FEASIBILITY STUDY OF
THE VERTICAL TRANSPORT OF
COAL BY PIPELINE

by
R.A. Hartman
and
J.R. Reed

An Investigation
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A special thanks is also given to the organizations and private firms listed in Appendix C who so generously provided a wealth of valuable information relative to this research. Without their cooperation this study would not have been possible.
# TABLE OF CONTENTS

<table>
<thead>
<tr>
<th>Section</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>ACKNOWLEDGMENTS</td>
<td>ii</td>
</tr>
<tr>
<td>LIST OF TABLES</td>
<td>v</td>
</tr>
<tr>
<td>LIST OF FIGURES</td>
<td>vi</td>
</tr>
<tr>
<td>NOMENCLATURE</td>
<td>vii</td>
</tr>
<tr>
<td>1. INTRODUCTION</td>
<td>1</td>
</tr>
<tr>
<td>2. VERTICAL HYDRAULIC TRANSPORT OF SOLIDS</td>
<td>3</td>
</tr>
<tr>
<td>2.1 Basic Characteristics of Vertical Flow</td>
<td>3</td>
</tr>
<tr>
<td>2.2 Systems of Transport</td>
<td>9</td>
</tr>
<tr>
<td>2.2.1 Types of Systems</td>
<td>9</td>
</tr>
<tr>
<td>2.2.2 Power Requirements</td>
<td>10</td>
</tr>
<tr>
<td>2.3 Transporting Media</td>
<td>15</td>
</tr>
<tr>
<td>2.4 Characteristics of Solids Being Transported</td>
<td>17</td>
</tr>
<tr>
<td>2.5 Pipe Requirements</td>
<td>22</td>
</tr>
<tr>
<td>3. CONVENTIONAL HOISTING OF COAL</td>
<td>26</td>
</tr>
<tr>
<td>3.1 Types of Mine Entries and Their Relative Use in Pennsylvania</td>
<td>26</td>
</tr>
<tr>
<td>3.2 Conveying Methods and Costs</td>
<td>27</td>
</tr>
<tr>
<td>3.3 Further Technical Considerations in the Application of Hydraulic Hoisting</td>
<td>32</td>
</tr>
<tr>
<td>4. HYDRAULIC HOISTING SYSTEMS</td>
<td>36</td>
</tr>
<tr>
<td>4.1 Successful Installations</td>
<td>36</td>
</tr>
<tr>
<td>4.2 Solids Feeding Systems</td>
<td>43</td>
</tr>
<tr>
<td>4.3 General Opinions from Literature and Industry</td>
<td>49</td>
</tr>
<tr>
<td>5. TECHNICAL FEASIBILITY AND DESIGN OPTIONS</td>
<td>52</td>
</tr>
<tr>
<td>5.1 Basis for Justification</td>
<td>52</td>
</tr>
<tr>
<td>5.2 System Alternatives</td>
<td>53</td>
</tr>
<tr>
<td>6. A CONCEPTUAL HYDRAULIC HOISTING SYSTEM</td>
<td>56</td>
</tr>
<tr>
<td>6.1 Basic Design</td>
<td>56</td>
</tr>
<tr>
<td>6.2 Economic Comparison with Conventional Hoisting</td>
<td>62</td>
</tr>
<tr>
<td>7. SUMMARY</td>
<td>68</td>
</tr>
<tr>
<td>7.1 Discussion of Results</td>
<td>68</td>
</tr>
<tr>
<td>7.2 Conclusion</td>
<td>73</td>
</tr>
<tr>
<td>7.3 Recommendations</td>
<td>75</td>
</tr>
<tr>
<td>BIBLIOGRAPHY</td>
<td>77</td>
</tr>
<tr>
<td>APPENDIX A:</td>
<td>HYDRAULIC SYSTEM COST DATA COMMON TO ALL OF THE PARTICULAR MINES STUDIED</td>
</tr>
<tr>
<td>------------</td>
<td>-------------------------------------------------------------------------</td>
</tr>
<tr>
<td>APPENDIX B:</td>
<td>ECONOMIC ANALYSES OF PARTICULAR MINES</td>
</tr>
<tr>
<td>APPENDIX C:</td>
<td>ORGANIZATIONS AND PRIVATE FIRMS WHO CONTRIBUTED INFORMATION TO THIS STUDY</td>
</tr>
</tbody>
</table>
## LIST OF TABLES

<table>
<thead>
<tr>
<th>Table</th>
<th>Description</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Conventional Mining Data</td>
<td>31</td>
</tr>
<tr>
<td>2</td>
<td>Comparison of Capital Costs of Conventional and Hydraulic Hoisting</td>
<td>63</td>
</tr>
</tbody>
</table>
# LIST OF FIGURES

<table>
<thead>
<tr>
<th>Figure</th>
<th>Description</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Power Requirements of Open and Circulating Hydraulic Hoisting Systems</td>
<td>14</td>
</tr>
<tr>
<td>2</td>
<td>Terminal Settling Velocities of Coal</td>
<td>19</td>
</tr>
<tr>
<td>3</td>
<td>Depths of Pennsylvania Bituminous Coal Mines with Shaft Type Entries</td>
<td>28</td>
</tr>
<tr>
<td>4</td>
<td>Lock-Hopper Coal Feeder</td>
<td>48</td>
</tr>
<tr>
<td>5</td>
<td>Underground Portion of a Hydraulic Hoisting System</td>
<td>57</td>
</tr>
<tr>
<td>6</td>
<td>Cost/Depth versus Output/Depth for Conventional and Hydraulic Hoisting</td>
<td>71</td>
</tr>
</tbody>
</table>
## NOMENCLATURE

<table>
<thead>
<tr>
<th>Symbol</th>
<th>Description</th>
<th>F-L-T Units</th>
<th>English Units</th>
</tr>
</thead>
<tbody>
<tr>
<td>(i_w)</td>
<td>pressure gradient of clear water</td>
<td>L/L</td>
<td>ft/ft</td>
</tr>
<tr>
<td>(i)</td>
<td>pressure gradient of coal-water mixture</td>
<td>L/L</td>
<td>ft/ft</td>
</tr>
<tr>
<td>(V)</td>
<td>water velocity</td>
<td>L/T</td>
<td>ft/sec</td>
</tr>
<tr>
<td>(V_w)</td>
<td>mixture velocity</td>
<td>L/T</td>
<td>ft/sec</td>
</tr>
<tr>
<td>(V_s)</td>
<td>settling velocity</td>
<td>L/T</td>
<td>ft/sec</td>
</tr>
<tr>
<td>(g)</td>
<td>acceleration of gravity</td>
<td>L/T²</td>
<td>ft/sec²</td>
</tr>
<tr>
<td>(L)</td>
<td>length of pipe</td>
<td>L</td>
<td>ft</td>
</tr>
<tr>
<td>(D)</td>
<td>diameter of pipe</td>
<td>L</td>
<td>ft</td>
</tr>
<tr>
<td>(d)</td>
<td>particle diameter</td>
<td>L</td>
<td>ft</td>
</tr>
<tr>
<td>(H)</td>
<td>static head</td>
<td>L</td>
<td>ft</td>
</tr>
<tr>
<td>(f)</td>
<td>friction factor</td>
<td>Dimensionless</td>
<td></td>
</tr>
<tr>
<td>(s_w)</td>
<td>specific gravity of mixture</td>
<td>Dimensionless</td>
<td></td>
</tr>
<tr>
<td>(s)</td>
<td>specific gravity of solids</td>
<td>Dimensionless</td>
<td></td>
</tr>
<tr>
<td>(s_o)</td>
<td>specific gravity of liquid</td>
<td>Dimensionless</td>
<td></td>
</tr>
<tr>
<td>(T)</td>
<td>coefficient of resistance of a free falling particle in water</td>
<td>Dimensionless</td>
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<tr>
<td>(P_s)</td>
<td>density of solids</td>
<td>FT²/L⁴</td>
<td>lb-sec²/ft⁴</td>
</tr>
<tr>
<td>(P_w)</td>
<td>density of liquid</td>
<td>FT²/L⁴</td>
<td>lb-sec²/ft⁴</td>
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<tr>
<td>(C_v)</td>
<td>volumetric transport concentration</td>
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<td></td>
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<tr>
<td>(C_{v'})</td>
<td>spatial concentration</td>
<td>Dimensionless</td>
<td></td>
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<tr>
<td>(Q)</td>
<td>flowrate of mixture</td>
<td>L³/T</td>
<td>ft³/sec</td>
</tr>
<tr>
<td>(Q_s)</td>
<td>flowrate of solids</td>
<td>L³/T</td>
<td>ft³/sec</td>
</tr>
<tr>
<td>(P_1)</td>
<td>power required to overcome friction loss in downflow pipe</td>
<td>FL/T</td>
<td>ft-lb/sec</td>
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</tbody>
</table>
### NOMENCLATURE CONTINUED

<table>
<thead>
<tr>
<th>Symbol</th>
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<th>English Units</th>
</tr>
</thead>
<tbody>
<tr>
<td>$P_2$</td>
<td>power required to introduce solids into pipe</td>
<td>FL/T</td>
<td>ft-lb/sec</td>
</tr>
<tr>
<td>$P_3$</td>
<td>power required to overcome friction loss in upflow pipe</td>
<td>FL/T</td>
<td>ft-lb/sec</td>
</tr>
<tr>
<td>$P_4$</td>
<td>power required to overcome pressure difference between bottom of downflow</td>
<td>FL/T</td>
<td>ft-lb/sec</td>
</tr>
<tr>
<td></td>
<td>pipe and bottom of upflow pipe</td>
<td></td>
<td></td>
</tr>
<tr>
<td>$P_t$</td>
<td>total power required by circulating system</td>
<td>FL/T</td>
<td>ft-lb/sec</td>
</tr>
<tr>
<td>$P_3'$</td>
<td>power required to pump coal-water mixture to surface in open system</td>
<td>FL/T</td>
<td>ft-lb/sec</td>
</tr>
<tr>
<td>$P_t'$</td>
<td>total power required by open system</td>
<td>FL/T</td>
<td>ft-lb/sec</td>
</tr>
<tr>
<td>$C_D$</td>
<td>coefficient of drag</td>
<td>Dimensionless</td>
<td></td>
</tr>
<tr>
<td>$R_E$</td>
<td>Reynolds number</td>
<td>Dimensionless</td>
<td></td>
</tr>
<tr>
<td>$K$</td>
<td>settling velocity correction coefficient</td>
<td>Dimensionless</td>
<td></td>
</tr>
<tr>
<td>$m$</td>
<td>coefficient dependent on $R_E$ and particle shape used in the determination</td>
<td>Dimensionless</td>
<td></td>
</tr>
<tr>
<td></td>
<td>of $K$</td>
<td></td>
<td></td>
</tr>
<tr>
<td>TDH</td>
<td>total dynamic head</td>
<td>L</td>
<td>ft</td>
</tr>
<tr>
<td>BTU</td>
<td>British Thermal Units</td>
<td></td>
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</tr>
</tbody>
</table>
SUMMATION OF RESULTS

The objective of this study is to investigate the technical and economic feasibility of the vertical transport (hydraulic hoisting) of coal by pipeline from underground mines. The scope of the study excludes horizontal pipeline transport within the mines, and from the mine surface to preparation plants. These possibilities are left to future consideration.

Initially, an extensive literature search provides a state-of-the-art basis for a discussion of the basic requirements and characteristics of vertical transport, as well as an indication of the success of past and present hydraulic hoisting installations. A brief survey of current manufacturing capabilities serves to substantiate the conclusion that hydraulic hoisting, as viewed herein, is technically feasible.

Because of the enormity involved in a complete economic analysis of hydraulic hoisting, that portion of the study has been limited to the consideration of one basic type of hydraulic system, namely, the transport of a maximum of four inch coal from the mining level to the surface in a transporting medium of water. The assumption is made that the coal would be introduced into the pipeline on the discharge side of the pumps, thus requiring only primary crushing facilities in the mine. The capital cost of this hydraulic system, including 10 percent of costs for contingencies, is then compared to the like capital costs of conventional hoisting for seven currently operating coal mines. The best cost data made available by industrial sources is interpreted as well as possible and used. Hopefully, the proprietary policies that some firms possess does not detract from this study. Only capital costs are
compared because operating cost data are unavailable for inclusion in the analyses. The results indicate hydraulic hoisting to be economically favorable for five of these mines, three out of three for shaft type entries, and two out of four for slope type entries. The total cost appears to be dependent on the output to depth ratio for each mine. The hydraulic system on the average appears to be favorable at ratio values less than two tph/ft, and is competitive at even higher ratios.

Although not included in the economic analysis, possible alternatives in a hydraulic system are discussed. Some of these alternatives are: the pumping of pulverized coal; using hydraulic hoisting only as a supplemental means of transport; the use of a conveying medium other than water; consideration of horizontal pipeline links within the mine and on the surface; and the pumping of coal using mine drainage water. An attempt to study the latter in this research was inconclusive because of insufficient dewatering data for conventional mines.

Given the conceptual hoisting system for this study, the next developmental step would appear to be a laboratory model study of the system with particular emphasis on the solids feeding device. However, it would seem appropriate that the above alternatives be selectively studied earlier because of their effect on the concept and economy of a system.
1. INTRODUCTION

This study is concerned with investigating the technical and economic feasibility of lifting coal by pipeline (hydraulic hoisting) from underground mines to the surface.

Thornton (1971) discussed the growth of interest in the hydraulic transport of solids through the use of a historical graph of publications since 1950 and a similar graph depicting the publications specifically relating to the transport of coal. In the past, interest in the pipelining of coal seems to have been limited to overland transport from mining areas to the consumer, generally a power plant. Research dealing with this form of hydraulic transport has been rather extensive and its success is illustrated by the Consolidation Coal Company pipeline in Ohio and the Black Mesa pipeline in Arizona. By comparison, however, interest in hydraulic hoisting as a possible means of reducing mine-to-surface transport costs has been rather limited in the United States, despite the construction of several pilot plants and operating installations in several foreign countries such as Japan, Poland, France, and Russia. The one pilot plant known to have existed in this country, as described by Gardner (1952), reportedly produced quite favorable results; however, no apparent use by the sponsoring company was made of the installation or its design beyond that of the pilot plant. In general, hydraulic hoisting has been acclaimed by researchers such as Chapus, Condolios, and Couratin (1962) to be an economical and practical alternative to conventional hoisting methods, while others express doubt concerning its applicability.
In examining the question of the feasibility of hydraulic hoisting, this study first discusses the current state of the art developed from an extensive literature search, with major emphasis centered on determining the optimal particle size and the maximum possible rate of transport. The state of the art, in conjunction with supplemental professional information necessary to design a complete hoisting system, then provides the basis for a discussion of the technical feasibility of hydraulic hoisting.

The complete analysis of economic feasibility is limited, due to the vast number of variations possible, to the comparison of one basic hydraulic hoisting system with the conventional hoisting costs (capital) of seven currently operating coal mines. Thornton (1971) indicates that a review of pilot plants involved in solids transport revealed a general tendency to underestimate costs. Therefore, an effort is made to maintain a conservative posture throughout this study. The best cost data that was made available by industrial sources is interpreted as well as possible and used. Hopefully, the proprietary policies that some firms possess will not detract from this study. Only capital costs are compared because operating cost data was unavailable for inclusion in the analyses. Also included herein is a discussion of possible economic alternatives as well as necessary considerations essential to their evaluation.
2. VERTICAL HYDRAULIC TRANSPORT OF SOLIDS

2.1 Basic Characteristics of Vertical Flow

In the flow of solid-liquid mixtures through pipelines, a number of distinguishing conditions which depend on the properties of the solids, the conveying medium, and the pipeline itself, are described by Zandi (1971). A discussion of these conditions will help to classify the problem.

The assumption is made in this study that the flow is heterogeneous. This condition occurs when solid particles are coarse, have a large relative density, and the velocity is such that it allows partial settling of the solid particles to form a zone of increased concentration near the pipe bottom, in the case of horizontal flow. The presence of these solids should also have little effect on the rheology of the fluid; i.e., the solid and liquid phases will essentially behave separately. Zandi further characterizes heterogeneous flow with the fact that particles do not interact chemically or electrically with the fluid, and that a concentration distribution exists across the pipe cross section with no deposition on the pipe bottom. The above characteristics apply to horizontal pipe flow; however, they can generally be applied to vertical flow also. The concentration distribution for vertical flow is different than for the horizontal case, and the reasons will be discussed later. It should be noted that despite numerous past and present publications concerning the conveying of solid-liquid mixtures, the major portion of what is known is concerned with horizontal pipes, a fact observed also by Gliddon (1957). Heterogeneous flow can be contrasted to the three other possible flow conditions: capsule,
homogeneous, and intermediate flow. Capsule flow, or the containerization of solids for piping, is used when the solids may react with the fluid, when the eventual separation of the solids would be too expensive, or when the solids are of a very abrasive nature. Capsule flow would not be a practical solution in this study. Homogeneous flow occurs when the solids are very fine and light, such that the solid and liquid phases may essentially behave as one. In this case the mixture usually exhibits characteristics unlike those of the conveying liquid and may be classified as a non-Newtonian fluid. Examples of this type of non-deposit flow are the transport of clays, drilling mud, sewage sludge, fine sand or coal, and other finely ground materials. Durand (1953) suggests that particles carried in this type of flow are usually under 30 microns in diameter. The intermediate condition occurs when both homogeneous and heterogeneous flow exist simultaneously. This is possible when the material being transported has a wide size distribution ranging from very fine to coarse particles. Although this characteristic is true of many materials being used in industry, Zandi (1971) indicated that very little work has been done in this area. The case being investigated in this study might fall within this intermediate range. However, classifying the flow as heterogeneous and using equations based on Durand's (1953) work, the pressure gradient may be more conservative (greater) as suggested by Bain and Bonnington (1970). They state that the use of a homogeneous suspension of fine particles as the conveying medium for coarser particles of the same material can result in pressure drops which are less than those encountered when the transporting medium is completely heterogeneous in nature. Therefore the heterogeneous assumption will not adversely affect any final conclusions.
The principles involved in the vertical transport by pipeline of a mixture appear very basic. To appreciate this fact one should first have some understanding of horizontal conveyance. Gliddon (1957) indicates that the flow characteristics of a horizontally conveyed solid-liquid mixture are primarily determined by the velocity of the liquid. The particles will remain almost uniformly distributed across the vertical cross section of the pipe at high velocities. As the velocity is reduced, the solids concentration will increase progressively towards the bottom of the pipe. Upon further reduction a sliding layer and eventually a stationary bed will develop. The depth of bed will increase progressively until complete blocking of the pipe results. These characteristics are discussed by Faddick (1970) and are illustrated by Toda (1969).

In the case of vertical transport, the problem is much simpler. Upward movement will always occur, provided the water velocity exceeds the settling velocity of the particles being transported. Durand (1953) conducted research involving the measurement of concentration and velocity over the cross section of a vertical pipe carrying a solid-liquid mixture. He found that the concentration is nearly uniform over the cross section and that the velocity profile is affected little by the solids. The most significant characteristic of vertical flow is that the frictional pressure gradient is essentially computed like that of clear water, except that the density used is that of the mixture. Durand (1953) indicates that this fact is true of homogeneous as well as heterogeneous mixtures. An explanation is given by Newitt, Richardson, and Gliddon (1961). They found, using photographs, that coarse particles, though randomly distributed at low velocities, actually tend to move
away from the walls at high velocities. This increases the concentration at the center of the pipe while leaving a clear annulus of water at the walls. At concentrations greater than 10 percent by volume, their measurements have shown that the central core of particles moves at a uniform velocity with a steep velocity gradient (boundary layer) near the wall. The presence of the clear annulus explains why the frictional pressure gradient can be determined in the same manner as that of clear water. Govier and Aziz (1972) indicate that the clear annulus will occur at velocities of about 11 fps, which is also confirmed by Toda (1969).

One possible explanation of the occurrence of the clear annulus is suggested by Glidden (1957). He indicates that because of the convex upward shape of the velocity profile in the pipe (the velocity component being a maximum in the center and decreasing to zero at the pipe wall), the velocity component acting on the side of a particle facing the wall would be less than the component acting on the side towards the center. This would result in a rotation of the particle in a direction away from the wall. Since the velocity profile changes most rapidly near the wall, the effect of the rotation may only be of a significant magnitude next to the wall and therefore be effective in maintaining a clear annulus.

Another possible explanation of this particle movement is the change in pressure over the cross section of the pipe as a result of the velocity distribution. From basic energy relationships, the pressure decreases with increasing velocity, thus the pressure will be higher at the walls than at the center. Therefore the formation of a center core of solids may be a result of particle movement from a high to a low pressure zone.

Pressure gradient relationships representing the flow of solid-liquid mixtures in vertical pipes generally involve two terms. The first
term is an expression of the pressure gradient in clear water which is
commonly represented by the Darcy-Weisbach equation,

\[ i_w = \frac{(fLV^2/2gD)}{L} \]  

(1)

where \( i_w \) is the pressure gradient, \( L \) is the length of pipe, \( V \) is the
average water velocity, \( f \) is the friction factor, \( D \) is the pipe diameter,
and \( g \) is the acceleration due to gravity. This equation is taken from
King and Brater (1963), although it can be found in any number of
hydraulics texts. The second term usually involves some type of correc-
tion factor which accounts for the increase in the specific gravity due
to the presence of the solids. Hughes, Hunt, and Pearn (1970) indicate
that by using the various methods available for predicting the velocities
or the resulting pressure drops for pipelining mixtures, estimates can
vary as much as 20 percent. This fact is one reason for attempting to
maintain conservative selections of basic design criteria which are to
be used in this study.

A number of formulas have been developed for determining the pressure
gradient relationships. Presumably, one of the earliest of such relation-
ships, based on experimental findings, has been developed by Durand
(1953). He had found that the head loss between two points in a
vertical pipe carrying a solid-liquid mixture is

\[ \Delta H = L_i + (s_v - 1)L \]  

(2)

where \( s_w \) is the specific gravity of the mixture. Bain and Bonnington
(1970) present a formula similar to Durand's. This formula is

\[ i = i_w (V_w/V)^2 + C_v'(s-1) \]  

(3)

where \( i \) is the pressure gradient of the mixture, \( V_w \) is the average
mixture velocity, \( C_v' \) is the spatial concentration of the solids, and \( s \)
is the specific gravity of the solids. It was found that \( V_w/V \) frequently
equals one, and thus $C_v'$ can be replaced by the delivered concentration, $C_v$. Sample calculations then show this equation and Durand's to produce essentially the same results. Another formula developed from experimental data and hydraulic transport theory is given by Frolov (1959) as

$$i = i_w s_w + \left( \sqrt{gD \left( s-s_o \right)} / 1.9V_T s_o \right) \ldots \ldots \ldots \ldots (4)$$

where $s_o$ is the specific gravity of the liquid, and $T$ is a coefficient of resistance of a free falling particle in water. For coal greater than 10 mm, this value equals $0.65\sqrt{0.66/(s-1)}$. This formula also results in a pressure gradient similar to the value obtained using Durand's formula, only not quite as large. One relationship, however, developed by Newitt, Richardson, and Gliddon (1961) shows a very substantial disagreement with results from the other formulas. This formula is

$$i = i_w + \left( 0.0037 (gD/V^2)^{1/2} (D/d) (p_s/p_w)^{1/3} \right) C_v i_w \ldots \ldots \ldots \ldots (5)$$

where $p_s$ and $p_w$ are the densities of the solids and the liquid respectively, and $d$ is the particle diameter. Using Bain and Bonnington's formula, the expression for the total pressure gradient is substituted into this equation and solved in terms of particle diameter. It is found that Newitt's equation provides the same results as the previous methods for very small particles only, approximately 0.05 mm. The equation given by Frolov was developed from experimental data for particles ranging from two to 60 mm or about 2.5 inches. No specific limitations were stated for the equations given by Bain and Bonnington or Durand. However, when referring to his research concerning hydraulic transport in general, Durand indicates the use of up to four inch particles. Therefore, in lieu of any information to the contrary, it is assumed that Durand's formula for vertical transport applies to particles of this size. From a review of past investigations concerning the basic
characteristics of vertical transport in pipes, it is felt that those premises originally established by Durand have been sufficiently substantiated by the work of others in his field to warrant their application to this study.

It was originally intended to provide the same type of review for equations representing horizontal flow. However, since such equations are not actually applicable to this study, let it suffice to indicate where they can be found. Zandi (1971) presents a rather complete chronological summary of the equations governing the horizontal flow of heterogeneous mixtures beginning with the earliest which was formulated by Durand in 1952. Other discussions on the topic are given by Bain and Bonnington (1970) and Govier and Aziz (1972). Bain and Bonnington also give a brief discussion on the pressure gradient in an inclined pipe, which is merely a vector sum of the gradients required for corresponding horizontal and vertical sections covering the same distance. A review of individual articles covering all types of flow is given in an annotated bibliography on the hydraulic transport of solids in pipelines by Thornton (1971).

2.2 Systems of Transport

2.2.1 Types of Systems

For a vertical hydraulic transportation system there are essentially two major variations to consider in a basic design. These variations are a circulating system and an open system, both as discussed by Leeman and Laubscher (1970). The circulating system consists of two pipes extending from the mining level to the surface with the pumping facilities being located at the surface. Clear water is pumped from
some type of receptacle down one pipe to the mining level. At this point the solids to be transported are introduced into the pipe and carried back up to the surface. Here the coal is separated from the water and the clear water is then recirculated. The open system, however, requires only one length of pipe extending from the mining level to the surface and all pumping facilities are located in the mine. Thus there is no circulation and all the water must come from seepage within the mine. For a mine with a very large coal output, the quantity of water required is rather significant.

2.2.2 Power Requirements

The actual power required to hoist the solids to the surface in a circulating system consists of four components which are described as follows: (1) the power required to overcome the frictional head loss in the downflow pipe; (2) the power necessary to introduce the solids into the pipe at the bottom of the system; (3) the power required to overcome the frictional head loss in the upflow pipe; and (4) the power required to lift the solids a given height H in the upflow pipe. These components are to be referred to, respectively, as \( P_1, P_2, P_3, P_4 \), the sum of which is the total power or \( P_t \). The power components in ft lbs/sec (convertible to horsepower) can be expressed in general terms as the product of the specific weight, flowrate, and head loss. The equations representing each of the components are expressed in terms of the flowrate of the solids and will be presented along with a brief explanation as to their origin. Leeman and Laubscher (1970) have compactly summarized these equations.

The first component, dependent on the friction loss of clear water, is given by the expression

\[
P_1 = \frac{\gamma}{2} \frac{Q^2}{D^2} \Delta H
\]
\[ P_1 = f(H/2D)p_w Q_s V^2(1-C_v)^3/C_v \]  \hspace{5cm} (6)

where \( f \) is the Darcy-Weisbach friction factor for clear water, \( H \) is the pipe length, \( D \) is the pipe diameter, \( p_w \) is the density of water, \( Q_s \) is the flowrate of solids, \( V \) is the mean velocity of the mixture, and \( C_v \) is the volumetric transport concentration equal to \( Q_s/Q \), where \( Q \) being the flowrate of the solid-liquid mixture. Although this equation represents the friction loss in clear water, the flowrate is in terms of the solids and the velocity is in terms of the mixture. These are converted to corresponding values for clear water by the presence of the \( C_v \) terms in the equation.

The second component, that which is required to introduce the solids into the pipe, must overcome the added pressure due to the solids mixture in the upflow pipe. It is expressed by

\[ P_2 = p_w gHQ_s (1+C_v')(s-1) \]  \hspace{5cm} (7)

where \( g \) is the acceleration due to gravity, \( C_v' \) is the spatial concentration equal to \((C_v/W)V\), where \( W \) being the mean upward velocity of the solids. The difference between \( C_v \) and \( C_v' \) is due to the "slipping" effect of the solids as they are being carried upwards by the water. The magnitude of this effect is dependent on the settling velocity of the solids.

The third component, which is necessary to overcome the friction in the upflow pipe, is essentially the same as in the downflow pipe except that the delivery rate is that of the total mixture instead of only clear water. This power is expressed as

\[ P_3 = f(H/2D)p_w V^2(Q_s/C_v) \]  \hspace{5cm} (8)

This term is based on the assumption that the presence of the solids does not add to the friction in the case of vertical flow, as established by Durand (1953) and confirmed by Chapus, Condolios, and Couratin (1962).
The value of $f$ is based on clear water although the actual liquid is not clear. Leeman and Laubscher (1970) indicate that this is permissible for vertical transport and use a value of 0.02. A value of $f$ of 0.02 is also given on a graph presented by Fontein (1958) for values of Froude number greater than 2.0. Represented on this graph is $f$ versus Froude number for the transport of coal in a vertical pipeline. Therefore the value of $f$ chosen for this study is 0.02.

The fourth component of power is that which is necessary to overcome the pressure difference between the bottom of the downflow pipe and the bottom of the upflow pipe due to the presence of the solids in the latter. This is expressed as

$$P_4 = C_V'(s-1)p_wgHQ_s(1-C_V)/C_V$$  

(9)

Combining the four power components and simplifying, the total power required for a circulating system can be expressed as

$$P_t = p_wgHQ_s(fV^2/2gD(1+(1-C_V)^3/C_V)+...$$

$$...+1+C_V'(s-1)/C_V$$  

(10)

A similar equation representing the power required by a circulating hydraulic system has been developed by Chapus, Condolios, and Couratin (1962). The calculation of the power requirements for hydraulically hoisting gold ore in a hypothetical mine using both of these equations shows that the two methods agree within approximately eight percent, Chapus, Condolios, and Couratin's value being the larger. One difference between the two methods was that Leeman and Laubscher used the mean mixture velocity in calculating the friction loss in the upflow pipe, whereas Chapus, Condolios, and Couratin assumed this loss to be the same as that of the downflow pipe. Although it is less conservative, Leeman and Laubscher's equation was chosen because they also develop a similar
equation for the open hoisting system, thus permitting calculations to be compared in a consistent way.

The power requirements for the open system consist of three components. The first two components are exactly the same as equations (7) and (8) for $P_2$ and $P_3$ in the circulating system and will not be repeated. The third component, required to pump the mixture to the surface, is given by

$$P_3' = (1+C_v'(s-1))p_w gH_0 (1/C_v-1)$$

Combining and simplifying the three terms, the expression for the total power required by an open hydraulic system can be expressed as

$$P_t' = p_w gH_0 (fV^2/2gDC_v + \ldots + (1+C_v'(s-1))/C_v)$$

A similar equation for the open system was not considered in the discussion by Chapus, Condolios, and Couratin (1962).

In comparing the two hydraulic systems it becomes readily evident that the primary differences between them are the quantity of water necessary for operation and the fact that an open system must be capable of lifting against a static head which is equivalent to the depth of the mine. Calculation of power requirements for varying delivery rates in a hypothetical mine for both a circulating and an open system are presented in Figure 1. This graph is only intended as a visual means of comparing relative power requirements of the two systems.

Leeman and Laubscher (1970) compare the power requirements for a circulating hydraulic hoist against those of conventional hoisting for three South African gold mines. They note that the results were approximately the same for gold ore 7/8 inch in diameter. They indicate that further crushing of the ore in the mine, however, would result in
Figure I. Power Requirements of Open and Circulating Hydraulic Hoisting Systems

- \( H = 1500 \) ft
- \( D = 12 \) in
- \( f = 0.02 \)
- \( s = 1.70 \)
- \( C_v = 0.25 \)
- \( C_v' = 0.30 \)
a reduction of the power required by the hydraulic hoist. Based on the results of Figure 1, this comparison also proves that an open system would not have been feasible in this situation from the standpoint of power.

It is the opinion of this study that other complex relations relating to flow and pressure gradient characteristics of solid-liquid mixtures in vertical pipes would not warrant consideration in an initial study of this scope when the resulting effect in relation to the overall economic outcome would likely be negligible. In a study such as this, it would be quite easy for one to become bogged down in overwhelmingly minute details and lose sight of one's true objective. It is pointed out by Wasp, Aude, Seiter, and Thompson (1971) that while knowledge of the pressure drop in a slurry pipeline is important, it is only a small portion of the information needed for the overall pipeline design.

2.3 Transporting Media

The only conveying medium being considered for application to this study is that of water. This is not to say that some other fluid might be very efficient, but rather that water is the only medium being considered in this investigation because, among other things, it is usually present in most mines. Of the past and present hydraulic hoisting installations reviewed, the majority of them use water for the conveying liquid. Ashe (1965), although not providing any support to the statement, indicates that the use of media other than water, such as pneumatic or oil systems, would not lend themselves to coal mining operations. However, the growing interest in pneumatic transport is indicated by the sponsorship of the first international conference on pneumatic
transport of solids in pipes by the British Hydromechanics Research Association in 1971.

A discussion presented by Frolov (1959) mentions that laboratory studies conducted in the Soviet Union between 1951 and 1957 show hydraulic transport in heavy liquids rather than water to be quite advantageous. In further studies involving the determination of the cheapest and most effective "heavy liquid", a calcium chloride solution with a specific gravity of 1.3 to 1.4 was chosen from a range of 11 inorganic and 12 organic liquids. It is also noted that the predicted total consumption of calcium chloride would not exceed approximately four to six pounds per ton of coal delivered in a circulating system. Upon further review of the properties of this chemical, it is stated by Perry, Chilton, and Patrick (1963), that it's corrosive effect on steel produced a loss of less than 0.02 in per year for a 60 percent solution with a pH greater than seven and a temperature less than 175°F. One advantage of calcium chloride is its resistance to freezing, an asset which would be invaluable where freezing might be a problem.

Pneumatic transport, offering such advantages as being combined with the ventilation system of a mine to aid in the removal of dust and methane, was recently the subject of a study by Konchesky and George (1971). Their discussion includes reference to some past and present pneumatic hoisting installations, and they express optimism that a practical pneumatic system could possibly be developed. It was indicated, however, that available sources furnished insufficient information from which to draw any conclusion as to its technical or economic feasibility. Therefore, the report includes only a discussion of one conceptual
pneumatic system along with a description of an experimental system which
was built by the United States Bureau of Mines for testing purposes.

Whether the use of calcium chloride, pneumatic transport, or
polymer additives such as those discussed by Poreh, Zakin, Brosh, and
Warshavsky (1970) can be economically justified, is a question beyond
the scope of this study. However, the possibility of investigating these
ideas in the future should not be overlooked.

2.4 Characteristics of Solids Being Transported

The unit weight of coal is quite variable and is dependent on the
form in which it is found. The American Institute of Steel Construction
(1970) indicates that lignite has a specific gravity from 1.1 to 1.4,
bituminous coal from 1.2 to 1.5, and anthracite coal from 1.4 to 1.7.
In order to maintain a conservative posture for design, a value of 1.70
is chosen for use in this study, although other studies such as the
Colorado School of Mines (1963) used lesser values such as 1.40. A lower
value dependent on actual data would most likely be used in a final
design.

The terminal settling velocity of coal is determined through the
application of a progression of graphical relationships, beginning with
the very basic log-log plot of drag coefficient ($C_D$) versus Reynolds
Number ($R_E$) for a smooth sphere in an infinite fluid. This graph can be
found in most books on fluid mechanics, e.g., Rouse (1961). The determi-
nation of settling velocity from this graph is implicit. Good average
classical data for this graph can be found in Zenz and Othmer (1960).
A transformed graph of $R_E$ versus $C_D R_E^2$, which makes the determination of
settling velocity explicit, was plotted for shape factors determined by
Albertson's method of shape factor correlation as given in Zenz and Othmer (1960). A shape factor of 1.0 corresponds to a smooth sphere and values less than 1.0 represent increasing degrees of angularity in particles. A maximum degree of angularity is indicated by a factor of 0.03. $C_D R_E^2$ can be shown to be a function only of particle and fluid properties, as indicated by Zenz and Othmer (1960), among others. Hence, it is possible for one to arbitrarily select a particle diameter and calculate a value of $C_D R_E^2$ for water. Entering the graph with this value of $C_D R_E^2$, a corresponding value of $R_E$ can be obtained, from which a particle settling velocity can be determined.

Using the procedure just described, curves of settling velocity versus particle diameter were plotted for smooth spheres of coal and for coal particles with maximum angularity. These curves are shown in Figure 2. The values on the curve for a smooth particle were compared to a similar graph in Bain and Bonnington (1970), which includes curves for a wide range of specific gravities. Results were nearly identical.

Three of the curves in Figure 2 represent single particles in an infinite fluid. In hydraulic hoisting, however, the particles are no longer falling singly, but can be considered as falling in an obstructing medium, as has been indicated by Chapus, Condolios, and Couratin (1962). The rate of fall is slower than in the case of an isolated particle, more so with increasing concentration. Chapus, Condolios, and Couratin (1962) illustrate the magnitude of this effect on a graph depicting a coefficient of fall velocity versus the solids concentration. For example, at a solids concentration of eight percent by volume, the rate of fall, under the conditions of the test, was reduced by 50 percent. The velocity coefficient for a concentration of 30 percent was applied to the average
Figure 2. Terminal Settling Velocities of Coal

Single Particle of Coal \( s = 1.70 \)

Infinite Water Environment at 50° F

---

**Terminal Fall Velocity (fps)**

<table>
<thead>
<tr>
<th>Particle Diameter (in)</th>
<th>0</th>
<th>1</th>
<th>2</th>
<th>3</th>
<th>4</th>
<th>5</th>
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<td>0</td>
<td>1</td>
<td>2</td>
<td>3</td>
<td>4</td>
<td>5</td>
</tr>
<tr>
<td>SMOOTH SPHERE</td>
<td>1</td>
<td>2</td>
<td>3</td>
<td>4</td>
<td>5</td>
<td>6</td>
</tr>
<tr>
<td>AVERAGE</td>
<td>2</td>
<td>4</td>
<td>6</td>
<td>8</td>
<td>10</td>
<td>12</td>
</tr>
<tr>
<td>MAXIMUM ANGULARITY</td>
<td>3</td>
<td>6</td>
<td>9</td>
<td>12</td>
<td>15</td>
<td>18</td>
</tr>
</tbody>
</table>

---

Figure 2. Terminal Settling Velocities of Coal
settling velocity curve on Figure 2. The resultant effect is represented by the lower curve on Figure 2. Because it is not known under what test conditions the influence of concentration on settling velocity was determined, other criteria were sought to provide a better basis upon which to establish a reasonable but conservative estimate as to the actual rate of fall.

Graf (1971) presents a discussion on the effect of a concentration of particles on the rate of fall in which he refers to an extensive study on the topic by Maude (1958). An equation given by Maude for a settling velocity correction coefficient is

\[ K = (1-C_v)^m \quad . . . . . . . \quad (13) \]

where \( m \) is a coefficient dependent on Reynolds Number and particle shape. From experimental data it is determined that for Reynolds Numbers less than one, \( m \) is about 4.5 and for values greater than \( 10^4 \), \( m \) is about 2.2, with \( m \) varying accordingly between these values. The equation is later used by Leeman and Laubscher (1970) in the study involving the transport of gold ore.

Although hydraulic hoisting systems in general can be designed to handle solids ranging in size from extremely fine to relatively coarse, it was decided that this study would be centered on the transport of coarse coal with a maximum diameter of four inches. A review of information pertaining to hydraulic hoisting installations indicates that, except for the installation discussed by Gardner (1952), this is approximately the maximum solids size which has been transported in the past. This size is chosen because it is typically the maximum size mined at the mine face and hence, would require a minimum of additional preparation facilities
located in the mine itself. At the same time it would maximize hoisting parameters, giving a conservative representation of costs.

Maude's equation was applied to a four inch particle of coal which has a settling velocity of 3.6 fps from the average curve on Figure 2. At a solids concentration of 30 percent, the resulting settling velocity is approximately 1.6 fps. Although not discussed, an additional reduction in settling velocity will result due to the effect of the pipe walls. However, this effect would only tend to make the assumed value more conservative.

The settling characteristics of slate and shale, which may also be present in a coal formation, were briefly examined. The American Institute of Steel Construction (1970) indicates that the maximum specific gravity of slate or shale is about 2.90. Using this specific gravity, the settling velocity of a smooth four inch sphere was determined from the graph previously referred to in Bain and Bonnington (1970). The approximate fall velocity in water would be about 7.9 fps. The actual value would be less than 7.9 fps due to particle roughness, concentration, and wall effects.

The solids concentration used in the preceding calculations is assumed to be 30 percent by volume. This value is based on a discussion presented by Bain and Bonnington (1970) which deals with the specific power consumption of a hydraulic transport system in relation to the concentration of solids being transported. The discussion indicates that the specific power consumption for a system pumping coal, at a specific gravity of 1.40, initially decreases rapidly with increasing concentration and then levels off to a minimum at a concentration of about 40 percent. It is indicated that without specific data pertaining
to the solids in question, this is a safe upper limit for the volumetric solids concentration. But it was also stated in this study that in most cases little would be gained by exceeding a concentration of 30 percent. Therefore a conservative figure of 30 percent was chosen for the spatial concentration of solids and is used throughout the analysis of the hydraulic hoisting system under consideration.

In the final design of the hydraulic hoisting system for this study it was decided that a minimum flow velocity of water of approximately 11 fps would be quite adequate. This is also the approximate velocity at which Govier and Aziz (1972) indicate the first occurrence of an almost clear annulus of water, the reason for assuming no increase in frictional losses due to the presence of the solids. Govier and Aziz also recommended a suitable minimum transport velocity as being twice the settling velocity. A velocity of 11 fps will result in a significant movement of coal whose settling velocity appears to be under 2 fps. It will also be capable of carrying almost any type of rock or other waste which may enter into the system.

2.5 Pipe Requirements

The sizing of pipe in this study is simply based on the maximum size of coal particles being transported, rather than using an overly refined technique described by Leeman and Laubscher (1970) to select the optimum diameter. The minimum size pipe, to avoid the risk of blockage, should have an inside diameter about equal to three times the size of the largest particle being transported. This statement is supported by Davies (1953), Bain and Bonnington (1970), and Singhal (1970). Hence, the resulting pipe diameter selected for use in this
study is 12 in, except in a few cases where the pipe size had to be limited to 10 in to maintain the desired velocity.

The wall thickness chosen for the pipes is 0.250 in. A Russian study by Turchaninov (1961) indicates that, for horizontal slurry pipelines, a wall thickness of at least 0.375 in be used. However, Gardner (1952) and Chapus, Condolios, and Couratin (1962) indicate that actual tests show the abrasion resulting from the vertical flow of solids is almost negligible. Therefore, an economical reduction in wall thickness from that recommended in the Russian study is deemed justified. United States Steel Corporation, the manufacturer of the pipe chosen for this study, indicates that its allowable pressure is greater than the maximum design pressure of 1000 psi which is based on requirements for a 1500 ft deep mine. The pipe is especially designed for transport of slurries and is more resistant to abrasion than conventional steel pipe. Although the abrasion in straight vertical pipe is minimal, this type of pipe would help resist abrasion in bends and any short horizontal sections that might be required.

The question arises that mine drainage water in a hydraulic hoisting unit might be highly corrosive where the water present in the mine is used to pump the coal, or because of the cumulative affects of the re-use of water in a circulating system. Bain and Bonnington (1970) presented a discussion concerning the causes of pipe corrosion and possible means by which it may be reduced. They point out that natural corrosion in a pipe conveying a liquid is rapidly reduced due to the formation of an oxide film on the inside of the pipe. The abrasion action of the solids mixture, however, can cause the continuous removal of this oxide film.
As a result, there may be little reduction in corrosion and a high loss of pipe material. This removal of the oxide film may possibly be reduced in the vertical flow because of the clear-liquid boundary layer.

Corrosion in pipes conveying a liquid is due to electrolysis and involves both cathodic and anodic processes. Corrosion inhibitors are therefore classified as anodic or cathodic, dependent on which portion of the reaction is inhibited. Because of the wide variety of chemical inhibitors available and the number of factors involved, it is difficult to recommend any type of inhibitor without knowing the particular conditions and first performing laboratory tests. Generally, however, Bain and Bonnington (1970) indicate that corrosion is greatly controlled by the amount of dissolved oxygen present in the system. By maintaining a high pH and reducing the available oxygen, corrosion in steel pipes is almost eliminated. Some minerals, such as coal, which may be transported are capable of absorbing some of the oxygen. As an example of the use of inhibitors and their effectiveness, comparative tests were conducted by Consolidation Coal Company. After preliminary tests, several inhibitors, sodium sulphite, sodium dichromate, and hexametaphosphate (commercially known as "Calgon") were selected for further investigation. Of these three, the overall effect of sodium sulphite was not completely understood, and the use of hexametaphosphate resulted in excessive pitting which did not occur when sodium dichromate was used alone. As a result of these studies, a pipe wear of three to five mils per year was chosen as desirable and this required a dosage rate of approximately 20 parts per million of sodium dichromate, which cost about 1.5 cents per ton of coal for the 108 miles of pipeline. In comparison, the rate of wear
without the inhibitor is estimated to be 114 mils or about 1/8 inch per year. The cost of a chemical inhibitor is not included in the economic analysis in this study because only capital costs are being examined. It should again be emphasized that operating costs were not overlooked but rather were excluded from this study because of the unavailability of necessary cost data. The type and required dosage rate of inhibitor would also necessitate an investigation of the expected corrosive characteristics of a coal-water mixture for the specific mine in which the use of a hydraulic hoisting system is being considered.

In the case of highly corrosive mine water, use could be made of corrosion resistant pipe. This would include pipes with some type of lining, such as rubber or the use of a metal such as stainless steel.

Costs were not obtained for rubber lined pipe but it was learned from United States Steel Company that an electrically welded stainless steel pipe, comparable in size to the slurry pipe, would cost approximately eight times as much. And a seamless stainless steel pipe would be nearly 16 times greater in cost. There would also be an added cost for installation due to the need for specialized welders. Thus, it may be cheaper to use corrosion inhibitors or simply replace the pipe when necessary.

Another point to consider is that of the expected life of the pipe. No information could be obtained as to a possible life expectancy of the unlined slurry pipe chosen for this study. This is a matter which is almost solely dependent on individual installation conditions. Wasp, Aude, Seiter, and Thompson (1971) indicate that an expected pipeline design life is usually 20 to 50 years.
3. CONVENTIONAL HOISTING OF COAL

3.1 Types of Mine Entries and Their Relative Use in Pennsylvania

From the latest available Annual Report of the Pennsylvania Department of Environmental Resources--Division of Coal, Oil, and Gas (1970), a total of 209 anthracite mines and 245 bituminous mines were reported in the Commonwealth of Pennsylvania. The bituminous mines are further defined by the type of mine opening. Of the 245 bituminous mines, 180 have a drift type opening and the remainder have either shaft or slope type entries. Of these three types of mine openings, the drift type mines are not being considered in this study. This type of opening is used to develop a coal seam which outcrops on the side of a hill or mountain. The mine entry can essentially be made horizontal into the seam, and the removal of coal will usually proceed in the same manner. Since this study is limited to vertical transport, no effort will be made at looking into the economics of a possible horizontal piping system. In terms of conventional mine transport systems, drift mines are the least costly in comparison to shaft and slope type mines. The system of transport used within a drift mine merely extends to the outcrop of the coal seam. It would appear impractical to install a horizontal hydraulic transport system unless the entire mine were to be operated hydraulically and all other means of coal transport were eliminated. Otherwise, costly equipment would be necessary to transfer from the conventional to hydraulic system when it would be much simpler to continue with the former for the entire length of the mine.
In terms of depth, most of the coal deposits of Pennsylvania are fairly shallow as illustrated by the large number of drift mines present. Depths for shaft type mines are taken from the 1970 Department of Environmental Resources report referred to earlier. These have been organized into the graphical form of Figure 3. This data does not include all the mines in Pennsylvania. Stefanko (1967) indicates that the deepest coal mine in the United States, at that time, was the Beatrice mine of Virginia, whose depth was 1350 ft.

The depth at which the coal deposit lies is the major factor in determining whether a slope or shaft type entry is used to develop a mine. One industrial source indicated that a slope mine is more economical at depths up to about 500 ft and shafts for depths greater than 1000 ft. In between it is debatable as to which type is better. It should be pointed out that this is not a firm rule, but rather a general guideline to which there is always an exception. When two large coal companies were asked to give what they thought was the limiting depth for slope entry mines, one replied with values of 500-600 ft and the other with 1000 ft. The decision of whether or not to use a sloping entry for these intermediate depths depends on the ability of the company to amortize the high cost of a slope belt by sufficient daily tonnage over a long life.

3.2 Conveying Methods and Costs

The coal conveying mechanism used in a shaft is commonly known as a skip hoist. This is an elevator type of arrangement with two cable held buckets or skips which are alternately loaded with coal in the mine and hoisted to the surface. A designer and manufacturer of mine
Figure 3. Depths of Pennsylvania Bituminous Coal Mines with Shaft Type Entries
shaft equipment indicated that a mine with a coal delivery rate of 1000 to 2000 tph would usually require two 18 ton skips. Johnston, Seerly, Short, and McKee (1969) mention that for depths over 1400 ft, a friction type hoist with 40 ton skips supported by two inch cables is used, and it was considered to be the largest hoist within capabilities of United States manufacturers. In addition to the hoists, auxiliary equipment such as a rail car dumping device, in the case of a mine with rail transport, a surge hopper for maintaining a constant delivery rate to the hoist, and weighing hoppers for metering the output is also necessary. Additional facilities would also be required for introducing coal to the hydraulic system being considered.

In the slope mine entry the common type of transport is an endless conveyor belt. The entry for the conveyor is generally constructed at a maximum of 18° from the horizontal in order to keep the coal from sliding on the belt. This means that for a given depth to a coal deposit, a slope length of approximately three times the depth would be required. Besides providing a continuous delivery rate, the conveyor, when also used throughout the mine, is merely an extension of the underground system to the surface. Such an extension uses a separate high tensile strength slope belt. There is no need for transferring the coal from one carrier to another, such as in the case of the shaft arrangement.

In a general comparison of costs between shaft and slope mines, a shaft costs about three times as much per foot to drive as a slope. However, in a situation where the ground is badly fractured, the slope would require much more in the way of structural support than the shaft. As for the transport system, the endless belt in the slope is rather expensive compared to the skip hoists in the shaft. These factors come
under the category of capital costs. Usually for depths under 1000 ft
the total capital costs for a slope are greater than those of a shaft.
However, there are exceptions to the rule, depending on the other condi-
tions unique to individual mines. The opposite is generally true for
the operating costs. With the skip hoist the maintenance and power
costs are higher when compared to those for a conveyor.

In order to determine the magnitude of the costs mentioned above
for use in this study, inquiries were sent out to a number of
Pennsylvania companies which had mines operating in Pennsylvania and
West Virginia. Each was asked if they would volunteer cost data on
some of their mining operations. The resultant data obtained is pre-
sented in Table 1. Represented in this data are mines having either
slope or shaft type entries with depths from 214 ft to 710 ft, and
having either rail or belt transport within the mine at delivery rates
ranging from 600 to 1500 tph. The companies indicated that these
capital costs include all necessary auxiliaries in accordance with good
and lawful mining practices. Values for the yearly volume of water
pumped from the mines were also given by some of the companies. However,

attempts to obtain both initial and operating costs for dewatering
facilities as well as for conventional hoisting operations proved to be
generally unsuccessful. The dewatering operating costs for mine F were
given as $45,000 in 1971. Throughout this study these mines will be
referred to simply as mine A through G. The name and owner of each mine
is given in Appendix B.

The total cost column in Table 2 later in this research is the sum
of the two cost columns of Table 1 plus an amount resulting from a five
percent per year compounding through 1972 as an estimate for inflation.
<table>
<thead>
<tr>
<th>Mine</th>
<th>Type of Entry</th>
<th>Vertical Depth (ft)</th>
<th>Cost of Driving and Lining Entry</th>
<th>Year</th>
<th>Cost of Purchasing and Installing Transport</th>
<th>Current Output (tph)</th>
<th>Water Removed (million gals/yr)</th>
<th>Gallons of Water per Ton of Coal</th>
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</thead>
<tbody>
<tr>
<td>A</td>
<td>Shaft</td>
<td>680</td>
<td>$638,251</td>
<td>1968</td>
<td>$1,260,447</td>
<td>1000</td>
<td>52.5</td>
<td>9</td>
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<tr>
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<td>Shaft</td>
<td>710</td>
<td>$965,396</td>
<td>1968</td>
<td>$1,340,119</td>
<td>1000</td>
<td>38.6</td>
<td>7</td>
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<td>C</td>
<td>Shaft</td>
<td>510</td>
<td>$536,000</td>
<td>1963</td>
<td>$904,000</td>
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<td>$160,000</td>
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<td>400.0</td>
<td>115</td>
</tr>
<tr>
<td>F</td>
<td>Slope</td>
<td>680</td>
<td>$2,292,000</td>
<td>*</td>
<td>$344,000</td>
<td>800</td>
<td>1222.6</td>
<td>787</td>
</tr>
<tr>
<td>G</td>
<td>Slope</td>
<td>300</td>
<td>$2,069,000</td>
<td>*</td>
<td>$441,500</td>
<td>600</td>
<td>uncertain</td>
<td>---</td>
</tr>
</tbody>
</table>

* Assumed to be 1972.
3.3 Further Technical Considerations in the Application of Hydraulic Hoisting

The purpose of this discussion will be to provide some insight into the nature of the considerations necessary in the application of hydraulic hoisting to conventional coal mining operations. Another intent will be to point out some of the options which could affect the overall economic feasibility of such a system. All the factors to be mentioned will not necessarily be included in the economic analyses of this study. In discussing the application of hydraulic hoisting, one should at least consider and account for operational variations in the individual mines before deciding on a possible type of hydraulic system that would be most suitable.

As has already been briefly discussed earlier, an important factor is that of the size of coal produced in the mining operations. In the past when many homes were heated by coal fired furnaces and the use of coal stoves was widespread, there was a great demand for large lump coal. The sizing was no problem because early mining methods resulted in the production of primarily coarse coal. The desire to keep the degradation of the coal to a minimum appeared fairly recently in design of the French hydraulic hoisting installation, as discussed by Chapus, Condolios, and Couratin (1962). Generally, this is no longer true, at least in the United States, because power plants and large industry are the primary recipients of the nation's coal. In these instances the coal is usually pulverized before use. With modern mining methods, such as continuous miners, many more fines are produced and most of the larger coal is about two inches. United States Steel Corporation indicated that less than
five percent of their coal mined is greater than four inches. However, the decision must be made as to whether the system will handle a fine slurry or lump coal as it comes from the mine. In the event that a fine slurry is used, complete crushing equipment must be provided underground and proper safety precautions taken to insure the control of dust. If the hydraulic system is designed to handle the run-of-the-mine coal, or approximately a maximum of four inch coal where continuous miners are used, the fact that minimal crushing facilities should be provided must not be overlooked. It was pointed out by one firm that crushing equipment should be capable of handling oversized pieces of coal or waste rock such as slate and shale which is often present in coal deposits.

If the hydraulic hoisting unit is the sole means of transporting coal from the mine, the water velocity in the transport line must be sufficient to carry the crushed rock which is of a higher specific gravity. In conventional mining operations it is sometimes the policy to hoist coal during the week and then remove the mined waste rock on weekends to avoid interference with other operations when possible, such as those of the surface preparation plant.

Another factor is the end product of the mine's coal and the necessary preparation required before sale. In some circumstances a company will screen all the fines from the coal as it comes from the mine and only the coarser coal is taken through the preparation plant. The fines, on the other hand, are then sold on a BTU per pound basis with no prior preparation. The BTU value is representative of the incombustible waste present and subsequently the coal's worth as a fuel. This method of handling and sale would not be as simple with a hydraulic hoisting installation. The fines, which make up a large portion of the
coal in modern mining methods, would be delivered by a hydraulic system to the surface in a slurry. At this point the fines would be ready for sale after screening under the conventional system. However, with the hydraulic system, the coal might also require separation, but in addition would require initial dewatering and final drying. This would have a definite influence on the overall cost and would require consideration in a comparative analysis of the two systems, if such a situation existed.

Still another consideration which may seem rather evident, but which would influence the ultimate design of the hydraulic hoist, is the water present in the mine. The two system options possible in the design, an open or a circulating hydraulic system, have already been presented in an earlier portion of this study. In conjunction with the quantity of water present is the water quality and its resultant effect on the piping system and any natural stream into which it may find its way. Naturally, the thought occurs that some device could be designed into a hydraulic system that would neutralize acid mine water, although such a device would have to meet the requirements of the Pennsylvania Clean Streams Act of 1965.

In designing a hydraulic hoisting system, it is also essential to include other auxiliary equipment necessary to meet the needs of the mine, e.g., an underground transport system to the hoist feeder. In the event that this transport is by rail, some means of unloading the railroad cars must be provided, such as a rotary dump or a unit for handling bottom dumping cars. Also, one should not overlook the possible need for a spare hydraulic hoisting unit to allow for the possibility of a breakdown in one of the operating units. The normal uninterrupted
operation of the mine is a point which cannot be taken lightly in the final economic analysis, whatever the type of hoisting system may be selected. The possibility arises that an old conventional hoist could serve as the back-up to a newly installed hydraulic hoist.
4. HYDRAULIC HOISTING SYSTEMS

4.1 Successful Installations

The progress being made in the field of hydraulic hoisting is best seen by reviewing the success of past and present hoisting installations. The following discussion covers a period of approximately 20 years and is based on the best information that was found in the literature search.

The Calumet and Hecla's zinc mine in Schullsburg, Wisconsin was the site of a full scale experimental hydraulic hoist as described by Gardner (1952). The hoist, also developed by Gardner and installed on a demonstration basis, has successfully conveyed four inch zinc ore (specific gravity approximately 4.0), a vertical distance of 365 ft through a standard 10 inch steel pipe with water. With five pumps working together to supply a total flow of 2000 gpm, the system could deliver 120 tph, the ore being fed in two-ton batches. Upon increasing the flow to 2500 gpm, a delivery rate of 240 tph could be maintained with a loading cycle of 30 seconds. The ore was introduced into the pipe by a lock hopper type feeder system on the discharge side of the pump. The feeder system consisted of two conical chambers in vertical tandem, with an interconnecting valve and a valve on the top of the upper chamber. The lower chamber emptied into the pipeline through a cut-out section of the pipe. In the operation, a batch of ore was dropped through a hopper into the upper chamber and the upper valve was closed. After the pressure in the two chambers is equalized through a high-pressure water bypass line, the lower valve opens and the ore drops by gravity through the lower chamber into the hoisting pipe. The cycle was automatically
activated by air pistons and electric relays. Experimentation with this prototype indicated no appreciable abrasion in the pipe. It was indicated that this may have been a result of the movement of ore particles to the center of the pipe at high velocities. It was also noted that ore particles as large as eight or nine inches and objects such as pipe sleeves, drill cores, and sledgehammer heads, also had been carried to the surface through the pipe. This system appears to be quite capable of handling substantial quantities of ore. While an output of 240 tph was handled at a loading cycle time of 30 seconds with one feeder, a feeder located on either side of the hoisting pipe would have provided a more continuous delivery rate and would also have boosted the capacity to nearly 500 tph. It should, however, be realized that this high capacity is primarily due to the advantage of the considerable weight of zinc ore in the feeder as pointed out by Borecki (1960). Coal particles, on the other hand, sink more slowly and would probably have a significant effect on the operation of this feeder, although requiring less power to transport in the pipe itself. It is not known whether the operation was an economical one or not. While it was technically a success, there was no apparent commercial use made of the installation. Attempts at further information from contact with the company proved fruitless.

In 1954 the Polish Mining Institute investigated solid-liquid feeder designs. Upon analyzing past means of introducing coal into pipelines, such as the Calumet and Hecla installation, it was decided that the type of feeders used were unsuitable for industrial application because of their low capacity. The result was the design of a one-chamber feeder, as discussed by Borecki and Radowiki (1958). The feeder
consisted of a pressure vessel which was separated from the transport line by a valve. After being emptied of water it was charged with a load of coal. After the inlet valve was closed and the transport line valve was opened, a worm gear in the vessel forced the charge of coal into the transport line. A prototype of the feeder was investigated on the surface under conditions closely approximating those in actual underground work. The installation was then installed in the Diebensko colliery in Poland at a depth of 310 meters. The feeder was successfully operated, and maintained an output of 100 tph, with a coal gradation in the zero to four inch range.

Chapus, Condolios, and Courtin (1962) discuss the results of their extensive research on hydraulic hoisting carried out by the Sogreah Company of Grenoble, France. From investigations on several projects in France, it was reported that hydraulic hoisting of solids is not only feasible, but that it can offer a number of attractive features such as: (1) small shaft space requirements, (2) low investment costs, (3) low operating costs resulting from good power efficiency, and (4) ease of combining hoisting with horizontal transport. As an example of possible output, the authors state that as much as 9000 tons of coal (size not given) can be hoisted daily through a 16 inch pipe based on continuous 24 hour pumping, this output is equivalent to 375 tph. It is not known whether this was achieved in an actual test or was merely an estimate. The operating principles of the Sogreah system were very similar to those presented by Gardner (1952) in that large particles of coal are fed into the transport line through a lock-hopper type arrangement. A hoisting installation of this type was also designed by the Sogreah Company and built in the Devillaine Coal Mine near St. Etienne, France. It consisted
of a 590 ft vertical lift plus a 215 ft horizontal section. The capacity of this circulating system was 50 to 60 short tons per hour of 3 1/4 inch sized coal or less. After a year of operation, wear on the pipes as well as the lock hopper type arrangement was very low. Valves experienced the most wear. The authors also quoted costs from the hoisting installation for 200 tph of minus four inch coal and 300 tph of minus 1 1/4 inch coal. However, comparable costs of conventional hoisting are not given. An alternative concept, an open system using high pressure pumps located in the bottom of the mine and pumping the coal-water mixture directly through the pumps, was rejected because of increased wear on the pumps. The Sogreah system is also described by Chapus, Condolios, and Couratin (1962) as being an ideal means of increasing the shaft output of a conventional hoisting operation. It is not likely that a hydraulic hoist would be used to increase the capacity of a conventional shaft hoist which operates at a pre-designed delivery rate. But it is quite possible that the hydraulic hoist may be used when the present mine has spread horizontally to such an extent that it is no longer economical to transport the coal underground to the existing shaft. In this case the cost might be reduced by hoisting the coal through a pipe in a bore hole at a more convenient location.

One of the more recent projects involving the hydraulic hoisting of coal is the Hitachi hydro-hoist, as discussed by Singhal (1970). Due to the large amounts of water present in Japanese mines, the concept of using drainage water as a means of transporting the coal to the surface has been under consideration since the early 1950's. It was not until 1962 that the first successful commercial installation, the Hitachi hydro-hoist, was installed in a Japanese mine. It received a
U.S. patent in 1969. The hoist was designed for the transport of coarse granular solids in a mixture with water over large vertical distances. Unlike the systems previously discussed, the hydro-hoist feeding device consists of a series of three pipes of the same diameter as the transport line. These pipes are horizontally arranged parallel to each other to serve as the discharge chambers of the feeder. Through the use of valves at both ends of each pipe and a type of bypass circuit tied into each of the pipes for introducing the mixture, the three pipes are filled and discharged in sequence into the transporting line to the surface. A high pressure clear water pump is linked directly with the feed pipes and provides the momentum needed to carry the coal to the surface. A low pressure sand pump is connected to the bypass circuits and supplies the necessary flowrate of coal-water mixture to the feed pipes. With proper coordination and timing of valves, a rather continuous flow of coal can be maintained. An advantage of the hydro-hoist is that it combines the capabilities of a high volume, low head centrifugal pump and a low volume, high head reciprocating plunger pump to operate in those head/flowrate ranges where neither centrifugal nor plunger pumps will serve. The hydro-hoist can operate successfully at heads up to 1000 meters or 3280 ft and at horizontal distances of up to 6.2 miles. In the two operating installations in Japan, the maximum size particle transported is 2.3 in at a delivery rate of 100 tph. Singhal indicates, however, that, because of the low delivery rate, widespread application of hydro-hoist may be hindered in cases where delivery rates of 500 tph or more are required. Although the hydro-hoist has clearly proven its effectiveness through actual operation, its consideration in a study such as this
would be minimal because most coal mines in the United States have a much larger required output.

Another type of hydraulic hoisting installation is in operation in the Bancroft Mines Ltd. in Zambia, as has been discussed by Ashe (1965). Copper ore, representing 30 percent of the total mined tonnage was successfully hoisted by centrifugal slurry pumps for a period of about six years at the time the article of Ashe was written. The conditions existing in the mine proved to be rather costly for the original mining system. The mine was very wet and presented safety problems due to "mud rushes" in storage piles at the shaft loading point. This in conjunction with numerous other problems resulting from excess water led the mine operators to the decision to incorporate into the system transport by hydraulic means. In the resulting arrangement, the raw ore underwent crushing, screening, and washing, after which the 70 percent coarse fraction was hoisted conventionally. The remaining ore with a maximum size of 3/16 in was then taken by launders to the pumping facilities where it was piped to the surface and dewatered. In this installation the solids pass through the pumps instead of being introduced into the discharge line. The first pumping station was located at the mining level, at a depth of 1317 ft. Located here were nine centrifugal pumps coupled in series and capable of pumping to an intermediate level located at a depth of 650 ft. Nine more pumps then hoisted the slurry to the surface. Pumps at the mining level discharge into an open sump at the 650 ft level, but they also have the operational capability of pumping the total distance to the surface, thereby eliminating the intermediate level. The system was designed to handle
160 tph at a pumping concentration of 40 percent solids by weight with the slurry having a specific gravity of approximately 1.3. Each of the 18 pumps is driven by a 90-hp motor and delivers a mixture of 1200 gpm. A second set of standby pumps were later installed to insure continuous operation without having to rely on the conventional system in the event of a breakdown.

Ashe discusses the application of the system used in the Bancroft mines to the pumping of coal. Approximate calculations are presented for pumping 300 tph of minus 2 1/2 inch coal versus minus 1/4 inch coal. The results indicate it to be less expensive to pump the finer material. The costs only represent the pumping equipment, and no discussion was given concerning crushing and abrasion. Chapus, Condolios, and Couratin (1962), have eliminated consideration of this type of system because excessive pump abrasion eventually reduced pumping efficiency. In the same article Ashe also reviews the use of hydraulic hoisting in general and discusses the factors to be considered in designing a hydraulic system. The opinion is expressed that technical factors involved in hoisting installations are well known and publicized, whether one is referring to a slurry pumping or a lock hopper type system. However, it is indicated that the economic factors relating to hydraulic transport and hoisting are not clearly understood, and each system must be examined on its individual merits.

The examples of successful hydraulic hoisting installations that have been discussed represent published information that is freely available. However, it must be understood that business firms, in their quest for a position of advantage, may restrict dissemination of information on all or part of an installation. A case in point might be a
study carried out by the British Hydromechanics Research Association on the hydraulic transport of lead slurries, as annotated in the Bulletin of Hydraulic Research (1966 & 1967). The results of the study were simply listed as confidential. Also, during the course of this study, correspondence was conducted with a large coal company in the United States working on hydraulic hoisting. However, no further information was made available. These two examples merely bring attention to the fact that this study can take into consideration only those technological advances for which information can be freely found.

4.2 Solids Feeding Systems

The delivery rate which hydraulic systems can maintain is dependent primarily on the capabilities of the solids feeding devices. A spokesman for the Bethlehem Mines Corporation indicated that a desirable delivery rate for a hydraulic hoisting unit would be approximately \sqrt[600]{600} \text{tph} to 1000 \text{tph}. The importance of this factor in considering hydraulic hoisting became evident in the last section. Essentially four different types of feeder arrangements emerge from the literature review. The first type is the basic lock-hopper feeder used in the Wisconsin zinc mine. This consists of two large interconnected chambers emptying by gravity into the smaller transport line which delivers a maximum of 240 \text{tph} of ore. It is claimed that this same type of feeder has delivered over 300 \text{tph} in the Sogreah installation. The second type is a one-chamber device with the coal being discharged into the transport line through the use of a worm gear. This feeder has successfully handled 100 \text{tph}. The third type is the Japanese hydro-hoist feeding system which delivers 100 \text{tph}. The feeder chambers are three parallel
pipes of the same diameter as the transport line. The fourth type is simply the pumping of a mixture of fine particles and water through the pumps. It does not actually require the use of a special feeder on the discharge side of the pump as in the other three cases, but rather involves a supply sump for handling the mixture. This system has handled 160 tph as indicated by Ashe (1965).

Other types of feeders are discussed by Bain and Bonnington (1970). They mention two main categories of feeders: first, different variations of the lock-hopper type, and second, moving pocket type feeders. The latter consists of a storage hopper which empties into a device having a rigid sliding section containing two open segments or pockets. In operation the sliding section is moved into a position where one pocket can be filled with coal from the storage hopper while the other is in line with a pipe delivering water from a high pressure pump. The water flowing through the pocket then carries the coal into the pipe line and to the surface. When the coal falls into these pockets by gravity, only a very low delivery can be achieved. However, delivery can be improved if the solids are forced into the pocket by a stream of low pressure water. One of the problems encountered in this design is leakage through seals around the pockets. These seals are subject to considerable wear. No definite statements are made concerning delivery rate. However, Bain and Bonnington present a graph of horsepower versus theoretical coal output and the output scale only extends to a value of 100 tph.

Frolov (1959) mentions several feeders which have been developed in the Soviet Union. These are primarily screw type feeders and one
of their advantages is referred to as being compactness. Illustrations are presented by Frolov. The capacities of these feeders are on the order of 30-40 tph.

In the first portion of a study presented by Oedjoe, Davies, and Buchanan (1963) the theoretical aspects of a new solids feeding device called the Hydro-Lift are discussed. The device involves the introduction of a high pressure jet of water into the conical shaped bottom of a closed cylindrical chamber containing the solids to be transported. The transport pipe, entering through the top of the chamber, is suspended a short distance above and directly over the incoming jet. As the particles travel into the path of the jet, the force of the jet acts to drive them up into the transport line and to the surface. Research on this system has only been done on a small scale so far, and no data is given as to the expected capacities or maximum size of particles. The system may have possibilities in the future upon further refinement and full scale testing.

Recent information from the Homer Research Laboratory of the Bethlehem Mines Corporation indicates that a hydraulic hoisting system is presently being employed in Poland which pipes coal directly from the mining face to the shaft and up to the surface in one step. The research laboratory is only presently beginning to investigate the system and had no information concerning its operation, capabilities, or success. This type of system would undoubtedly present numerous technical problems, such as facilities for sizing the coal in addition to a feeder of sufficient mobility capable of keeping up with the mining operations. In the case of modern continuous miners used in the United States, the required equipment for a hoisting system may be of substantial size.
If, however, these problems can or have been overcome, this type of system would be a more efficient application of hydraulic transport to mining. It would eliminate the use of two transport systems, i.e. rail or belt transport from the face to the shaft, and hydraulic hoisting from the shaft bottom to the surface.

Of the feeding systems discussed for which sufficient information was available, none could be considered to have delivery rates which would make them attractive to modern large scale mining operations. The feeder system eventually chosen for this study is a lock-hopper type as described by Fontein (1958). It is based on studies done by the Mining Research Establishment of the Dutch State Mines. In these studies it was decided that a lock-hopper system should be capable of handling 700 tph of coal in a size range from 0-80 mm (about 3.1 in). The feeder developed by Gardner (1952) was initially considered as a likely possibility. However, this design was discarded by the researchers because of its relatively small capacity due to rather long waiting periods between charges of coal. It was found that this was generally true of most feeders with large conical bottoms. Heavier construction was also required for chambers of larger diameter. Experiments were then conducted on Perspex type models as a possibly more efficient alternative. In the Perspex type of lock-hopper feeder, the chambers are of small diameter, approaching or essentially equal to the diameter of the transporting line. Such chambers are presumably capable of large capacities, from 700 to over 1000 tph. It should be noted that this information is based on model studies only. Reduction in the time required to complete the loading cycle is achieved through the use of a locking chamber which consists of a cylinder of a given diameter inside
a cylinder of a larger diameter which makes up the chamber exterior. This concept is illustrated for this study in Figure 4. The inner cylinder is also of a shorter length so that it opens into the outer chamber at the lower end. In operation, the mixture of coal and water drops by gravity into the inner pipe causing the clear water present from the previous cycle to be forced out the bottom and up the space between the two walls, leading to an extinction tank. This tank is sized so that it becomes filled just as the coal approaches the bottom of the main chamber (A and B). The remaining water is then forced out of this tank through a small diameter pipe, thus reducing the momentum of the coal charge as it passes the upper valve, allowing this valve to be closed immediately. If this means of dampening were not provided, the charge of coal and water, because of its inertia, tends to rebound upon reaching the bottom of the chamber and thus increase the time which must elapse before the upper valve can be closed. It is estimated by the researchers that with proper coordination and timing of controls, the cycle of a feeder with a capacity of about 1400 tph could be completed in less than 10 seconds. Again one should keep in mind that these figures are based on assumptions relative to model studies and not an actual full scale installation. In a full scale facility the overall length of the feeder may be as large as 80 ft, depending on the capacity involved and the diameter of the chamber. Fontein's paper also includes criteria used in the modelling. It was assumed that the influence of wall friction in the feeder could be neglected, and hence, the influence of the Reynolds Number would be minor. The ratio of the specific gravity of the liquid to the solids, and the ratio of the particle diameter to the pipe diameter are the same for the prototype and the model. Thus,
Figure 4. Lock-Hopper Coal Feeder
the Froude numbers, based on the mixture velocity, will also be the same. These criteria were then used in determining the full scale capacities from the results of tests on a 1:5 model.

The coal feeding system developed by the Dutch State Mines was chosen for use in this study because it is representative of the current state of the art in hydraulic hoisting. The reason is due to its potential as indicated by those who researched it relative to other systems, although a prototype feeder has not been used in actual operation. Therefore, an effort was made to keep assumptions concerning the feeder's capabilities rather conservative so as not to overestimate its potential and unjustifiably influence the results of the economic evaluation. Hence, the capacity of the feeder was estimated as being 500 tph.

4.3 General Opinions From Literature and Industry

In order to provide a more complete picture of hydraulic hoisting, it seems only appropriate to provide a brief review of facts and opinions expressed in the literature as well as those encountered through personal contact. These ideas have not been included in earlier discussions.

Although not considered in this study, Maurier and Lemke (1968) discuss the application of a complete hydraulic mining system as investigated and used in the Soviet Union. This system includes the removal of coal from the mining face by hydraulic means, the transport of coal through the mine in open flumes to a shaft, and the lifting of the coal to the surface by pipeline. The paper discusses the use of such a
system and its limitations relative to the characteristics of the coal seam being mined.

Johnston, Seerley, Short, and McKee (1969) indicate that when dealing with large coal mines having a delivery rate of approximately 40,000 tpd or greater, transport systems such as the slurry or pneumatic type are technically impractical. Their discussion may have been referring to systems of mines because mines usually approach a maximum of 20,000 tpd.

Frolov (1959) discusses an earlier study in the Soviet Union which indicates that hydraulic hoisting should be 1/2 to 1/3 of the cost of hoisting coal by conventional means. This estimate is presumably based only on mining in the Soviet Union.

Harper (1970), who is associated with a firm involved in the manufacture of mine haulage equipment, indicates that the capital costs for a new hydraulic hoisting installation are comparable to those of a new conventional hoisting installation. However, the operating costs of the hydraulic system would seem unattractive unless compared with the combined cost of hoisting the coal and dewatering the mine in a conventional operation. When an inquiry was made by the writer concerning the justification of this statement, no support could be given. But reference was made to the Hitachi Corporation, likely referring to the Hitachi hydro-hoist in Japan, as was discussed earlier.

In gathering data for the economical analysis in this study, several different viewpoints were encountered from persons in industry. A representative of a large Pennsylvania corporation was very enthusiastic about the future of hydraulic transport in mines and indicated that they were currently in the process of beginning investigations to determine
its applicability to their operations. It was also learned that another
mining company in Pennsylvania is currently conducting similar research.
The opinion was expressed by another person that the cost for the initial
commercial hydraulic hoisting installation in the United States would
probably run about twice as much as a comparable conventional installa-
tion because such a system is not yet perfected. This person was an
engineer associated with a corporation involved in the design and build-
ing of mine production shafts.

As can be seen from the preceeding remarks, the general consensus
concerning hydraulic hoisting is rather varied. It is the opinion of
the author that before any hydraulic hoisting system reaches the point
where it can be applied economically to commercial mining operations,
it must first show it can successfully compete with a system such as the
conventional skip hoist. In the United States, the skip hoist has proven
to be very reliable because it has benefitted from many years of techno-
logical refinement.
5. TECHNICAL FEASIBILITY AND DESIGN OPTIONS

5.1 Basis for Justification

Before any type of economical analysis can be meaningful, the technical feasibility of hydraulic hoisting must first be established. From a review of past efforts, there appears to be little doubt that hoisting is technically possible, and probably to any degree.

As an illustration, technical feasibility was investigated for a hypothetical 1500 ft deep coal mine from which it was assumed that four inch coal was hydraulically hoisted under a maximum pressure gradient (pumps down in the mine). The depth was chosen as being greater than the deepest known coal mine in the United States. The conclusion with regard to technical feasibility was based on the availability of the primary components necessary for the system. These components are the pumps, the transporting pipes, and the means of feeding the coal into the pipe on the discharge side of the pump. Using the method of calculation for an open pumping system as given by equation (12) for an arbitrary delivery rate of 500 tph, a transport concentration of 25 percent, a pipe size of 12 in, and a specific gravity of coal of 1.70, the flowrate and head requirements could be determined. A flowrate of 3520 gpm of water at a corresponding total head of 2570 ft or a pressure of approximately 1100 psi was required. Since it was found that the Union Pump Company in Michigan could supply multi-stage centrifugal pumps capable of flowrates up to a maximum of 4000 gpm under a corresponding total head of approximately 5500 ft, commercially supplying a capable pump presents no real problem. Obtaining pipe with adequate pressure capabilities also presents no problem as indicated in the
section on pipe requirements. Hence, a coal feeding device is all that is needed to establish technical feasibility. The 500 tph was chosen as a reasonably desirable delivery rate in order to determine the flow-rate and pressure gradient required. The feeders of some past and present hydraulic hoisting installations have already operated under conditions more extreme than those mentioned above. The installation discussed by Gardner (1952) pumped a heavier material with a larger maximum particle size and the installation discussed by Singhal (1970) operated under a greater head. It is assumed that the feeder in Figure 4 can also be designed to meet the above conditions. It should be noted that the question of technical feasibility is essentially independent of the delivery rate of the coal feeder. The delivery rate per feeder will, however, influence the economic feasibility.

5.2 System Alternatives

It seems only appropriate within this discussion of technical feasibility to include a brief summation of possible design options, some of which have been discussed in detail before. Hydraulic hoisting may be the sole means of hoisting coal to the surface or it may be combined with existing conventional equipment to provide a supplemental means of lifting coal, as indicated by Chapus, Condolios, and Couratin (1962) and Singhal (1970). A mine with hydraulic hoisting for transporting coal to the surface might use some other means such as rail to get the coal to the shaft. Or it could have a completely hydraulic transport system from the mine face to the surface. In addition a system could be designed to handle any range of gradation, from raw coal with waste rock mined at the mine face to a finely pulverized material using
complete crushing equipment underground to form a fine slurry for transport, possibly through the pump instead of discharge line feeding.

The conveying medium used to transport the solids can also be varied. Although the use of water is most common, perhaps in the form of mine drainage, the use of other fluids such as inexpensive solutions in water or even air may present possibilities. Chemicals added to water to increase buoyancy may also reduce the corrosiveness. Even friction reducing polymers might be used. There is also the possibility of floating the coal to the surface in a column of liquid with a specific gravity greater than that of coal. Fluids capable of this phenomena might be rather expensive, and the delivery rate per pipe would most likely be small, although there would probably be a savings in power and possibly in the cost of handling the fluids. Also, with pumps down in the mine in an open system, dewatering is achieved as an advantage in addition to the hoisting of the coal. In the closed system for a dry mine the pumping facilities are located on the surface and the water carrying the coal is clarified and recirculated. If the mine requires dewatering, but the water in the mine is insufficient for the desired delivery rate, the water could be pumped to the surface by a separate system as in a conventional mine, treated, and then used as needed to recharge a closed hydraulic system. Another possibility in the same situation is pumping the mine water directly into the main chambers of the coal feeder during discharge of the coal. The water may, however, require some treatment in the mine before pumping. Consideration can also be given to pumping in stages at mine level and at mid-mine level, if the gradation is a slurry that can pass through pumps.
Although a complete economic analysis would certainly involve the thorough investigation of all of the above possibilities, it is beyond the scope of this research to do so. However, it was the author's intention to at least point out alternatives as documentation of possible future study.
6. A CONCEPTUAL HYDRAULIC HOISTING SYSTEM

6.1 Basic Design

The general layout of the system (excluding pumping facilities) which provides the basis for an economic comparison is shown in Figure 5. The units shown on this diagram represent a conceptual system for a mine with a delivery rate of approximately 1000 tph. This will be treated as an alternative to conventional hoisting operations in a coal mine. It extends from the point where conventional transport within the mine, whether rail or conveyor belt, delivers the coal to the base of the shaft or slope entry to the point where the coal is taken into the preparation plant on the surface. Figure 5 also does not include clarification equipment for the recirculating system which would be located on the surface. An itemization of costs for these items as well as other details are presented in Appendix A. The costs of those items for which a particular trade name or manufacturer was not cited must be estimated as items of special fabrication. This also includes installation costs of individual units. Assistance with these costs was obtained from appropriate consulting or manufacturing firms. An effort was also made to standardize the sizes of these units so that they could be applied singly or in multiples of the basic unit. As an example, two 500 tph feeders (standard size) would be required in a 700 tph mine.

To accommodate a hydraulic hoisting unit in conventional underground mining operations, a surge hopper for converting the output from the mine into a constant delivery rate for the hydraulic hoist would be required. Such a hopper is also used in conventional skip-hoisting installations.
Surge Hopper receiving coal from rotary dump or belt transport in mine.

Weighing Hoppers

Crusher (receiving +4 in coal)

Crushed coal to conveyor

Conveyor

Reciprocating Feeder

Vibrating Screen (passing +4 in coal)

Figure 5. Underground Portion of a Hydraulic Hoisting System
Where belt transport is used from the mine face, the belts can empty directly into the hopper. For mines A and B, transport in the mine is by rail. The company indicates that in both these mines the railroad cars are emptied by a rotary dumping device, the cost of which has been included in the total cost of the conventional hoisting installation. Also necessary in this type of operation is a car haul which automatically positions the cars for dumping. The cost of these two items are therefore included in the hydraulic system of mines A and B. As for the surge hopper, a consulting firm involved in the design of mine production shafts recommends a 500 ton hopper as an appropriate capacity and indicates that any greater variation in delivery rate be controlled in the actual mining operations.

The coal from the surge hopper is emptied onto a conveyor belt at a constant rate by means of a reciprocating feeder. One reciprocating feeder will be included in the cost estimate for each operating lock-hopper feeder unit required by the mine. A spare reciprocating feeder is provided for the mines which require only one operating lock-hopper unit. In the other mines one of the remaining feeders would be capable of providing enough coal for two lock-hopper units in the event of a breakdown.

The "run of the mine" coal is taken by conveyor to the vibrating screens which selects the plus four inch coal for the crushing unit. As in the case of the reciprocating feeders, one screen is provided for each lock-hopper unit. The crushing unit is capable of handling up to 112 tph of any type of rock and can reduce it to less than four inches. Because only about five percent of the coal from the mines is greater than four inches, only one crusher is provided for each of the
mines. For mines D and E where three operating lock-hopper units are required, the three sets of screens and conveyors would be placed in a parallel fashion similar to the arrangement shown in Figure 5. Only in these mines would all the screens divert the oversize coal to one crusher which would follow the center screen and discharge the crushed coal onto the center belt. The rate at which the coal is fed to the three screens can be regulated, depending on the amount of oversize coal, in order that the total output is evenly distributed among the three lock-hopper units.

The coal is then carried by conveyor belt to the two weighing hoppers required for each of the lock-hopper units. The weighing hoppers alternately release measured batches of coal (approximately 2 tons) into the feeder supplying the transporting pipeline. The company which provided the information on the weighing hoppers indicates that the system could be based on either batches of a constant weight or a constant volume. With the type of lock-hopper feeder being used, batches of constant volume are most suitable because sufficient clearance is needed to close the valves of the main chambers after they are filled with coal. A coal charge of a constant volume would insure proper operation of the lock-hopper unit. The cost of the weighing hopper quoted by the company is for one much larger than required. An estimate was made for an appropriate hopper based on the cost of the larger sized unit.

The lock-hopper feeder unit is illustrated in Figure 4. The movement of coal and water is shown by the corresponding arrows. The size of the chambers is based upon a total delivery rate of the unit of 500 tph and a time of 20 seconds for one chamber to complete a cycle. A value of 80 lbs per ft$^3$ was used as the bulk weight of the coal to
estimate the required volumes. The resulting length of the main chambers (A and B) is 19 ft and the upper chambers (C and D) are 20 ft. An estimate of the complete cost of the feeder unit, including construction, installation, and controls, was roughly approximated by one company and appears in Appendix A.

The length of pipe required by the recirculating system for each lock-hopper unit is twice the depth to the coal seam plus 50 ft. The addition of the 50 ft is to account for the lock-hopper unit setting below the level of the seam. Another 150 ft is added to this total to account for possible lengths of horizontal pipe which may be required. The 150 ft is a conservative extra rather than part of the transport system to the shaft entrance. The installation cost includes welding and erection. The cost of laterally supporting the pipe in the shaft (to prevent buckling) was estimated at $20 to install a brace every ten feet. This cost was applied only to the vertical sections of pipe, not the 150 ft added to the pipe required for each unit. The length of pipe required per lock-hopper unit for the dewatering system is simply equal to the depth to the coal seam plus the same 200 ft as above. Installation costs are also the same.

The selection of the pumping units is based on the required capacity and pumping heads for each mine. Subsequently, the calculated power requirements are given by equation (10) for a recirculating system and equation (12) for a dewatering system. The data which varied from mine to mine are given for each mine in Appendix B. It is noted that terms such as coal flowrate, required horsepower, flowrate of water, and pumping head are per lock-hopper feeder or hoisting unit. The operating range of the pumps examined were such that one basic type of pump could
be applied to the recirculating systems (type A, Appendix A). The only difference for the individual mines is a change in impeller diameter which has no influence on the overall costs. One other pump (type B, Appendix A) was selected to accommodate the dewatering system proposed for mine F.

For the recirculating hydraulic hoisting system, the cost of providing adequate clarification for the water returning to the mine was inclined in the analysis. A description of the clarifiers is given in Appendix A. Such equipment is not included in the cost of the dewatering system or its comparable conventional system, the dewatering cost of which are unknown anyhow. It is assumed for the sake of cost comparison that, because of the large volumes of mine drainage involved, the conventional installation would already have adequate clarification equipment capable of removing coal particles and treating the water. With the closed system, the mine may have much less water and the conventional system clarification facilities would be much smaller. However, the closed hydraulic system still recirculates a large volume of water and additional clarification facilities will be necessary.

The final item in each comparison will be the cost of the shaft. This term is included in the total cost of both the conventional and the hydraulic systems, so that all comparisons will be on a similar basis. In the case of a hydraulic system in a shaft mine, the costs of the shafts are assumed identical.

One point not included in the cost analysis of the hydraulic system is that of possible increased underground excavation costs. This cost may only be significant in a mine with a large output requiring a number of hydraulic hoisting units. Simply as a matter of information,
one firm indicated that costs of underground excavation may be as high as $34 a cubic yard. The consideration of excavation costs in the overall economic analysis, however, would require a more exacting design of the hydraulic system to determine necessary space requirements.

6.2 Economic Comparison with Conventional Hoisting

The question of economic feasibility, taking all possibilities into consideration, can be rather complex. The true complexity of the problem was not fully anticipated at the onset of this study. It was eventually concluded that in order to accomplish a meaningful economic analysis, the study would have to limit itself to the investigation of one basic type of hydraulic hoisting system adapted to existing mines for which data was available. The subsequent analysis consists of a comparison of the initial costs (including installation) of this basic type of system with comparable costs of seven existing coal mines with conventional hoisting facilities. It should also be noted that no attempt was made to unitize the cost of hydraulic hoisting in terms of the total amount of coal mined, such as the cost of hoisting per ton of coal. This technique would have required some knowledge of operating and depreciation costs of components of the system. Because such information could not be obtained, the capital costs of the hoisting facilities were related to the size of the mine, in terms of the delivery rate and the depth.

The results of the economic comparison between conventional and hydraulic hoisting systems are presented in Table 2. The estimated total comparison cost for each of the components required for a complete hydraulic hoisting system have been presented in Appendix B for the mines
### Table 2. Comparison of Capital Costs of Conventional and Hydraulic Hoisting

<table>
<thead>
<tr>
<th>Mine</th>
<th>Type of Entry</th>
<th>Depth (ft)</th>
<th>Current Output (tph)</th>
<th>Total Cost of Conventional Hoisting System</th>
<th>Estimated Cost of Hydraulic Hoisting System</th>
<th>Maximum Capacity of Lock-Hopper Feeder</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>500 tph/Feeder</td>
<td>600 tph/Feeder</td>
</tr>
<tr>
<td>A</td>
<td>Shaft</td>
<td>680</td>
<td>1000</td>
<td>$2,307,700</td>
<td>$2,257,400</td>
<td></td>
</tr>
<tr>
<td>B</td>
<td>&quot;</td>
<td>710</td>
<td>1000</td>
<td>$2,802,200</td>
<td>$2,699,300</td>
<td></td>
</tr>
<tr>
<td>C</td>
<td>&quot;</td>
<td>510</td>
<td>600</td>
<td>$2,232,800</td>
<td>$2,028,400</td>
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<tr>
<td>D</td>
<td>Slope</td>
<td>214</td>
<td>1500</td>
<td>$560,300</td>
<td>$1,800,400</td>
<td>+221</td>
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<tr>
<td>E</td>
<td>&quot;</td>
<td>240</td>
<td>1400</td>
<td>$960,000</td>
<td>$1,836,400</td>
<td>+91</td>
</tr>
<tr>
<td>F</td>
<td>&quot;</td>
<td>680</td>
<td>800</td>
<td>$2,635,800*</td>
<td>$2,030,400*</td>
<td>-23</td>
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<tr>
<td>G</td>
<td>&quot;</td>
<td>300</td>
<td>600</td>
<td>$2,510,500</td>
<td>$1,506,500</td>
<td>-40</td>
</tr>
</tbody>
</table>

* Does not include capital cost of a separate dewatering system.
examined. After determining the total cost for each hydraulic system, an arbitrary ten percent was added to this total to account for any contingencies which may have been overlooked. It should also be noted that the number of significant figures in the individual costs for each mine is not significant, but rather "carried along" for rounding off in the final results. A negative in the % Diff column is favorable to hydraulic hoisting relative to conventional hoisting.

Although operating costs have not been included in this analysis, it was realized that they could prove to be an important consideration in establishing economic feasibility. An example of this can be seen in the selection of conventional hoisting facilities, namely, the alternative of either installing a sloping entry using belt transport or a shaft entry with a skip hoist. As has been mentioned, the total capital costs for a slope entry are generally greater than those in a shaft entry for depths down to 1000 ft. Examining only capital costs in this case would probably be misleading since the operating costs of a conveyor system in a slope entry are usually less than those of a skip hoist in a comparable shaft entry. The operating costs for a hoisting installation are dependent on the number of personnel necessary to operate the facilities, the power required, the cost of maintenance and repair or replacement, and the cost of any expendables. For the hydraulic hoisting system being considered, the magnitudes of the primary power components are known, and it would probably be possible to make an estimate as to the personnel requirements. However, it would be rather difficult to make a reasonable estimate as to the expected maintenance and repair without actual data from an operating installation. Even if reasonable estimates were made, comparable operating cost data for conventional hoisting systems are
unavailable. The same situation exists for both capital and operating costs of the necessary dewatering equipment of the existing coal mines under study.

In order to complete an economic analysis of the hydraulic hoisting system, it was necessary to make the following reasonable assumptions. The majority of the assumptions have been previously justified.

(1) The capacity of one lock-hopper coal feeder is 500 tph.
(2) The maximum size of coal being transported is four inches.
(3) There is a minimal amount of waste rock present in the coal seams of the mines and it can be adequately handled by the crushing facilities included in the hydraulic hoisting system.
(4) The same shaft is used in both the conventional and the hydraulic system. The cost of driving and lining per foot of shaft for the conventional shaft mine is used to estimate the similar cost in the hydraulic system when comparison is made with conventional slope mines.
(5) The hydraulic system is assumed to transport the entire output of the mine.
(6) The systems being compared require essentially the same preparation (coal washing) facilities, and hence this cost is not included in the analysis.
(7) Additional clarification equipment is required only in the case of a closed hydraulic hoisting system.
(8) The transport concentration of coal is 25 percent by volume.
(9) Sufficient space can be made available within the mine for the installation of coal feeders and auxiliary equipment.
(10) The hydraulic system only provides for the hoisting of the coal from the mine level to the surface and not for transport within the mine itself.

(11) All drive units on pumps are assumed to be of the constant speed type, and adequate throttling of flow can be obtained by use of in-line valves.

(12) The sole transporting medium is water.

(13) All transport lines are considered to be vertical except for relatively short horizontal sections at the coal feeders and on the surface. The flow velocity is assumed to be sufficient to transport the coal through these horizontal sections.

(14) Abrasion in vertical pipes is considered minimal and pipe walls 1/4 in thick are sufficient.

(15) Operating costs are assumed to be traded off equally, and therefore, are not included in the costs being considered.

The hydraulic hoisting system for this study has been limited in scope by the above assumptions. A broadening of the analysis beyond these assumptions will depend on future research and possibly pilot type studies.

Coeuillet and Veillet (1955) present an investigation somewhat similar to this study, where the costs of hydraulic hoisting systems were compared to those of conventional hoisting systems for several coal mines. The results of their study indicate that in most cases the cost of hydraulic extraction was comparable with that of the most up-to-date conventional haulage installations. Much the same could be said for this research. It was mentioned also that hydraulic methods would one day catch up to conventional hoisting in the degree of technical
development. Nearly 20 years later the hydraulic hoisting of coal has managed to achieve some degree of success outside the United States, particularly in Europe. No successful commercial hydraulic hoisting installation exists in the United States for the sole purpose of hoisting coal from underground mines. The feasibility of European hydraulic hoisting is based upon its attractiveness in comparison to conventional European hoisting systems. However, the present need is a comparison of hydraulic hoisting with current conventional hoisting systems in the United States which are reportedly more advanced than similar European techniques. Such a comparison is the primary purpose of the economic analysis.
7. SUMMARY

7.1 Discussion of Results

Each mine was examined on the basis of a recirculating hydraulic system with a maximum capacity of 500 tph per hoisting unit. From the information obtained on the amount of water present in each mine, the quantity of water pumped from mine F was sufficient to combine the normal dewatering with the hoisting system. For mines C and G the advantage of increasing the capacity of the coal feeding system above 500 tph per unit was investigated. For mines A, B, and C with shaft type entries, the recirculating hydraulic system estimate was less than conventional costs by about $50,000 to $200,000. The economic advantage of the recirculating hydraulic system in mine C could probably be attributed to a type of transport used within the mine itself. Had the mine transport been by rail instead of belt, the increase in cost due to the need for rail car pumping facilities would have been over $200,000 for a rotary dumping unit.

Hydraulic hoisting is favored slightly but consistently in the shaft mines A, B, and C; however, for mines D, E, F, and G, with sloping entries, the differences between the conventional and the recirculating hydraulic hoisting system costs are somewhat uncertain and more pronounced. In the case of mines D and E, the hydraulic system is more costly by approximately $1,000,000. Mines F and G, on the other hand, show the hydraulic system to be less expensive than the conventional system. It is believed that the great difference in costs for mine G is primarily due to geologic conditions involved in the construction of the slope.
entry. The depth of mine F was more than twice that of mine G, yet the costs of the shaft were nearly the same. In view of this fact, an advantage of installing a hydraulic system is that the required shaft would be easier to construct under adverse conditions than a comparable sloping entry.

As indicated earlier, only one of the mines contained enough drainage water to be considered for a dewatering type of hydraulic hoisting system. Based on a 25 percent volumetric concentration of coal, approximately 748 gallons of water are required to remove each ton of coal from the mine, although figures for the yearly output of coal and the total volume of water pumped resulted in a value of approximately half of that figure. The assumption was made that the water was adequate, in order to compare the initial costs of the dewatering system with those of the recirculating system. The result was that the recirculating and dewatering systems only differed by about $42,000, the dewatering system being the higher of the two. The actual cost difference will be less than this if the cost of a separate dewatering system is included in the recirculating system cost. An advantage of the recirculation system is a reduction in power costs as is illustrated by Figure 1. An advantage of the dewatering system is that the capital and operating costs of normal dewatering would be included in the hoisting costs. The operating costs of mine F for dewatering totaled $45,000 in 1971. Based solely on the results of this comparison, it would be difficult to establish the economic advantage of the recirculating system over the dewatering system or vice versa.

In the analysis of mines C and G, both with current outputs of 600 tph, the recirculating hydraulic hoisting system required three hoisting
units in order to handle the output of either mine. This was based on a maximum feeder capacity of 500 tph and the provision of one spare hoisting unit. Because the mine output in each case was only 100 tph greater than the assumed capacity of one hoisting unit, the economic advantage of providing a feeder capable of 600 tph was investigated. This would mean only two hoisting units would be required by the hydraulic system. The additional cost of building a feeder capable of handling 600 tph is considered relatively negligible since it would only involve an increase in the length of the feeder chambers. This also involves the assumption that a delivery rate of 600 tph is physically possible. This comparison indicates that in both mines this increase in feeder capacity would result in a reduction in costs of approximately $250,000. This study has not ignored the problem of physical limitations of past and present coal feeders, but rather, it attempts to provide some insight concerning the economic feasibility of hydraulic hoisting, assuming a better feeder can be developed. The feeder in Figure 4 has not tested against the physical limitations of a high delivery rate coal feeder, but it has exhibited a potential beyond that of other coal feeders of this type.

In an attempt to determine any general conclusions that could be drawn from the results of this study, numerous ways of expressing the results of the economic comparisons in graphical terms were examined. A regression analysis was not deemed justified for this study because of the small number of data being considered. The graphical illustration which best displayed a relationship between conventional and hydraulic hoisting costs is presented in Figure 6, which is a plot of the total hoisting system cost per ft versus the ratio of output per ft.
Figure 6. Cost/Depth versus Output/Depth for Conventional and Hydraulic Hoisting
The points on this graph only represent the costs of the conventional and recirculating hydraulic hoisting systems. The cost axis was divided by the depth for each mine so that the mines could be examined on a comparable basis. Figure 6 defines limiting bounds of costs for each of the methods of hoisting coal (conventional and hydraulic). The upper boundary for the conventional data was extended to account for effect of the one extreme point which represents the conventional cost of mine G. It should be realized, however, that the large cost of mine G may be due to certain localized conditions and not truly representative of mining costs in general. From the intersection of bands on Figure 6 it also appears as if conventional hoisting is more economically attractive in most cases. However, the data in Table 2 indicates that hydraulic hoisting is favored. The reason for this possible misunderstanding is that Figure 6 is a plot also representing the output and depth characteristics of the mines and is not indicative of the relative number of mines for which hydraulic hoisting is economically favored. After the somewhat arbitrary determination of the Figure 6 boundaries, it was hoped to obtain some indication of the circumstances under which the costs of either system would become competitive with one another. Based on the limited data which was available for consideration, such a statement is perhaps overly optimistic, although some degree of optimism may be warranted due to the conservative nature of the cost estimates. An attempt to define an indicator of feasibility is made in the next section.
7.2 Conclusions

From a state-of-the-art review of information dealing with the vertical hydraulic transport of solids in pipelines, the following conclusions are drawn:

(1) Researchers are generally in agreement concerning the basic characteristics of the vertical hydraulic transport of solids in pipelines, although the actual means of determining pressure gradients may vary somewhat.

(2) Based on the number of successful hydraulic hoisting installations in existence, and possessed of the knowledge that appropriate equipment essential to the operation of the practical system used in this study can be made available, one can conclude that the hydraulic hoisting of coal from underground mines is definitely a technical feasibility, perhaps to any degree.

(3) Although there have been a number of successful installations, the economic feasibility of hydraulic hoisting is relatively unstudied and uncertain.

(4) A factor which has hindered the application of hydraulic hoisting to conventional coal mining operations is the apparent limited delivery capacity of hoisting units.

Although an all encompassing economic analysis of hydraulic hoisting including all of the various system options was beyond the scope of this study, the results that were obtained from the specific systems compared indicate certain trends between the available capital costs of conventional mine hoisting methods and those of a comparable hydraulic hoisting system. Data on operating costs were unavailable, but its inclusion in
the analyses would have been a valuable addition to the study. The conclusions determined from the capital cost comparisons are as follows:

(1) For coal mines with shaft type entries, hydraulic system costs are slightly less than those of the existing conventional systems.

(2) In coal mines having slope type entries, the hydraulic hoisting system appears to be more feasible with increasing mine depth and decreasing coal output, e.g. the hydraulic system could not compete with a conventional system for a mine at a depth of 200 ft with coal output of 1500 tph.

(3) Conventional and hydraulic hoisting systems may be economically competitive for output to depth ratios between two to four tph per ft depth; hydraulic hoisting appears favored below a ratio of two.

(4) An increase in the coal delivery rate per hydraulic hoisting unit increases the economic desirability of the hydraulic system.

The hydraulic hoisting system selected in this study was based on a coal feeding device with a capacity of 500 tph. Although this feeding system has not been evaluated under full scale operating conditions, scale test results have shown it to have a potential exceeding prototype hydraulic hoisting facilities.

The primary objective of this study has not only been to provide some insight into the economic feasibility of the hydraulic hoisting of coal from underground mines, but also to define the state of the art including avenues of investigation open to future researchers. It is concluded that the hydraulic hoisting of coal exhibits some degree of economic feasibility relative to capital costs when applied to certain underground mines in the United States. Thus the feasibility should be
determined on an individual basis for any mine in question. The feasibility would also depend on the actual operating costs of the hydraulic system. Because of the energy crisis which this country is beginning to experience, there will most likely be a corresponding tendency towards "less margin of safety" design methods in many types of systems. This philosophy may also tend to brighten further the outlook of hydraulic hoisting as viewed in this study in that the design of a hoisting system would be less conservative. It is felt, however, that while hydraulic hoisting shows favorable indications of becoming a means of mine to surface transport of coal, it will be some time before it attains the degree of dependability required by commerical mining operations.

7.3 Recommendations

Although the lock-hopper feeding device in this study appears to possess attractive operating characteristics, it is not recommended that the next step in further research be the construction of a full scale pilot installation. It is the opinion of this study that further verification of the operating characteristics and capabilities of this device and the system be done again on a scale model basis. In this way basic design data for the feeder could be developed, along with an experience factor so necessary in undertaking a full scale pilot plant.

If a full scale pilot plant were to be built, the hoisting unit consisting of a pump, pipe, coal feeder, weighing hoppers, crusher, screen, surge hopper, reciprocating feeder, conveyor belts, and clarification equipment for a 500 tph unit in a mine of, for example, 500 ft, would cost about $450,000. This figure assumes an existing mine not in
use, but conventionally equipped, could be made available at no cost. If the mine were abandoned and all the coal removed, test coal could be used. Pumps could be located in the mine and/or on the surface, depending on the type of system one wished to investigate—recirculating and/or dewatering. A dewatering system (pumps located in the mine) could be built in which the mixture would be pumped to the surface and back in a continuous pipe, but then separation of the coal from the water would have to be done in the mine, rather than at the surface as in the recirculating system. Unless it was known exactly what mine would be used and the conditions involved, a pilot plant cost would be difficult to estimate accurately. In general, a pilot plant could range well beyond $500,000; therefore, the decision to undertake such a task should be well justified.

Future research involving hydraulic hoisting as conceived in this study should first be concentrated on the investigation of present as well as new concepts of feeding devices. Efforts should also be aimed at the evaluation of other alternatives in hydraulic hoisting which could actually affect the concept and economy of a system. These include such options as pumping pulverized coal (optimum size to be determined), the use and possibly treatment of acid mine drainage water within the system, the use of other fluid media such as inexpensive solutions in water to increase the buoyancy of the coal and possibly reduce corrosiveness, the use of hydraulic hoisting as a supplemental system to conventional hoisting, and the use of a completely hydraulic transport system from mine face to preparation plant.
BIBLIOGRAPHY


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APPENDIX A

HYDRAULIC SYSTEM COST DATA COMMON TO ALL
OF THE PARTICULAR MINES STUDIED

Power Requirements:

Circulating and Dewatering Systems: Equations (10) and (12)

\[ P_W = 1.94 \frac{\text{lb-sec}^2}{\text{ft}^4}, \ g = 32.2 \frac{\text{ft}}{\text{sec}^2}, \ f = 0.02 \]

\[ C_V = 0.25, \ C_V' = 0.30 \]

Flowrate of Water: \((Q-Q_s)\)

Determined from following equation: \((Q-Q_s) = (Q_s)(1/C_V)-Q_s\)

Units: cfs, Conversion to gpm: 1 cfs = 449 gpm

Pumping Head: TDH

Determined from following equation: \(TDH = (hp \times 3960)/Q-Q_s\)

Units: ft

Pumps and Auxiliary Equipment:

Type A: Circulating system pump

Manufacturer: Union Pump Co., Type: single stage centrifugal,

Head and Flow: 700 ft and 4500 gpm

Speed: 3550 rpm, Impeller Diameter: 9 to 13 in

Cost: $8350

Auxiliary Equipment (costs) - Motor (constant speed): $9200

Mounting Base: $1800, Coupling: $400, Reduced Voltage Starter: $2400

Installation: 4800 lbs per unit @ $0.40 per lb

Total Cost per Pumping Unit: $24,070
Type B: Open or dewatering system pump

Manufacturer: Union Pump Co., Type: single stage centrifugal,
Head and Flow: 800 to 2000 ft and 3200 gpm, Speed: 3550 rpm,
Impeller Diameter: 11 to 13 in, Cost: $24,600
Auxiliary Equipment (costs) - Motor (constant speed): $17,000,
Mounting Base: $2000, Coupling: $600, Reduced Voltage Starter: $2400, Explosion Proof Construction of Motor: 25 percent or $4250
Installation: 9000 lbs per unit @ $0.40 per lb
Total Cost per Pumping Unit: $54,450

Piping:

Type: Slurry pipe, Manufacturer: U.S. Steel Corp.

Wall Thickness: 0.250 in

Cost: 10 in diameter - $345.80 per 100 ft
12 in diameter - $422.91 per 100 ft

Installation Cost: $1800 per 100 ft, Lateral Support - $200 per 100 vertical ft (vertical length = total length minus 150 ft per hoisting unit)

Total Cost per 100 ft (minus lateral support):
10 in diameter - $2146
12 in diameter - $2223

Lock-Hopper Coal Feeders:

Total Cost per Feeder: $100,000

This cost is only an estimate, a contract with a capable manufacturer would be necessary for actual purchase.
Weighing Hopper:

No. Required per Lock-Hopper Feeder: 2, Capacity: 2 tons
Cost per Hopper: $15,000 (Includes $4600 loading cell)
Installation: 15,000 lbs per unit @ $0.40 per lb
Total Cost per Weighing Hopper: $21,000

Crusher:

Type: 24 x 42 in Rockmaster, Manufacturer: McLanahan and Stone Corp., Motor: 75 hp, 5-8 percent slip, Speed: 1200 rpm,
Costs - Crusher: $38,675, Gear Guard and Oil Bath: $1495,
V-Belt Drive: $1075, 8 D300 Belts: $750, Motor (explosion proof): $900, Installation: 40,500 lbs per unit @ $0.40 per lb
Total Cost of Crusher: $59,095

Screens:

Type: 6 x 14 ft, vibrating, Tilt: 25°
Manufacturer: Roberts and Schaffer Inc.
Capacity: 500-600 tph, Required Headroom: 25 ft
Total Cost per Screen: $25,000

Conveyors:

Type: 42 in wide belt
Manufacturer: Irwin-Sensenich Corp.
Cost: (including drive motor and supporting structure) $55 per ft
Installation: $15 per ft
Total Cost per ft of Conveyor: $70
Surge Hopper:

Capacity: 500 tons, Cost: $25,000
Installation: 60,000 lbs per unit @ $0.40 per lb
Total Cost of Surge Hopper: $49,000

Reciprocating Feeders: (no. required depends on output of mine), Cost per Feeder: $15,000
Installation: 16,000 lbs per unit @ $0.40 per lb
Total Cost per Feeder: $21,400

Rail Car Dumping Facilities:

Car Haul: 200 hp, Cost: $82,690
Installation: 80,000 lbs per unit @ $0.40 per lb
Rotary Dump Cost: $60,000
Installation: 100,000 lbs per unit @ $0.40 per lb
Total Cost of Dumping Facilities: $214,690

Clarification Equipment:

Type: Circular Clarifier, Depth: 12 ft, Capacity: 4 mgd,
Diameter: 80 ft
Concrete Structure: $150 per yd³ = $44,700
Mechanical Equipment: $40,000
Installation: 1/3 of mech. cost = $13,300 (estimate)
Excavation: 3 ft deep @ $2 per yd³ = $1120
Pipe: Estimated as 200 ft of 12 in pipe = $844
Total Cost of 4 mgd Clarifier: $99,964

Capacity: 5 mgd, Diameter: 90 ft
Concrete Structure: $54,400
Mechanical Equipment: $40,000
Installation: $13,300
Clarification Equipment: (continued)

Excavation: $1400
Pipe: $844

Total Cost of 5 mgd Clarifier: $109,944

Shaft:

In mines with shaft type entries, cost of the shaft was assumed to be the same for the hydraulic system as for the conventional system. Where necessary, costs were adjusted to 1972 prices at an inflation rate of five percent compounded annually. In the case of mines with slope type entries, the cost per ft of shaft was based on the shaft costs of mines A and B. The estimated cost of the shaft where the conventional entry is a slope is $1100 per ft.
APPENDIX B
ECONOMIC ANALYSES OF PARTICULAR MINES

Mine A

Name: Blacksville #1  
Owner: Consolidation Coal Co.

Type of Entry: Shaft  
Transport in Mine: Rail

Current Output: 1000 tph  
Depth: 680 ft

Proposed Hydraulic Hoisting System

Type: Circulating with pumps on surface

Static Head: 730 ft  
Coal Flowrate: 7.84 cfs

Feeder Capacity-Maximum: 500 tph  
Operating: 500 tph

Required Power: 504 hp  
Flowrate of Water: 3520 gpm

Pumping Head: 568 ft

<table>
<thead>
<tr>
<th>Item</th>
<th>Cost</th>
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<tbody>
<tr>
<td>Pumping Units: No. Reqd. 3 Type-A</td>
<td>$72,240</td>
</tr>
<tr>
<td>Piping: Size-12 in Length-4830 ft</td>
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<td>Lock-Hopper Feeders: No. Reqd. 3</td>
<td>$300,000</td>
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<tr>
<td>Weighing Hoppers: No. Reqd. 6</td>
<td>$126,000</td>
</tr>
<tr>
<td>Crusher: No. Reqd. 1</td>
<td>$59,095</td>
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<tr>
<td>Screens: No. Reqd. 2</td>
<td>$50,000</td>
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<tr>
<td>Conveyor Belts: Total Length-380 ft</td>
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<tr>
<td>Surge Hopper:</td>
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<tr>
<td>No. of Hoppers-1 No. of Feeders-2</td>
<td>$91,800</td>
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<td>Rail Car Dumping Facilities</td>
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<td>Clarification Equipment:</td>
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<tr>
<td>No. of Units-2 Size-5 mgd</td>
<td>$219,888</td>
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<td>Shaft:</td>
<td>$775,751</td>
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Estimated Cost: $2,052,194

Plus 10 percent for Contingencies: $205,219

Estimated Total Cost of Hydraulic System: $2,257,413
**Mine B**

Name: Blacksville #2  
Owner: Consolidation Coal Co.

Type of Entry: Shaft  
Transport in Mine: Rail

Current Output: 1000 tph  
Depth: 710 ft

**Proposed Hydraulic Hoisting System**

Type: Circulating with pumps on surface

Static Head: 760 ft  
Coal Flowrate: 7.84 cfs

Feeder Capacity-Maximum: 500 tph  
Operating: 500 tph

Required Power: 526 hp  
Flowrate of Water: 3520 gpm

Pumping Head: 593 ft

<table>
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Estimated Cost $2,453,899

Plus 10 percent for Contingencies $245,389

Estimated Total Cost of Hydraulic System $2,699,288
Mine C

Name: Coffeen  Owner: Consolidation Coal Co.
Type of Entry: Shaft  Transport in Mine: Belt
Current Output: 600 tph  Depth: 510 ft

Proposed Hydraulic Hoisting System

Type: Circulating with pumps on surface
Static Head: 560 ft  Coal Flowrate: 1.58 cfs
Feeder Capacity-Maximum: 500 tph  Operating: 300 tph
Required Power: 220 hp  Flowrate of Water: 2130 gpm
Pumping Head: 408 ft

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<td>Clarification Equipment:</td>
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<tr>
<td>No. of Units- 2 Size- 4 mgd</td>
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Estimated Cost $1,844,003
Plus 10 percent for Contingencies $184,400
Estimated Total Cost of Hydraulic System $2,028,403
Mine C

Name: Coffeen
Owner: Consolidation Coal Co.

Type of Entry: Shaft
Transport in Mine: Belt

Current Output: 600 tph
Depth: 510 ft

Proposed Hydraulic Hoisting System

Type: Circulating with pumps on surface

Static Head: 560 ft
Coal Flowrate: 3.16 cfs

Feeder Capacity-Maximum: 600 tph
Operating: 600 tph

Required Power: 492 hp
Flowrate of Water: 4240 gpm

Pumping Head: 460 ft

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<td>No. of Units-2 Size-4 mgd</td>
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Estimated Cost $1,612,183

Plus 10 percent for Contingencies $161,218

Estimated Total Cost of Hydraulic System $1,773,401
Mine D

Name: McElray  Owner: Consolidation Coal Co.

Type of Entry: Slope  Transport in Mine: Belt

Current Output: 1500 tph  Depth: 214 ft

Proposed Hydraulic Hoisting System

Type: Circulating with pumps on surface

Static Head: 264 ft  Coal Flowrate: 2.63 cfs

Feeder Capacity-Maximum: 500 tph  Operating: 500 tph

Required Power: 182 hp  Flowrate of Water: 3520 gpm

Pumping Head: 206 ft

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<td>Piping: Size-12 in Length-2712 ft</td>
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<tr>
<td>Weighing Hoppers: No. Reqd. 8</td>
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<td>Crusher: No. Reqd. 1</td>
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<tr>
<td>Screens: No. Reqd. 3</td>
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<tr>
<td>Surge Hopper: No. of Hoppers-1 No. of Feeders-3</td>
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Estimated Cost $1,636,777

Plus 10 percent for Contingencies $163,677

Estimated Total Cost of Hydraulic System $1,800,455
Mine E

Name: Robena
Owner: U.S. Steel Corp.

Type of Entry: Slope
Transport in Mine: Belt

Current Output: 1400 tph
Depth: 240 ft

Proposed Hydraulic Hoisting System

Type: Circulating with pumps on surface

Static Head: 290 ft
Coal Flowrate: 2.63 cfs

Feeder Capacity-Maximum: 500 tph
Operating: 467 tph

Required Power: 184 hp
Flowrate of Water: 3300 gpm

Pumping Head: 221 ft

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<th>Item</th>
<th>Cost</th>
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<tr>
<td>Pumping Units: No. Reqd. 4 Type-A</td>
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<td>Piping: Size- 12 in Length- 2920 ft</td>
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<td>Conveyor Belts: Total Length- 570 ft</td>
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<td>Rail Car Dumping Facilities:</td>
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<td>Shaft:</td>
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Estimated Cost $1,669,497

Plus 10 percent for Contingencies $166,950

Estimated Total Cost of Hydraulic System $1,836,447
Mine F

Name: Concord
Owner: U.S. Steel Corp.

Type of Entry: Slope
Transport in Mine: Belt

Current Output: 800 tph
Depth: 680 ft

Proposed Hydraulic Hoisting System

Type: Circulating with pumps on surface

Static Head: 730 ft
Coal Flowrate: 2.10 cfs

Feeder Capacity-Maximum: 500 tph
Operating: 400 tph

Required Power: 383 hp
Flowrate of Water: 2830 gpm

Pumping Head: 535 ft

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<th>Item</th>
<th>Cost</th>
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<tbody>
<tr>
<td>Pumping Units: No. Reqd. 3 Type-A</td>
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<tr>
<td>Piping: Size- 12 in Length- 4830 ft</td>
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<td>Lock-Hopper Feeders: No. Reqd. 3</td>
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</tr>
<tr>
<td>Weighing Hoppers: No. Reqd. 6</td>
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<tr>
<td>Crusher: No. Reqd. 1</td>
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<tr>
<td>Screens: No. Reqd. 2</td>
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<tr>
<td>Conveyor Belts: Total Length- 380 ft</td>
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<tr>
<td>Surge Hopper: No. of Hoppers- 1 No. of Feeders- 2</td>
<td>$ 91,800</td>
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<td>Rail Car Dumping Facilities:</td>
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<tr>
<td>Clarification Equipment: No. of Units- 2 Size- 4 mgd</td>
<td>$ 199,928</td>
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<td>$ 804,000</td>
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Estimated Cost $ 1,845,793

Plus 10 percent for Contingencies $ 184,579

Estimated Total Cost of Hydraulic System $ 2,030,372
Mine F

Name: Concord  Owner: U.S. Steel Corp.
Type of Entry: Slope  Transport in Mine: Belt
Current Output: 800 tph  Depth: 680 ft

Proposed Hydraulic Hoisting System

Type: Open (Dewatering) with pumps in mine
Static Head: 730 ft  Coal Flowrate: 2.10 cfs
Feeder Capacity-Maximum: 500 tph  Operating: 400 tph
Required Power: 876 hp  Flowrate of Water: 2830 gpm
Pumping Head: 1224 ft

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<td>Conveyor Belts: Total Length- 380 ft</td>
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<td>Rail Car Dumping Facilities</td>
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Estimated Cost $1,883,813
Plus 10 percent for Contingencies $188,381
Estimated Total Cost of Hydraulic System $2,072,194
Mine G

Name: Mine No. 60  Owner: Bethlehem Mines Corp.

Type of Entry: Slope  Transport in Mine: Belt

Current Output: 600 tph  Depth: 300 ft

Proposed Hydraulic Hoisting System

Type: Circulating with pumps on surface

Static Head: 350 ft  Coal Flowrate: 1.58 cfs

Feeder Capacity-Maximum: 500 tph  Operating: 300 tph

Required Power: 144 hp  Flowrate of Water: 2130 gpm

Pumping Head: 268 ft

<table>
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<th>Item</th>
<th>Cost</th>
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<tbody>
<tr>
<td>Pumping Units: No. Reqd. 3  Type- A</td>
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<td>Piping: Size- 10 in Length- 2550 ft</td>
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<td>No. of Units-2 Size- 4 mgd</td>
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Estimated Cost $1,369,533

Plus 10 percent for Contingencies $136,953

Estimated Total Cost of Hydraulic System $1,506,486
Mine G

Name: Mine No. 60  Owner: Bethlehem Mines Corp.

Type of Entry: Slope  Transport in Mine: Belt

Current Output: 600 tph  Depth: 300 ft

Proposed Hydraulic Hoisting System

Type: Circulating with pumps on surface

Static Head: 350 ft  Coal Flowrate: 3.16 cfs

Feeder Capacity-Maximum: 600 tph  Operating: 600 tph

Required Power: 308 hp  Flowrate of Water: 4240 gpm

Pumping Head: 288 ft

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<th>Item</th>
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Estimated Cost $1,146,853

Plus 10 percent for Contingencies $114,685

Estimated Total Cost of Hydraulic System $1,261,538
APPENDIX C

ORGANIZATIONS AND PRIVATE FIRMS WHO CONTRIBUTED INFORMATION TO THIS STUDY

American Iron and Steel Institute, Washington, D.C.
Bechtel Corp., San Francisco, Calif.
Breon's Electric Motor Sales, State College, Pa.
Consolidation Coal Co., Library, Pa.
Connellsville Corp., Connellsville, Pa.
Gulf Publishing Co., Houston, Texas
Heyl and Patterson Inc., Pittsburgh, Pa.
Irwin-Sensenich Corp., Irwin, Pa.
Pennsylvania Department of Environmental Resources, Harrisburg, Pa.
Roberts and Schaffer Inc., Pittsburgh, Pa.
Taylor Forge Division, Bellwood, Ill.
Union Pump Co., Battle Creek, Mich.
STATEMENT OF TRANSMITTAL

Special Research Report SR-97 transmitted herewith has been prepared by the Coal Research Section of the College of Earth and Mineral Sciences Experiment Station. Each of the Special Reports listed below presents results obtained in connection with one of the research projects supported by the Departments of Environmental Resources and Commerce of the Commonwealth of Pennsylvania or a technical discussion of related research. The following is a list of Special Research Reports to date:

SR-1 The Crushing of Anthracite May 31, 1957
SR-2 Petrographic Composition and Sulfur Content of a Column of Pittsburgh Seam Coal August 1, 1958
SR-3 The Thermal Decrepidation of Anthracite September 15, 1958
SR-4 The Crushing of Anthracite with a Jaw Crusher November 1, 1958
SR-5 Reactions of a Bituminous Coal with Sulfuric Acid February 1, 1959
SR-6 Laboratory Studies on the Grindability of Anthracite and Other Coals April 1, 1959
SR-7 Coal Characteristics and Their Relationship to Combustion Techniques April 15, 1959
SR-8 The Crushing of Anthracite with an Impactor-Type Crusher April 25, 1959
SR-9 The Ignitibility of Bituminous Coal (A Resume of a Literature Survey) May 4, 1959
SR-10 Effect of Gamma Radiation and Oxygen at Ambient Temperatures on the Subsequent Plasticity of Bituminous Coals May 6, 1959
<table>
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<tbody>
<tr>
<td>SR-11</td>
<td>Properties and Reactions Exhibited by Anthracite Lithotypes Under Thermal Stress</td>
<td>May 11, 1959</td>
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<tr>
<td>SR-12</td>
<td>Removal of Mineral Matter from Anthracite by Chlorination at High Temperatures</td>
<td>June 22, 1959</td>
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<tr>
<td>SR-13</td>
<td>Radiation Stability of a Coal Tar Pitch</td>
<td>June 25, 1959</td>
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<tr>
<td>SR-14</td>
<td>The Effect of Nuclear Reactor Irradiation During Low Temperature Carbonization of Bituminous Coals</td>
<td>July 31, 1959</td>
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<tr>
<td>SR-15</td>
<td>Effect of Anthracite and Gamma Radiation at Ambient Temperature on the Subsequent Plasticity of Bituminous Coals</td>
<td>August 5, 1959</td>
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<tr>
<td>SR-16</td>
<td>The Isothermal Kinetics of Volatile Matter Release from Anthracite</td>
<td>August 25, 1959</td>
</tr>
<tr>
<td>SR-17</td>
<td>The Combustion of Dust Clouds: A Survey of the Literature</td>
<td>November 30, 1959</td>
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<tr>
<td>SR-18</td>
<td>The Ignitibility of Bituminous Coal</td>
<td>June 15, 1960</td>
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<td>SR-19</td>
<td>Changes in Coal Sulfur During Carbonization</td>
<td>August 1, 1960</td>
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<td>SR-20</td>
<td>The Radiation Chemistry of Coal in Various Atmospheres</td>
<td>September 12, 1960</td>
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<td>SR-21</td>
<td>Reaction of Bituminous Coal with Concentrated Sulfuric Acid</td>
<td>October 1, 1960</td>
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<td>SR-23</td>
<td>A Phenomenological Approach to the Batch Grinding of Coals</td>
<td>January 20, 1961</td>
</tr>
<tr>
<td>SR-24</td>
<td>The Unsteady State Diffusion of Gases from Anthracite at High Temperatures</td>
<td>January 21, 1961</td>
</tr>
<tr>
<td>SR-25</td>
<td>Some Advances in X-Ray Diffractometry and Their Application to the Study of Anthracites and Carbons</td>
<td>February 24, 1961</td>
</tr>
<tr>
<td>SR-26</td>
<td>The Filtration of Coal Solutions</td>
<td>March 17, 1961</td>
</tr>
<tr>
<td>SR-27</td>
<td>A Preliminary Investigation into the Application of Coal Petrography in the Blending of Anthracite and Bituminous Coals for the Production of Metallurgical Coke</td>
<td>May 1, 1961</td>
</tr>
<tr>
<td>SR-28</td>
<td>Preparation and Properties of Activated Carbons Prepared from Nitric Acid Treatment of Bituminous Coal</td>
<td>August 15, 1961</td>
</tr>
<tr>
<td>SR-29</td>
<td>The Reactions of Selected Bituminous Coals with Concentrated Sulfuric Acid</td>
<td>August 31, 1961</td>
</tr>
<tr>
<td>SR-31</td>
<td>Mineral Matter Removal from Anthracite by High Temperature Chlorination</td>
<td>March 26, 1962</td>
</tr>
<tr>
<td>SR-32</td>
<td>The Effect of Crusher Type on the Liberation of Sulfur in Bituminous Coal</td>
<td>April 29, 1962</td>
</tr>
<tr>
<td>SR-33</td>
<td>Investigation of the Circular Concentrator - Flotation Circle System for Cleaning Fine Coal</td>
<td>September 10, 1962</td>
</tr>
<tr>
<td>SR-34</td>
<td>Reactions of Coal with Atomic Species</td>
<td>September 24, 1962</td>
</tr>
<tr>
<td>SR-36</td>
<td>A Study of the Burning Velocity of Laminar Coal Dust Flames</td>
<td>November 5, 1962</td>
</tr>
<tr>
<td>SR-37</td>
<td>Molecular Sieve Material From Anthracite</td>
<td>November 16, 1962</td>
</tr>
<tr>
<td>SR-38</td>
<td>Studies of Anthracite Coals at High Pressures and Temperatures</td>
<td>April 29, 1963</td>
</tr>
</tbody>
</table>
SR-39  Coal Flotation of Low-Grade Pennsylvania Anthracite Silts  May 13, 1963
SR-41  Some Aspects of the Chemistry of Sulfur in Relation to Its Presence in Coal  August 20, 1963
SR-42  The Unsteady State Diffusion of Gases from Coals  February 15, 1964
SR-43  The Effect of Concentration and Particle Size on the Burning Velocity of Laminar Coal Dust Flames  March 1, 1964
SR-44  The Electroknetic Behavior of Anthracite Coals and Lithotypes  May 25, 1964
SR-46  The Utilization of Coal Refuse for the Manufacture of Lightweight Aggregate  September 1, 1964
SR-47  A Simulation Model on the Optimal Design of Belt Conveyor Systems  March 5, 1965
SR-48  Beneficiation of Fly Ash  April 12, 1965
SR-49  Application of Linear Programming Methods of Mine Planning and Scheduling  July 10, 1965
SR-50  Petrographic Composition and Sulfur Content of Selected Pennsylvania Bituminous Coal Seams  August 2, 1965
SR-51  Preliminary Investigations of Fog Disposal Methods Applicable to Greater Pittsburgh Airport  August 20, 1965
SR-52  Subsurface Disposal of Acid Mine Water by Injection Wells  August 10, 1965
SR-53  Roof Bolt Load and Differential Sag Measurements  September 3, 1965
SR-54  A Study of the Reactions Between Coal and Coal Mine Drainage  November 22, 1965


SR-56  Computer Simulation of Materials Handling in Open Pit Mining  June 6, 1966

SR-57  The Evaluation of Anthracite Refuse as a Highway Construction Material  July 30, 1966

SR-58  An Investigation of the Cleaning of Bituminous Coal Refuse Fines by an Experimental Hydrocyclone  August 15, 1966

SR-59  Chlorination and Activation of Pennsylvania Anthracites  October 24, 1966

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SR-61  Investigations of the Cyclone Washing of Fine Coal in Water  December 12, 1966

SR-62  Linear Programming Short Course  May 1, 1967

SR-63  Planning Belt Conveyor Networks Using Computer Simulation  May 15, 1967

SR-64  The Economic Importance of the Coal Industry to Pennsylvania  August 1, 1967


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SR-71  The Revegetation of Highly Acid Spoil Banks in the Bituminous Coal Region of Pennsylvania  February 10, 1969
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SR-75  A Complete Coal Mining Simulation  November 10, 1969
SR-76  An Investigation of the Natural Beneficiation of Coal Mine Drainage  May 15, 1970
SR-77  Application of a Continuous Mining System in a Medium Pitching Anthracite Bed of Northeastern Pennsylvania  May 31, 1970
SR-78  Evaluation of a Monorail Mine Haulage System  February 1, 1971
SR-81  Coal Mine Refuse Disposal in Great Britain  March 31, 1971
SR-82  Prevention of Coal Mine Drainage Formation by Well Dewatering  April 15, 1971
SR-83  Pennsylvania Anthracite Refuse: A Literature Survey on Chemical Elements in Coal and Coal Refuse  April 30, 1971
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SR-87  Crushing Anthracite Refuse  July 30, 1971

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SR-91  A Study of the Concrete Block Industry: A National and Regional Approach  May 15, 1972

SR-92  Growth of Tree Seedlings and Use of Amendments on Bituminous Refuse  September 30, 1972

SR-93  Bulk Transport of Anthracite Refuse  January 30, 1973

SR-94  Operation Anthracite Refuse  January 15, 1973

SR-95  Simulation of Quantity and Quality Control in Mining Ventilation  February 25, 1973

SR-96  The Utilization of Incinerated Anthracite Mine Refuse as an Aggregate in Bituminous Mixes for Surfacing Highways  April 1, 1973

William Spackman, Director
Coal Research Section
Office of Coal Research Administration