
27	6	10	2.770	725
28	6	10	2.770	740
29	6	10	2.770	760
30	6	10	2.770	750
31	6	10	2.770	710
32	6	10	2.770	725
33	6	10	2.770	715
34	6	10	2.770	675
35	6	10	2.770	735
36	6	10	2.770	745
37	6	10	2.770	730
38	6	10	2.770	710
39	6	10	2.770	740
40	6	10	2.770	800
41	6	10	2.770	750
42	6	10	2.770	675
43	6	10	2.770	725
44	6	10	2.770	750
45	6	10	2.770	700
46	6	10	2.770	680

TABLE 7 CONTINUED

TEST NUMBER	UNDERFLOW VOLUME IN MILLILITERS	TIME IN SECONDS	WEIGHT TAGGED MATERIAL OVERFLOW IN GRAMS
21	150	7.0	0.1894
22	100	5.5	0.0834
23	75	5.0	0.2484
24	90	4.0	0.2030
25	75	5.5	0.2821
26	100	5.5	0.3212
27	80	5.0	0.2653
28	100	6.0	0.2695
29	95	5.8	0.5365
30	175	6.6	0.5447
31	100	5.6	0.1469
32	100	5.9	0.1844
33	100	6.0	0.3309
34	125	6.0	0.2741
35	120	6.9	0.3534
36	110	6.0	0.3122
37	100	6.3	0.6712
38	100	6.3	0.6126
39	105	6.3	0.4424
40	110	6.9	0.4065
41	110	6.6	0.4931
42	105	6.9	0.4927
43	105	6.1	0.6468
44	100	6.9	0.8540
45	100	6.0	0.4401
46	105	6.0	0.9621

TABLE 7 CONTINUED

TEST NUMBER	WEIGHT TAGGED MATERIAL IN UNDERFLOW IN GRAMS	PARTITION COEFFICIENT PER CENT
21	0.1230	60.6
22	0.1548	64.9
23	0.1477	62.7
24	0.1203	62.7
25	0.1232	69.6
26	0.1573	67.2
27	0.0524	83.5
28	0.0632	81.0
29	0.2489	68.3
30	0.2672	67.5
31	0.1469	42.5
32	0.1731	51.6
33	0.3369	49.6
34	0.3254	45.4
35	0.1982	64.1
36	0.1792	63.5
37	0.3128	68.2
38	0.3294	65.0
39	0.2754	61.8
40	0.2152	65.4
41	0.3920	55.6
42	0.3239	60.3
43	0.5508	54.0
44	0.4828	63.8
45	0.1833	70.6
46	0.2417	79.0

APPENDIX C

OPTIMIZATION DATA FOR POCAHONTAS #3 COAL, HAGY COAL, NEW MEXICO COAL, AND POCAHONTAS #3 COAL WITH 30 PER CENT LIMESTONE ADDED

TABLE 8 - OPTIMIZATION DATA FOR POCAHONTAS #3 COAL

TEST NUMBER	COAL NUMBER	SPECIFIC GRAVITY UNDER INVESTIGATION	MESH	BODY LENGTH IN INCHES
47	1	4.8 - 5.1	65 x 100	3.0
48	1	4.8 - 5.1	65 x 100	3.0
49	1	4.8 - 5.1	65 x 100	3.0
50	1	4.8 - 5.1	65 x 100	3.0
51	1	4.8 - 5.1	65 x 100	3.0
52	1	4.8 - 5.1	65 x 100	3.0
53	1	4.8 - 5.1	65 x 100	3.0
54	1	4.8 - 5.1	65 x 100	3.0
55	1	4.8 - 5.1	65 x 100	3.0
56	1	4.8 - 5.1	65 x 100	3.0
57	1	4.8 - 5.1	100 x 150	3.0
58	1	4.8 - 5.1	100 x 150	3.0
59	1	4.8 - 5.1	100 x 150	3.0
60	1	4.8 - 5.1	100 x 150	3.0
61	1	4.8 - 5.1	100 x 150	3.0
62	1	4.8 - 5.1	100 x 150	3.0
63	1	4.8 - 5.1	100 x 150	3.0
64	1	4.8 - 5.1	100 x 150	3.0
65	1	4.8 - 5.1	100 x 150	3.0
66	1	4.8 - 5.1	100 x 150	3.0
67	1	4.8 - 5.1	150 x 200	3.0
68	1	4.8 - 5.1	150 x 200	3.0
69	1	4.8 - 5.1	150 x 200	3.0
70	1	4.8 - 5.1	150 x 200	3.0
71	1	4.8 - 5.1	150 x 200	3.0
72	1	4.8 - 5.1	150 x 200	3.0
73	1	4.8 - 5.1	150 x 200	3.0
74	1	4.8 - 5.1	150 x 200	3.0
75	1	4.8 - 5.1	150 x 200	3.0
76	1	4.8 - 5.1	150 x 200	3.0
77	1	4.8 - 5.1	minus 200	3.0
78	1	4.8 - 5.1	minus 200	3.0
79	1	4.8 - 5.1	minus 200	3.0
80	1	4.8 - 5.1	minus 200	3.0
81	1	4.8 - 5.1	minus 200	3.0
82	1	4.8 - 5.1	minus 200	3.0
83	1	4.8 - 5.1	minus 200	3.0
84	1	4.8 - 5.1	minus 200	3.0
85	1	4.8 - 5.1	minus 200	3.0
86	1	4.8 - 5.1	minus 200	3.0

TABLE 8 CONTINUED

TEST NUMBER	INLET DIAMETER IN INCHES	CONE ANGLE IN DEGREES	VORTEX FINDER IN INCHES	APEX DISCHARGE IN INCHES
47	0.375	40	0.250	0.145
48	0.375	40	0.250	0.145
49	0.375	40	0.250	0.145
50	0.375	40	0.250	0.145
51	0.125	40	0.250	0.145
52	0.250	40	0.250	0.145
53	0.375	40	0.250	0.145
54	0.375	40	0.250	0.145
55	0.375	20	0.250	0.145
56	0.375	60	0.250	0.145
57	0.375	40	0.250	0.145
58	0.375	40	0.250	0.145
59	0.375	40	0.250	0.145
60	0.375	40	0.250	0.145
61	0.125	40	0.250	0.145
62	0.250	40	0.250	0.145
63	0.375	40	0.250	0.145
64	0.375	40	0.250	0.145
65	0.375	20	0.250	0.145
66	0.375	60	0.250	0.145
67	0.375	40	0.250	0.145
68	0.375	40	0.250	0.145
69	0.375	40	0.250	0.145
70	0.375	40	0.250	0.145
71	0.125	40	0.250	0.145
72	0.250	40	0.250	0.145
73	0.375	40	0.250	0.145
74	0.375	40	0.250	0.145
75	0.375	20	0.250	0.145
76	0.375	60	0.250	0.145
77	0.375	40	0.250	0.145
78	0.375	40	0.250	0.145
79	0.375	40	0.250	0.145
80	0.375	40	0.250	0.145
81	0.125	40	0.250	0.145
82	0.250	40	0.250	0.145
83	0.375	40	0.250	0.145
84	0.375	40	0.250	0.145
85	0.375	20	0.250	0.145
86	0.375	60	0.250	0.145

TABLE 8 CONTINUED

TEST NUMBER	PRESSURE IN PSIG	SLURRY DENSITY IN PER CENT	VORTEX FINDER POSITION IN INCHES	OVERFLOW VOLUME IN MILLILITERS
47	6	10	2.470	580
48	6	10	2.770	640
49	6	10	3.070	560
50	6	10	3.770	850
51	10	10	3.370	560
52	10	10	3.370	550
53	10	10	3.370	660
54	15	10	3.370	700
55	10	10	3.370	700
56	10	10	3.370	830
57	6	10	2.470	720
58	6	10	2.770	800
59	6	10	3.070	700
60	6	10	3.370	750
61	10	10	3.370	460
62	10	10	3.370	650
63	10	10	3.370	740
64	15	10	3.370	570
65	10	10	3.370	800
66	10	10	3.370	750
67	6	10	2.470	720
68	6	10	2.770	580
69	6	10	3.070	660
70	6	10	3.370	730
71	10	10	3.370	780
72	10	10	3.370	610
73	10	10	3.370	570
74	15	10	3.370	620
75	10	10	3.370	650
76	10	10	3.370	640
77	6	10	2.470	520
78	6	10	2.770	740
79	6	10	3.070	640
80	6	10	3.370	570
81	10	10	3.370	600
82	10	10	3.370	570
83	10	10	3.370	630
84	15	10	3.370	650
85	10	10	3.370	650
86	10	10	3.370	640

TABLE 8 CONTINUED

TEST NUMBER	UNDERFLOW VOLUME IN MILLILITERS	TIME IN SECONDS	WEIGHT TAGGED MATERIAL OVERFLOW IN GRAMS
47	150	4.8	0.0097
48	130	4.8	0.0200
49	110	5.3	0.0037
50	105	6.6	0.0139
51	70	12.6	0.0098
52	90	5.4	0.0102
53	100	4.3	0.0066
54	150	4.7	0.0081
55	120	4.3	0.0328
56	120	4.2	0.0031
57	160	5.4	0.1031
58	120	5.4	0.0073
59	140	6.0	0.0184
60	140	6.6	0.1163
61	50	9.6	0.0293
62	100	6.6	0.0074
63	140	5.4	0.1608
64	170	3.3	0.0359
65	160	5.9	0.0334
66	100	4.8	0.0264
67	90	4.9	0.0042
68	125	5.4	0.0107
69	160	5.4	0.0064
70	180	4.8	0.0152
71	80	15.6	0.0440
72	90	6.0	0.0169
73	100	4.1	0.0168
74	105	3.5	0.0254
75	130	3.9	0.0090
76	120	4.3	0.0099
77	100	4.1	0.3992
78	140	5.7	0.2950
79	100	4.3	0.2358
80	110	4.2	0.3666
81	60	11.4	0.2371
82	90	4.8	0.2046
83	145	4.1	0.6247
84	110	3.3	0.7117
85	150	4.9	0.3305
86	100	4.8	0.3904

TABLE 8 CONTINUED

TEST NUMBER	WEIGHT TAGGED MATERIAL IN UNDERFLOW IN GRAMS	PARTITION COEFFICIENT PER CENT
47	0.0594	14.0
48	1.1081	1.7
49	0.0394	8.6
50	0.1237	10.1
51	0.0046	31.9
52	0.1283	7.4
53	0.0820	7.5
54	0.0692	10.5
55	0.1478	12.6
56	0.0065	32.2
57	0.2967	25.8
58	0.0864	7.8
59	0.0924	16.3
60	0.0250	17.6
61	0.0478	38.0
62	0.0793	8.5
63	0.0264	14.1
64	0.1262	22.2
65	0.1219	21.5
66	0.1086	19.5
67	0.0083	33.6
68	0.0742	12.7
69	0.0108	37.2
70	0.0104	59.3
71	0.0431	50.5
72	0.0697	19.5
73	0.0411	29.2
74	0.0223	53.2
75	0.0067	57.3
76	0.0153	39.3
77	0.0898	81.6
78	0.1256	70.1
79	0.0743	76.4
80	0.0746	83.0
81	0.0803	74.4
82	0.0826	71.4
83	0.2301	74.2
84	0.2404	74.4
85	0.0890	78.8
86	0.1180	76.8

TABLE 9 - PARTITION CURVE DATA, POCAHONTAS #3 COAL

TEST NUMBER	COAL NUMBER	SPECIFIC GRAVITY UNDER INVESTIGATION	MESH	BODY LENGTH IN INCHES
87	1	1.2 - 1.3	65 x 100	3.0
88	1	1.3 - 1.4	65 x 100	3.0
89	1	1.4 - 1.5	65 x 100	3.0
90	1	1.5 - 1.6	65 x 100	3.0
91	1	1.6 - 1.7	65 x 100	3.0
92	1	1.7 - 1.8	65 x 100	3.0
93	1	1.2 - 1.3	100 x 150	3.0
94	1	1.3 - 1.4	100 x 150	3.0
95	1	1.4 - 1.5	100 x 150	3.0
96	1	1.5 - 1.6	100 x 150	3.0
97	1	1.6 - 1.7	100 x 150	3.0
98	1	1.7 - 1.8	100 x 150	3.0
99	1	1.2 - 1.3	150 x 200	3.0
100	1	1.3 - 1.4	150 x 200	3.0
101	1	1.4 - 1.5	150 x 200	3.0
102	1	1.5 - 1.6	150 x 200	3.0
103	1	1.6 - 1.7	150 x 200	3.0
104	1	1.7 - 1.8	150 x 200	3.0
105	1	1.2 - 1.3	minus 200	3.0
106	1	1.3 - 1.4	minus 200	3.0
107	1	1.4 - 1.5	minus 200	3.0
108	1	1.5 - 1.6	minus 200	3.0
109	1	1.6 - 1.7	minus 200	3.0
110	1	1.7 - 1.8	minus 200	3.0

TABLE 9 CONTINUED

TEST NUMBER	INLET DIAMETER IN INCHES	CONE ANGLE IN DEGREES	VORTEX FINDER IN INCHES	APEX DISCHARGE IN INCHES
87	0.375	40	0.250	0.145
88	0.375	40	0.250	0.145
89	0.375	40	0.250	0.145
90	0.375	40	0.250	0.145
91	0.375	40	0.250	0.145
92	0.375	40	0.250	0.145
93	0.375	40	0.250	0.145
94	0.375	40	0.250	0.145
95	0.375	40	0.250	0.145
96	0.375	40	0.250	0.145
97	0.375	40	0.250	0.145
98	0.375	40	0.250	0.145
99	0.375	40	0.250	0.145
100	0.375	40	0.250	0.145
101	0.375	40	0.250	0.145
102	0.375	40	0.250	0.145
103	0.375	40	0.250	0.145
104	0.375	40	0.250	0.145
105	0.375	40	0.250	0.145
106	0.375	40	0.250	0.145
107	0.375	40	0.250	0.145
108	0.375	40	0.250	0.145
109	0.375	40	0.250	0.145
110	0.375	40	0.250	0.145

TABLE 9 CONTINUED

TEST NUMBER	PRESSURE IN PSIG	SLURRY DENSITY IN PER CENT	VORTEX FINDER POSITION IN INCHES	OVERFLOW VOLUME IN MILLILITERS
87	6	10	2.770	740
88	6	10	2.770	600
89	6	10	2.770	870
90	6	10	2.770	700
91	6	10	2.770	800
92	6	10	2.770	580
93	6	10	2.770	650
94	6	10	2.770	600
95	6	10	2.770	675
96	6	10	2.770	650
97	6	10	2.770	770
98	6	10	2.770	660
99	6	10	2.770	630
100	6	10	2.770	680
101	6	10	2.770	650
102	6	10	2.770	720
103	6	10	2.770	800
104	6	10	2.770	650
105	6	10	2.770	640
106	6	10	2.770	700
107	6	10	2.770	660
108	6	10	2.770	740
109	6	10	2.770	825
110	6	10	2.770	750

TABLE 9 CONTINUED

TEST NUMBER	UNDERFLOW VOLUME IN MILLILITERS	TIME IN SECONDS	WEIGHT TAGGED MATERIAL OVERFLOW IN GRAMS
87	150	5.4	0.0779
88	100	4.2	0.0708
89	200	6.8	0.0522
90	110	6.0	0.0182
91	160	6.0	0.0087
92	140	5.4	0.0308
93	110	5.4	0.0554
94	130	5.2	0.0805
95	120	5.1	0.0359
96	110	4.9	0.0203
97	150	5.9	0.0238
98	120	5.8	0.0360
99	190	5.4	0.0868
100	120	5.4	0.0362
101	140	5.0	0.0628
102	130	5.4	0.0362
103	160	6.6	0.0368
104	150	6.0	0.0268
105	150	5.4	0.0084
106	100	5.7	0.0449
107	135	4.9	0.0409
108	150	5.7	0.0438
109	180	6.3	0.0489
110	130	5.9	0.0351

TABLE 9 CONTINUED

TEST NUMBER	WEIGHT TAGGED MATERIAL IN UNDERFLOW IN GRAMS	PARTITION COEFFICIENT PER CENT
87	0.0450	63.3
88	0.0700	50.3
89	0.0574	47.6
90	0.0339	34.9
91	0.0193	31.1
92	0.1106	21.8
93	0.0296	65.0
94	0.0494	61.8
95	0.0301	54.3
96	0.0228	47.7
97	0.0387	38.2
98	0.0791	31.3
99	0.0290	74.9
100	0.0144	71.5
101	0.0427	59.2
102	0.0361	50.0
103	0.0409	47.4
104	0.0220	45.1
105	0.0014	85.7
106	0.0134	77.0
107	0.0167	75.0
108	0.0188	69.9
109	0.0280	62.2
110	0.0265	56.9

TABLE 10 - POCAHONTAS #3 COAL WITH 30 PER CENT LIMESTONE DATA

TEST NUMBER	COAL NUMBER	SPECIFIC GRAVITY UNDER INVESTIGATION	MESH	BODY LENGTH IN INCHES
111	1	1.2 - 1.3	65 x 100	3.0
112	1	1.3 - 1.4	65 x 100	3.0
113	1	1.7 - 1.8	65 x 100	3.0
114	1	1.2 - 1.3	100 x 150	3.0
115	1	1.3 - 1.4	100 x 150	3.0
116	1	1.7 - 1.8	100 x 150	3.0
117	1	1.2 - 1.3	150 x 200	3.0
118	1	1.3 - 1.4	150 x 200	3.0
119	1	1.7 - 1.8	150 x 200	3.0
120	1	1.2 - 1.3	minus 200	3.0
121	1	1.3 - 1.4	minus 200	3.0
122	1	1.7 - 1.8	minus 200	3.0

TABLE 10 CONTINUED

TEST NUMBER	INLET DIAMETER IN INCHES	CONE ANGLE IN DEGREES	VORTEX FINDER IN INCHES	APEX DISCHARGE IN INCHES
111	0.375	40	0.250	0.145
112	0.375	40	0.250	0.145
113	0.375	40	0.250	0.145
114	0.375	40	0.250	0.145
115	0.375	40	0.250	0.145
116	0.375	40	0.250	0.145
117	0.375	40	0.250	0.145
118	0.375	40	0.250	0.145
119	0.375	40	0.250	0.145
120	0.375	40	0.250	0.145
121	0.375	40	0.250	0.145
122	0.375	40	0.250	0.145

TABLE 10 CONTINUED

TEST NUMBER	PRESSURE IN PSIG	SLURRY DENSITY IN PER CENT	VORTEX FINDER POSITION IN INCHES	OVERFLOW VOLUME IN MILLILITERS
111	6	10	2.770	680
112	6	10	2.770	720
113	6	10	2.770	750
114	6	10	2.770	750
115	6	10	2.770	780
116	6	10	2.770	790
117	6	10	2.770	730
118	6	10	2.770	790
119	6	10	2.770	690
120	6	10	2.770	780
121	6	10	2.770	740
122	6	10	2.770	750

TABLE 10 CONTINUED

TEST NUMBER	UNDERFLOW VOLUME IN MILLILITERS	TIME IN SECONDS	WEIGHT TAGGED MATERIAL OVERFLOW IN GRAMS
111	160	6.0	0.0412
112	110	5.7	0.0740
113	180	6.1	0.0656
114	170	6.3	0.0511
115	160	6.0	0.0706
116	175	6.1	0.0294
117	120	5.2	0.0745
118	160	6.1	0.1070
119	160	6.0	0.0857
120	100	5.7	0.9070
121	150	6.0	0.0920
122	130	5.9	0.1145

TABLE 10 CONTINUED

TEST NUMBER	WEIGHT TAGGED MATERIAL IN UNDERFLOW IN GRAMS	PARTITION COEFFICIENT PER CENT
111	0.0220	65.2
112	0.0520	59.6
113	0.1184	35.1
114	0.0136	78.9
115	0.0396	72.1
116	0.0340	46.3
117	0.0202	78.6
118	0.0358	74.8
119	0.0530	61.7
120	0.0076	92.3
121	0.0127	87.8
122	0.0404	74.5

TABLE 11 - OPTIMIZATION DATA FOR HAGY COAL

TEST NUMBER	COAL NUMBER	SPECIFIC GRAVITY UNDER INVESTIGATION	MESH	BODY LENGTH IN INCHES
123	2	4.8 - 5.1	65 x 100	3.0
124	2	4.8 - 5.1	65 x 100	3.0
125	2	4.8 - 5.1	65 x 100	3.0
126	2	4.8 - 5.1	65 x 100	3.0
127	2	4.8 - 5.1	65 x 100	3.0
128	2	4.8 - 5.1	65 x 100	3.0
129	2	4.8 - 5.1	65 x 100	3.0
130	2	4.8 - 5.1	65 x 100	3.0
131	2	4.8 - 5.1	65 x 100	3.0
132	2	4.8 - 5.1	65 x 100	3.0
133	2	4.8 - 5.1	100 x 150	3.0
134	2	4.8 - 5.1	100 x 150	3.0
135	2	4.8 - 5.1	100 x 150	3.0
136	2	4.8 - 5.1	100 x 150	3.0
137	2	4.8 - 5.1	100 x 150	3.0
138	2	4.8 - 5.1	100 x 150	3.0
139	2	4.8 - 5.1	100 x 150	3.0
140	2	4.8 - 5.1	100 x 150	3.0
141	2	4.8 - 5.1	100 x 150	3.0
142	2	4.8 - 5.1	100 x 150	3.0
143	2	4.8 - 5.1	150 x 200	3.0
144	2	4.8 - 5.1	150 x 200	3.0
145	2	4.8 - 5.1	150 x 200	3.0
146	2	4.8 - 5.1	150 x 200	3.0
147	2	4.8 - 5.1	150 x 200	3.0
148	2	4.8 - 5.1	150 x 200	3.0
149	2	4.8 - 5.1	150 x 200	3.0
150	2	4.8 - 5.1	150 x 200	3.0
151	2	4.8 - 5.1	150 x 200	3.0
152	2	4.8 - 5.1	150 x 200	3.0
153	2	4.8 - 5.1	minus 200	3.0
154	2	4.8 - 5.1	minus 200	3.0
155	2	4.8 - 5.1	minus 200	3.0
156	2	4.8 - 5.1	minus 200	3.0
157	2	4.8 - 5.1	minus 200	3.0
158	2	4.8 - 5.1	minus 200	3.0
159	2	4.8 - 5.1	minus 200	3.0
160	2	4.8 - 5.1	minus 200	3.0
161	2	4.8 - 5.1	minus 200	3.0
162	2	4.8 - 5.1	minus 200	3.0

TABLE 11 CONTINUED

TEST NUMBER	INLET DIAMETER IN INCHES	CONE ANGLE IN DEGREES	VORTEX FINDER IN INCHES	APEX DISCHARGE IN INCHES
123	0.375	40	0.250	0.145
124	0.375	40	0.250	0.145
125	0.375	40	0.250	0.145
126	0.375	40	0.250	0.145
127	0.125	40	0.250	0.145
128	0.250	40	0.250	0.145
129	0.375	40	0.250	0.145
130	0.375	40	0.250	0.145
131	0.375	20	0.250	0.145
132	0.375	60	0.250	0.145
133	0.375	40	0.250	0.145
134	0.375	40	0.250	0.145
135	0.375	40	0.250	0.145
136	0.375	40	0.250	0.145
137	0.125	40	0.250	0.145
138	0.250	40	0.250	0.145
139	0.375	40	0.250	0.145
140	0.375	40	0.250	0.145
141	0.375	20	0.250	0.145
142	0.375	60	0.250	0.145
143	0.375	40	0.250	0.145
144	0.375	40	0.250	0.145
145	0.375	40	0.250	0.145
146	0.375	40	0.250	0.145
147	0.125	40	0.250	0.145
148	0.250	40	0.250	0.145
149	0.375	40	0.250	0.145
150	0.375	40	0.250	0.145
151	0.375	20	0.250	0.145
152	0.375	60	0.250	0.145
153	0.375	40	0.250	0.145
154	0.375	40	0.250	0.145
155	0.375	40	0.250	0.145
156	0.375	40	0.250	0.145
157	0.125	40	0.250	0.145
158	0.250	40	0.250	0.145
159	0.375	40	0.250	0.145
160	0.375	40	0.250	0.145
161	0.375	20	0.250	0.145
162	0.375	60	0.250	0.145

TABLE 11 CONTINUED

TEST NUMBER	PRESSURE IN PSIG	SLURRY DENSITY IN PER CENT	VORTEX FINDER POSITION IN INCHES	OVERFLOW VOLUME IN MILLILITERS
123	6	10	2.470	580
124	6	10	2.770	730
125	6	10	3.070	710
126	6	10	3.370	700
127	10	10	3.370	420
128	10	10	3.370	725
129	10	10	3.370	640
130	15	10	3.370	930
131	10	10	3.370	650
132	10	10	3.370	830
133	6	10	2.470	630
134	6	10	2.770	720
135	6	10	3.070	725
136	6	10	3.370	660
137	10	10	3.370	570
138	10	10	3.370	570
139	10	10	3.370	830
140	15	10	3.370	1030
141	10	10	3.370	750
142	10	10	3.370	660
143	6	10	2.470	780
144	6	10	2.770	460
145	6	10	3.070	650
146	6	10	3.370	740
147	10	10	3.370	390
148	10	10	3.370	750
149	10	10	3.370	760
150	15	10	3.370	730
151	10	10	3.370	700
152	10	10	3.370	670
153	6	10	2.470	710
154	6	10	2.770	700
155	6	10	3.070	700
156	6	10	3.370	720
157	10	10	3.370	400
158	10	10	3.370	570
159	10	10	3.370	600
160	15	10	3.370	710
161	10	10	3.370	650
162	10	10	3.370	760

TABLE 11 CONTINUED

TEST NUMBER	UNDERFLOW VOLUME IN MILLILITERS	TIME IN SECONDS	WEIGHT TAGGED MATERIAL OVERFLOW IN GRAMS
123	105	4.8	0.0067
124	150	6.0	0.0017
125	140	5.9	0.0058
126	140	7.5	0.0132
127	50	11.0	0.0039
128	100	6.8	0.0022
129	130	6.2	0.0028
130	150	6.0	0.0184
131	135	4.1	0.0050
132	190	5.3	0.0044
133	120	6.0	0.0050
134	150	5.9	0.0083
135	110	6.6	0.0040
136	110	6.0	0.0084
137	50	12.0	0.0021
138	70	6.0	0.0029
139	160	6.0	0.0056
140	240	6.0	0.0057
141	135	5.7	0.0047
142	100	5.1	0.0049
143	170	6.6	0.0082
144	140	4.7	0.0014
145	140	5.3	0.0016
146	150	6.9	0.0026
147	50	8.4	0.0008
148	100	6.1	0.0032
149	150	5.1	0.0042
150	140	4.2	0.0023
151	150	5.0	0.0126
152	140	4.2	0.0055
153	150	6.0	0.0186
154	130	6.0	0.0035
155	140	5.4	0.0033
156	130	5.1	0.0111
157	50	12.0	0.0068
158	80	6.0	0.0125
159	130	4.5	0.0120
160	130	3.9	0.0199
161	130	4.2	0.0056
162	150	4.8	0.0110

TABLE 11 CONTINUED

TEST NUMBER	WEIGHT TAGGED MATERIAL IN UNDERFLOW IN GRAMS	PARTITION COEFFICIENT PER CENT
123	0.0456	12.8
124	0.0406	4.0
125	0.0407	12.4
126	0.0500	20.9
127	0.0194	16.8
128	0.0309	6.6
129	0.0659	4.1
130	0.0861	17.6
131	0.0430	10.4
132	0.0430	9.2
133	0.0103	32.7
134	0.0345	19.4
135	0.0144	21.7
136	0.0282	22.9
137	0.0082	20.4
138	0.0221	6.4
139	0.0349	13.8
140	0.0344	14.2
141	0.0134	25.9
142	0.0244	16.7
143	0.0113	42.0
144	0.0042	25.0
145	0.0024	40.0
146	0.0020	56.6
147	0.0029	21.6
148	0.0134	19.3
149	0.0137	24.3
150	0.0049	31.9
151	0.0164	43.3
152	0.0141	28.0
153	0.0212	46.8
154	0.0050	42.4
155	0.0035	48.5
156	0.0096	53.6
157	0.0073	48.2
158	0.0353	26.2
159	0.0249	32.5
160	0.0129	60.7
161	0.0057	49.5
162	0.0149	42.5

TABLE 12 - PARTITION CURVE DATA, HAGY COAL

TEST NUMBER	COAL NUMBER	SPECIFIC GRAVITY UNDER INVESTIGATION	MESH	BODY LENGTH IN INCHES
163	2	1.3 - 1.4	65 x 100	3.0
164	2	1.4 - 1.5	65 x 100	3.0
165	2	1.5 - 1.6	65 x 100	3.0
166	2	1.6 - 1.7	65 x 100	3.0
167	2	1.7 - 1.8	65 x 100	3.0
168	2	1.3 - 1.4	100 x 150	3.0
169	2	1.4 - 1.5	100 x 150	3.0
170	2	1.5 - 1.6	100 x 150	3.0
171	2	1.6 - 1.7	100 x 150	3.0
172	2	1.7 - 1.8	100 x 150	3.0
173	2	1.3 - 1.4	150 x 200	3.0
174	2	1.4 - 1.5	150 x 200	3.0
175	2	1.5 - 1.6	150 x 200	3.0
176	2	1.6 - 1.7	150 x 200	3.0
177	2	1.7 - 1.8	150 x 200	3.0
178	2	1.3 - 1.4	minus 200	3.0
179	2	1.4 - 1.5	minus 200	3.0
180	2	1.5 - 1.6	minus 200	3.0
181	2	1.6 - 1.7	minus 200	3.0
182	2	1.7 - 1.8	minus 200	3.0

TABLE 12 CONTINUED

TEST NUMBER	INLET DIAMETER IN INCHES	CONE ANGLE IN DEGREES	VORTEX FINDER IN INCHES	APEX DISCHARGE IN INCHES
163	0.375	40	0.250	0.145
164	0.375	40	0.250	0.145
165	0.375	40	0.250	0.145
166	0.375	40	0.250	0.145
167	0.375	40	0.250	0.145
168	0.375	40	0.250	0.145
169	0.375	40	0.250	0.145
170	0.375	40	0.250	0.145
171	0.375	40	0.250	0.145
172	0.375	40	0.250	0.145
173	0.375	40	0.250	0.145
174	0.375	40	0.250	0.145
175	0.375	40	0.250	0.145
176	0.375	40	0.250	0.145
177	0.375	40	0.250	0.145
178	0.375	40	0.250	0.145
179	0.375	40	0.250	0.145
180	0.375	40	0.250	0.145
181	0.375	40	0.250	0.145
182	0.375	40	0.250	0.145

TABLE 12 CONTINUED

TEST NUMBER	PRESSURE IN PSIG	SLURRY DENSITY IN PER CENT	VORTEX FINDER POSITION IN INCHES	OVERFLOW VOLUME IN MILLILITERS
163	6	10	2.770	700
164	6	10	2.770	810
165	6	10	2.770	760
166	6	10	2.770	730
167	6	10	2.770	710
168	6	10	2.770	770
169	6	10	2.770	810
170	6	10	2.770	740
171	6	10	2.770	740
172	6	10	2.770	800
173	6	10	2.770	700
174	6	10	2.770	675
175	6	10	2.770	750
176	6	10	2.770	700
177	6	10	2.770	720
178	6	10	2.770	690
179	6	10	2.770	680
180	6	10	2.770	760
181	6	10	2.770	750
182	6	10	2.770	750

TABLE 12 CONTINUED

TEST NUMBER	UNDERFLOW VOLUME IN MILLILITERS	TIME IN SECONDS	WEIGHT TAGGED MATERIAL OVERFLOW IN GRAMS
163	100	5.7	0.0542
164	115	6.5	0.0619
165	110	6.0	0.0679
166	100	5.9	0.0391
167	90	6.0	0.0215
168	90	8.2	0.0913
169	105	6.0	0.0618
170	110	6.0	0.0459
171	140	5.9	0.0332
172	110	6.8	0.0436
173	100	6.0	0.0478
174	80	6.0	0.0930
175	110	6.6	0.0502
176	90	5.9	0.0135
177	75	5.8	0.0540
178	95	5.7	0.0271
179	100	6.0	0.0506
180	105	5.8	0.0473
181	100	6.5	0.0554
182	130	6.7	0.0665

TABLE 12 CONTINUED

TEST NUMBER	WEIGHT TAGGED MATERIAL IN UNDERFLOW IN GRAMS	PARTITION COEFFICIENT PER CENT
163	0.0427	55.9
164	0.0793	43.7
165	0.1058	39.1
166	0.0871	30.9
167	0.0638	25.2
168	0.0494	64.8
169	0.0487	55.9
170	0.0490	48.3
171	0.0490	40.3
172	0.0853	33.8
173	0.0248	65.8
174	0.0542	63.1
175	0.0334	60.1
176	0.0135	59.5
177	0.0515	51.2
178	0.0135	66.7
179	0.0258	66.2
180	0.0223	67.9
181	0.0288	65.7
182	0.0505	56.8

TABLE 13 - OPTIMIZATION DATA FOR NEW MEXICO COAL

TEST NUMBER	COAL NUMBER	SPECIFIC GRAVITY UNDER INVESTIGATION	MESH	BODY LENGTH IN INCHES
183	3	4.8 - 5.1	65 x 100	3.0
184	3	4.8 - 5.1	65 x 100	3.0
185	3	4.8 - 5.1	65 x 100	3.0
186	3	4.8 - 5.1	65 x 100	3.0
187	3	4.8 - 5.1	65 x 100	3.0
188	3	4.8 - 5.1	65 x 100	3.0
189	3	4.8 - 5.1	65 x 100	3.0
190	3	4.8 - 5.1	65 x 100	3.0
191	3	4.8 - 5.1	65 x 100	3.0
192	3	4.8 - 5.1	65 x 100	3.0
193	3	4.8 - 5.1	100 x 150	3.0
194	3	4.8 - 5.1	100 x 150	3.0
195	3	4.8 - 5.1	100 x 150	3.0
196	3	4.8 - 5.1	100 x 150	3.0
197	3	4.8 - 5.1	100 x 150	3.0
198	3	4.8 - 5.1	100 x 150	3.0
199	3	4.8 - 5.1	100 x 150	3.0
200	3	4.8 - 5.1	100 x 150	3.0
201	3	4.8 - 5.1	100 x 150	3.0
202	3	4.8 - 5.1	100 x 150	3.0
203	3	4.8 - 5.1	150 x 200	3.0
204	3	4.8 - 5.1	150 x 200	3.0
205	3	4.8 - 5.1	150 x 200	3.0
206	3	4.8 - 5.1	150 x 200	3.0
207	3	4.8 - 5.1	150 x 200	3.0
208	3	4.8 - 5.1	150 x 200	3.0
209	3	4.8 - 5.1	150 x 200	3.0
210	3	4.8 - 5.1	150 x 200	3.0
211	3	4.8 - 5.1	150 x 200	3.0
212	3	4.8 - 5.1	150 x 200	3.0
213	3	4.8 - 5.1	minus 200	3.0
214	3	4.8 - 5.1	minus 200	3.0
215	3	4.8 - 5.1	minus 200	3.0
216	3	4.8 - 5.1	minus 200	3.0
217	3	4.8 - 5.1	minus 200	3.0
218	3	4.8 - 5.1	minus 200	3.0
219	3	4.8 - 5.1	minus 200	3.0
220	3	4.8 - 5.1	minus 200	3.0
221	3	4.8 - 5.1	minus 200	3.0
222	3	4.8 - 5.1	minus 200	3.0

TABLE 13 CONTINUED

TEST NUMBER	INLET DIAMETER IN INCHES	CONE ANGLE IN DEGREES	VORTEX FINDER IN INCHES	APEX DISCHARGE IN INCHES
183	0.375	40	0.250	0.145
184	0.375	40	0.250	0.145
185	0.375	40	0.250	0.145
186	0.375	40	0.250	0.145
187	0.125	40	0.250	0.145
188	0.250	40	0.250	0.145
189	0.375	40	0.250	0.145
190	0.375	40	0.250	0.145
191	0.375	20	0.250	0.145
192	0.375	60	0.250	0.145
193	0.375	40	0.250	0.145
194	0.375	40	0.250	0.145
195	0.375	40	0.250	0.145
196	0.375	40	0.250	0.145
197	0.125	40	0.250	0.145
198	0.250	40	0.250	0.145
199	0.375	40	0.250	0.145
200	0.375	40	0.250	0.145
201	0.375	20	0.250	0.145
202	0.375	60	0.250	0.145
203	0.375	40	0.250	0.145
204	0.375	40	0.250	0.145
205	0.375	40	0.250	0.145
206	0.375	40	0.250	0.145
207	0.125	40	0.250	0.145
208	0.250	40	0.250	0.145
209	0.375	40	0.250	0.145
210	0.375	40	0.250	0.145
211	0.375	20	0.250	0.145
212	0.375	60	0.250	0.145
213	0.375	40	0.250	0.145
214	0.375	40	0.250	0.145
215	0.375	40	0.250	0.145
216	0.375	40	0.250	0.145
217	0.125	40	0.250	0.145
218	0.250	40	0.250	0.145
219	0.375	40	0.250	0.145
220	0.375	40	0.250	0.145
221	0.375	20	0.250	0.145
222	0.375	60	0.250	0.145

TABLE 13 CONTINUED

TEST NUMBER	PRESSURE IN PSIG	SLURRY DENSITY IN PER CENT	VORTEX FINDER POSITION IN INCHES	OVERFLOW VOLUME IN MILLILITERS
183	6	10	2.470	720
184	6	10	2.770	720
185	6	10	3.070	710
186	6	10	3.370	650
187	10	10	3.370	770
188	10	10	3.370	710
189	10	10	3.370	630
190	15	10	3.370	750
191	10	10	3.370	750
192	10	10	3.370	730
193	6	10	2.470	750
194	6	10	2.770	760
195	6	10	3.070	730
196	6	10	3.370	810
197	10	10	3.370	900
198	10	10	3.370	690
199	10	10	3.370	710
200	15	10	3.370	750
201	10	10	3.370	740
202	10	10	3.370	860
203	6	10	2.470	770
204	6	10	2.770	770
205	6	10	3.070	820
206	6	10	3.370	760
207	10	10	3.370	870
208	10	10	3.370	670
209	10	10	3.370	870
210	15	10	3.370	970
211	10	10	3.370	830
212	10	10	3.370	800
213	6	10	2.470	650
214	6	10	2.770	750
215	6	10	3.070	670
216	6	10	3.370	800
217	10	10	3.370	770
218	10	10	3.370	650
219	10	10	3.370	650
220	15	10	3.370	720
221	10	10	3.370	750
222	10	10	3.370	770

TABLE 13 CONTINUED

TEST NUMBER	UNDERFLOW VOLUME IN MILLILITERS	TIME IN SECONDS	WEIGHT TAGGED MATERIAL OVERFLOW IN GRAMS
183	145	5.4	0.0720
184	150	6.0	0.0604
185	150	5.7	0.0237
186	150	6.0	0.1482
187	100	15.6	0.0022
188	120	6.6	0.0439
189	145	4.8	0.1222
190	160	4.2	0.0859
191	150	4.5	0.0810
192	100	4.8	0.0832
193	190	6.0	0.0913
194	180	6.0	0.1064
195	160	5.9	0.0920
196	190	6.7	0.1672
197	160	18.0	0.0561
198	140	6.7	0.0533
199	135	6.3	0.1003
200	140	4.2	0.0697
201	130	5.6	0.2948
202	150	5.7	0.1943
203	180	6.6	0.0904
204	200	7.4	0.1948
205	200	6.6	0.2135
206	175	6.5	0.2704
207	125	18.0	0.0453
208	150	6.1	0.0570
209	200	6.0	0.1775
210	100	4.9	0.1705
211	170	6.0	0.3244
212	160	4.6	0.1424
213	150	6.0	0.0432
214	350	6.0	0.1143
215	130	5.0	0.3431
216	200	6.0	0.2180
217	100	14.0	0.5849
218	100	6.0	0.1033
219	650	4.8	0.1139
220	160	4.2	0.7412
221	115	5.0	0.0771
222	120	4.8	0.0851

TABLE 13 CONTINUED

TEST NUMBER	WEIGHT TAGGED MATERIAL IN UNDERFLOW IN GRAMS	PARTITION COEFFICIENT PER CENT
183	0.1916	27.3
184	0.2386	23.8
185	0.1683	12.3
186	0.4368	25.0
187	0.0402	5.2
188	0.3456	11.2
189	0.5584	18.1
190	0.5971	12.5
191	0.1833	30.6
192	0.2763	21.5
193	0.2083	30.5
194	0.4812	16.1
195	0.3163	25.1
196	0.3583	31.6
197	0.7376	7.1
198	0.2984	15.1
199	0.2826	26.1
200	0.2204	24.1
201	0.6144	32.4
202	0.5663	25.6
203	0.1390	39.4
204	0.4703	29.3
205	0.6998	23.4
206	0.3733	42.0
207	0.7642	5.6
208	0.2273	20.5
209	0.2974	37.3
210	0.3828	30.8
211	0.4366	42.6
212	0.2120	40.1
213	0.0595	42.1
214	0.0770	59.7
215	0.3419	50.1
216	0.0875	71.3
217	0.3800	60.6
218	0.0499	67.4
219	0.0522	68.5
220	0.7988	48.1
221	0.0352	75.3
222	0.3407	71.4

TABLE 14 - PARTITION CURVE DATA, NEW MEXICO COAL

TEST NUMBER	COAL NUMBER	SPECIFIC GRAVITY UNDER INVESTIGATION	MESH	BODY LENGTH IN INCHES
223	3	1.2 - 1.3	65 x 100	3.0
224	3	1.3 - 1.4	65 x 100	3.0
225	3	1.4 - 1.5	65 x 100	3.0
226	3	1.5 - 1.6	65 x 100	3.0
227	3	1.6 - 1.7	65 x 100	3.0
228	3	1.7 - 1.8	65 x 100	3.0
229	3	1.2 - 1.3	100 x 150	3.0
230	3	1.3 - 1.4	100 x 150	3.0
231	3	1.4 - 1.5	100 x 150	3.0
232	3	1.5 - 1.6	100 x 150	3.0
233	3	1.6 - 1.7	100 x 150	3.0
234	3	1.7 - 1.8	100 x 150	3.0
235	3	1.2 - 1.3	150 x 200	3.0
236	3	1.3 - 1.4	150 x 200	3.0
237	3	1.4 - 1.5	150 x 200	3.0
238	3	1.5 - 1.6	150 x 200	3.0
239	3	1.6 - 1.7	150 x 200	3.0
240	3	1.7 - 1.8	150 x 200	3.0
241	3	1.2 - 1.3	minus 200	3.0
242	3	1.3 - 1.4	minus 200	3.0
243	3	1.4 - 1.5	minus 200	3.0
244	3	1.5 - 1.6	minus 200	3.0
245	3	1.6 - 1.7	minus 200	3.0
246	3	1.7 - 1.8	minus 200	3.0

TABLE 14 CONTINUED

TEST NUMBER	INLET DIAMETER IN INCHES	CONE ANGLE IN DEGREES	VORTEX FINDER IN INCHES	APEX DISCHARGE IN INCHES
223	0.375	40	0.250	0.145
224	0.375	40	0.250	0.145
225	0.375	40	0.250	0.145
226	0.375	40	0.250	0.145
227	0.375	40	0.250	0.145
228	0.375	40	0.250	0.145
229	0.375	40	0.250	0.145
230	0.375	40	0.250	0.145
231	0.375	40	0.250	0.145
232	0.375	40	0.250	0.145
233	0.375	40	0.250	0.145
234	0.375	40	0.250	0.145
235	0.375	40	0.250	0.145
236	0.375	40	0.250	0.145
237	0.375	40	0.250	0.145
238	0.375	40	0.250	0.145
239	0.375	40	0.250	0.145
240	0.375	40	0.250	0.145
241	0.375	40	0.250	0.145
242	0.375	40	0.250	0.145
243	0.375	40	0.250	0.145
244	0.375	40	0.250	0.145
245	0.375	40	0.250	0.145
246	0.375	40	0.250	0.145

TABLE 14 CONTINUED

TEST NUMBER	PRESSURE IN PSIG	SLURRY DENSITY IN PER CENT	VORTEX FINDER POSITION IN INCHES	OVERFLOW VOLUME IN MILLILITERS
223	6	10	2.770	670
224	6	10	2.770	790
225	6	10	2.770	800
226	6	10	2.770	830
227	6	10	2.770	800
228	6	10	2.770	830
229	6	10	2.770	910
230	6	10	2.770	750
231	6	10	2.770	710
232	6	10	2.770	900
233	6	10	2.770	770
234	6	10	2.770	640
235	6	10	2.770	660
236	6	10	2.770	700
237	6	10	2.770	730
238	6	10	2.770	810
239	6	10	2.770	880
240	6	10	2.770	750
241	6	10	2.770	680
242	6	10	2.770	800
243	6	10	2.770	660
244	6	10	2.770	900
245	6	10	2.770	760
246	6	10	2.770	740

TABLE 14 CONTINUED

TEST NUMBER	UNDERFLOW VOLUME IN MILLILITERS	TIME IN SECONDS	WEIGHT TAGGED MATERIAL OVERFLOW IN GRAMS
223	150	4.8	0.0095
224	170	6.1	0.0872
225	190	5.9	0.0698
226	200	6.0	0.0636
227	200	6.0	0.0487
228	180	5.4	0.0519
229	220	6.6	0.0707
230	160	5.4	0.1142
231	175	5.4	0.0810
232	190	6.0	0.1337
233	175	5.4	0.0747
234	140	4.5	0.0480
235	110	5.4	0.0680
236	180	5.4	0.0805
237	150	5.4	0.0846
238	140	5.7	0.1101
239	150	5.7	0.0943
240	150	5.4	0.0664
241	160	4.8	0.0152
242	140	5.8	0.0660
243	150	4.2	0.0710
244	190	6.0	0.1184
245	130	5.4	0.0950
246	120	4.6	0.0604

TABLE 14 CONTINUED

TEST NUMBER	WEIGHT TAGGED MATERIAL IN UNDERFLOW IN GRAMS	PARTITION COEFFICIENT PER CENT
223	0.0023	80.5
224	0.0580	64.5
225	0.0775	47.3
226	0.0980	40.0
227	0.0845	36.5
228	0.1042	33.2
229	0.0172	80.9
230	0.0340	77.0
231	0.0563	60.5
232	0.1196	52.7
233	0.0853	46.7
234	0.0636	43.0
235	0.0140	83.0
236	0.0200	80.0
237	0.0413	66.7
238	0.0968	53.2
239	0.0868	52.0
240	0.0654	50.5
241	0.0010	93.8
242	0.0155	81.0
243	0.0230	75.5
244	0.0551	68.2
245	0.0638	59.8
246	0.0424	58.7

APPENDIX D

PYRITIC SULFUR REJECTION CALCULATIONS

CALCULATIONS OF PYRITIC SULFUR REJECTION

Hagy coal

	Feed	Overflow	Underflow
Pyritic sulfur in per cent	1.19	0.61	1.62

Pyritic sulfur in product in per cent

$$(0.66 \times 0.61)/1.0 \times 1.19 = 33.8$$

Pyritic sulfur rejection in per cent = 100.00 - 33.8 = 66.2

Pittsburgh coal

	Feed	Overflow	Underflow
Pyritic sulfur in per cent	1.22	0.39	0.72

Pyritic sulfur in product in per cent

$$(0.66 \times 0.39)/1.0 \times 1.22 = 21.1$$

Pyritic sulfur rejection in per cent = 100.00 - 21.1 = 78.9

CALCULATION OF THEORETICAL PYRITIC SULFUR REJECTION

Mesh	Pyritic Sulfur Rejection Coefficient	Weight Pyrite Per Cent	Pyrite in Underflow Per Cent	Pyrite in Overflow Per Cent	Cumulative Pyrite in Underflow Per Cent
65 x 100	95.98	42.3	95.98	4.02	40.6
100 x 150	80.60	17.8	80.60	19.40	55.0
150 x 200	75.00	24.4	75.00	25.00	73.3
minus 200	57.60	15.5	57.60	42.20	82.2

The theoretical pyritic sulfur rejection is 82.2 per cent.

VITA

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Lynn V. Amundson

RESPONSE TO PARAMETER VARIATION OF A ONE-INCH

DIAMETER HYDROCYCLONE FOR PYRITIC SULFUR REMOVAL

by

Lynn Vinzant Amundson

(ABSTRACT)

The use of a 1.0" diameter hydrocyclone was investigated for the removal of pyritic sulfur from coal. The hydrocyclone was constructed to permit rapid changes in cone angle, inlet diameter, vortex finder position, vortex finder diameter, and apex discharge diameter.

Synthetic material which duplicated the actual coal with respect to size, shape, and specific gravity distribution was used during this investigation. The synthetic material provided close control of the specific gravity distribution and eliminated the problem of material degradation during the course of the investigation.

A new technique for obtaining float-sink data was developed. Magnetically tagged material, identical to coal with respect to size, shape, and specific gravity was introduced into the system. The weight of the tagged material in the overflow and underflow samples permitted rapid calculation of the partition coefficient.

Refuse material was introduced into the system. It was found that this increased the yield of clean coal.

An economic analysis of a hydrocyclone system designed to process 100 tons per hour of coal was performed.