A mining research contract report
DECEMBER 1983

COAL INJECTOR FOR COARSE SLURRY TRANSPORT

Bureau of Mines Open File Report 190-84

Contract JO333914

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BUREAU OF MINES
UNITED STATES DEPARTMENT OF THE INTERIOR

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50272-101

REPORT DOCUMENTATION PAGE	1. REPORT NO. BuMines OFR 190-84	2.	3. Recipient's Accession No. PB85 149441/AS

4. Title and Subtitle	5. Report Date
Coal Injector for Coarse Slurry Transport	Dec. 1983
	6.
7. Author(s) Mackenzie Burnett, Leslie S. Rubin, Andrew Harvey, John C. Roberge, Robert L. Cardenas, and Hans Hug	8. Performing Organization Rept. No.
9. Performing Organization Name and Address Foster-Miller, Inc. 350 2d Ave. Waltham, MA 02254	10. Project/Task/Work Unit No. 11. Contract(C) or Grant(G) No. (C) J0333914 (G)
12. Sponsoring Organization Name and Address Office of Assistant Director--Mining Research Bureau of Mines U.S. Department of the Interior Washington, DC 20241	13. Type of Report & Period Covered Contract research, 7/1/75--4/30/83 14.

15. Supplementary Notes

Approved for release October 1, 1984.

16. Abstract (Limit: 200 words)

An inducer-type pump, capable of injecting dry coal into a hydraulic transport pipeline was developed and installed in a specially designed and fabricated slurry injector vehicle. The full-scale prototype injector system can deliver up to 11 ton/min of coal into an 8-in pipeline at pressures up to 95 lb/in^2. The helical inducer injector operates on centrifugal principles with flow passing axially through a helical-pitched conical rotor. Water is delivered to the pump through an annular slot at the atmospheric pressure coal inlet. The spinning rotor maintains an air-core vortex of water that pressurizes the discharge housing. A screw feeds coal through the open inlet into the vortex. The inner radius of the vortex contracts as pressure is increased, giving a limiting maximum pressure that is proportional to the square of the rotor speed. Below this limit, flow rate, discharge pressure, and rotor speed are independent variables. The self-powered, 42-in-high slurry injector vehicle follows and receives run-of-mine coal from a continuous miner, reduces the top size to 3-in, and delivers the sized coal to the injector feed screw for slurry formation and pumps it through a mobile hose attached to the vehicle's outby end.

17. Document Analysis a. Descriptors

Mining research
- Coarse slurry transport
- Hydraulic coal transport
- Ventilated helical injector
- Slurry injector vehicle

b. Identifiers/Open-Ended Terms

c. COSATI Field/Group 08I

18. Availability Statement	19. Security Class (This Report)	21. No. of Pages
Release unlimited by NTIS.	Unclassified	262
	20. Security Class (This Page) Unclassified	22. Price

(See ANSI-Z39.18) See Instructions on Reverse OPTIONAL FORM 272 (4-77) (Formerly NTIS-35) Department of Commerce

FOREWORD

This report was prepared by Foster-Miller, Inc., (FMI), of Waltham, MA, under Bureau of Mines Contract No. J0333914. It was administered under the technical direction of the Pittsburgh Research Center. The Technical Project Officer was Mr. Anthony J. Miscoe. Mr. Joseph A. Gilchrist was the Contracting Officer. This report is the result of work carried out during the period from July 1975 to April 1983.

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EXECUTIVE SUMMARY

Continuous hydraulic haulage of run-of-mine (ROM) from the mine face to the coal preparation or dewatering plant has been determined to be one of the most promising techniques for increasing productivity and reducing haulage hazards associated with conveyor belts, shuttle cars and rail systems.

For hydraulic transport to be used in thin seams, it is necessary to consider an injector other than a conventional centrifugal slurry pump which is too large to be used on a machine only 42-in in height. The low machine requirement also makes slurry mixing prior to injection impractical because of the level control problem associated with shallow hoppers and the height required for gravity feed.

In addition, the hydraulic haulage system must be capable of handling the random flows and variable flow rates which result from the cyclic operation of continuous miners. A typical solution to this problem employs variable speed motor drives on the centrifugal pumps. These variable speed drives are controlled by an expensive and sophisticated, computer-based feedback-logic system.

In an effort to eliminate the need for controls and because thin seams present substantial new problems for face haulage of coarse coal, the Bureau contracted FMI to design and devleop an injector mounted on a vehicle which accepts dry coal and produces a pressurized slurry for pipeline transport out of the mine.

Program Objective

The primary objective of this program was to conceive, design, fabricate and test a pumping device/vehicle which can receive ROM coal from primary underground mining machinery in a 4-ft seam and inject it into a hydraulic transport line. Ideally, this coarse coal injector for initiating coarse slurry transport shall receive dry coal and water directly and independently, without pre-slurry mixing tanks. Detailed objectives relative to the design and performance of the injector and injector vehicle include:

Major Injector Specifications

a. Work in coal seams as thin as 4-ft

b. Inject dry coal into a hydrotransport pipeline at pressures in excess of 80 lb/in^2 and at coal flow rates of 0.4 tons/min or greater

c. Handle a coal top size which is one-third the transport line diameter

d. Feed coal into the slurry transport system at concentrations of up to 50% by volume

e. Process coal with 30% refuse by weight in normal operation and 100% refuse at half rate for cleanup work

f. No water leakage or air ingestion

g. Design applicable to line sizes from 6- to 18-in in diameter

h. Handle variable coal flow rates without extraneous controls

Major Vehicle Specifications

a. Size limitation 42 in, 27 ft and 9 ft (height, length and width)

b. Have sufficient mobility to follow and receive coal from a continuous miner - Tram speed of 70 ft/min

c. 20,000 lb drawbar pull

d. Receive ROM coal and reduce top size to one-third the pipeline diameter

e. Fail safe operation - no water leakage

f. One ton ROM storage

g. Permissible design.

Coarse Coal Injector Description

The coarse coal helical injector is an adaption of a flow inducer that is typically used when net positive suction head (NPSH) is very low, particularly if the flow contains a major fraction of air or vapor. Figure 1 represents a cross sectional view of the coarse coal injector assembly depicting its four major components: housing, rotor, water inlet ring and solids feed screw. The outer housing is formed from two sections - one cylindrical and the other an open ended cone. The internal rotor is driven by a shaft through the back end of the pump. This rotor is constructed from a helical

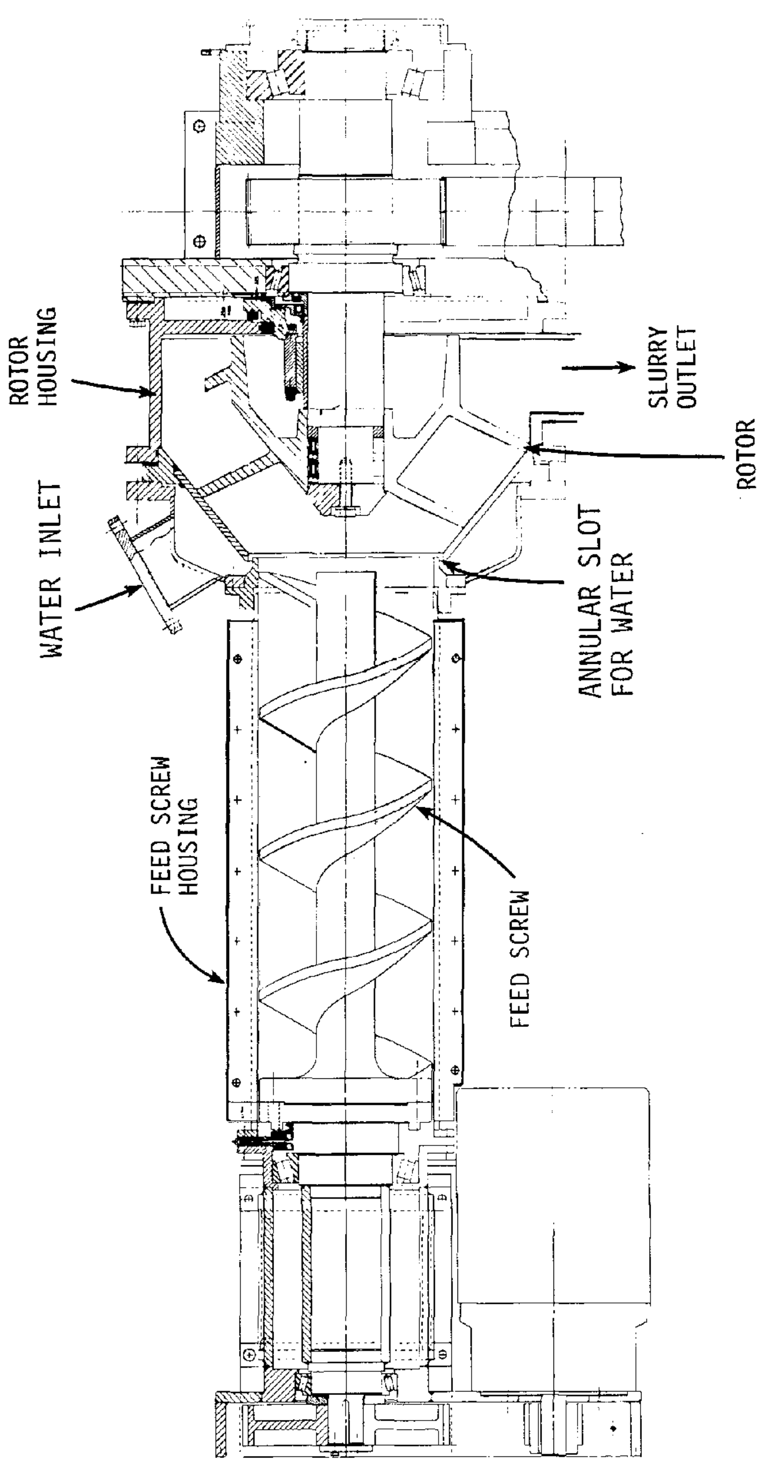

FIGURE 1. - Ventilated coarse coal injector assembly.

flight attached to a truncated conical shaped hub. The contour of both the rotor blade tip and rotor hub increases in diameter from pump inlet to outlet. Water is delivered through an annular slot around the atmospheric pressure solids inlet at the front of the rotor. The turning rotor accelerates the incoming water forming a centrifugal pool of rotating liquid with an inner air core. (This rotating liquid ring pressurizes the discharge housing.) Dry solids are directed by means of the feed screw into the ventilated inlet where they are picked up by the rotor blade, flung outward and submerged in the water vortex. Fluid level inside the inducer adjusts automatically in response to the required discharge pressure. As a higher discharge pressure is required, due to increased flow or solids concentration, the inner radius of the air vortex shrinks. As discharge pressure requirements decrease, the air core expands. For a constant speed, the injector's discharge pressure will primarily vary with the product of the slurry's density and the difference between the square of the revolving liquid ring's outer radius (radius of cylindrical housing) and the square of the inside radius to the liquid ring's free surface. The maximum discharge pressure occurs when the air core of the vortex has contracted to the diameter of the atmospheric solids inlet. Thus, the coarse coal injector has two operational limits - one due to inlet flooding caused by high back pressure and one due to outlet aeration at low back pressures. The rotor geometry (passage size, blade lead) and operating speed provide a pump flow capacity significantly greater than the total coal and water flow metered to the inducer.

In a centrifugal pump, inlet flow is unrestricted and is determined by the particular pump characteristic curve, a relationship between back pressure, impeller speed and diameter. With the coarse coal injector, flow and back pressure are each independent variables determined by the system and *not* the inducer. Figure 2 shows the system performance curves for water and maximum solids concentration superimposed on the coarse coal injector performance (shaded area). Any point on the shaded area is a possible head-flow condition for constant speed injector operation. When introduced into a hydraulic transport system, the injector follows the system performance curve. If water is metered into the system at a constant rate and only solids flow is varied, then performance will fall along line (a-b). Such operation is not possible with constant speed centrifugal pumps. (Centrifugal pumps require variable speed drives and feedback logic and controls to handle varying flows.)

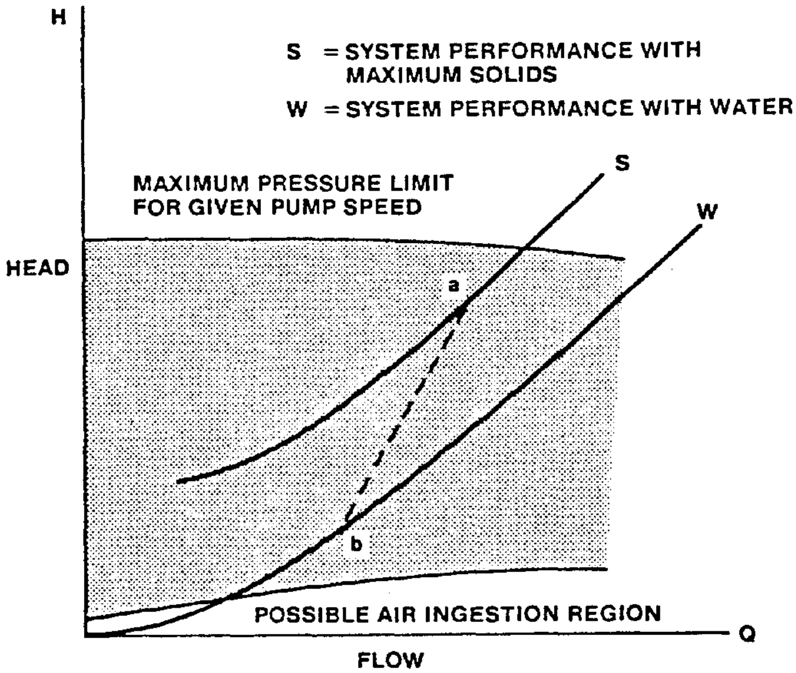

FIGURE 2. - Typical coarse coal injector performance and system head-flow curves.

Mechanically, the helical rotor injector is particularly suited for handling coarse solids in a restricted space. The axial flow permits the discharge housing inside diameter to be almost as small as the rotor outside diameter. This is not possible in a radial flow design which requires a scroll casing diameter much larger than the rotor diameter.

Solids enter the rotor inlet at a very low axial velocity and swirl compared to the peripheral velocity of the rotor inlet at its maximum diameter. The rotor blade tip speed at the inlet is approximately 60 ft/s. This is comparable to current slurry pump operating practice. The rotor blade and passage design is very different from that of centrifugal slurry pumps. It is similar to conical inducer pumps previously referred to for low NPSH operation, but significant modifications have been made in order to minimize the effects of solids impact.

The helical pitch of the rotor gives a small ratio of axial flow velocity to rotor speed, so that over the entire range of flow rates, the solids have initial trajectories that hit the blade pressure surface at a shallow glancing angle. Axial impact velocity is consequently small; and thus, wear of the blade surface is minimized. The sickle-shaped blade reduces the effect of the high relative velocity of solids to the blade leading edge. Impact near the tip is with an edge angled approximately 60° from the relative velocity; and thus, impact velocity is half of that for the normal impact in a typical pump.

Injector Testing

Development of the coarse coal injector resulted in the construction of two differently sized units. The first injector was a scale model unit designed for operation with a 3-in-diam pipeline. The second injector was a full scale prototype designed for operation with an 8- to 10-in-diam pipeline. Rotor diameters were, respectively, 10.5 in and 31.5 in. Coal inlet diameters were, respectively, 5 in and 15 in.

Testing of both units were conducted at slurry facilities specially constructed for injector testing. Figures 3 and 4 conceptually illustrate the model scale and full scale test facilities, respectively. The small scale slurry facility was a batch operated system. The full scale facility was designed for continuous operation.

Testing of both injectors demonstrated their ability to receive dry coal and produce a slurry of up to 50% coal by volume. Discharge pressures of up to 95 lb/in^2 could be produced by both pumps when operating at rotor tip speeds of 137 ft/s. A summary of both injector's process performance is presented in Table 1. More importantly, both injectors could operate at any combination of flow and discharge pressure[1] proving their ability to self-regulate to any downstream requirement without motor speed control and computer feedback logic.

The one major problem which still exists is a rotor cone wear problem. Initial tests with mild steel and later tests with hard facings and exotic coatings did not provide acceptable parts life. The very nature of the relative movement between rotor tip and cone is analogous to a cone crusher. If a theoretical zero clearance between rotor tip and cone could be maintained, rotor tip wear could be prevented. Since this is not possible, a rotor and cone were fabricated from HC-250, a state-of-the-art material for wear resistance. Testing/evaluation of this material was not completed due to the exhaustion of project funds.

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[1] Below or at its maximum pressure of 95 lb/in^2.

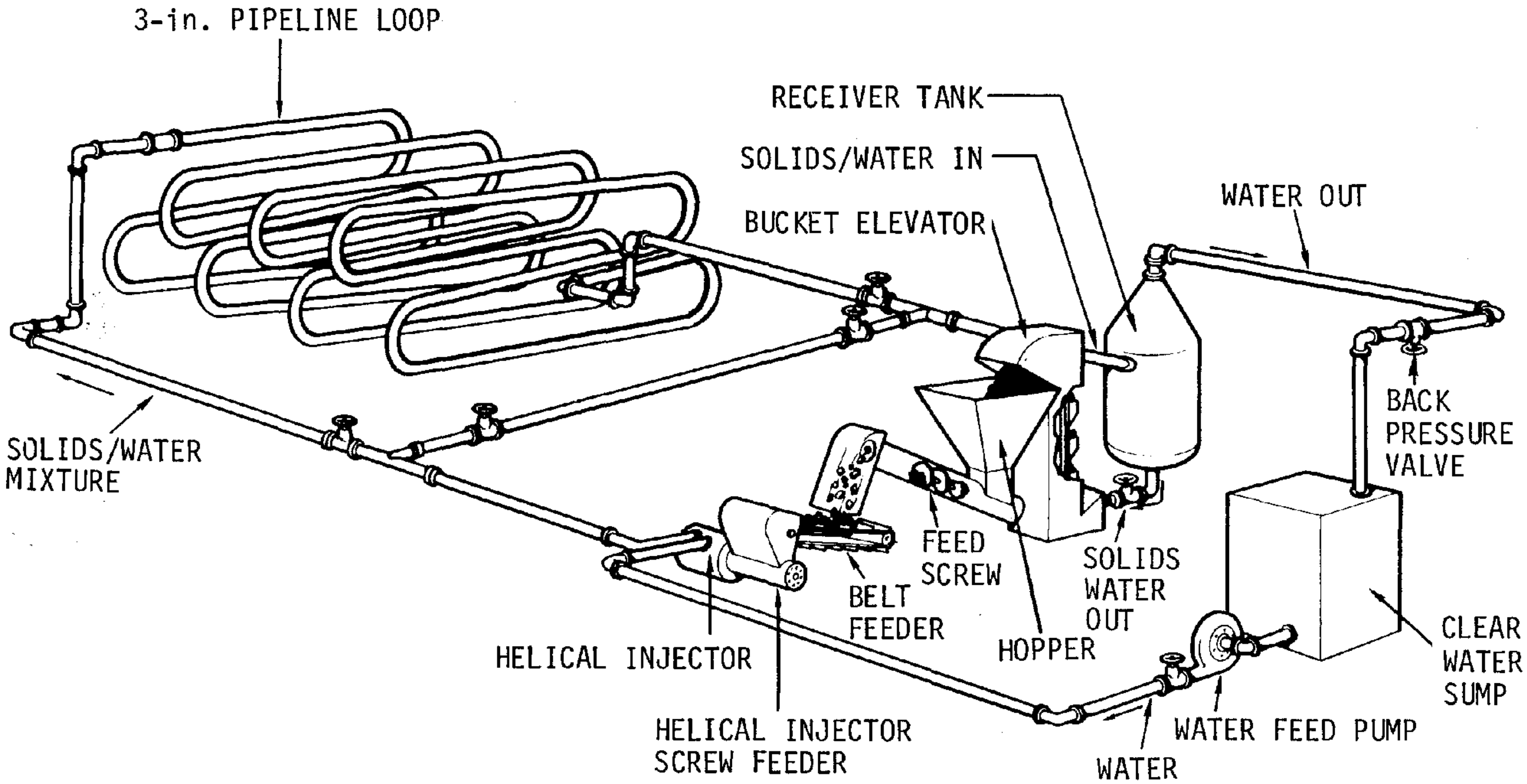

FIGURE 3. - Model scale slurry test facility.

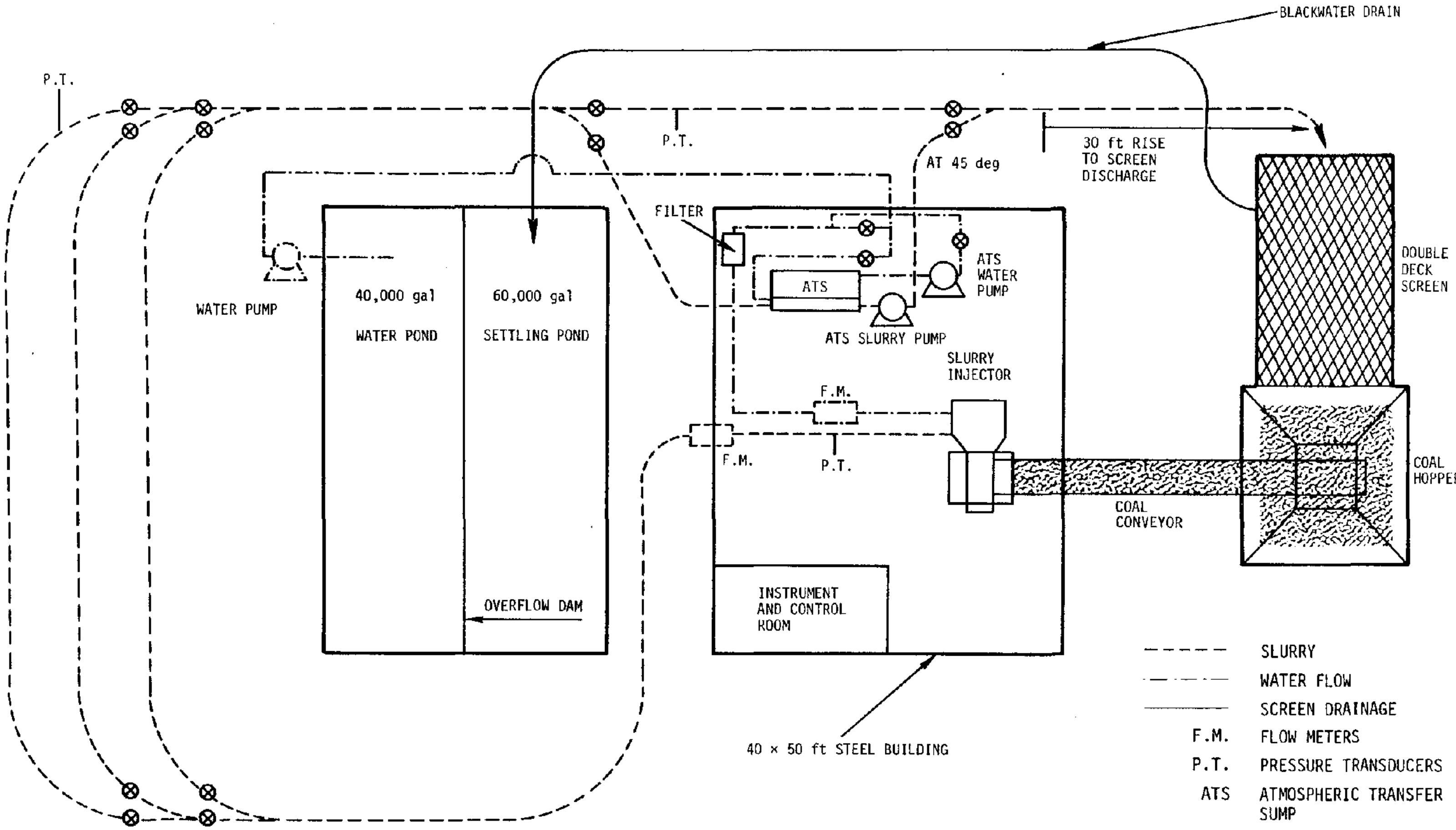

FIGURE 4. - Slurry test facility conceptual layout.

TABLE 1. - Injector performance

Parameter	Model scale	Prototype
Coal injection rate	2000 lb/min	11.4 tons/min
Maximum discharge pressure (lb/in^2)	94	95
High end horsepower (hp)	41	375
Rotor speed (rpm)	3000	1000
Rotor tip speed (ft/s)	137	137
Maximum lump size	1 in	3.5 in
Maximum slurry concentration (volume)	50%	48%

Mobile Slurry Injector Vehicle

The slurry injector vehicle developed during this program originated from a Jeffrey hardrock crawler loader stripped to an empty vehicle chassis with track assembly and associated planetary transmission. The complete vehicle is pictured in figures 5 and 6, the inby end and discharge end, respectively.

In operation, ROM coal from the continuous miner is delivered to the vehicle's inby end (fig. 5) where it is transported by chain conveyor to the scalper assembly for size sorting. Minus 3-in particles pass through the scalper while plus 3-in solids are delivered to a hooked tooth double roll crusher. Crushed and scalped solids are then transported by chain conveyor to the coarse coal injector feed screw (fig. 6). The separately delivered water and solids are combined into a slurry within the injector pump and discharged at pressures up to 95 lb/in^2. The valving arrangement shown in figure 6 provides the capability for bypassing the injector and maintaining slurry flow through the hydrotransport system in the event of injector failure. Figure 7 shows a side view of the injector with an operator at the vehicle control station.

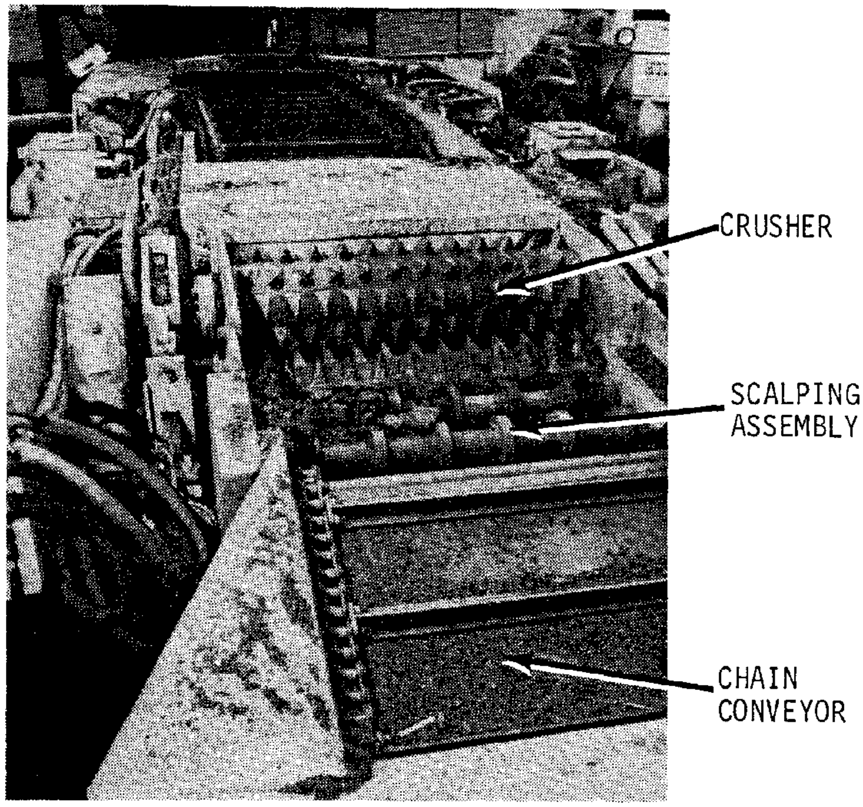

FIGURE 5. - Injector vehicle (inby end).

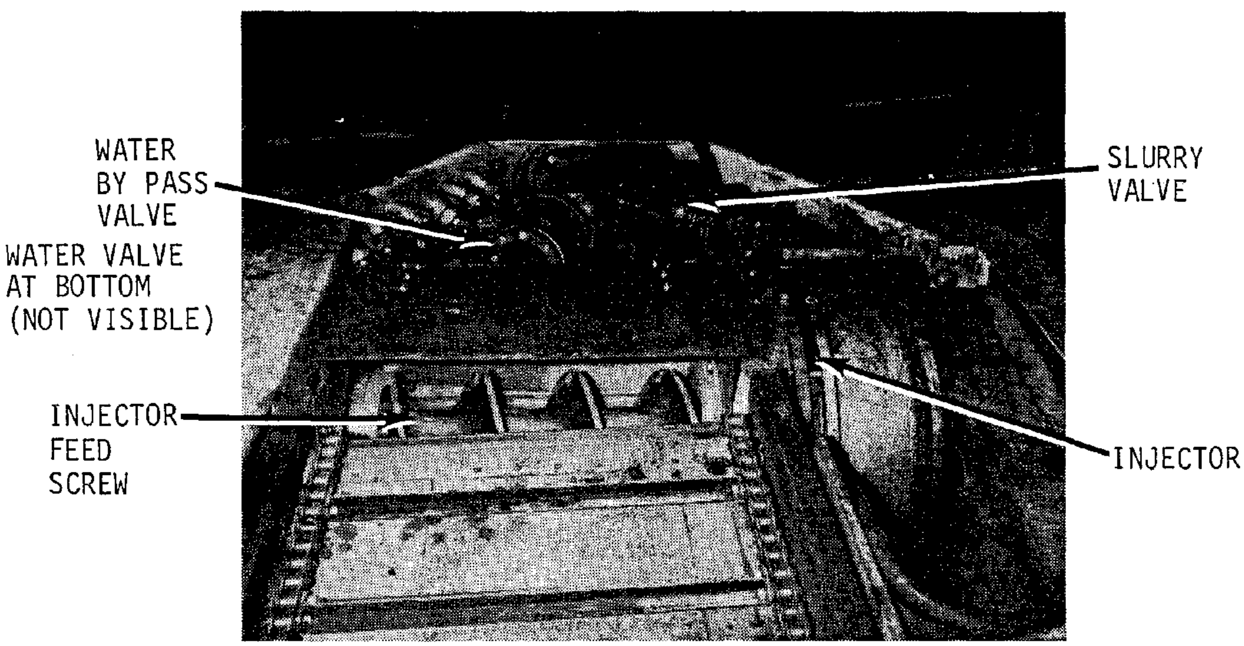

FIGURE 6. - Injector vehicle (discharge end).

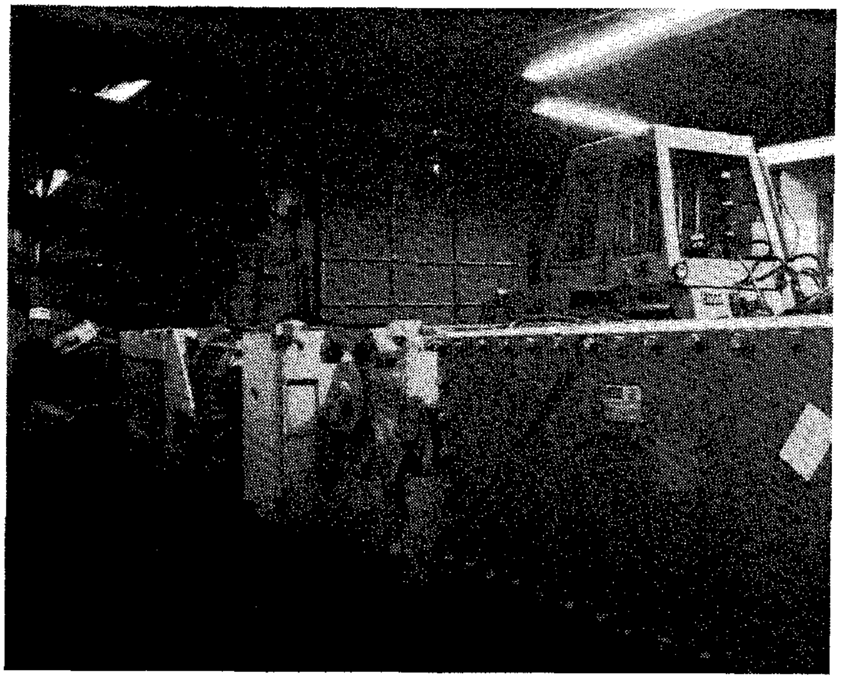

FIGURE 7. - Injector vehicle (side view).

Conclusions

An injector pumping device/vehicle suitable for use in a 4-ft seam and sized for an 8- to 10-in diam pipeline was designed, developed and tested by FMI in accordance with the contracted Statement of Work. Results of the Bureau's sponsored program relative to the prototype helical injector's and vehicle's performance can be highlighted as follows:

a. Coal[1] handling capability in excess of 11 tons/min at concentrations up to 50% by volume

b. Maximum injector discharge pressure is proportional to pump speed squared. At 1000 rpm, the prototype injector developed a maximum of 98 lb/in^2

c. Pump efficiency at normal operating levels is in the 45 to 60% range

d. Capable of handling any combination of flow and pressure below a discharge pressure of 98 lb/in^2

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[1]Anthracite at 1.7 SG.

e. Totally self-regulating to changes in volumetric flow and discharge pressure requirements

f. Some injector air ingestion was experienced at low discharge pressures. This parameter was not quantified

g. Acceptable parts life for the rotor and cone has not been demonstrated to date

h. All vehicle components except the roll tooth crusher, chain conveyor, minor hydraulic circuitry, and slurry valve performed satisfactorily. The non-enclosed crusher's speed was too high causing coal spit-back. All other component failures were due to improper installation of flaws or fabrication.

Recommendations

a. Endurance test the injector with the HC-250 cone and rotor to determine wear characteristics

b. Reduce speed of crusher or provide an enclosure to prevent coal spit-back.

c. Quantify air ingestion-discharge pressure relationship and evaluate effects of three phase flow

d. Evaluate a shrouded rotor design to determine hydraulic and mechanical characteristics.

1. INTRODUCTION

Mining of coal in modern underground mines is a highly mechanized operation. Most coal is cut by large continuous miners. These machines have large rotating cutting drums with multiple tungsten carbide bits which are capable of cutting coal at peak rates exceeding 150 kg/s (10 ton/min).

The miner gathers coal as it is cut and transports the coal from the mine face on a short, integral conveyor. This conveyor enables the miner to load the coal directly into a shuttle car. Space limitations allow only one shuttle car to be in the loading area at a time. This limits the rate at which coal can be hauled away from the mine face and interrupts the continuous miner cutting operations. When the shuttle car is filled, it carries the coal to the tail piece of a conveyor belt, which is the first stage of a conveyor system to transport the coal out of the mine, or it delivers the coal to a rail haulage system. The batch nature of the shuttle car system limits the duty cycle of the continuous miner. Accidents associated with the haulage equipment cause a significant percentage of coal mine fatalities (17% for 1960-65 in the United States) (1).

Haulage systems having a minimum number of transfer points will tend to be safer and less likely to malfunction. The use of a continuous rather than a batch type system will lead to high utilization of the mining equipment and resultant improvements in the productivity of mining operations.

Continuous hydraulic haulage from the mine face to the coal preparation plant has been determined to be a feasible technique for increasing productivity and reducing haulage hazards (2).

For hydraulic transport to be used in low coal seams, it is necessary to consider an injector other than a conventional slurry pump which is too large to be used on a machine only 42-in in height. The low machine requirement also makes slurry mixing prior to injection impractical because of the level control problem associated with shallow hoppers and the height required for qravity feed.

In addition, for ROM hydraulic haulage to be competitive with other means of transport such as conveyor belts, rail cars or trucks, it must be capable of handling random flows and variable flow rates. One particular example is haulage of coal and rock in underground mining where variable solids flow rate is the normal mode of operation.

Hydraulic transport of solids under continually varying rates, concentration and density can present a monumental control problem for a system using conventional state-of-the-art pumps. Hydraulic transport lines several thousand meters long with centrifugal pumps appropriately located along the line are subject to severe water hammer and pump cavitation if the input conditions (solids concentration, solids density and solids or fluid flow rate) change. Cavitation and water hammer can occur when the system contains some sections with high concentrations of solids and others with little or no solids. These sections of fluid tend to move at different velocities due to inertia effects causing separation of the column and potential pump failure. In addition, transient effects can be particularly disastrous if the pump at the input point ventilates or cavitates due to inadequate control of sump conditions.

There are many control schemes which can or have been devised to reduce these problems. Variable speed drives on the pumps and/or intermediate sumps at each pump location have been used. Both, however, require elaborate control systems to insure that the line dynamics remain within the systems' capability. A typical head-flow curve for a simple system is shown in figure 8. A control system can be devised to provide constant head or constant

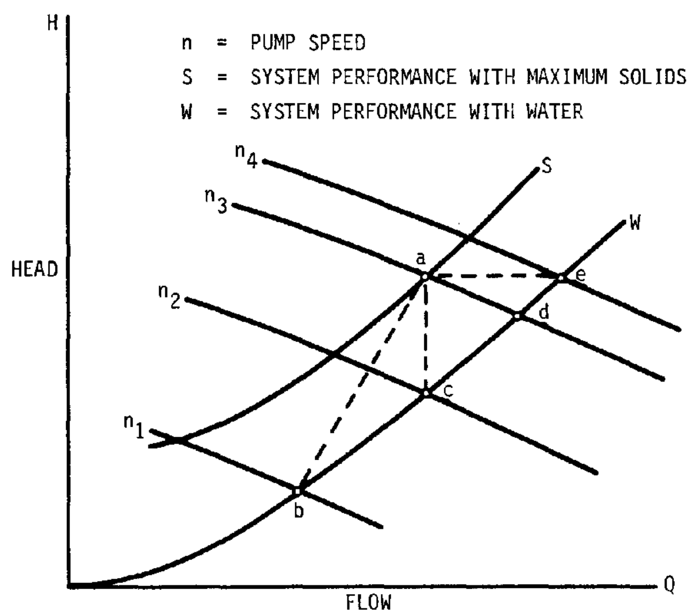


FIGURE 8. - Typical centrifugal pump and system head-flow curves.

flow by varying pump speed. Point (a) represents H-Q for maximum solids flow and points (b), (c), (d) and (e) are potential operating points for water-only conditions. If constant pump speed is desired, then the corresponding H-Q at water-only flow is represented by point (d). Point (b) represents the constant water flow, the solids being the only flow variable. Constant Q and thus constant line velocity is approached by operating from points (a) to (c). Point (e) represents a constant head operation. In each case except point (b), water flow must be varied for proper system operation. Also, in each case except point (d), the pump speed must be varied. In long systems with several boost pumps, it may be difficult if not impossible to operate a reliable system without variable speed pumps and/or variable water flow to the system (3).

In an effort to eliminate these control problems and because thin seams present substantial new problems for face haulage of coarse coal, the Bureau contracted Foster-Miller to design and develop a mobile injector which accepts dry coal and produces a pressurized slurry for pipeline transport out of the mine.

The program for developing a coarse coal injector/ vehicle for thin seam slurry transport which is self-regulating for changing solids flow rates was divided into three phases.

Phase I - Concept Development

Phase II - Scale-Model Injector Development and Testing

Phase III - Prototype Injector/Vehicle Development and Testing.

Details of Phase I and Phase II results have been documented in separate reports dated December 1975 and June 1977, respectively. A summary of major Phase I and Phase II highlights and details of the Phase III program results are presented in this final report.

REFERENCES

1. Kleysteuber, W.K. and Miscoe, A.J., "Unusual Concepts for Coal Mine Haulage," Proc. 4th and 5th Annual Seminars on Bulk Materials Handling, University of Pittsburgh, (December 1971-1972).

2. Link, J.M., Allan, A. and Faddick, R.R., "Feasibility of Hydraulic Transportation in Underground Coal Mines," Colorado School of Mines Research Institute, USBM Contract Report HO133037.

3. Gregory, F.W., "Hydrotransport of Solids Underground," Mining Technology Clearing House, December 1977.

2. PHASE I SUMMARY - INJECTOR CONCEPT DEVELOPMENT

2.1 INJECTOR SPECIFICATIONS

The original coarse coal slurry injector contract provided a detailed set of design criteria for development of a coal injector system. For the most part, the FMI conceptual design effort had been directed towards meeting the criteria presented in the original contract. In some cases, preliminary investigations indicated that deviations from these criteria might be appropriate. The original design criteria and recommended variations are described in the following two subsections.

2.1.1 Original Contract Design Criteria

2.1.1.1 Application

The solids injector shall be designed to accept coal from primary mining machinery in the three principal coal mining systems - continuous, conventional and longwall.

2.1.1.2 Overall Dimensions

The solids injector shall have the following maximum dimensions: Length, 27 ft; width, 9 ft; and height, for operation in a 4-ft coal seam. Smaller dimensions are highly desirable.

2.1.1.3 Mobility

The injector may be either self-powered for locomotion, or it may be towable by the primary machinery.

The solids injector shall be sufficiently mobile to follow all operating maneuvers of the primary machinery of the systems listed under section 2.1.1.1. Mobility is here defined as tramming speed (if powered), turning radius and steering response speed.

The injector shall be able to travel on difficult terrain - clay floors in coal mines. Clay floors may be hard-packed or soft and loose, either wet or dry, and undulating in short or long waves.

2.1.1.4 Capacities

The injector shall be able to absorb the discharge of the primary mining machinery and inject it into the pipeline. This may run as high as 12 tons/min during short periods (up to 3 min in a 4-ft seam) for continuous miners.

The injector shall accept coal in sizes generated by the primary mining machinery. No more than 10% of the coal to be hydraulically transported shall be as large as one-third of the pipeline diameter; therefore, oversize must be controlled by crushing or breaking.

The injector shall be capable of handling coal having up to 30% refuse in normal operation and 100% refuse (at half-rate) for clean-up work.

Protection must be provided to prevent ingestion of metal scrap (such as roof bolts and cutting bits) and other trash such as cribbing, curtain scraps, clothing or personal safety equipment.

The injector shall be designed for 6- to 18-in standard pipe. If design work indicates that more than one model is necessary, the conclusion must be justified. Sizes for future testing shall be for 6-in pipe as a small-scale prototype and 10- or 12-in pipe for a full-scale prototype.

Provision shall be made for clearing a jammed injector without affecting the pipeline flow and with minimum downtime of the injector. For example, hatchways could be installed at critical points in the solids path.

The injector shall be designed for use with standard Schedule 40 pipe and pipe fittings and within all standard codes governing working pressures, temperatures and dimensional tolerances for such pipe. Specifications for design changes required for pressure ratings for 250, 500, and 1,000 psig shall be provided along with the standard design. If more than one model is necessary, the conclusions must be justified.

2.1.1.5 Controls

Easily identifiable, simple controls and read-out devices (where necessary) shall be specified for the operator. These shall include the injection controls for the coal as well as power-assist devices and locomotion controls if required. Programmable or automatic systems shall be specified separately since initial interest is in manual control. They are, however, desirable for future application.

The injector shall control the rate of injection of coal. This shall be controllable for a range of coal concentrations from zero to 50% by volume in the pipeline. Response time shall be appropriately fast to prevent either plugging or inefficient low concentrations.

Controls shall be specified for coordinating multiple injectors operating on the same haulage pipeline.

2.1.1.6 Service and Economy Requirements

The injector shall be designed to meet the same service specifications of all underground coal mining machinery. All components shall be heavy-duty with minimum maintenance requirements.

Delicate or precision-adjustment mechanisms shall not be specified.

Reliability (in the technical sense) shall be equal to that of continuous miners or face haulage equipment.

Wearing parts in contact with the coal shall be abrasion resistant.

Provision shall be made for repairs or preventive maintenance in the underground environment. For example, lubrication points shall be easily accessible, the use of large or heavy parts shall be minimized, and it shall be possible to change parts with minimum dismantling of the machine.

Standard available parts and service equipment shall be used wherever possible to minimize mine parts inventory. For example, mining machinery oils, greases, bearings, fastenings, etc., shall be specified.

Lowest possible capital and operating costs shall be the primary goal. They can be extended to operator and mechanic skill requirements. It would be useless to the Government to provide a machine that nobody can afford to buy.

2.1.1.7 Safety

The injector shall meet the requirements of all applicable Federal legislation, especially permissibility.

The injector shall fail in a safe manner. That is, upon failure, it should not release a flood of water, parts should not shatter explosively, nor should it bury itself, and fill the entry with coal, or any other material.

The injector shall not create any new or additional hazard in coal mining.

2.1.2 Contract Design Criteria - Further Considerations

Task 3 of Phase I, Refine Design Criteria, involved evaluating and refining the specifications presented in the contract. In this effort, FMI consulted the Bureau personnel, CONOCO personnel, our consultant, F.L. Smith, and made extensive use of the recently completed Colorado School of Mines Research Institute (CSMRI) Study, Feasibility of Hydraulic Transportation in Underground Coal Mines (2). The impact of this effort on the basic system specification is discussed below.

2.1.2.1 Application

There was general agreement between FMI and the Bureau personnel that the design of a solids injector for a continuous coal mining system represented the most difficult injector design problem. It was agreed that an injector which is compatible with a continuous mining system would be readily adaptable to conventional and longwall systems. Therefore, the program effort was directed to the design of an injector system for application to a continuous mining system.

2.1.2.2 Capacity

In specifying the capacity of an injector system, a tradeoff is required between the continuous injection rate of the device and the capacity of the machine to absorb high rate surges. A study of probable miner output and a tradeoff study of injection rate versus surge capacity established a minimum capacity specification of a 6.4 tons/min injection rate with a 1 ton surge capacity. This specification was followed for the injector designs presented in section 2.2. However, it should be noted that both of the recommended injector designs could be upgraded to a 12 tons/min injection rate based upon centrifugal pump scaling laws or by proportionally increasing the rotor speed.

2.1.2.3 Injector Input Sizing

The contract specifications require that "no more than 10% of the coal to be hydraulically transported shall be as large as one-third of the pipeline diameter." Minimum pipeline size is specified as 8 in. Combining these requirements, and considering the desire for a universal injector system, a specification of 3 in as the maximum dimension of coal injected into the pipeline has been established.

Therefore, the injector system must be capable of accepting ROM coal, but the maximum coal dimension must be reduced to 3 in by the time the coal reaches the pipeline. Based on an analysis of coal size distribution from continuous miners presented in the CSMRI report (see table 2), the percentage of ROM coal which will have to be reduced in size to meet this specification will be lower than the specified 8.7 to 22% range.

2.1.2.4 Injector Input Sorting

The FMI design concepts provide for sorting magnetic scrap, but sorting of nonmagnetic scrap (cribbing, curtain scraps, clothing, etc.) was determined to be technically infeasible within system space and economic constraints.

2.1.2.5 Injector Input Pressure

The required input pressure for a mine face injector system was determined by consulting the Bureau personnel and CONOCO Research and Development personnel, and by studying the CSMRI report. A useful operating pressure range was established at 75 to 125 psig.

TABLE 2. - Typical coal size distributions from continuous miners

Coal size	A	B	C	D	E
Plus 2 in	14.4	9.9	8.7	11.7	22.0
2 by 1-1/4 in	7.8	8.4	9.8	8.1	22.0
1-1/4 by 3/4 in	12.7	12.5	17.6	12.1	22.0
3/4 by 1/4 in	29.8	22.6	35.7	30.0	20.0
1/4 in by 20 m	29.8	38.5	21.8	28.8	28.5
28 m by 0	5.5	8.1	6.4	9.3	7.5

2.2 INJECTOR CONCEPTS

Injector concepts considered during Phase I were divided into two categories: units that accept only pre-slurried or submerged coal, and units that can accept dry or surface wet coal. Within these two feed categories (wet and dry), the injector pump can be classified further according to its pressuring mechanism. The three general categories explored during Phase I include:

a. Inertial pressurization, e.g., centrifugal and jet pumps

b. Volumetric pressurization, e.g., piston and gear pumps

c. Volumetric transfer, e.g., pocket feeders and lock hoppers.

Several potentially attractive concepts were considered in each functional category. Concepts given serious consideration included:

a. Helical injector
b. Sealed flight screw injector
c. Centrifugal solids injector
d. Progressive cavity injector
e. Jet pump
f. Pocket feeder
g. Floating piston pocket feeder.

A qualitative comparison of each concept based on the functional and dimensional requirements of a face haulage injector is summarized in table 3. All concepts had some merit. All but two, the helical injector and sealed flight screw injector, were rejected for reasons such as size, pressure capability or overall complexity.

2.3 ANALYSIS/TECHNICAL DISCUSSION OF CANDIDATE INJECTORS

2.3.1 Coarse Coal Helical Injector

2.3.1.1 General Description

The helical injector is a device that operates on centrifugal principles, but is designed to accept dry solids in the inlet while maintaining a pressure against water at the discharge. Solids that are fed at atmospheric pressure without water to the inlet pass through an air-water interface in the rotor passages and are discharged to a high pressure water vortex in the rotor housing. The rotor provides the energy to inject

TABLE 3. - Comparison and evaluation of various injector concepts

Injectors	Helical injector 2-blade	Sealed flight screw	Centrifugal Solids	Progressive Cavity (Moyno)	Jet pump	Jet driven vortex	Pocket Feeder	Floating piston pocket feeder
Design coal injection rate (tons/min)	6.4	8	6.4	5.8 (3 units)	6.4	6.4	6.4	6.4
Design injection pressure (lbs/in.^{2})	80	100	80	100	15	80	100	100
Estimated power (BHP)	250	200	300	225	140	400	280	280
Injector size h × w × l (feet)	2.5 × 3.5 × 6	2 × 4 × 10	3 × 7 × 11	1 × 4 × 10 (3 units)	2 × 2 × 6 Not including pressure pump	2.5 × 2.5 × 3	Not designed	3'4" × 9 × 18
Maximum injection size coal top size (in.)	2	3.5	2	2	2	Limited only by slurry line size	2	2
Coal degradation	Substantially less than slurry pumps	Minimal	No worse than slurry pumps	Minimal	Minimal	Minimal	Minimal	Minimal
Coal condition at inlet and method of feeding	Dry-screw conveyor	Dry-self fed by extension of screw	Dry-flat belt	Slurry 25% by volume screw conveyor	Slurried inlet	Dry	Dry	Dry-Chain Conveyor
Air purge	Centrifugal separation	Air leaks back thru screw - replaced by water leakage	Centrifugal separation	Coal premixed with water at inlet	Coal premixed with water at inlet	Centrifugal separation	Displaced by water in pocket	Displaced by water in cylinder
Water leaks during operation	None	Negligible	None	None	None	None	Minimal	Minimal
Water leakage with power off without a check valve	Full line flow - approx. 1000 gpm if mine system designed for 50% slurry and 6.4 ton/min	Approximately 540 gpm - estimate based on screw/wall clearance	Same as mixed flow	Negligible	Full slurry line back flow	Full slurry line back flow	None	None
Pressure flow characteristics	Positive displacement until inlet floods at 100 psi for 2 blade rotor	Positive displacement	Positive displacement until inlet floods at 100 psi	Positive displacement	Flow drops with pressure increase	Flow drops with pressure increase	Positive displacement	Positive displacement
Ingestion of refuse and tramp iron (assuming the condition of trash separator failure)	Passes material or jams at inlet-blade damage or internal jam possible	Passes material or stalls - little damage can be cleared without teardown by reversing screw	Same as mixed flow	Passes material or stalls-little damage-tear down to clear jam	Passes material or plugs - no damage	Passes material or plugs no damage	Passes material or stalls-damage to seal surface possible	Passes material or stalls-damage to seal surfaces
Starting with load of coal in unit	Possible	Possible	Possible	Will jam	Possible	Possible	Possible	Possible
Maximum slurry concentration	Only limit is mine system design	Only limit is mine system design	Only limit is mine system design	25% by volume	25% by volume	25% by volume	50% by volume	50% by volume

TABLE 3. - Comparison and evaluation of various injector concepts--Continued

Injectors	Helical injector two-blade	Sealed flight screw	Centrifugal solids	Progressive cavity (Moyno)™	Jet pump	Jet driven vortex	Pocket feeder	Floating piston pocket feeder
State-of-the-Art	Is state-of-art for fluid pumping	Combination of known machine elements.	Is state-of-art for fine slurry pumping.	Is state-of-art for fine slurry pumping.	Is state-of-art for dredging gravel.	No	Process known	Combination of known machine elements.
Ease of Maintenance in Mine Entry	Fair-access on both sides - inlet must be removed to reach rotor.	Good - all major parts accessible from end of machine.	Poor - difficult removal of injector in mine	Good - easy access to all components.	Good to excellent	Good	Poor	Poor
Critical Development Factors and Risk	Three-phase flow a possible problem. Risk, small.	Traveling flight seal design - reduction of leakage and wear. Risk, small.	Same as mixed flow plus packaging design (a space problem). Risk, small.	This device must keep coal in slurry to function. Risk, unknown.	Little risk at the low pressure configuration.	Control of water in vortex to avoid flooding. Risk, high.	Feeding of pockets. Risk, high.	Coal must be slurried to push out exit. Risk, high.
Design Expansion Higher Flows	Two or three units possible in low seam, larger unit in higher seam.	-2 screw design -higher speeds -larger size all possible given envelop.	Limited by seam height.	Limited - unless larger systems become available.	Larger unit.	Larger unit.	Larger unit limited by seam height.	Larger unit limited by seam height.
Design Expansion Higher Pressures	Limited by coal degradation and inlet wear.	Approximately 300 psi within reasonable size (no basic limit).	No	175 psi is available on current non-coarse slurry applications.	Pressures limited by maximum allowable nozzle diameter	No	Strength of components only restriction.	Strength of components only restriction.
Reasons for Rejection			Space limited in 4-ft mine entry.	1) Slurried inlet. 2) Not an injector. 3) Low capacity. 4) Ability to handle coal unknown. 5) Low slurry concentration.	1) Slurried inlet. 2) Low pressure. 3) Low slurry concentration.	1) High power. 2) Control problem. 3) Low slurry concentration.	1) Space limitations. 2) Complicated mechanical design. 3) Seal wear.	1) Space limitations. 2) Complicated mechanical design. 3) Seal wear. 4) Problems in empty cylinders.

the coal and to maintain the vortex. The air-water interface in the rotor adjusts naturally to pressure variations in the slurry line, within limits, such that the device offers some positive displacement characteristics with respect to control.

The horizontal orientation of the helical injector axis makes it very adaptable to coal mine height and space limitations. It is simple, rugged, and has flow passages that can pass coarsely broken coal. Because of its simple, rugged construction the helical injector will be a very reliable machine.

Its speed and power consumption are nearly comparable to centrifugal slurry pumps for the same mass flow and discharge pressure, while its tendency to break or degrade coal is probably less. Because of its compact design, parallel units can be placed on the same injector vehicle if required for very high coal flow rates.

2.3.1.2 Design Concept Description

The concept of the recommended device is an adaption of a well-known type flow-inducer that is typically used when NPSH is very low, particularly if the flow contains a major fraction of air or vapor. Due to its highly overlapping blades and gradually expanding diameter, inlet to outlet, this type of rotor maintains a nearly constant pressure differential between inlet and outlet as long as there is some fraction of liquid flow to diffuse and fill the rotor.

The present concept is to restrict or "starve" the inlet of water flow even further, to essentially zero, and to operate at somewhat reduced outlet back pressure, so that the rotor partially empties itself, from the inlet outward radially, due to centrifugal force on the contained swirling water. Then, conceptually, dry solids can be stuffed or thrown into the empty or "ventilated" inlet where they will be picked up by the rotor blades, flung outward and submerged in the water due to their higher specific gravity and the centrifugal field. After the coal is thrown from the rotor into the vortex generated in the cylindrical housing, slurry transport water flowing through the housing (not via the rotor inlet) will carry off the solids.

Design and operation of the helical inducer for dry solids or starved inlet conditions can be described further with reference to sketches and a preliminary design layout.

Figure 9 is a simple sketch showing how almost any rotor or paddle wheel will maintain an annular vortex of water in a cylindrical barrel. Heavy solids or coal introduced in the air core of the vortex will gravitate to the outside of the water vortex. It is more efficient and positive to accelerate the solids up to vortex speed with rotor blades or passages than with a jet vortex.

Figure 10 shows the cross section of a conical rotor having outwardly spiralling blades and passages. The rotor has a front, outer shroud and rear inner shroud attached to the outwardly spiralling blades. Figure 11 shows the spiral blades on the rotor in the housing with the front of the housing removed. The leading edges of the blades are swept sharply forward like sickle blades. This prevents coal from hitting the edges except at a very shallow, glancing angle. The long spiral blades provide for gradual peripheral acceleration of the coal in the rotor. The helical injector has two operational limits, one due to inlet flooding at high back pressure, and one due to outlet aeration at low back pressure. Within these two limits, dry coal can be fed to the machine without regard to back pressure variation. Thus, within limits, the device functions as a positive-displacement injector of dry coal into a transport line.

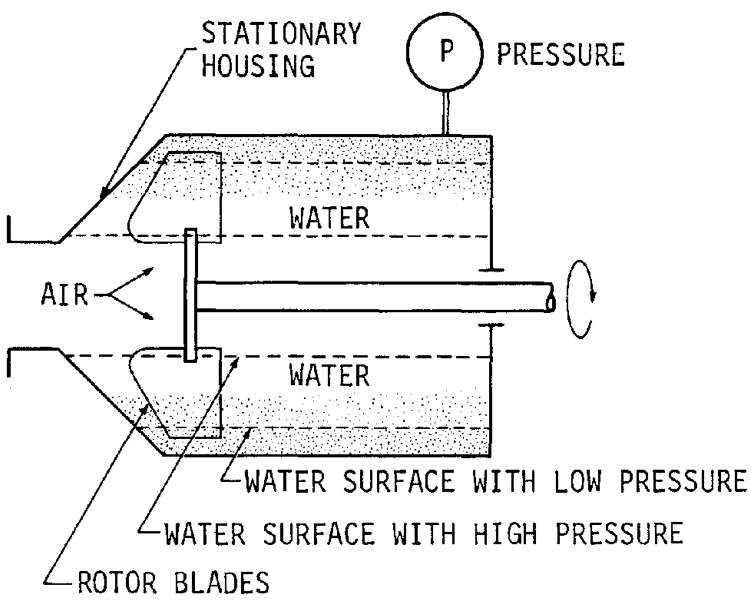


FIGURE 9. - Sketch of annular water vortex driven by a rotor causing a ventilated core.

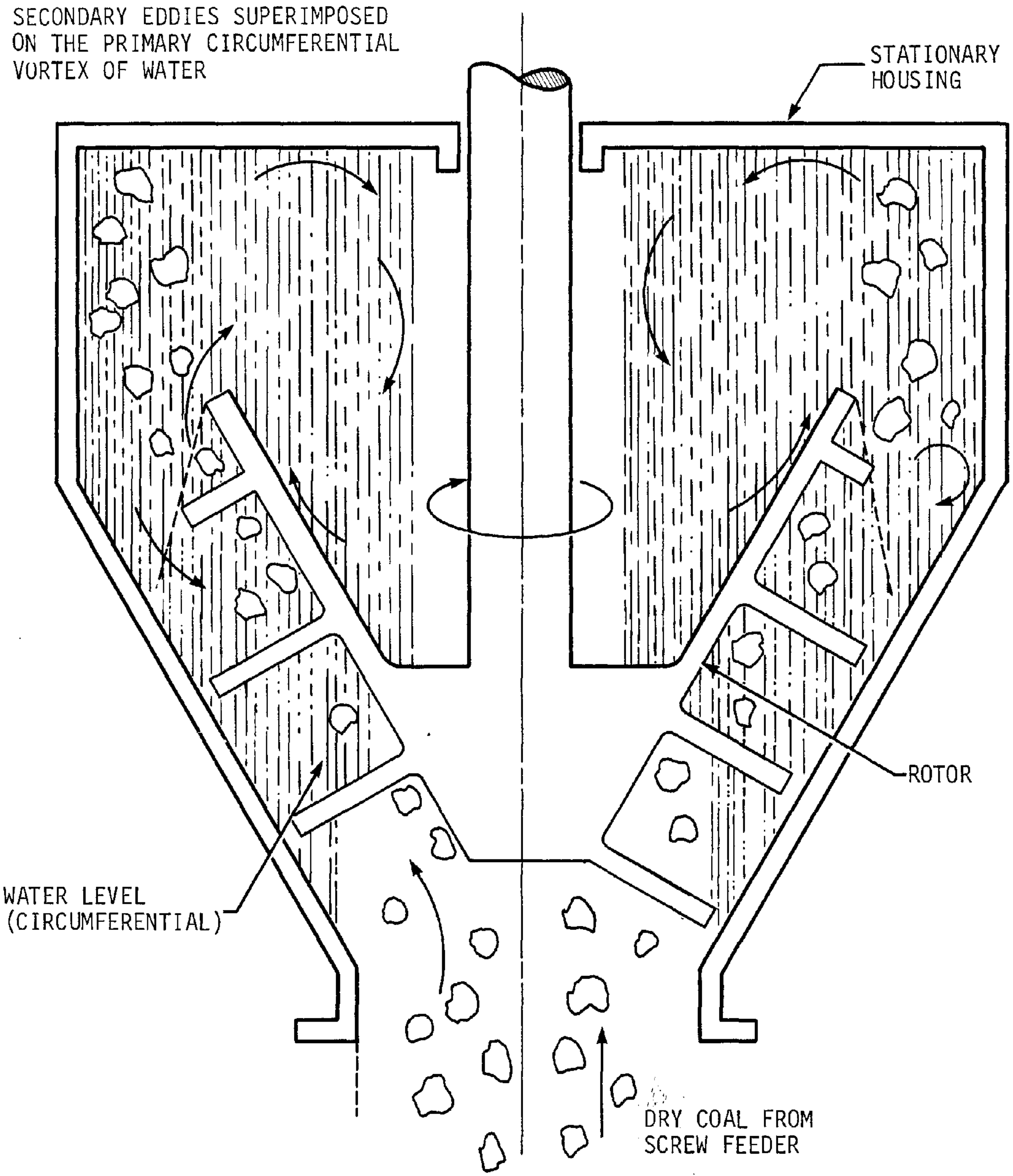


FIGURE 10. - Schematic axial section view of helical injector maintaining a water vortex having a ventilated core or inlet.

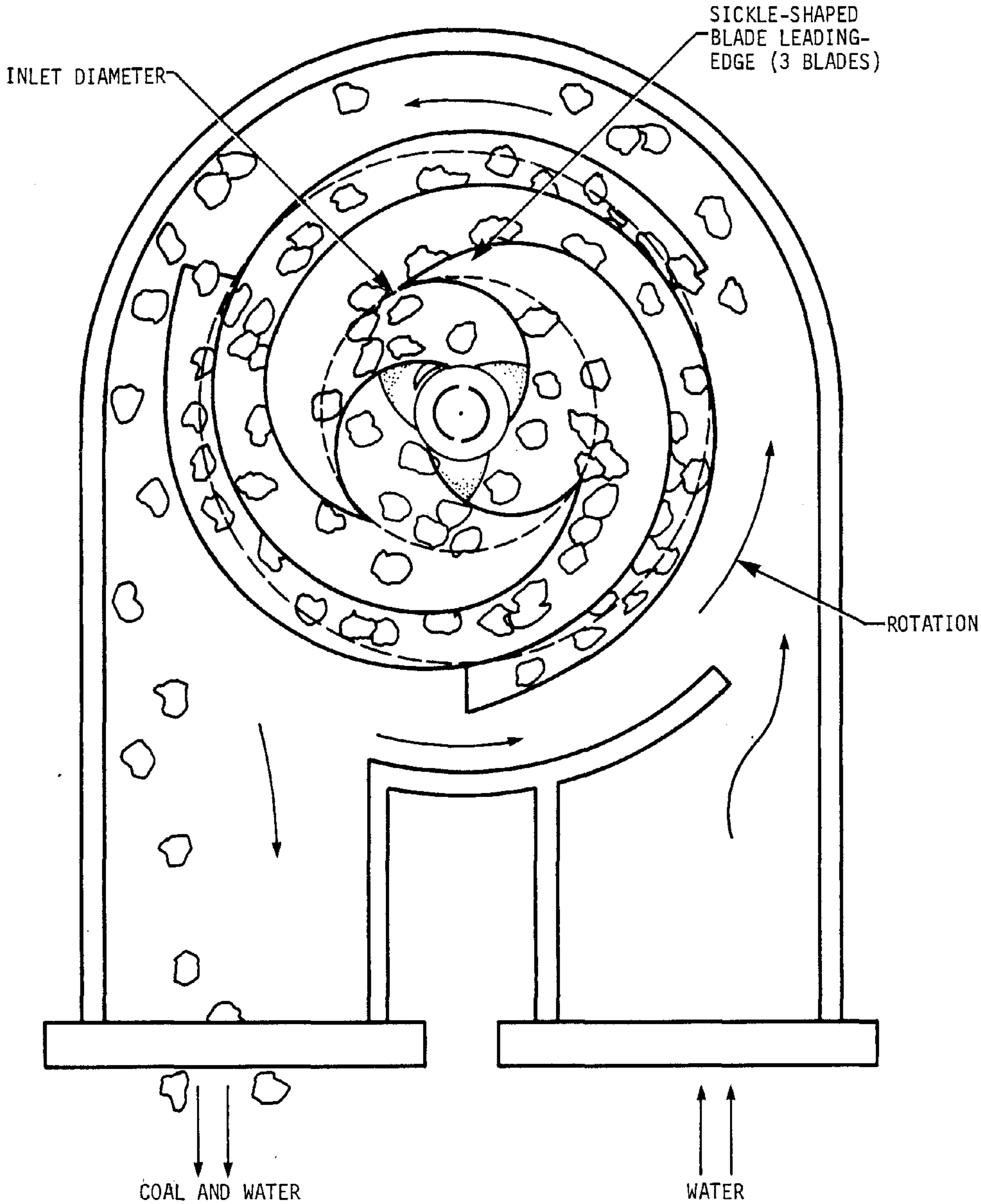


FIGURE 11. - Schematic axial view of helical injector with front housing and rotor shroud removed.

2.3.1.3 Dynamic Design

The dynamic design of the helical injector can be described best by tracing the flow path of the coal as it is fed to the inducer inlet, picked up by the rotor blades, passed through to the vortex in the housing, and finally transported away in the slurry line.

Inlet Feeding with a Screw

Top-sized dry coal must be directed at the inlet and accelerated to an average velocity suitable for the bulk volume rate. This is accomplished best in a screw conveyor that has a tubular or shrouded delivery section that mates with the inducer inlet eye. The screw conveyor receives coal from a crusher or feeder-breaker conveyor and so will have a trough-type input section. Coal will be fed to the screw at a varying rate, depending on cutting operations. At low rates, the screw will accelerate the coal; at high rates, the screw performs mostly a gathering function because the velocity from the feeder is significant, about 5 ft/s. At the design rate of 6.4 tons/min, the axial velocity of coal from the screw discharge is about 7 ft/s, which is high compared to industrial screw conveyors but necessary in order to "match" a desired inducer inlet of about 12-in-diam.

Pick-Up of Solids in the Rotor Inlet

Solids enter the rotor inlet at a very low axial velocity and swirl compared to the peripheral velocity of the rotor inlet at its maximum diameter. With a 12-in inlet and 1200 rpm rotor speed, rotor inlet tip speed is comparable to, but not more than, the highest found in slurry pump practice. However, the rotor inlet shroud, blade and passage design is very different from that of centrifugal slurry pumps. It is similar to conical inducer pumps previously referred to for low NPSH operation, but modifications are made in order to minimize the effects of solids impact and to minimize entrainment of air.

Inducer pump practice for low NPSH has shown that air or vapor ingestion, which in that application is desirable, is increased with a backward swept blade leading edges and the absence of a front rotor shroud.

A view of the rotor is shown in figure 12, which is a half-sectioned assembly drawing of the preliminary design of the entire injector unit. A view directly in the inlet also shows the blade leading edges (for two or three blades in alternative designs). Perhaps difficult to visualize, the blade edges are best described as

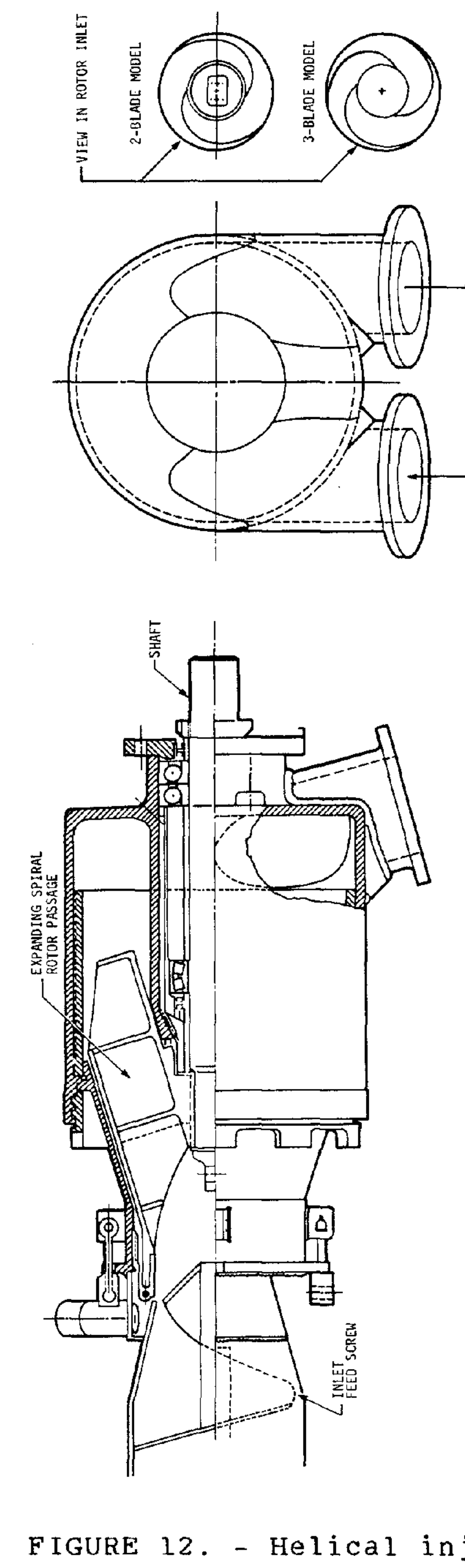


FIGURE 12. - Helical injector.

sickle-shaped, with the sickle tip at the forward, outermost inlet diameter and the shank of the sickle attached to the rearward, innermost hub of the rotor. The blade surfaces are helical and pitched at a shallow angle or small axial lead with respect to the inlet periphery or the tangential direction.

This helical pitch is suited to the small ratio of axial flow velocity to rotor speed, so that over the entire range of flow rates from zero to maximum, the solids have initial trajectories that hit the blade "pressure" surface at a shallow glancing angle, 20^{o} or less. Even more important, the leading or "cutting" edge of the blade makes an angle of less than 10^{o} with the peripheral relative trajectory of solids near the tip, increasing to only 25^{o} near the blade inner root, or rotor hub, which is at lower velocity. Although the solids-to-blade relative velocity is as high as 130 ft/s with this design, the impact velocity normal to the blade edge or surface is as much as one-third that of the particle glancing velocity.

As solids glance off the rotor blades, they acquire axial velocity and tangential velocity which direct them in a conical "spray" matching the rotor envelope.

The area for flow increases substantially as the solids turn from the inlet and flow out the rotor (meridional flow area). The solids maintain a high velocity relative to the rotor and "hug" one side of the blade passage near the hub until they are fully submerged in the water.

Water fills the rotor from the housing forward and inward in the rotor passages to some point near the inlet determined by housing back pressure.

Submergence of Coal in Water

The average air-water surface of the vortex core in the impeller is an axial cylinder. The outwardly spiralling rotor flow passages intersect the mean-vortex surface at an acute angle. Generally, the solids slide into the water on one side of the passage, and air is left to circulate on the other side. Actually, the process is turbulent, with strong recirculation flows of water and air in the rotor passages. However, the action of centrifugal and coriolis forces on the different media cause segregation according to density and particle size. Air that is entrained in the wake of larger coal lumps tends to float inward radially and return to the inlet while more dense coal and water are forced outward by the centrifugal force.

Rotor Discharge of Coal

As coal is fed to the rotor, a dense slurry tends to form and "slide" through the outer portion of the flow passage while clearer water will recirculate in a counter-flow in the inner part of the flow passage. The coal exits the rotor with a high rearward velocity relative to the rotor, hence, a reduced tangential velocity relative to impact with the housing.

Proper rotor design is essential to satisfactory operation. The rotor shroud cone angle and the blade helix angle approaching the discharge must be designed to be "steep" enough to ensure migration of solids. At the same time, the rotor-blade angles should be minimized in order to minimize the contact forces between solids and rotor, and solids and housing. There is an optimum rotor-blade shape for minimizing wear and coal degradation. This optimum rotor shape was evaluated in the testing program.

It is important to point out that the solids discharge from the housing is significantly different from that in slurry pumps. The solids leaving the rotor do not impinge directly on a scroll "cut-water" or discharge pipe; but rather, they slow down, mix more with water and roll or flow out the discharge pipe. This should minimize wear of the injected discharge and reduce wall degradation.

Estimation of Performance

Pressure-flow-power relationships for the helical inducer are estimated according to the same principles that apply to inertial pumps for liquids. The major difference in computation is that most of the angular momentum imparted, or power applied, is due to the coal. The input water flow is reduced beyond the point where water will fill the entire rotor. At the same time, the back pressure of the system is reduced to a range that will not result in net back-flow through the rotor and out the inlet. In other words, the rotor passes solids, but is "stalled" with respect to water flow. It operates at a discharge head less than that achievable if the machine were operated as a water pump.

The estimated performance shown in figure 13 is for the design shown in figure 12 operating at 1,200 rpm, near its estimated maximum practical speed. The upper limit of flow and head at this speed is the operating characteristic of the unit acting as a pump with a flooded inlet.

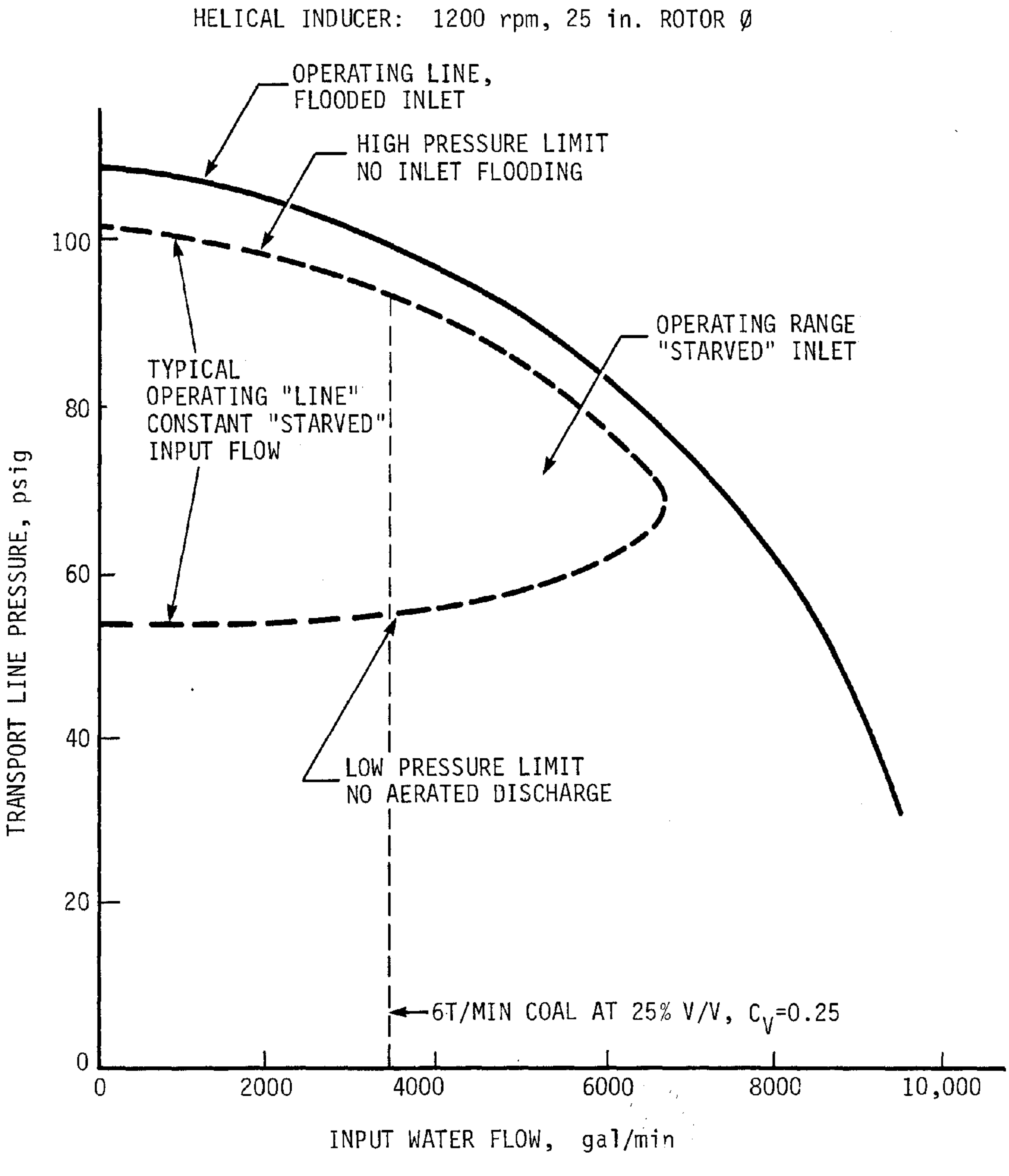


FIGURE 13. - Estimated operating characteristics of proposed helical injector under "starved" or ventilated input conditions.

If the total input flow of water and coal is less than the flooded pumping flow at a given slurry line back pressure, then the unit operates as an injector having a venitlated or "atmospheric" inlet condition. In other words, any point under the head-flow curve is a possible operating point, within limits that were mentioned previously.

If the injector is operating at a low total flow rate, say 6 tons/min of coal and less than 1,000 gal/min of entrained water, and the back pressure is <u>less</u> than about 100 lb/in², the inlet will not be flooded. If the back pressure is <u>more</u> than about 50 lb/in², the inducer should not pass air into the slurry line. Within these limits, varying back pressure does not affect input flow.

If the design or average operating pressure of the slurry line is 80 lb/in² with flow supplied by the return-line pump, back-pressure can fluctuate up and down 25% without affecting coal input as determined by conveyor feed. Similarly, coal feed can be terminated or water flow (sluicing the feeder-breaker and screw-feeder) increased to 2,000 gal/min or more without causing inlet flooding.

Figure 13 shows a dotted line around the estimated range of input and pressure conditions for which the Helical Inducer will operate as an injector having "flow" and "head" independent of each other. The actual range of values will have to be established by experiment.

Effect of Operating Speed

The pressure-flow characteristics of any inducer design operating under these conditions is still dependent on operating speed, similar to the behavior of centrifugal pumps. The various head-flow "curves" are nearly self-similar, with maximum flow proportional to speed, stall head proportional to the square of speed, and power roughly proportional to the multiple of flow times the square of speed.

Thus, a given design of the helical injector may be operated at various speed levels selected to match the pressure level of the slurry transport system and the output of the continuous miner. In addition, different rotors having different numbers and shapes of blades can be used in the same housing to offer different matches of flow and head.

2.3.1.4 Mechanical Design

A cross section of the preliminary mechanical design has been shown in figure 12. Mechanically, the design is very different from most slurry pumps in being more axially-disposed and smaller in diameter. Specific novel or attractive features are described as follows.

a. The unit is coupled to the screw feeder by a quick disconnect flange. The inlet labyrinth seal or wear ring is replaceable without further disassembly. A rotor shroud ring can also be replaced at the same time.

b. The front conical housing that covers the rotor is locked to the housing body by a single breech-lug ring. The front housing also locks a wear-insert in the main housing outboard of the rotor. This can be replaced after pulling the rotor frontward off the shaft.

c. The main housing is a casting or partial weldment containing the bearing supports internally. Thus, the rotor over-hand is comparatively short. There is ample room for an extremely stiff shaft and high-load, long life bearings. Considering high shock-load factors and moderate speed, the operating life is several thousands of hours.

d. The close-coupled design is made possible by the use of a mechanical face-seal and guarding lip-sealer. Current face seals using carbide or ceramic rings (tungsten carbide, alumina, silicon carbide, modified graphite) have shown an operating life of more than a year if properly installed and supplied with a moderately clean coolant-purge flow. The seal is serviceable from the rotor end, without disturbing the shaft-bearing set.

e. The unit can be driven conveniently direct-drive or with a pulley-belt system.

2.3.1.5 Machine Wear and Coal Degradation

The helical injector design has several features that should result in less machine wear and coal degradation than in slurry pumps.

a. The size and maximum operating speed are comparable to slurry pumps.

b. The shape of the blade leading edges is forward-swept, sickle-like, which is expected to reduce the impact of solids to a glancing angle with greatly reduced normal-impact or rebound energy. This is quite different from typical slurry pumps in which the solids must turn sharply and impact normally to the blade leading edges.

c. The conical shape of the rotor and helically overlapping blades lead to a more gradual peripheral acceleration of solids over a longer path and greater blade area. All of these features should lead to longer life and less coal degradation in the rotor compared to slurry pumps.

d. The cylindrical or barrel-shaped housing offers advantages. Solids leaving the rotor tangentially do not impinge directly on the region where the discharge pipe blends into the housing. By the time large solid lumps reach the discharge end of the housing, they are probably moving slower and against the cylindrical wall where they can roll or slide out the discharge pipe. This is in contrast to slurry pumps where solids leave the rotor and impact directly on a volute-discharge transition.

The inducer inlet shroud ring and a housing guard ring outboard of the rotor are easily replaceable inserts. They can be made of steel-reinforced elastomeric pieces which are expected to extend life and reduce impact degradation of coal. Alternatively, the lines can be made of hard-cast or hard-faced material if excessive gouging of the elastomer is observed.

The rotor can be made either as a casting or as a weldment, with a separately fabricated and attached shroud. Several alternatives leading to long life are possible, for example, Ni-Hard cast inner rotor; deep-case hardened shroud; and a thick lay-up of hard-face welded to the leading edges of the blades.

Based on these special design features, we expect the unit to provide near-design performance and acceptably low coal degradation over an operating life delivering several hundred thousand tons of coal. This should be economically attractive.

2.3.1.6 Summary of Performance

The estimated performance of a helical injector operating at maximum practical speed is summarized in table 4.

2.3.2 Sealed Flight Screw Injector

2.3.2.1 General Operational Concept

The sealed flight screw injector (fig. 14) is designed to take dry coal from a hopper and inject it into a pressurized water line without leakage of water or injection of air. The screw injector is a positive displacement device due to the novel concept of traveling flight seals which prevent gross leakage back through the material contained in the screw flights as would happen in a simple screw. The machine is fail-safe; a flood of water is not released if the injector stops.

The unit is self-feeding from a dry hopper at one end. Coal travels through the housing in the pockets defined by the screw flights and traveling seals to a pressurized water chamber. The water/slurry chamber has an inlet and outlet pipe to provide slurry system water flow through the chamber. Coal carried by the screws is flushed from the screws in the chamber and carried out the slurry transport line in a slurry.

The design concept is not limited by line pressure; pressures up to 300 lb/in^2 are realistic. The unit is a very compact, low profile design.

2.3.2.2 Mechanical Design

Figure 14 is a concept drawing of the sealed flight screw injector. It consists of two counter-rotating screws. Turning of the screws causes solids in the hopper to be carried to the water chamber at the right end. A simple screw is not a positive displacement device due to the continuous path through the spiraled flights. Therefore, to create a positive displacement device, a seal must be provided to prevent blow-out through the spiraled flights.

In this concept, a series of individual plates, called seal plates, fit between successive flights, creating pockets defined by the flights and consecutive seal plates. The seal plate rides in a slot in the housing and is carried along by the synchronized rotation of the two screws. (The same as the action of a rotating screw on a non-rotating nut.) The plate rides through the screw flights until it reaches the end of the flights. At this

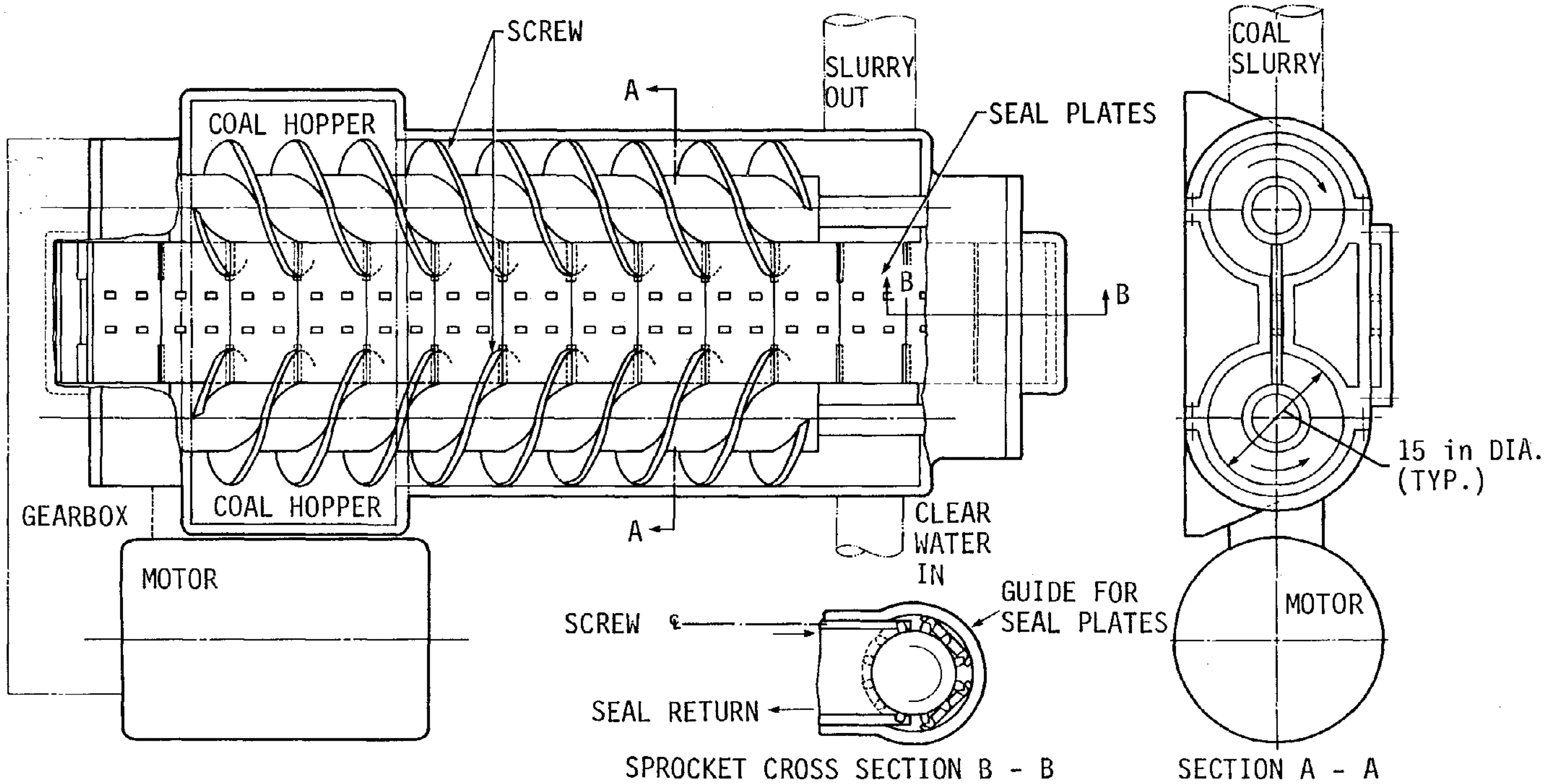


FIGURE 14. - Sealed flight screw injector.

TABLE 4. - Performance of the helical injector

Flow = 6.4 tons coal/min
Normal operating pressure = 80 lb/in^2
Positive displacement pressure range = 50 to 100 lb/in^2
Power = 250 hp (39 hp/ton/min)
Rotor speed = 1,200 rpm
Average maximum lump size = 3 in
Inlet = dry coal, screw fed
Air purge = centrifugal

point, it engages with a sprocket (see cross-section of sprocket in fig. 14). The powered sprocket is synchronized with the screws. It carries the seal plate through a 180^{o} turn and into a return slot in the injector housing. The seal plates are pushed by the sprocket in a continuous train through the return slot to the other end, where an identical powered sprocket engages the seal plate and turns it through another 180^{o} turn and in position to be engaged with the flights of the screws.

There are several methods to insure engagement without precision maching of parts. One such method is to undercut the sides of the first flight of engagement to allow worn plates to engage without interference. The problem of correct engagement of the plates with the sprockets can be handled in a similar way by enlarging the fore and aft sprocket holes on each seal plate which allows the plate to engage even when out of position due to wear.

The seal plates engage the screw prior to entering the hopper feed area to insure that the solids do not become caught betwen the seal and the screw. Likewise, the plates do not turn into the return slot until they have passed through the water/slurry chamber and the coal has been washed from the screw.

2.3.2.3 Fluid Dynamic Design

Coal and refuse in the hopper is picked up dry by the screws and transported through the injector in the pockets to the water/slurry chamber and carried away by the water. Because the solids are dry at the inlet, air must be purged prior to injection into the water line. This is accomplished by leakage of water between the flight tips and the housing and also between the seal plates and the flights and shaft of the screw.

Figure 15 shows a pictorial view of air purging to displace the air, a leakage rate of about 1,000 gal/min is required at the 8 tons/min coal flow rate. If the average leakage gap is 0.025 in, the leakage rate is estimated to be 540 gal/min, which is less than the required water to purge the air from the coal. Additional leakage can be attained in the last two flights to insure all air is purged.

The leakage figure of 540 gal/min was calculated as if the screw were not turning. When the screw turns, it displaces about 2,000 gal/min; therefore, no leakage reaches the hopper. By-pass valves route the water around the mixing chamber when the injector is not in use.

Due to the forward motion of the coal, an equilibrium point is reached where the coal motion is equal to the relative water leakage speed, creating an air/water interface as shown in figure 15. This equilibrium point is quite stable and is not very sensitive to line pressure changes. Doubling the line pressure from 100 lb/in^2 to 200 lb/in^2 would result in the leakage rate increasing by only 40%, causing the air/water interface to move toward the hopper end of the screw. In fact, the proposed screw design is capable of operating against a system pressure of 300 lb/in^2 without leakage to the hopper. At this operating condition, the screw would be fully flooded with water.

The sealed flight screw could be designed for higher pressure operation by increasing the length and mechanical redesign to withstand the increased system pressure.

2.3.2.4 Design for Wear

The screw is a very stiff and relatively short (for screw feeders) element. It is mounted in lubricated bearings at each end outboard of mechanical seals. The design clearance between the housing, seal plates and screw flights is about 0.025 in. The screw would best be made of deep-case, hardened, high strength steel, possibly hard-faced locally for wear resistance. The housing in which the screws fit is also of a wear resistant material, such as cast Ni-Hard. It is split housing, allowing for easy access to the screws and all seal plates.

The seal plate material may be chosen of a softer material than the housing and screw to minimize wear on

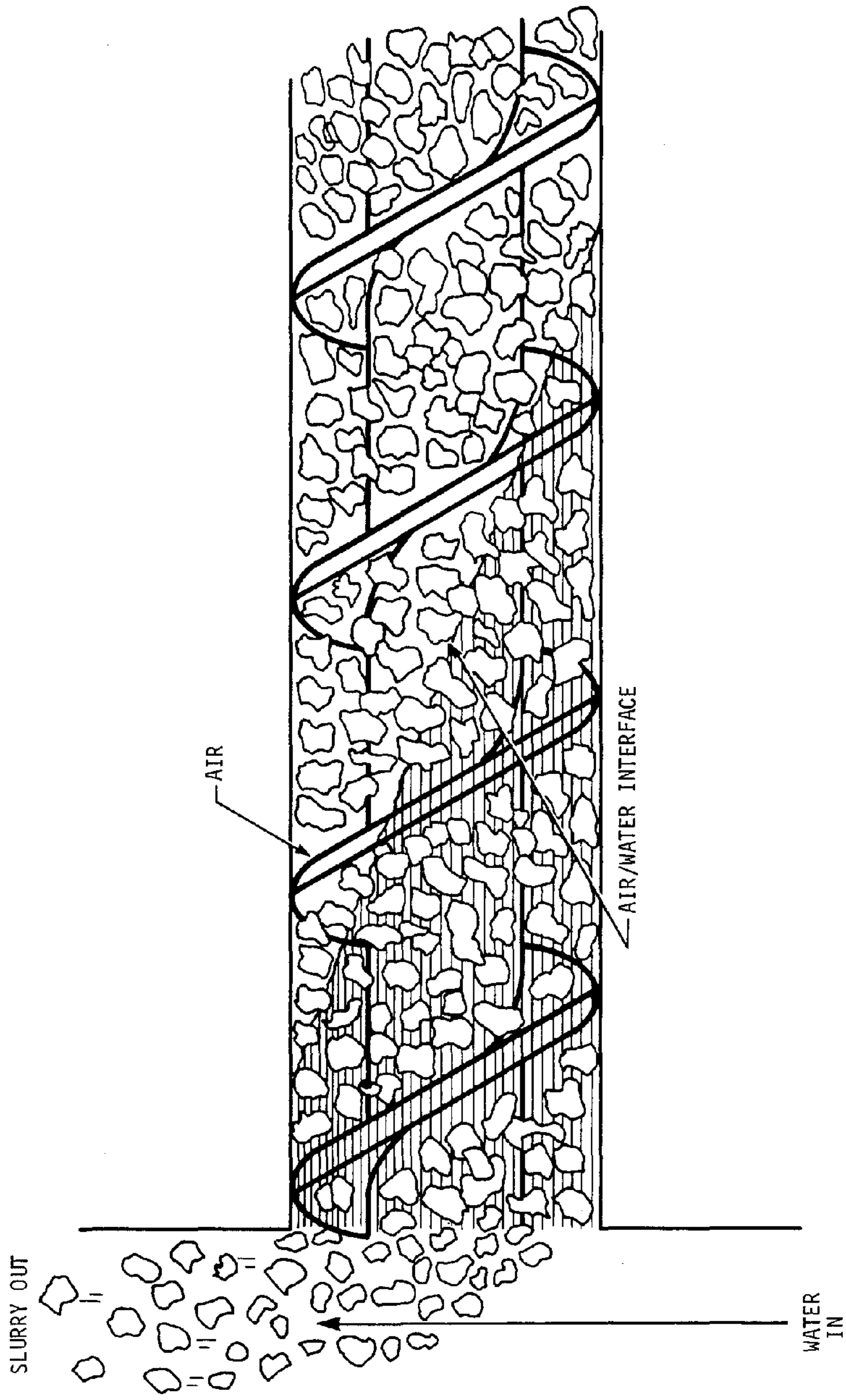


FIGURE 15. Schematic of air purged from coal by leakage water.

the screw. Special wear surfaces (for example, brake lining material) or lubrication will be provided where the seal plates slide flat on the housing between the screws. The seal plates are captured within the guided path and can easily be removed one at a time by removing an access plate at one end and cycling the machine.

It is expected that wear on the screw and housing will be minimum. A total of 250,000 tons of coal may be injected before wear would require refurbishment.

Screw conveyors built by Jeffrey in coal service are known to run six months to a year (two or three shifts) at similar speeds before requiring refurbishment (4).

Calculations based on data from a nip roll crusher indicate that unhardened nips would process 5,000 tons (5) while wearing 0.01 in.

By hardening, the wear rate would be decreased by a factor of 60 (6). This would indicate that 300,000 tons of coal might be processed.

These numbers indicate that wear is likely not to be a major problem in the development of the screw injector.

2.3.2.5 Maintenance

The compact design lends itself to convenient access of all injector parts. The housing is of split design to provide access to the screws and seal plates. Individual seal plates can be replaced through an access hatch one at a time by cycling the machine.

The injector will require a minimum of maintenance. All maintenance including major repairs can be carried out in the mine shaft.

2.3.2.6 Performance Summary

The estimated performance of the sealed flight screw injector is summarized in table 5. In summary, this injector has the following special features:

a. Higher flows to 20 tons/min by increased screw diameter

b. Pressures to 300 lb/in^2

c. Positive displacement

d. Fail-safe - stoppage of screw does not release large quantity of water

e. Dry hopper feed

f. Slurry concentration controllable from zero to maximum limit of mine transport system

g. Low profile design.

TABLE 5. - Performance of the sealed flight screw injector

Flow - 8 tons/min (can be increased to 20 tons/min)
Pressure - 100 lb/in^2 nominal (300 or greater possible)
Power - 200 hp (25 hp/ton/min)
Speed - 130 rpm
Screw diameter - 15 in
Size - 2 ft high x 4 ft wide x 10 ft long
Maximum lump size - 3.5 in
Inlet - dry coal
Air purge - water infiltration through screw
Coal feed - self feed from hopper

2.4 CONCEPT COMPARISON AND EVALUATION

Table 6 is a comparison summary of some of the critical factors of the two injectors, and table 7 provides a rating of the performance of these two injectors against the design criteria listed in the contract specifications. The two machines, although performing the same function, are very different in their method of accomplishing this function. The helical injector is a fluid dynamic machine, while the sealed flight screw is a positive displacement machine.

Both injectors meet the specifications of the coarse slurry face haulage system. Both take dry coal and inject it into a pressurized line. The major areas in which they differ are itemized below.

TABLE 6. - Comparison of helical injector and sealed flight screw injector

Design parameters	Helical injector	Sealed flight screw injector
Coal injection rate	6.4 tons/min at 80 lb/in^2	8 tons/min at 100 lb/in^2
Maximum coal injection rate possible in allowed vehicle space	20 tons/min at 80 lb/in^2 (requires 3 units)	20 tons/min at 100 lb/in^2 (requires 24 in screws instead of the 15 in shown)
Design pressure	80 lb/in^2	100 lb/in^2
Pressure-flow characteristics	Positive displacement in operating range (50 to 100 lb/in^2)	Positive displacement
Power	250 hp at 6.4 tons/min	200 hp at 8 tons/min
Design speed	1200 rpm Rotor tip speed: Inlet - 60 ft/sec Outlet - 120 ft/sec	130 rpm Screw tip speed 8.5 ft/sec
Injector dimensions	6 ft long, 3 ft wide, 2.5 ft high	10 ft long, 4 ft wide, 2 ft high
Minimum passage size	6 in	3.5 in
Maximum coal lump size	2 in (will take occasional lumps to 4 in)	3.5 in (8 tons/min machine)
Coal degradation	less than slurry pump	Minimal
Coal feeding	Conveyor meters coal to short screw feeder. Feeds dry coal to rotor inlet	Conveyor meters dry coal to exposed section of sealed flight screw
Air purge	Centrifugal action separates air within rotor	Leakage of water around screw flights displaces air
Water leakage to hopper during operation	None	None
Water leakage with power off	Bypass valves required full line flow without valves	Bypass valves required - 500 gal/min without valves at 100 lb/in^2 line pressure
Ingestion of refuse (assuming magnetic separator failure)	Passes material or jams at inlet - blade damage or internal jams possible	Passes material or stalls - little damage - can be cleared without tear down by reversing screw
Starting with load of coal in unit	Possible	Possible
Maximum slurry concentration	Only limit is mine system design	Only limit is mine system design
State-of-the-art	Is state of the art for fluid pumping	Combination of known machine elements
Ease of maintenance in mine entry	Unit disassembles horizontally for easy access	Split housing allows access to all major parts from top or side. Hatch allows easy access to seal plates for replacement.
Critical development factors and consequence	Three phase flow may present stability problems - no applicable data available - may result in a reduction of coal injection rate	Reduction of jamming and wear may require precision machining and special hard face coatings could result in high cost

TABLE 6. - Comparison of helical injector and sealed flight screw injector--Continued

Design parameters	Helical injector	Sealed flight screw injector
Design expansion to higher flows	Multiple units in parallel (up to 3 in envelope dimensions 20 tons/min)	1. 20 percent higher speed possible 2. larger diameter screws - up to 20 tons/min possible
Design expansion to higher pressure	Multiple units in series possible (2 in series provide about 200 psi)	Approximately 300 lb/in^2 within reasonable size (strength and bulk of unit is only real limit)
Estimated wear rate effect of wear on performance	No performance change due to wear. Estimated life on rotor in excess of 150,000 tons coal injected	No performance change due to wear until leakage rate exceeds pumping rate - estimated to be 250,000 tons coal injected. Screw can be refurbished
Estimated unit manufacturing cost with motor and drive, limited production	$26,000	$45,000

TABLE 7. - Design criteria injector concept comparison

Design consideration	Helical injector	Sealed flight screw
APPLICATION The solids injector shall be designed to accept coal from primary mining machinery in the three principal coal mining systems - continuous, conventional and longwall.	Compactness of design makes it useable in all three configurations	Compactness of design makes it useable in all three configurations
OVERALL DIMENSIONS The solids injector shall have the following maximum dimensions: Length, 27 ft; width, 9 ft; and height, for operation in a 4-ft coal seam. Smaller dimensions are highly desirable.	Three-foot machine height allows good clearance for 4-ft mine	Has potential for use in 3-ft or less mine height
MOBILITY The injector may be either self-powered for locomotion or it may be towable by the primary machinery.	Self-powered design is based on state-of-the-art feeder breakers such as Owens and Stamler	Self-powered design is based on state-of-the-art feeder breakers such as Owens and Stamler
The solids injector shall be sufficiently mobile to follow all operating maneuvers of the primary machinery of the systems listed under application. Mobility is here defined as tramming speed (if powered), turning radius, and steering response speed.	Same as above	Same as above
The injector shall be able to travel on difficult terrain-clay floors in coal mines. Clay floors may be hardpacked or soft and loose, either wet or dry, and undulating in short or long waves.	Same as above	Same as above
CAPACITIES The injector shall be able to absorb the discharge of the primary mining machinery and inject it into the pipeline. This may run as high as 12 tons/min during short time periods (up to 3 min in a 4-ft seam) for continuous miners.	Envelope dimensions allow room for two helical inducers which can meet required flow rate at 80 psi or one unit can meet flow rate at 40 psi	Injector can be designed for the high flow and remain within the required dimensional envelope
The injector shall accept coal in sizes generated by the primary mining machinery. No more than 10 percent of the coal to be hydraulically transported shall be as large as one-third of the pipe-line diameter, therefore, oversize must be controlled by crushing or breaking.	Breaker/Crusher unit can size coal to meet requirements.	Breaker/Crusher unit can size coal to meet requirements.
The injector shall be capable of handling coal having up to 30 percent refuse in normal operation and 100 percent refuse (at half-rate) for clean-up work.	Design will accommodate	Design will accommodate
Protection must be provided to present ingestion of metal scrap (such as roof bolts and cutting bits) and other trash such as cribbing, curtain scraps, clothing, or personal safety equipment.	Electromagnet ahead of breaker/ crusher removes magnetic items	Electromagnet ahead of breaker/ crusher removes magnetic items
The injector shall be designed for 6- to 18-in. standard pipe. If design work indicates that more than one model is necessary, the conclusion must be justified. Sizes for future testing shall be for 6-in. pipe as a small-scale prototype and 10- or 12-in. pipe for a full-scale prototype.	Machine can be sized for high flows and used at reduced rate on 6-in. line	Machine can be sized for high flows and used at reduced rate on 6-in. line

TABLE 7. - Design criteria injector concept comparison--Continued

Design considerations	Helical injector	Sealed flight screw
Provision shall be made for clearing a jammed injector without affecting the pipeline flow and with minimum downtime of the injector. For example, hatchways could be installed at critical points in the solids path.	Inducer comes apart easily at rotor for easy access to internal passages	Easy access to injector is provided. Also, injector can be reversed to extract jammed items. By-pass valves reroute water around injector.
<u>CONTROLS</u> Easily identifiable, simple controls and read-out devices (where necessary) shall be specified for the operator. These shall include the injection controls for the coal as well as power-assist devices and locomotion controls if required. Programmable or automatic systems shall be specified separately since initial interest is in manual control. They are, however, desirable for future application.	System requires minimal controls	System requires minimal controls.
The injector shall control the rate of injection of coal. This shall be controllable for a range of coal concentrations from zero to 50 percent by volume in the pipeline. Response time shall be appropriately fast to prevent either plugging or inefficient low concentrations.	Conveyor speed controls amount of coal injected into line	Conveyor speed controls amount of coal injected into line
Controls shall be specified for coordinating multiple injectors operating on the same haulage pipeline.	Control design allows for multiple injectors	Control design allows for multiple injectors
<u>SERVICE AND ECONOMY REQUIREMENTS</u> The injector shall be designed to meet the same service specifications of all underground coal mining machinery. All components shall be heavy-duty with minimum maintenance requirements.	Heavy duty slurry pumping techniques used. All parts accessible in mine	Heavy duty parts assure low maintenance. All parts accessible in mine
Delicate or precision-adjustment mechanisms shall not be specified.	Machine does not require precision parts or adjustments	Machine does not require precision parts or adjustments
Reliability (in the technical sense) shall be equal to that of continuous miners or face haulage equipment.	Comparable to mine equipment	Comparable to mine equipment
Wearing parts in contact with the coal shall be abrasion resistant.	All parts in contact with coal will be wear resistant	All parts in contact with coal will be wear resistant
Provision shall be made for repairs or preventive maintenance in the underground environment. For example, lubrication points shall be easily accessible, the use of large or heavy parts shall be minimized, and it shall be possible to change parts with minimum dismantling of the machine.	Compact design allows for maintenance in mine	Compact design allows for maintenance in mine
Standard available parts and service equipment shall be used wherever possible to minimize mine parts inventory. For example, mining machinery oils, greases, bearings, fastenings, etc., shall be specified.	Will comply	Will comply

TABLE 7. - Design criteria injector concept comparison--Continued

Design consideration	Helical injector	Sealed flight screw
Lowest possible capital and operating costs shall be a primary goal. This can be extended to operator and mechanic skill requirements. It would be useless to the Government to provide a machine that nobody can afford to buy.	Design reflects this requirement	Design reflects this requirement
SAFETY The injector shall meet the requirements of all applicable Federal legislation, especially permissibility.	Will be met	Will be met
The injector shall fail in a safe manner. That is, upon failure it should not release a flood of water, parts should not shatter explosively, nor should it bury itself, and fill the entry with coal, or any other material.	By-pass valves required to prevent leakage of water. Will not shatter explosively due to low speed	Minimum leakage of water without shutting valves manual by-pass valves provide seal. Will not fail dangerously
The injector shall not create any new or additional hazard in coal mining.	None	None

a. Mechanically: The helical injector is by far the simplest construction with only one moving part - the rotor. The mechanical design can be based on state-of-the-art technology of aerospace helical inducer fuel pumps and slurry pumps used in the process industries. It does not require the mating or sliding of precise sealing surfaces to develop resistance to leakage flow.

   The sealed flight screw has about forty moving parts. Sealing is accomplished by the mating of the seal plates with the surfaces of the screw flights. A special, precisely machined screw would be required for this purpose. A conventional screw-conveyor is not precise enough for operation. This extra complexity of mechanical design is reflected in the higher cost of this injector.

b. Fluid dynamically: The positive displacement sealed flight screw is the simplest fluid dynamic machine. We have a high degree of confidence that the design will feed the design rate of coal into the pressurized slurry line. It is possible that estimates of the leakage rate may be in error, and higher than estimated leakage flows may result. This may require redesign to reduce clearances, or lengthening the screw to include more flights - design changes which can be made relatively easily.

   The helical injector is more complicated fluid dynamically. It has a three-phase flow situation in the rotor with air, coal, and water passing through or being separated. For any given speed, there are fluid dynamic limits on the coal feed rate and pressure for water-starved inlet operation. Coal feed rates higher than the design maximum could result in air entrainment in the output slurry. Slurry line back pressure greater than the maximum capability of the inducer at constant speed would result in leakage. Both of these performance requirements could be increased by increasing the operating speed. However, higher operating speed is likely to lead to greater coal degradation. The best operating speed and injection capacity would have to be determined by testing.

c. Pressure capability: The sealed flight screw has a higher potential pressure capability which is only limited by size, strength, and wear of its parts. Strength can be increased relatively easily by increasing thickness in most cases. The present design could probably operate up to about 300 lb/in². Leakage can be reduced by increasing the screw length and number of sealed flights.

The helical injector is at its practical pressure limit of approximately 100 lb/in². Higher pressure would require increased rotor tip speed with a possible increase in coal degradation. Two units could also be connected in series for increased pressure operation up to 200 lb/in².

d. Coal injection rate: The sealed flight screw design has a higher capacity of 8 tons/min as compared to 6.4 tons/min of the helical inducer. This general design appears capable of 12 tons/min with an increased size that is still within vehicle limits. Two helical injectors can be used in parallel for more than 12 tons/min, the total space required being less than that of the sealed flight screw. The capacity of both machines can be increased in proportion with speed if higher speeds are found to be practical in actual operation.

e. Coal degradation: The sealed flight screw has a much lower operating tip speed than the helical injector, 8 ft/s versus 137 ft/s, but the screw can "crunch" coal between mating parts, while the high velocities in the inducer are at glancing angles and submerged in water. On balance, the coal particle velocities are substantially lower in the screw than in the helical injector; therefore, coal degradation should be less. The screw has smaller passage width than the inducer, 3.5 versus 6 in, but the inducer is more sensitive to the possibility of two lumps jamming together. Coal should be top-sized to 2 in for both machines, but both machines will accept occasional larger lumps. Coal degradation is likely to get worse with higher pressure operation of the helical injector while pressure should have little effect on coal degradation with the screw. Thus, the sealed flight screw should demonstrate superior performance, at higher pressures (greater than 100 lb/in²) than the helical injector.

f. Fail-safe operation: The screw is estimated to leak at a rate of 540 gal/min when not turning and with full line pressure of 100 lb/in² applied at the outlet. The helical injector offers no resistance when not turning and would leak the full open line flow rate. In both cases, safety shut-off valves would have to be provided for fail-safe operation though the effect of a power failure would be less rapid with the screw.

g. Effect of wear on performance: Wear of the seal plates and the screw flights will result in increased leakage through the sealed flight screw. However, actual leakage to the hopper could be reduced by increasing the rotating speed of the screws with increasing wear.

The helical injector performance is estimated to be extremely insensitive to wear of the rotor passages and blades. It should change little with time of operation until wear becomes severe enough to cause ultimate breakage of the rotor parts.

h. Maintenance: Both the injectors have been designed for easy in-mine maintenance. Because the sealed flight screw has more moving parts and sliding surfaces, it is expected to require greater maintenance than the helical inducer. Based on the preliminary design effort expended to date, we have a high degree of confidence that the helical injector will be able to achieve extended periods of reliable and continuous operation at the face. A similar assessment for the sealed flight screw could only be made after the completion of an extensive in-mine test program.

i. Cost: The helical injector is in the same price class as currently produced slurry pumps. The sealed flight screw is a higher cost unit, estimated to be about 1.7 times the cost for the helical injector.

Downtime for maintenance is expected to be higher for the screw. Thus, both first cost and operation costs are estimated to be higher for the screw than the helical injector.

j. Development risk: Since the two injectors are so different in operation, the risk associated with their development is markedly different.

The development risk with the helical injector lies in its fluid dynamic and coal degradation performance. Once this has been demonstrated, there is a high degree of confidence that the unit will operate reliably underground with a continuous miner. This confidence is reinforced by the strong interest expressed by CONOCO in this design. They have had good success with the use of a centrifugal pump with their underground hydraulic transport system. It is their considered opinion that a fluid dynamic device such as the helical injector could be made to work in a mine if its fluid dynamic performance was demonstrated by laboratory testing.

As mentioned earlier, our confidence in the design performance of the sealed flight screw is high; however, the success of such a device could be evaluated satisfactorily only after extensive underground operation has demonstrated that the unit can operate continuously and reliably in the severe underground environment. This comment is generally true of all positive displacement types of feeders. For example, the Russian screw feeder design (7) discussed in Appendix A showed good laboratory performance and initial testing in a mine. The lack of any further reports on the device is a strong indication of the failure of this design in underground production. CONSOL, though interested in the design, has expressed reservations on its capability to operate satisfactorily underground.

k. Conclusion: In summary, the development risk associated with the helical injector would be lower since a good evaluation of its potential could be made from an aboveground test program. Such a program can be better defined and controlled than in-mine tests. It should be shorter in time and considerably less expensive than a full scale underground test series. Based on the results of a laboratory test program, a go/no-go decision could be made on whether to proceed further with the development of the injector, or whether to consider the development of the more expensive sealed flight screw.

The sealed flight screw, if made to work reliably, would offer superior performance especially for higher pressure operation. Thus, the device is a strong contender as a high pressure feeder for long distance hydraulic transport in the main slurry lines and for hydraulic hoisting. A development program should be initiated for this type of a device at some appropriate time. It is likely to be costly and extensive time-wise.

since it would depend on a full-scale prototype construction and life testing.

Since the pressure capability of the helical injector appears to be sufficient for face haulage and it is likely to involve a lower development risk, we recommend it as our primary candidate for face haulage.

The sealed flight screw is our secondary choice for face haulage but a primary candidate for the development of a higher pressure coal injector.

2.5 CONCLUSIONS AND CONCEPT RECOMMENDATIONS

The principal conclusions of the Phase I effort on the preliminary design of a dry coal injector for underground hydraulic transport are:

a. There is no single state-of-the-art device which can meet the specified requirements.

b. The concept generation and evaluation effort of this phase of the program has led to the identification of two viable injector concepts: the helical injector and the sealed flight screw injector.

c. Based on a detailed evaluation of the two devices, we have selected the helical injector as the primary candidate for face haulage where the pressure requirement is moderate, up to 100 lb/in^2. The helical injector is simpler mechanically than the screw and, therefore, has the potential of operating more reliably in face haulage applications.

d. The sealed flight screw performance is potentially superior to the helical injector at pressures higher than 100 lb/in^2. Thus, it would be a prime candidate for injecting coal at higher pressures into the main slurry line for long distance transport and for vertical hoisting. However, the greater mechanical complexity of the screw system and the potential reliability problems associated with this complexity lead us to select this concept as a back-up candidate for face haulage.

e. The development risk associated with the helical injector lies in proper optimization of the rotor design to achieve the necessary fluid dynamic performance. Once the proper fluid dynamic design has been established, we have a high degree of confidence that the unit will operate satisfactorily at the mine face.

f. Injection performance of the sealed flight screw is relatively assured. Its success will depend on achieving high reliability and long wear life in operation at the mine face. Accordingly, an extensive in-mine test program would be required to prove this system.

g. The conceptual sealed flight screw injector offers such sufficient promise that the development of this injector, particularly for higher pressure operation, should be initiated at an appropriate time.

REFERENCES

4. Mr. Gingras-Jeffrey.

5. Materials Problems and Research Opportunities in Coal Conversion, 16-18 April 1974, Columbus, Ohio - The National Science Foundation and the Office of Coal Research, L.G. Austin, Vol. II, p. 58.

6. Same as (5), W.L. Silence, Vol. II, p. 455.

7. Improvement of the Mine Station, Hoisting and Underground Transportation through Application of High Pressure Hydraulic Transport, Frolov, A.G.

3. PHASE II SUMMARY - HELICAL INJECTOR SCALE MODEL DEVELOPMENT AND TESTING

3.1 MODEL SCALE INJECTOR PERFORMANCE OBJECTIVES

A reduced scale (one-third linear scale) hardware development program was considered to be the most effective and economical method for evaluating the helical injector concept. This reduced scale program was confined to the design and development of the helical injector and feed screw assembly. All other injector vehicle systems include state-of-the-art capabilities and, therefore, were not studied on a reduced scale.

The performance goals of the model scale injector were based on the Phase I performance estimates for the full-scale helical injector. The full-scale and model scale injector's performance goals and linear scale factor are presented in table 8.

3.2 MODEL SCALE COARSE COAL INJECTOR SYSTEM DESIGN

The helical injector shown in figure 16 is composed of two major elements, the helical injector and feed screw. The helical injector one-third scale prototype was designed to provide flexibility in testing a wide range of configurations. Figure 17 is a cross section drawing of the helical injector. The prototype features are the ability to change rotors. The rotor may be quickly removed from the front without removing the bearings or shaft, allowing various rotors to be tested on the same unit.

The shaft and its bearings are designed to provide a very stiff unit which can withstand heavy unbalance forces. Duplex bearings on a 1-3/4 in diam shaft provide the thrust carrying capability, while a spherical roller bearing on a 1.60-in diam shaft carries the rotor end of the shaft. The bearings are lubricated with grease through external fittings. The shaft, bearings, seals and their housing are assembled as a unit and may be removed without disturbing the outer housing and associated pipes.

The injector was belt driven by a 30-hp, 1,800-rpm, three-phase electric induction motor. A variable speed "V" belt pulley drive allows inducer speeds from 500 rpm to 3,600 rpm. Four speeds are also available through a cog-belt drive. The 3-in diam slurry exit and a 2-in back water inlet are arranged so that flow enters and leaves the cylindrical chamber tangentially behind the back plane of the rotor. A "front water" inlet provides clear water through an annular ring to the front of the rotor.

TABLE 8. - Performance goals

Parameter	Full scale helical inducer	Scale factor for 1/3 linear	One-third scale helical inducer
Coal injection rate	6.4 ton/min	1/9	1400 lb/min
Normal operating pressure lb/in^2	80	1	80
Discharge pressure range lb/in^2	50 - 100	1	50 - 100
Power - BHP	350	1/9	40
Rotor speed rpm	1000	3	3000
Rotor tip speed - ft/s	125	1	125
Maximum lump size - in	3	1/3	1
Inlet	Dry coal screwfeed		Dry coal screwfeed
Purge	Centrifugal		Centrifugal
Maximum slurry ratio by volume	50%	1	50%
Coal degradation	Minimum		Minimum
Trash injection	Coal with 30% refuse and 100% refuse at 3/2 ton/min	1/9	700 lb gravel/min

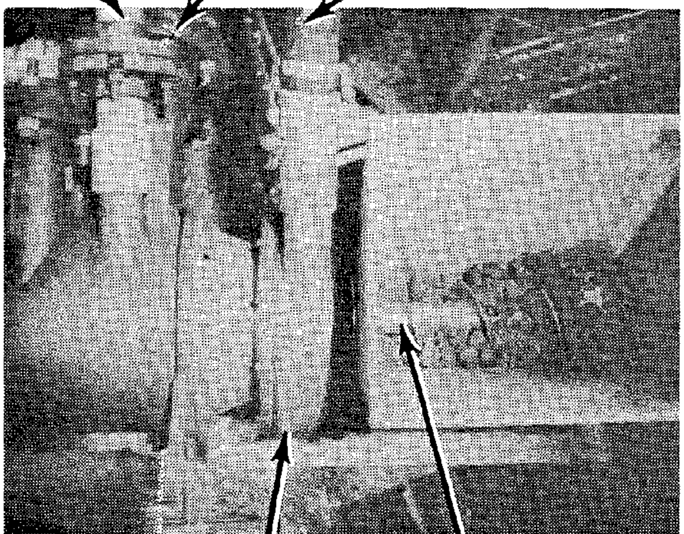


FIGURE 16. - Helical injector.

In development of the prototype, an additional back water inlet and slurry outlet were added and a baffle plate installed to effectively shorten the unit.

3.2.1 Feed Screw Design

The high flow rates achievable by the helical inducer require a system capable of moving the solids quickly into the inducer inlet.

For the inlet diameter of 5 in and the solids throughput of up to 2,000 lb/min, the design chosen is a high speed screw with maximum flow capacity twice that of typical screw conveyors. Figure 18 shows the basic construction of the feed screw.

At 1,600 rpm, the screw is approximately half filled at maximum solids loadings. The 9-ft/s axial particle velocity is 80% of that within the rotor. Such a close match of axial velocities minimizes impact on the back of the rotor tip. The coal particles also rotate in the barrel and this produces two advantageous effects. The rotation is in the same direction as that of the rotor and this lessens the impact between particles and the rotor leading edge. The tangential particle velocity at the

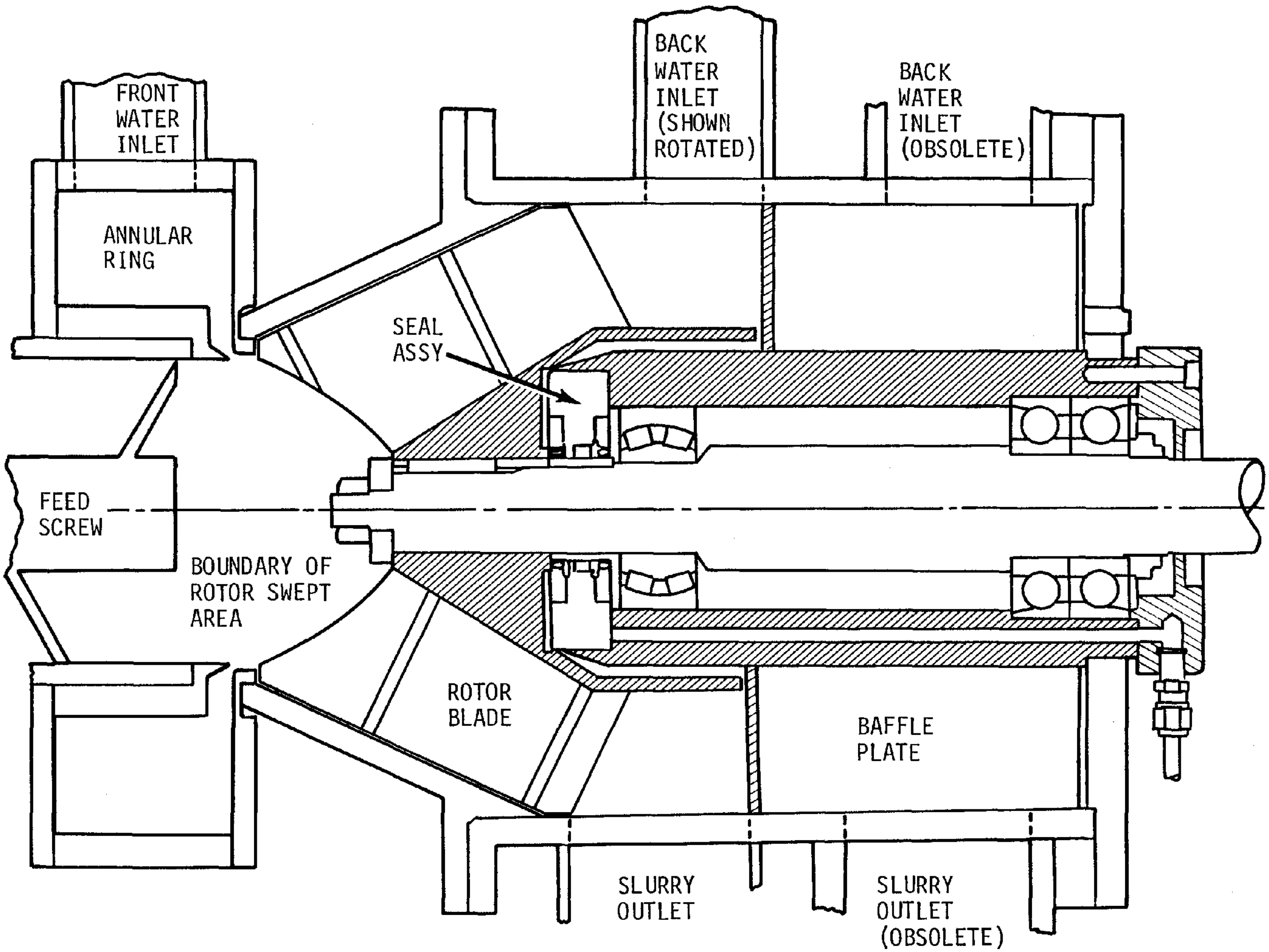


FIGURE 17. - Helical injector with single blade rotor.

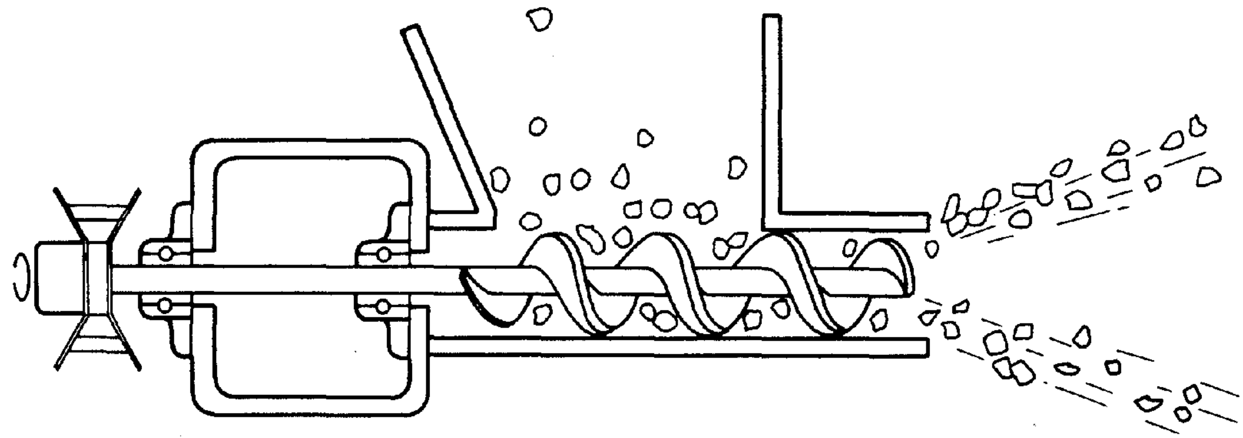


FIGURE 18. - Feed screw.

exit of the barrel results in a flare of the coal stream roughly matching the flow path in the rotor housing. Wear of injector parts is minimized by low impact velocities and non-abrupt changes in direction of particle motion.

The feed screw is self-feeding from a dry hopper. The prototype screw was made from aluminum with one forward-leaning flight and a 4-in pitch.

The screw is thermally shrunk onto a 1-7/8 in steel shaft and located by two ball bearings. The screw is cantilevered from the drive end. This allows independent operation of the screw and the inducer and eliminates the need for bearings in the solids flow region. This very stiff, short feed screw is vibration free up to 150% of the 1,600 rpm design speed.

The screw is driven by a 5-hp, 1,800 rpm, three-phase motor. A high velocity chain drive provides for speeds from 400 rpm to over 2,400 rpm. The feed screw used less than 1 hp, as measured with a three-phase wattmeter.

3.3 MODEL SCALE SLURRY TEST FACILITY

A 3-in-diam pipeline hydrotransport test facility was constructed for the purpose of testing the model scale helical injector. The facility's prime components include the helical injector, a pipeline of up to 600 ft of 3-in-diam PVC pipe, and a water solids separation system at the end of the pipeline. The system was designed to operate on a batch basis for the solids and a continuous flow system for water.

Figure 19 is a simplified schematic of the model scale slurry test facility. Water is fed to the injector by a centrifugal pump equipped with a throttling valve on its discharge side. Coal is delivered to the belt feeder-feed screw assembly by a 12-in-diam metering screw located at the bottom of the coal storage hopper. The water-solids mixture is discharged from the injector into one of two 3-in pipelines (a 600-ft pipeline or a 40-ft pipeline). The 40-ft pipeline was used for mapping the injector's performance over a wide range of conditions. The 600-ft pipeline was used for evaluating pipeline transients and their affect on the injector. At the end of either pipeline, the water-solids mixture is tangentially discharged into a 100-ft^3 cylindrical pressure vessel. All particles above approximately 0.03-in drop out as water flow discharges at the top and returns to the water tank. Back pressure is generated in the slurry pipeline by a throttling valve downstream of the pressurized receiver tank. Between runs, coal is returned to the metering screw/storage hopper by a bucket elevator which receives coal through a 6-in ball valve located at the bottom of the receiver tank.

Basic injector testing and performance mapping was achieved by gradually raising the line back pressure in increments until the limiting pressure was reached as indicated by fluid filling the feed screw hopper.

3.4 HELICAL INJECTOR MODEL SCALE DEVELOPMENT

3.4.1 Rotor/Injector Development

Two basic rotor configurations were developed and tested during Phase II. The first basic rotor geometry tested, shown on the right side of figure 20, has two vanes with 5-in pitch on a 35 $^{\circ}$ cone. The second rotor geometry tested, shown on the left side of figure 20, was used for most performance tests conducted during Phase II. It has a single vane of 3-in pitch on a 68° cone with the blade outer edge on a 52° cone. A common casing and shaft assembly was used with both basic rotor geometries and front housing components.

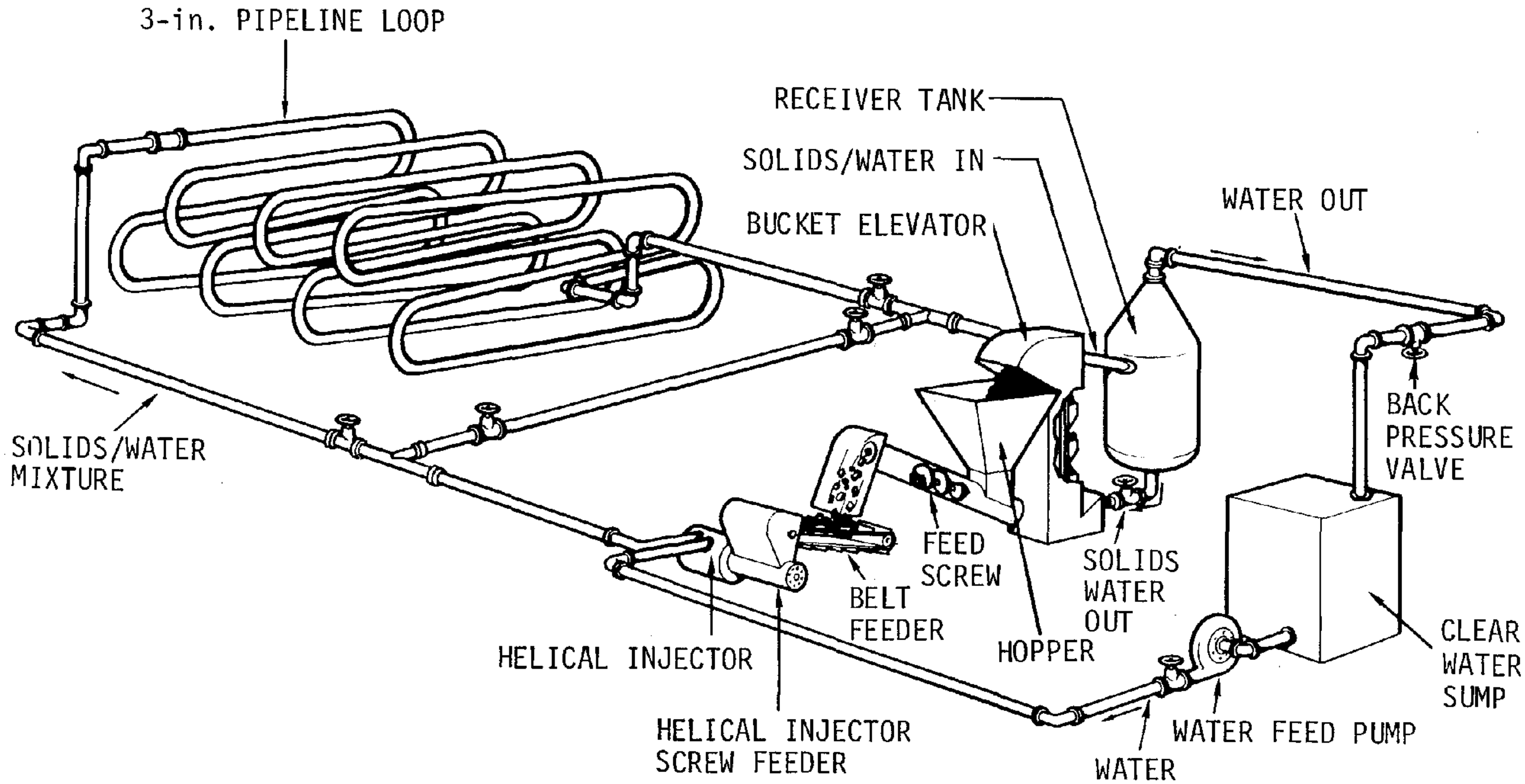


FIGURE 19. - Model scale slurry tests facility.

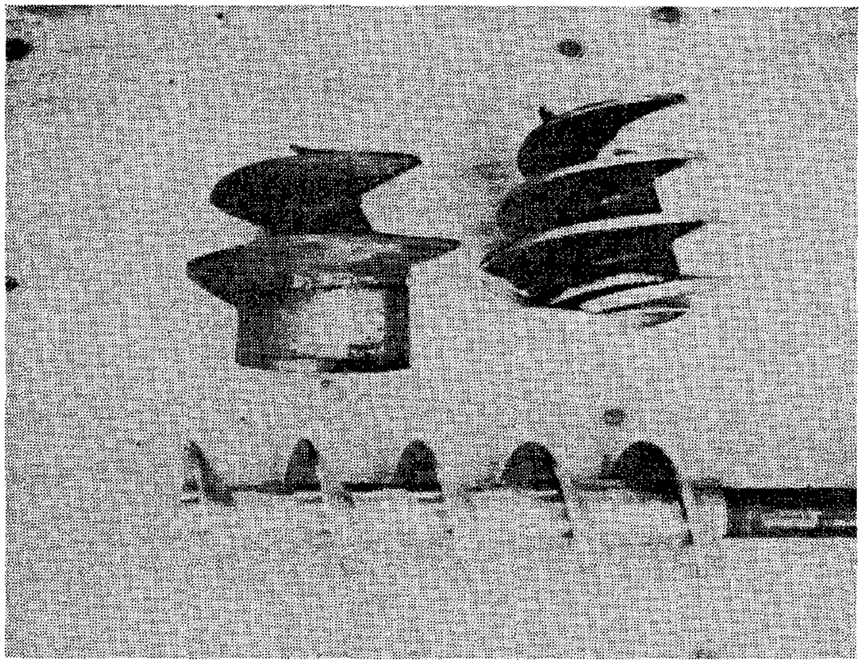
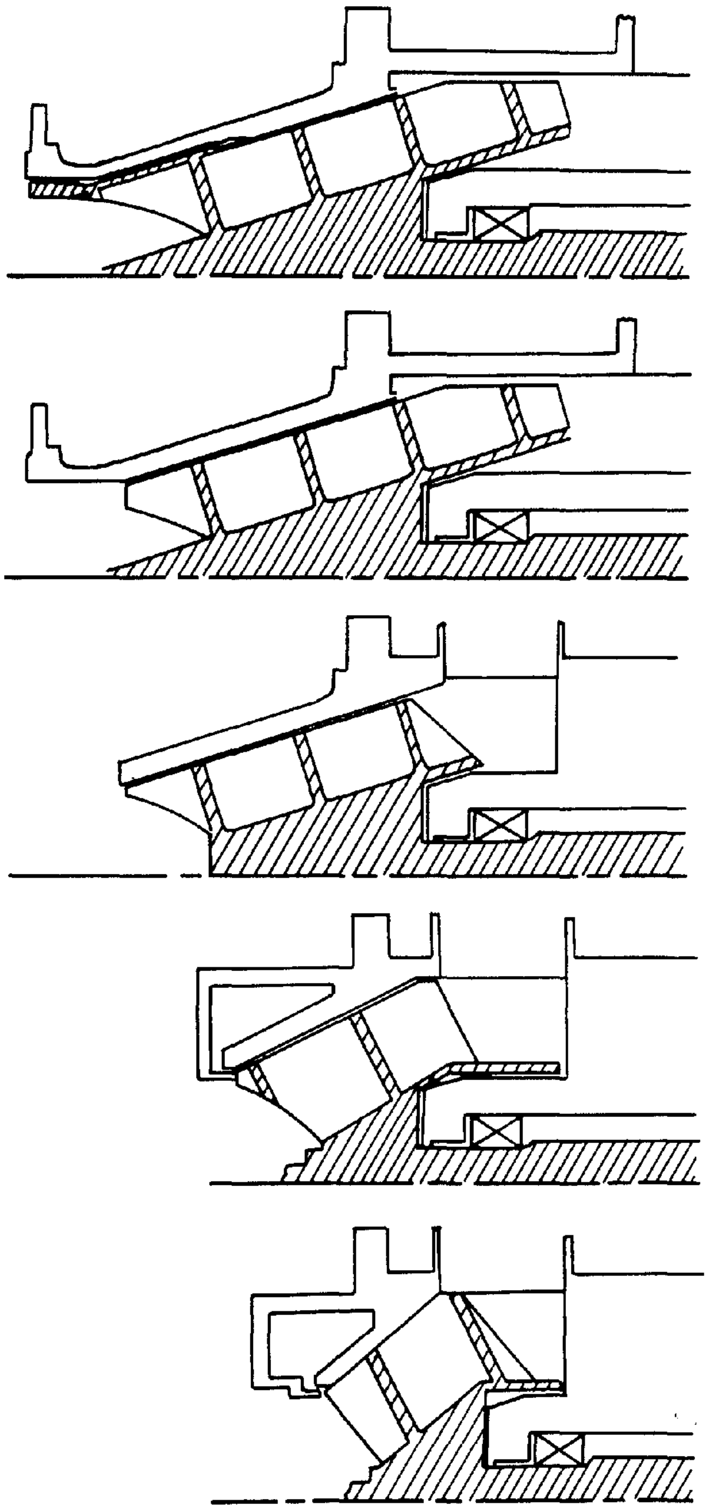


FIGURE 20. - Test rotors and feed screw.

A pictorial chronology of the rotor/front housing development is illustrated in figure 21. The original design, figure 21, included a conical shroud for the purpose of maximizing pressure capability. In fact, it served as a centrifugal trap for coal and totally blocked flow. The shroud was removed (fig. 21b) and a close running clearance between vane and housing resulted in successful injection of coal. Successive trimmings led to the final two vane configurations shown in figure 21c. The vanes were shortened circumferentially and ports relocated for a reduction in power consumption. The inlet was opened up to permit use of a larger diam feed screw.

A single-vaned unshrouded rotor was built to further incorporate beneficial features found with the two vaned rotors. This design is shown in figure 21d. The steeper design permits reduced length and thus less friction. Passage size was increased, providing coarser coal handling capabilities. The pitch was reduced, resulting in lower axial velocity and minimized wear and coal degradation. Power consumption at a given coal flow and back pressure was slightly less than that of the two vaned rotor. A significant portion of Phase II testing was



(A)
TWIN FLIGHTS
2 in LEAD
17.5 DEGREE TAPER
INITIAL VANE
CONFIGURATION

(B)
TWIN FLIGHTS
2 in LEAD
17.5 DEGREE TAPER
TRIMMED VANES

(C)
TWIN FLIGHTS
2 in LEAD
17.5 DEGREE TAPER
TRIMMED VANES AND
HOUSING

(D)
SINGLE FLIGHT
3 in LEAD
26 DEGREE TAPER

(E)
SINGLE FLIGHT
2.6 in LEAD
40 DEGREE TAPER

FIGURE 21. - Helical inducer evolution.

Based on these gained improvements in process performance, the rotor was again modified as shown in figure 21e. Three trimmings of this 40° rotor as shown in figure 22 were evaluated along with the 26° rotor, for data comparability. Results of these tests are presented in figures 23 through 26. MOD I yielded less power per back pressure than MOD II. The best combination of power consumption and pressure capability resulted from the MOD II version which has 6 in of vane at the maximum rotor diam of 10.5 in. Pressure capability of this 40° rotor is virtually identical to that of the previous 26° design and there was a power savings which ranged from 10 to 15%. It is this 40° rotor design which provided the basis for the full-scale injector.

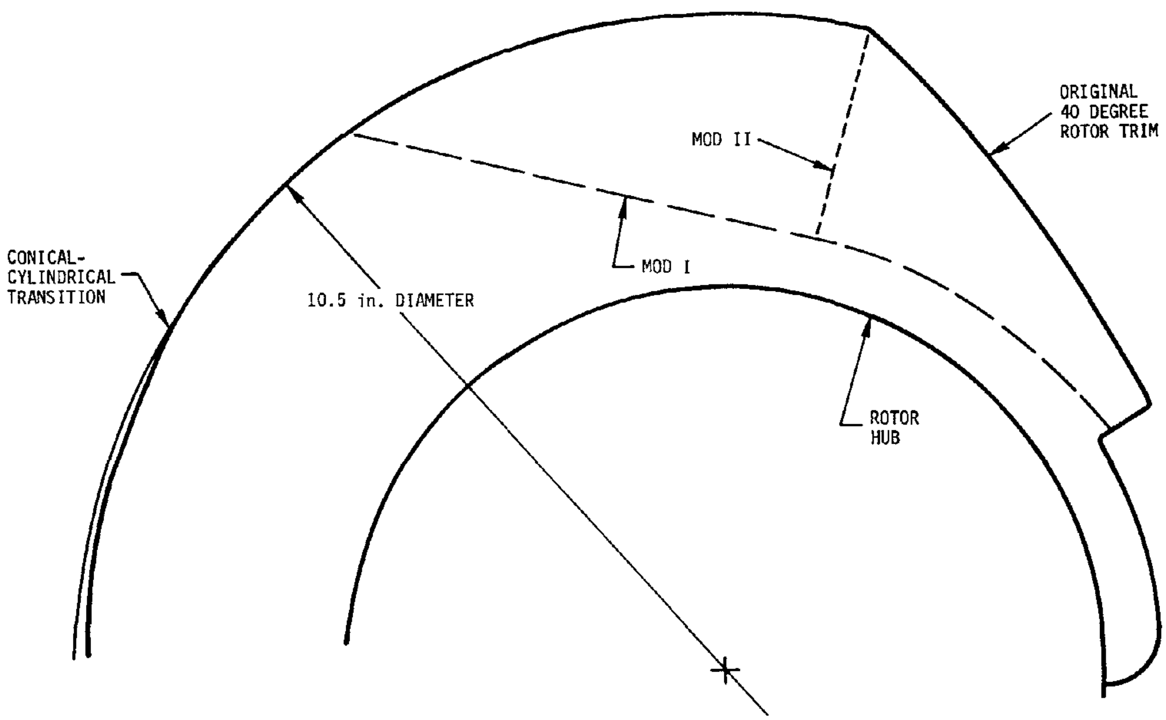


FIGURE 22. - Forty degree rotor trimmings (viewed through coal inlet).

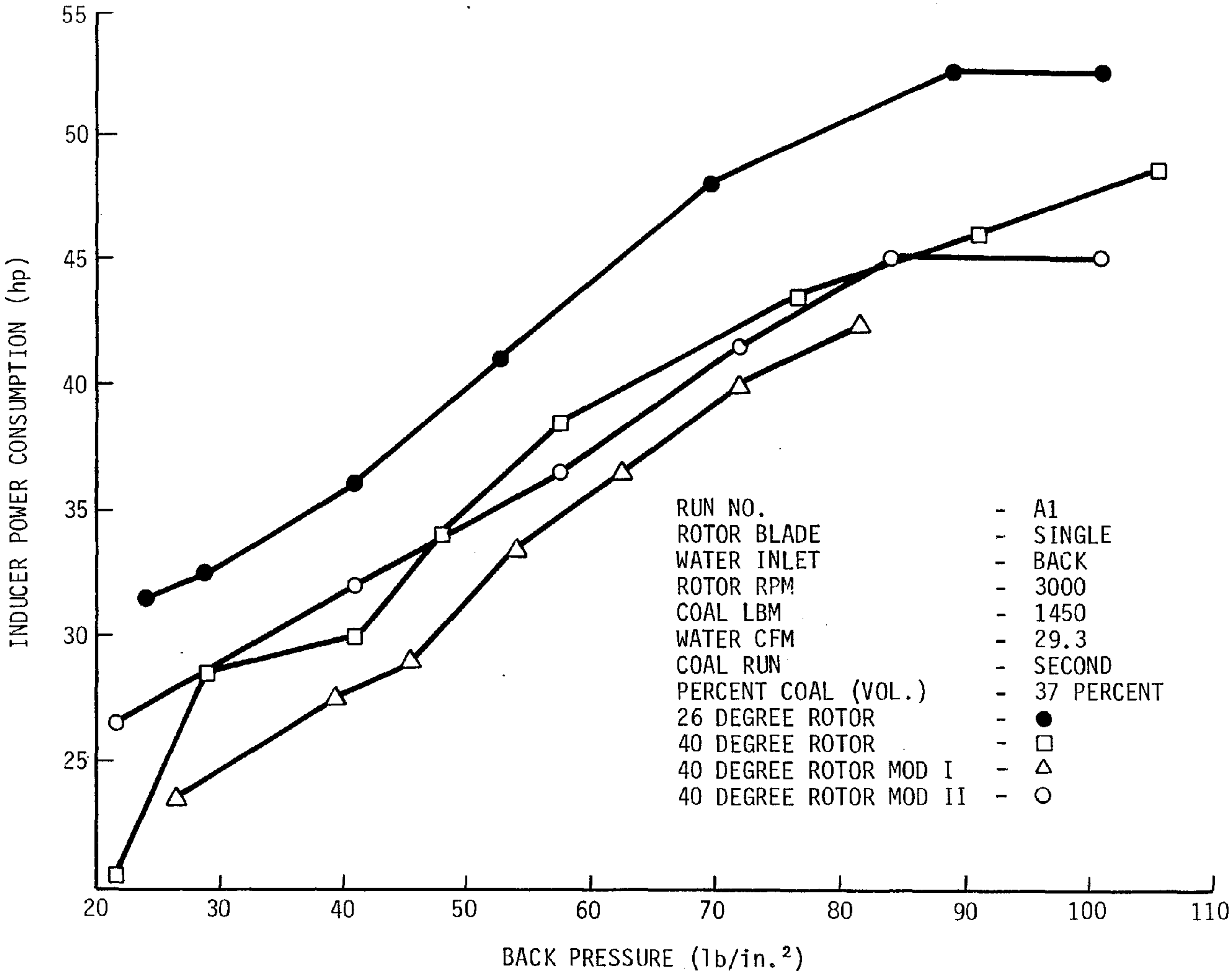


FIGURE 23. - Inducer proof tests, A1.

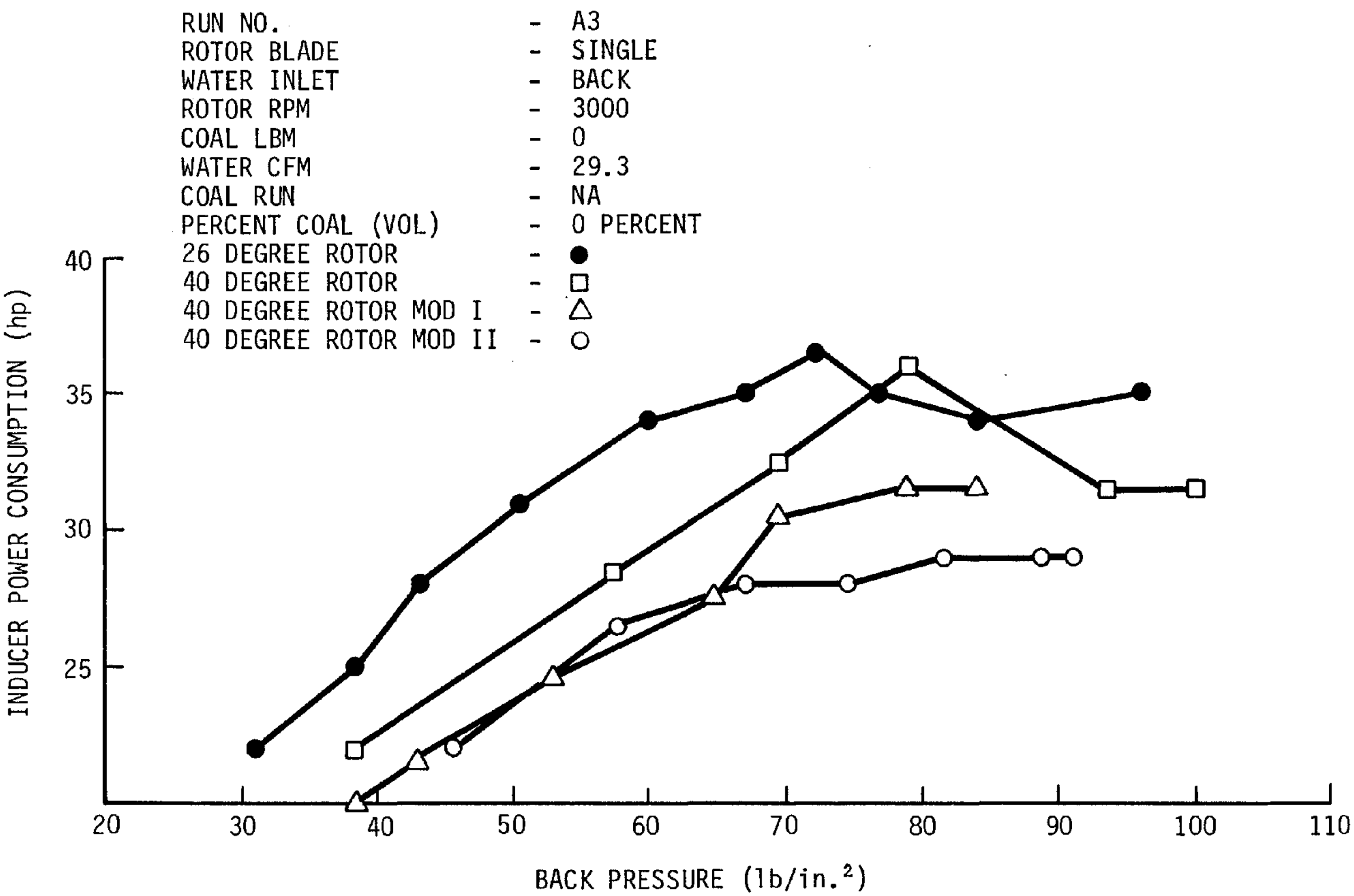


FIGURE 24. - Inducer proof tests, A3.

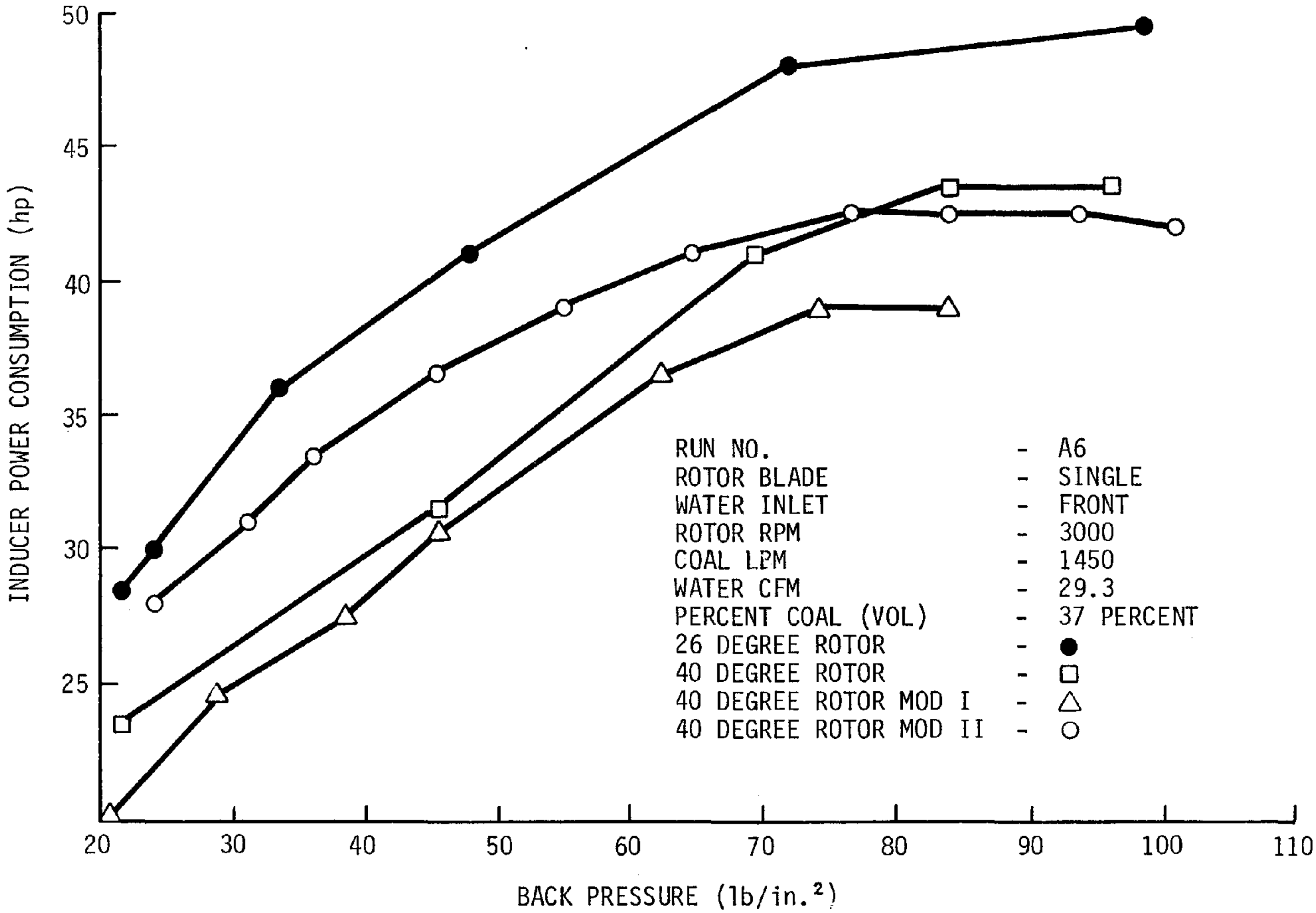


FIGURE 25. - Inducer proof tests, A6.

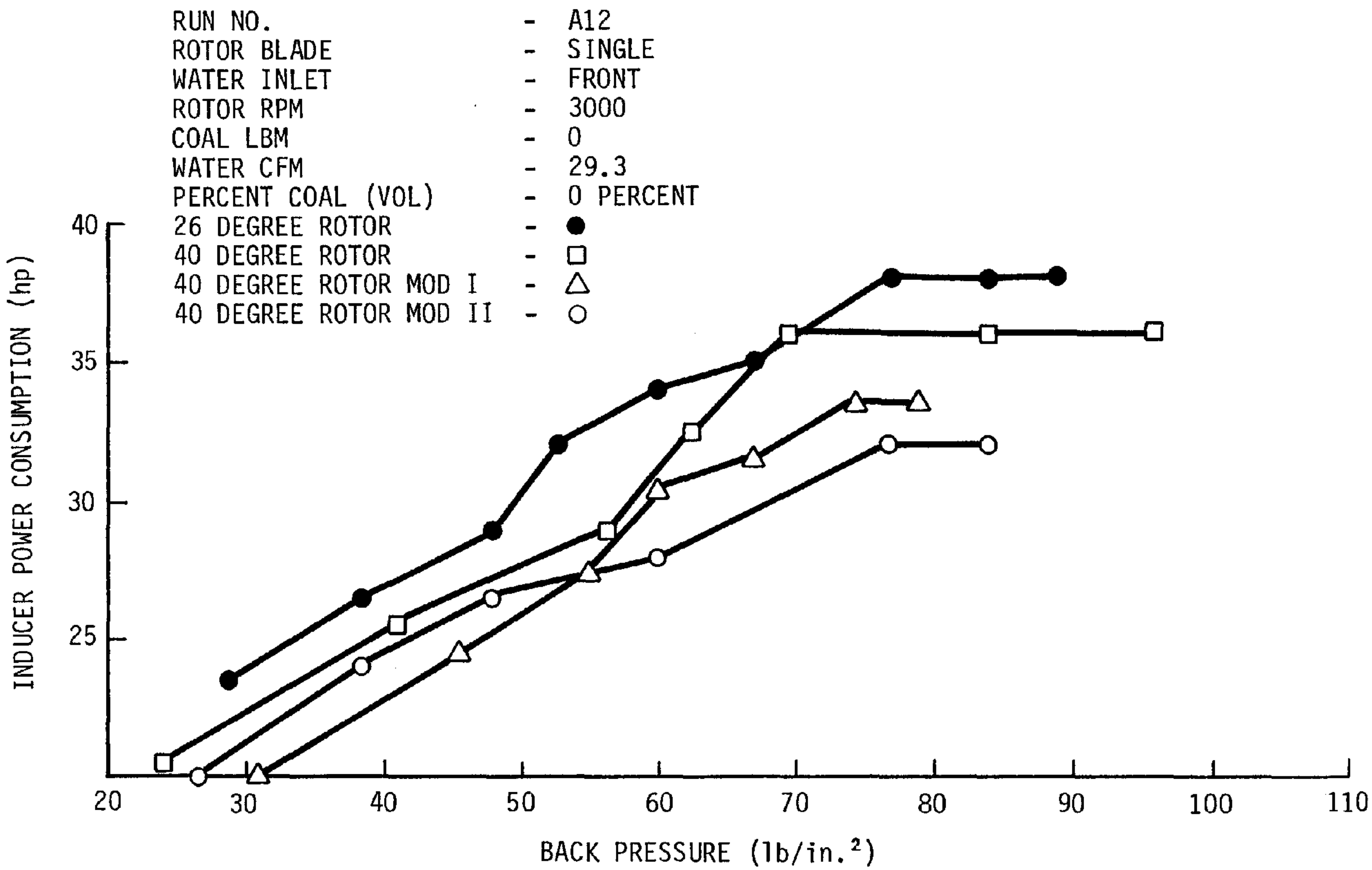


FIGURE 26. - Inducer proof tests, A12.

3.4.2 Feed Screws

Three screw feeder configurations were tested during the development program. Performance increased with each design from 1,200 lb/min of coal for the first screw to 2,000 lb/min for the final design.

Four-Inch-Diameter Single Flight Screw

This 4-in-diam screw with a 4-in pitch was limited to about 1,200 lb/min. It was operated up to 2,400 rpm to determine the optimum speed for maximum flow. Its peak performance of 1,200 lb/min was achieved at 1,300 rpm ± 100 rpm.

Five-Inch-Diameter, Double Flight Screw

This design also had a 4-in pitch. The extra flight was added with the intention of machining it off later to determine its effect on performance. The flights were canted forward to aid in filling the screw at high speed. The forward cant tends to hold the coal lumps in the screw rather than flinging them radially. The optimum flow of 1,500 lb/min was achieved at 1,600 rpm.

Five Inch Diameter, Single Flight Screw

This screw was the same as the previous screw with one flight removed. Removal of the flight caused flow to increase to 2,000 lb/min at the same 1,600-rpm rotational speed.

3.4.3 Horizontal Conveyor

A conveyor was designed and tested which delivers coal horizontally to the feed screw. Its purpose is to simulate coal delivery to the full-scale injector operating in low coal seams. A feed rate of 1,700 lb/min was tested with the conveyor belt surface slightly above the screw centerline and a rough, covered belt. Figure 27 is a simplified top view of the test setup. Figure 28 is a sectional view showing five geometries used in the evolution of the conveyor-screw barrel configuration. Maximum coal flow is printed below each geometry. Performance appeared insensitive to changing of screw speed above 1,500 rpm and conveyor speed above 550-ft/min. Conveyor power consumption was less than 2 hp and did not vary noticeably with coal rate.

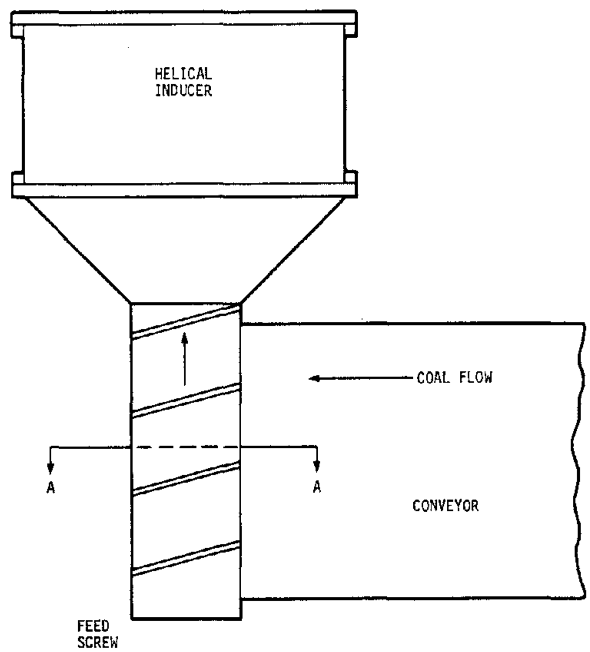


FIGURE 27. - Top view of horizontal conveyor.

3.5 MODEL SCALE INJECTOR TEST RESULTS

The majority of model scale injector tests were conducted using the single vane rotor (fig. 21d) and the 5-in, single flight screw. A test matrix was established which included the following:

a. Maximum coal flow and back pressure

b. A highly varied slurry concentration (by volume)

c. Front and back water operation

d. Runs with changes in only one variable.

Table 9 illustrates this test matrix which included 38 combinations of 3 rotor speeds, 2 inlet water locations, 5 coal flows (including zero) and 2 water flows. Results of each of these tests were reported in the Phase II report as individual performance curves giving inducer power and efficiency as a function of back pressure (pump discharge pressure). Four sets of injector characteristic curves were generated from the performance curves. These are discussed in the following subsection.

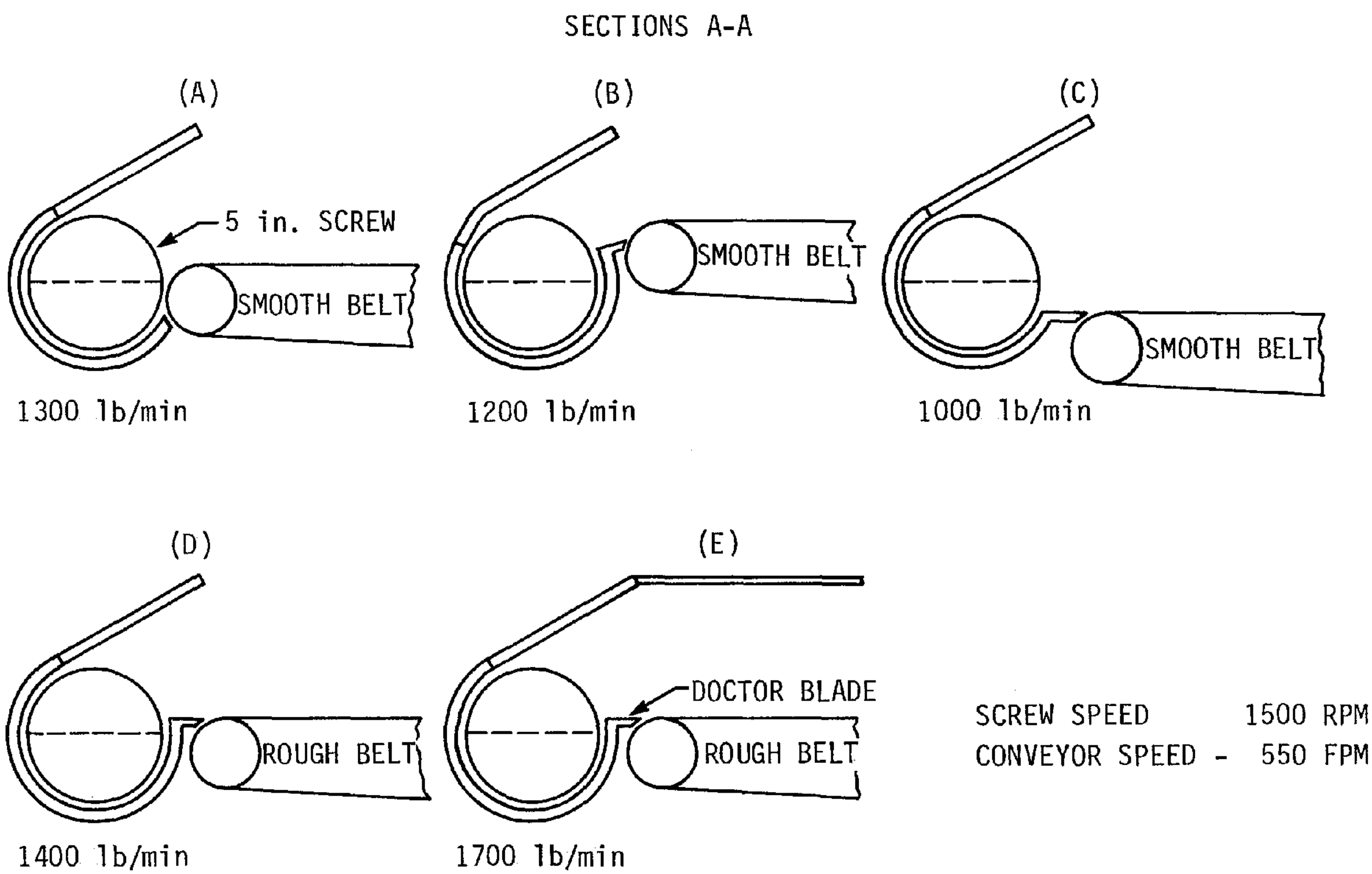


FIGURE 28. - Sectional views of horizontal conveyor.

TABLE 9. - Test matrix

Rotor Speed (RPM) Water Inlet (Location)		Coal Flow (LBM) Slurry Ratio (Percent Coal by Volume) Water Flow (CFM)									
		2000 LBM		1450 LBM		920 LBM		560 LBM		0 LBM	
		45 Percent 29.3 CFM	20.9 CFM	37 Percent 29.3 CFM	45 Percent 20.9 CFM	27 Percent 29.3 CFM	34 Percent 20.9 CFM	18 Percent 29.3 CFM	24 Percent 20.9 CFM	0 Percent 29.3 CFM	0 Percent 20.9 CFM
3000 RPM	Back			A1	A2					A3	A4
	Front	A5		A6	A7	A8	A9	A10	A11	A12	A13
2700 RPM	Back	A14		A15	A16					A17	A18
	Front	A19		A20	A21	A22	A23	A24	A25	A26	A27
2000 RPM	Back			A28	A29					A30	A31
	Front	A32		A33	A34		A35		A36	A37	A38

3.5.1 Injector Characteristics Curves

Figure 29 shows the maximum pressure and power consumption relationship to injector speed for 2,000- lb/min coal flow and 29.3-ft^3/min front water flow. The pressure curve confirms the speed squared relationship predicted by theory (see Appendix A). Figure 30 is a similar type plot but at 1,450 lb/min coal flow and with two water flows through front and back water inlet. The bunching of the maximum pressure curves shows the independence of maximum pressure to parameters other than speed. Power consumption for front water operation is 13 to 48% lower than for back water, the difference is greater at the lower speed range.

Figure 31 illustrates the independence of the injector's maximum pressure capability to coal flow and water flow. Injector discharge pressure is independent of flow and automatically adjusts to the system requirements.

Figure 32 shows power consumption versus coal flow for three speeds and pressures and two front water flows. At 35 lb/in^2, power for the injection of 2,000 lb/min of coal was 70% greater than for water only. At 90 lb/in^2, 2,000 lb/min of coal required 28% greater power than water alone. For 70 and 90 lb/in^2, doubling of the coal rate from 1,000 to 2,000 lb/min, required a 10% increase in power.

3.5.2 Injector Transients

Tests were conducted on the 3-in pipeline to determine the magnitude of pressure changes caused by sudden start-up and shut-down of coal flow rate, and whether an inlet water control valve is needed to operate under these conditions. Figure 33 is a plot of line pressure and flow during a sudden stoppage of coal injection and a sudden start-up of coal injection. The tests were run with a constant water flow, coal flow being the only variable. On increasing coal rate, the pressure pulse rises over a 1- or 2-s interval. The duration of the pressure pulse is short.

The decreasing coal rate produces a negative pressure surge of smaller magnitude than the positive surges. Pressure does not drop low enough for air ingestion by the inducer. Figure 33 is a good representation of the interaction of injector and line. A slurry line is not characterized by a value of pressure, but rather, the pressure varies significantly with the rate of coal injection, the change in rate of the injection, and the duration of the injection.

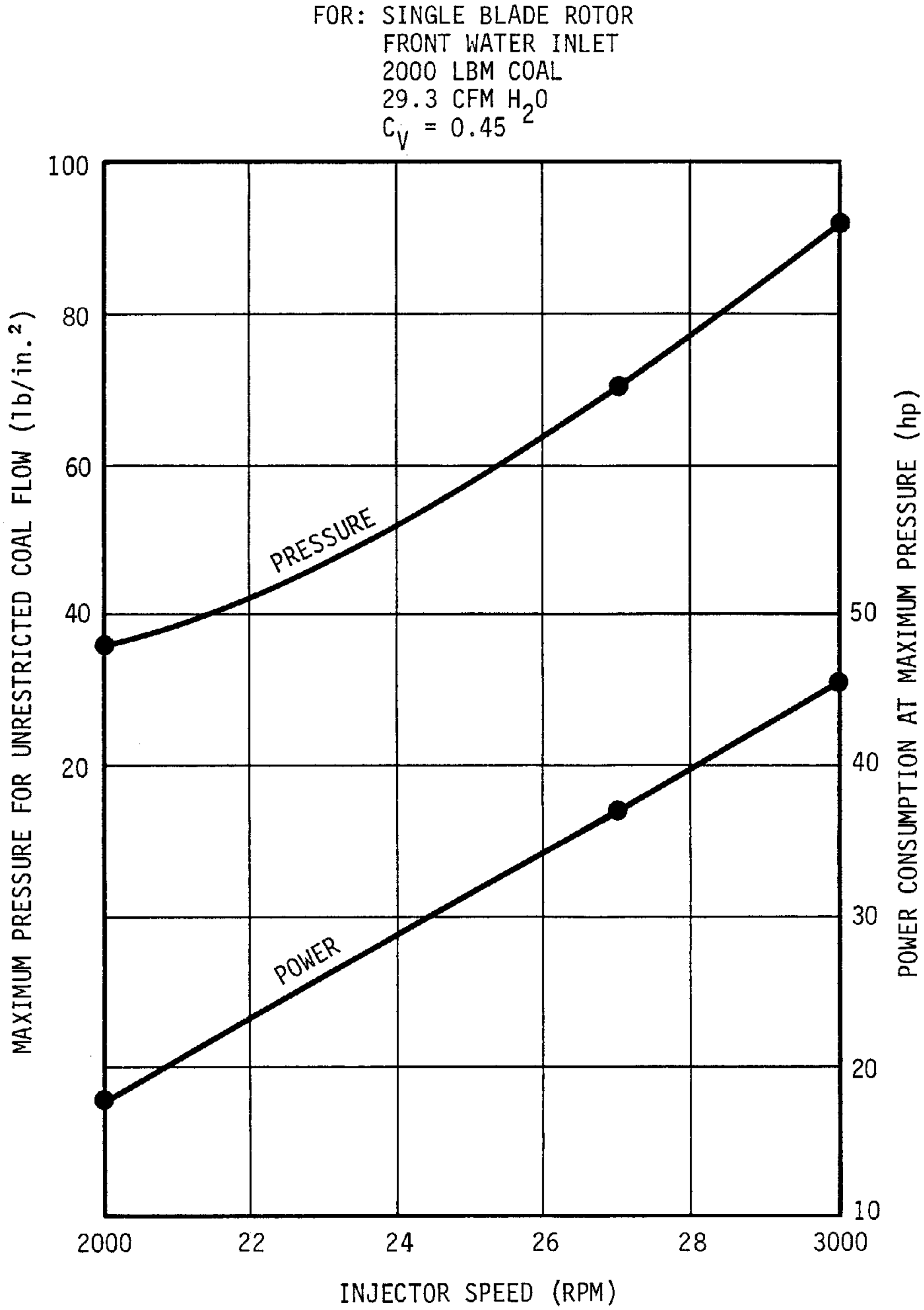


FIGURE 29. - Maximum back pressure and power consumption versus injector speed.

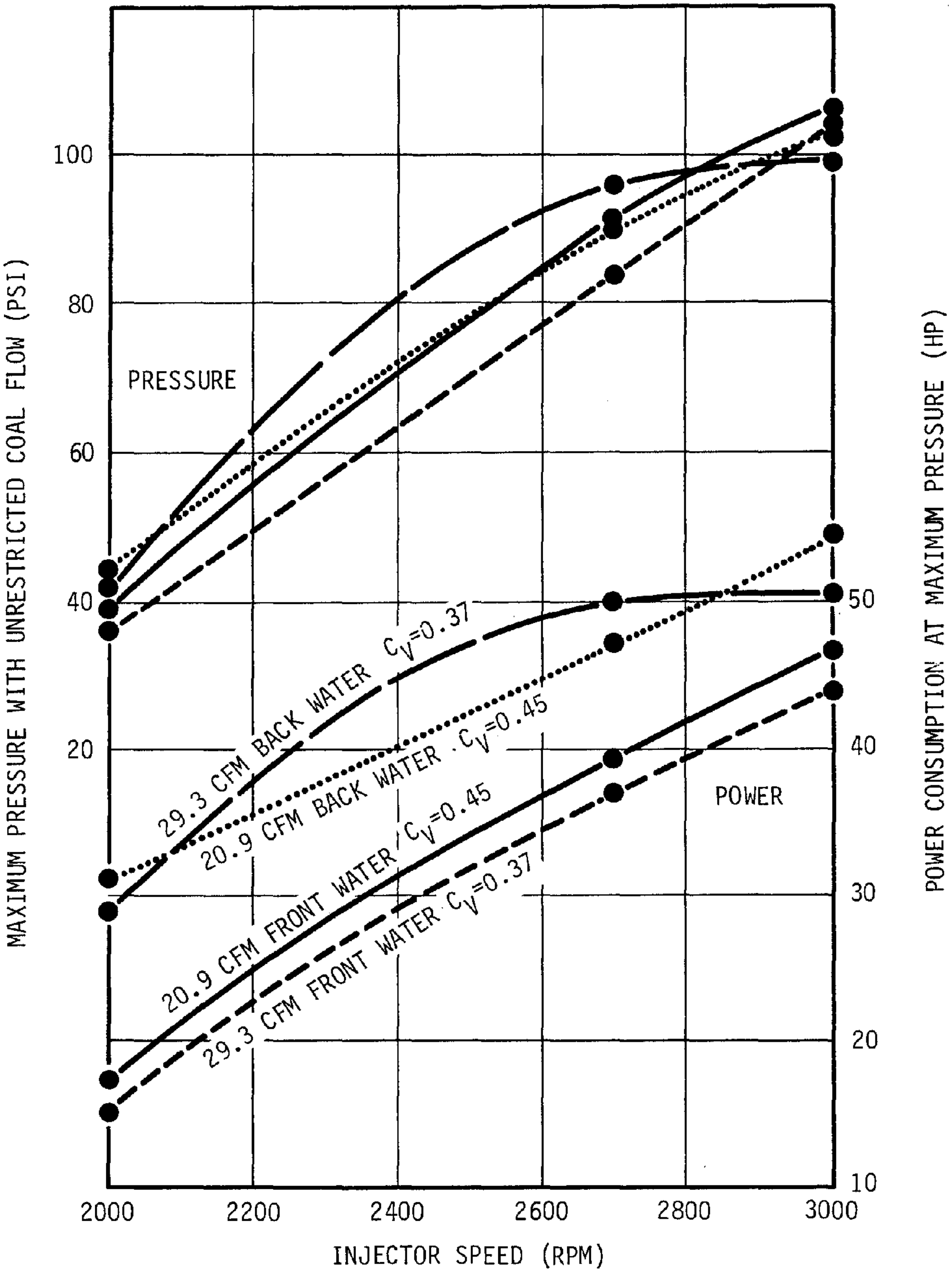


FIGURE 30. - Maximum back pressure and power consumption versus injector speed.

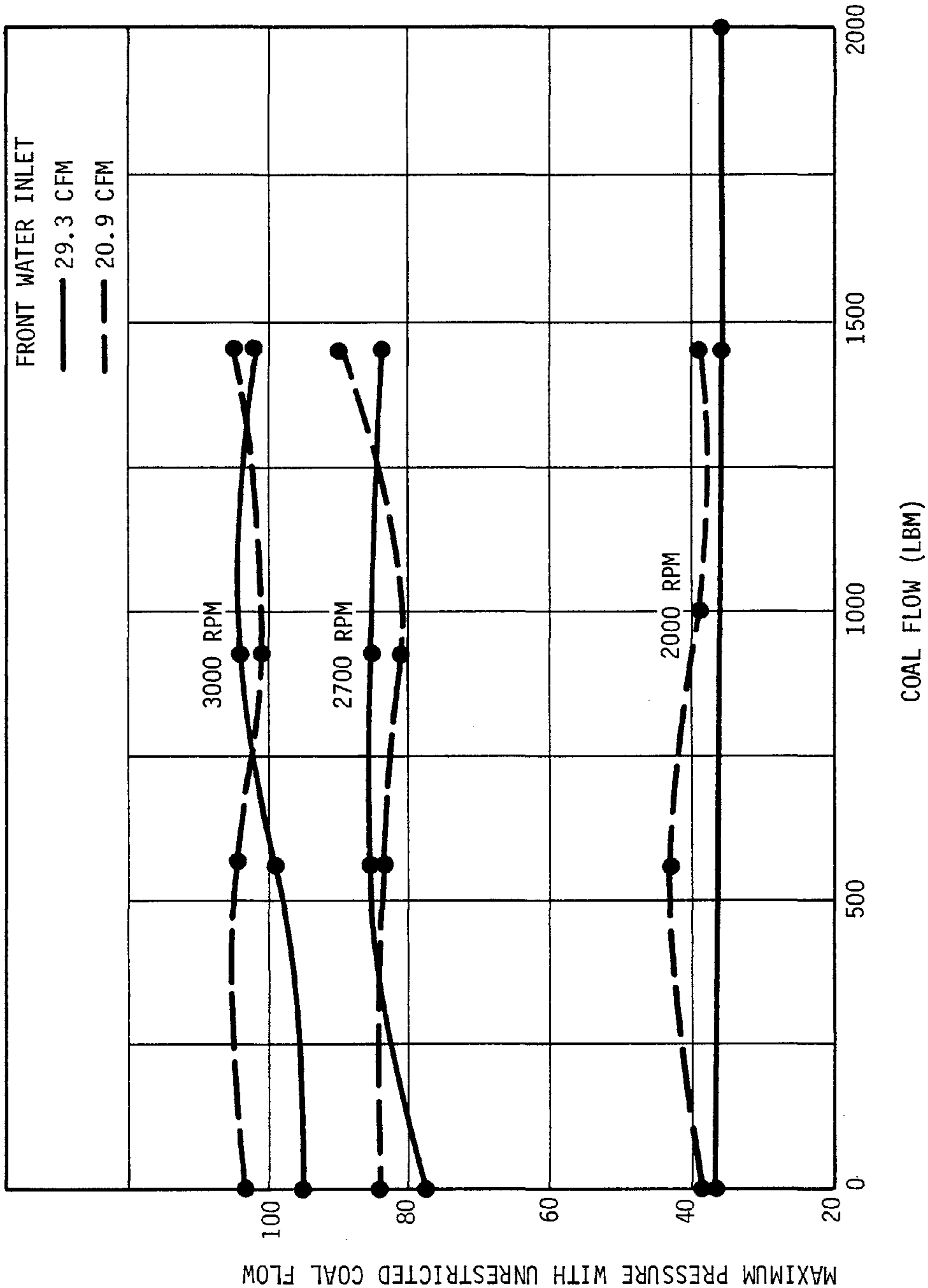


FIGURE 31. - Maximum pressure versus coal flow for single blade rotor.

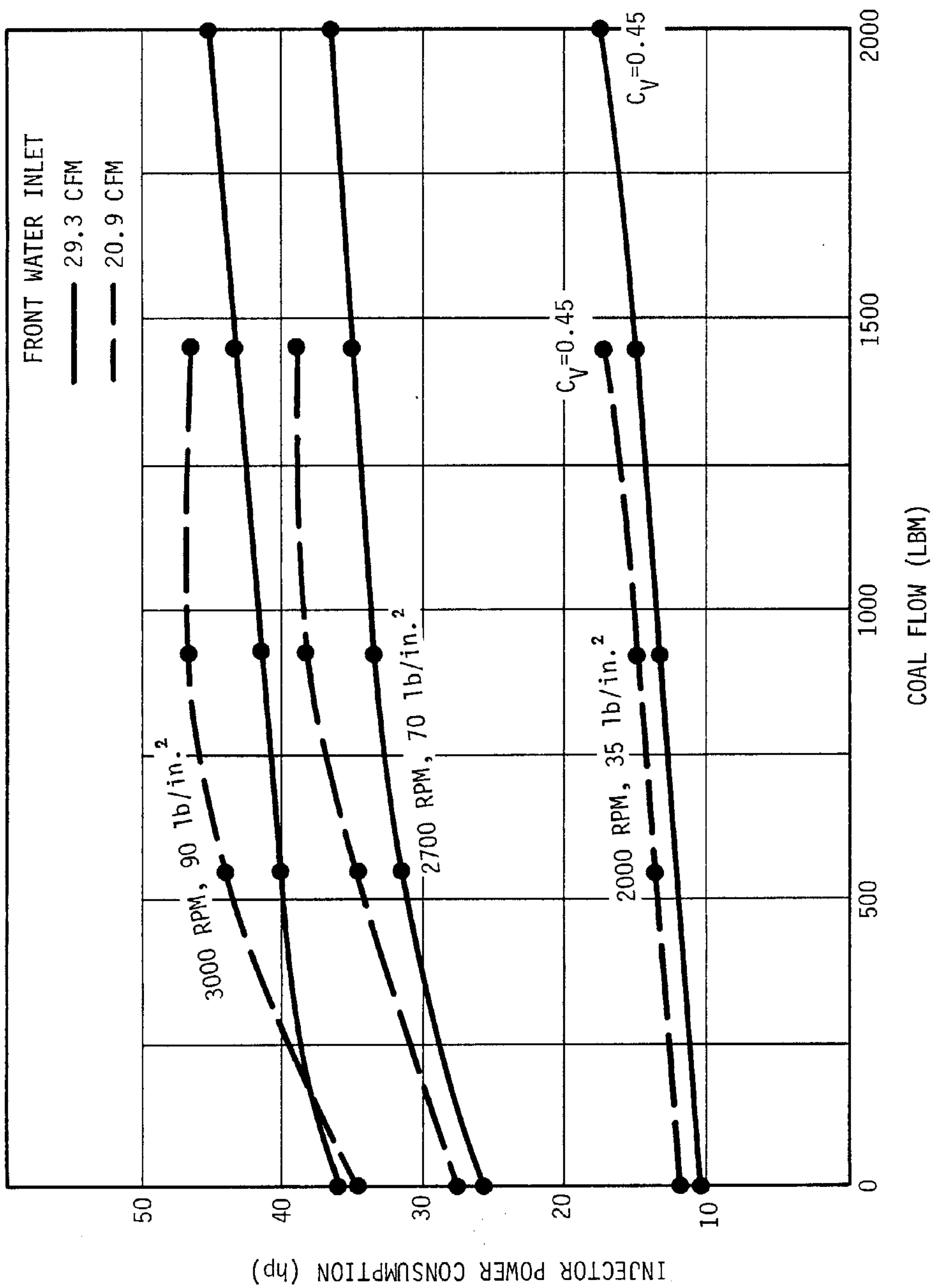


FIGURE 32. - Power versus coal flow for single blade rotor.

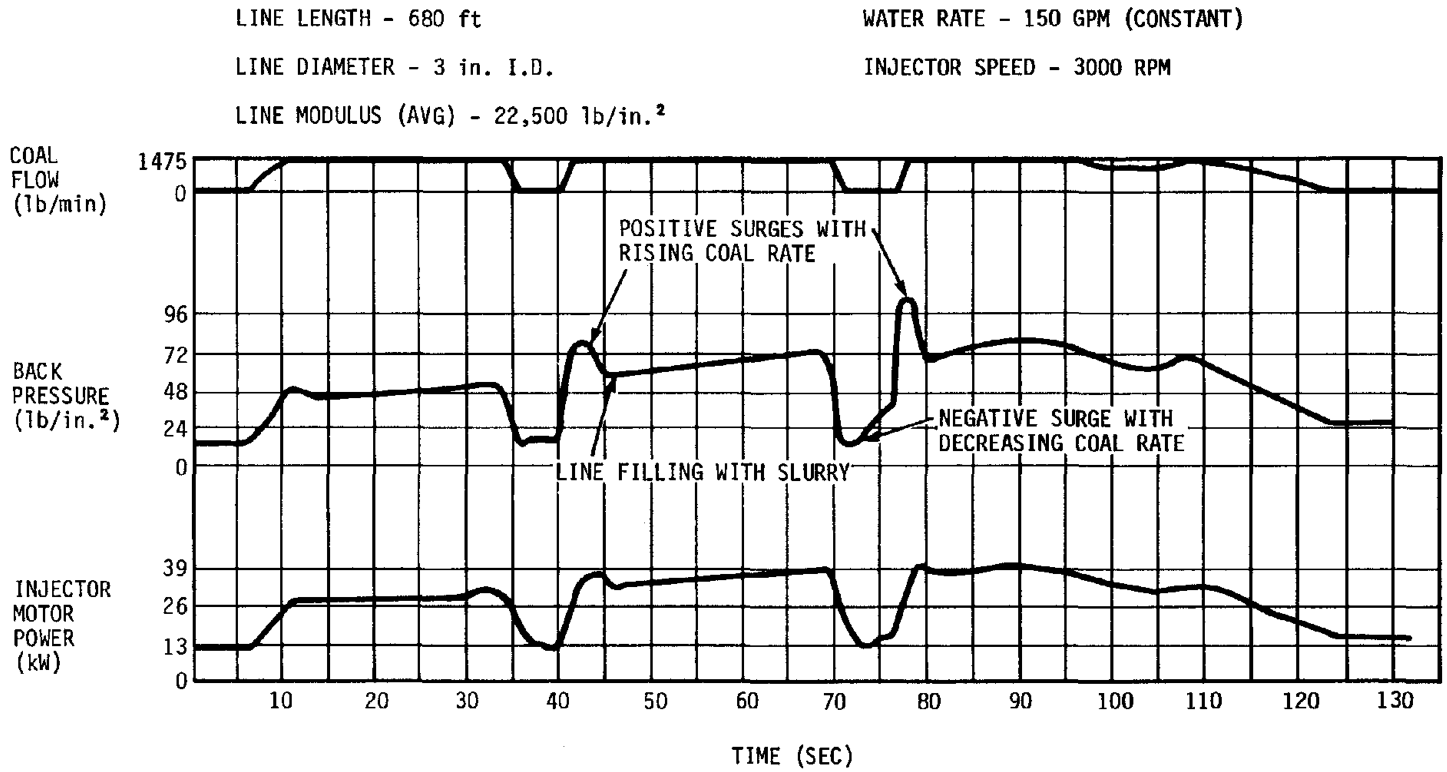


FIGURE 33. - Transient coal injection into slurry line.

Flow control may be applied to the inlet water to provide a more constant back pressure. However, it is impossible to maintain both back pressure and slurry flow. Experiments with water control and analysis of the entire face haulage subsystem are encouraging for the use of a face slurry line in which *flow is uncontrolled.*

4. PHASE III INJECTOR PROTOTYPE SCALE DEVELOPMENT AND TESTING

Before the injector could be incorporated into the vehicle design, extended testing of the feed screw injector assembly was conducted to validate performance and assure proper operation for integration into the vehicle assembly. During this portion of Phase III, the feed screw injector assembly was fabricated and mounted on an 8- by 16-ft test skid. The injector and feed screw skid was evaluated at an 8-in-diam pipeline slurry facility specifically constructed for this test program. The prototype injector and feed screw design, slurry test facility and test results are detailed in this section.

4.1 PERFORMANCE GOALS

The performance goals and operating specifics of the prototype injector and feed screw assembly are summarized in table 10. In addition to these quantitative goals, the injector pumping system is expected to:

TABLE 10. Prototype injector performance goals and operating specifics

Parameter	Full scale helical inducer for 4 ft entry
Coal injection rate	6.4 tons/min
Normal operating pressure lb/in.^{2}	80
Positive displacement pressure range lb/in.^{2}	50 to 100
Power - bhp	350
Rotor speed rpm	1000
Rotor tip speed - ft/sec	125
Maximum lump size - in.	3
Inlet	Dry coal screw feed
Purge	Centrifugal
Maximum slurry ratio by volume	50 percent
Coal degradation	Minimum
Trash injection	Coal with 30 percent and 100 percent refuse at 3.2 tons/min

a. Minimize coal degradation

b. Have a parts life requiring no additional maintenance over state-of-the-art pumping devices

c. Respond automatically to pipeline pressure requirements

d. Have no water leakage

e. Have no or minimal air ingestion

f. Design applicable to 18- to 24-in-diam pipeline

g. Handle peak coal surges to 9 to 10 tons/min.

4.2 PROTOTYPE INJECTOR-FEED SCREW DESIGN

The prototype injector and feed screw assembly as designed for installation in the vehicle is shown in figure 34. The injector's overall diameter or height is 33.5 in. It's 31.5-in-diam rotor is a single-vaned, scaled up version of the one-third scale model rotor shown in figure 21 e. As part of the rotor design, a computer program has been written to provide the templates for the vane of the helical rotor. This program is provided in appendix B. The feed screw is a 15-in-diam, single flight, 12-in pitch helix. Details of the injector's design are discussed in section 5.3.1.

The injector and feed screw assembly used for concept testing, figure 35, did not utilize the gearbox arrangement illustrated in figure 34. Specially designed bearing assemblies and supportive structures were fabricated for belt and sheave drives. This configuration provided the flexibility for operating both the injector and feed screw at various rotational speeds. Drive power to the injector was supplied by a 1,800-rpm, 350-hp, 1000-V motor. Drive power to the feed screw was supplied by a 900-rpm, 50-hp, 440-V motor. The gearbox arrangement in figure 34 is intended for use in the production model which would have a fixed speed.

Material of construction for pump rotor, cone, plenum and housing ranged from mild steel to ABEX HC-250. During the testing, various hard facings and coatings were applied to the rotor and cone. Details of material wear are discussed in section 4.4.

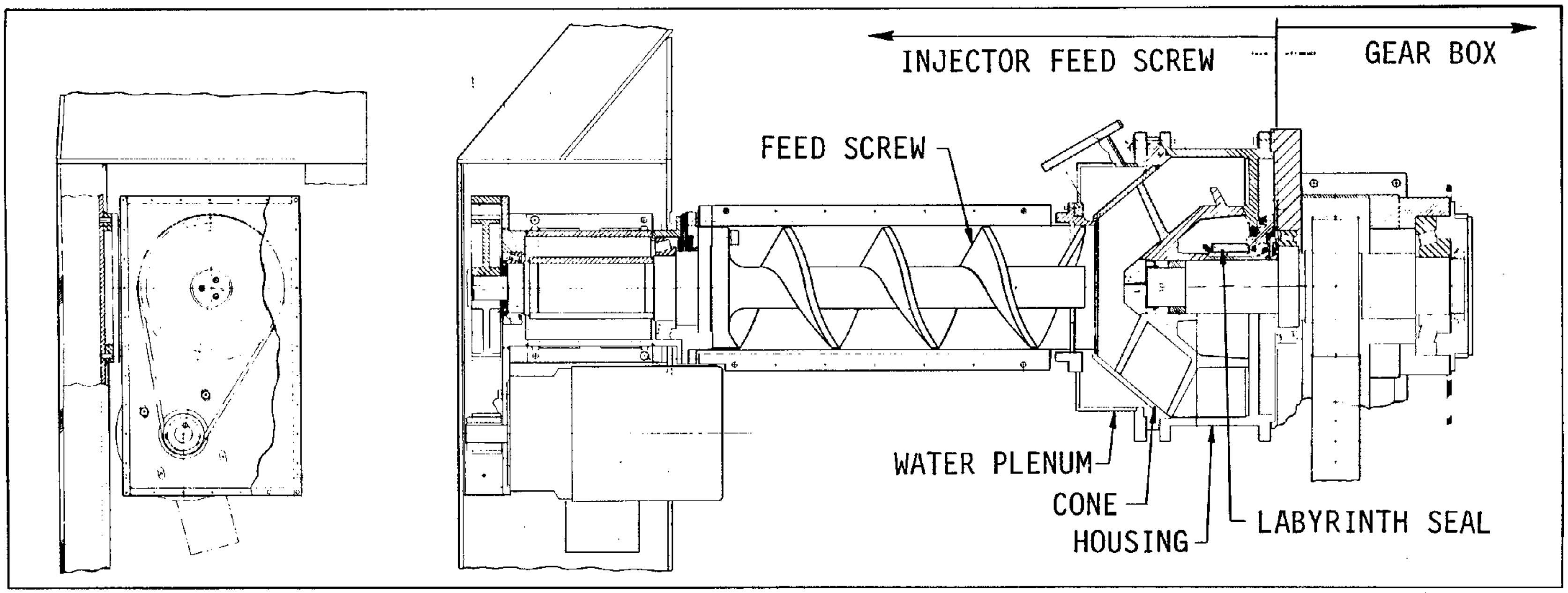


FIGURE 34. - Injector and feed screw assembly.

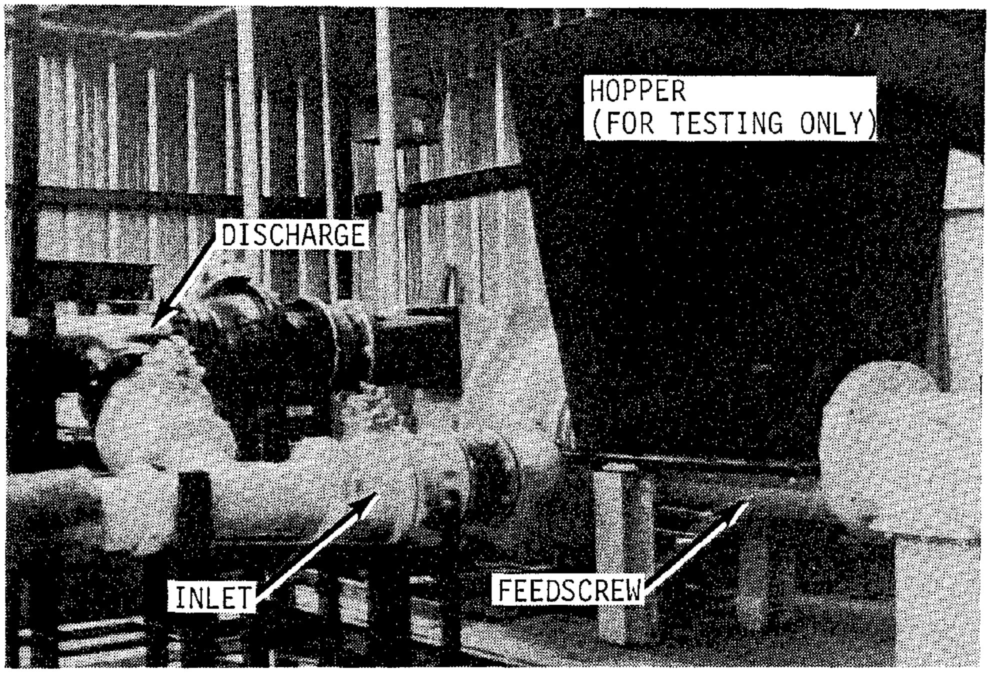


FIGURE 35. - Prototype injector and feed screw assembly for development testing.

4.3 FULL SCALE SLURRY TEST FACILITY

4.3.1 System and Equipment

A special 8-in-diam pipeline Slurry Test Facility (STF) was constructed at the FMI field test site at Saxonville, MA to evaluate the helical injector and follow on slurry injector vehicle. This 5 acre facility provides continuous coal and water flow at peak rates of 12 tons/min and 2400 gal/min, respectively. A layout of the STF including elevations is presented in figure 36. A conceptual layout which simplifies the flow paths is shown in figure 37. Major components include:

a. Pre-engineered clear span metal building
b. Helical injector skid (injector and feed screw)
c. Atmospheric transfer sump (ATS) skid
d. Transformer shed
e. Settling pond and clear water pond
f. Three 8-in-diam pipelines
g. One 6-in-diam pipeline
h. One 4-in-diam pipeline (water only)
i. Water pump
j. Water filter
k. Dewatering screen
l. Coal hopper

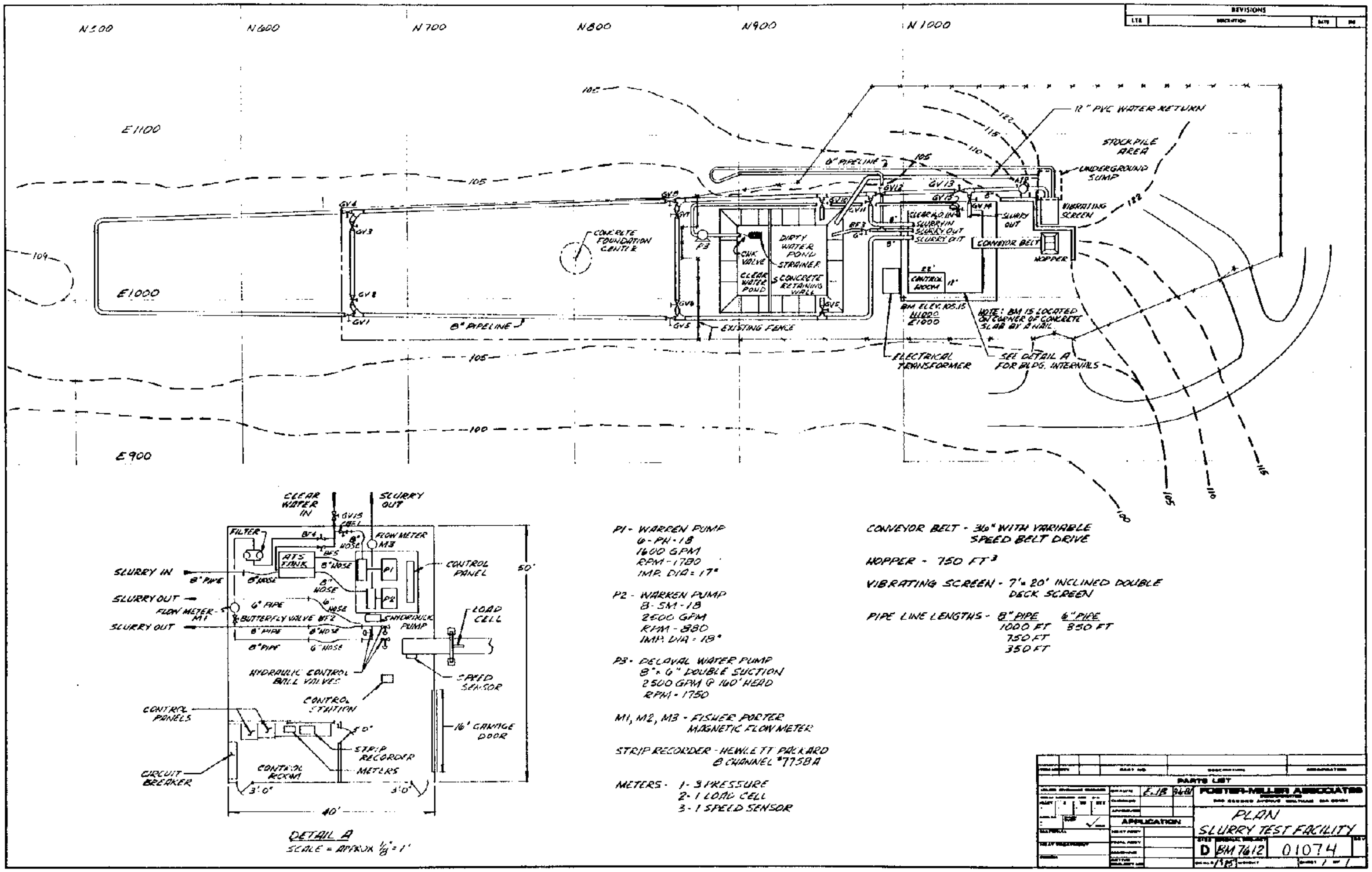


FIGURE 36. - Slurry test facility.

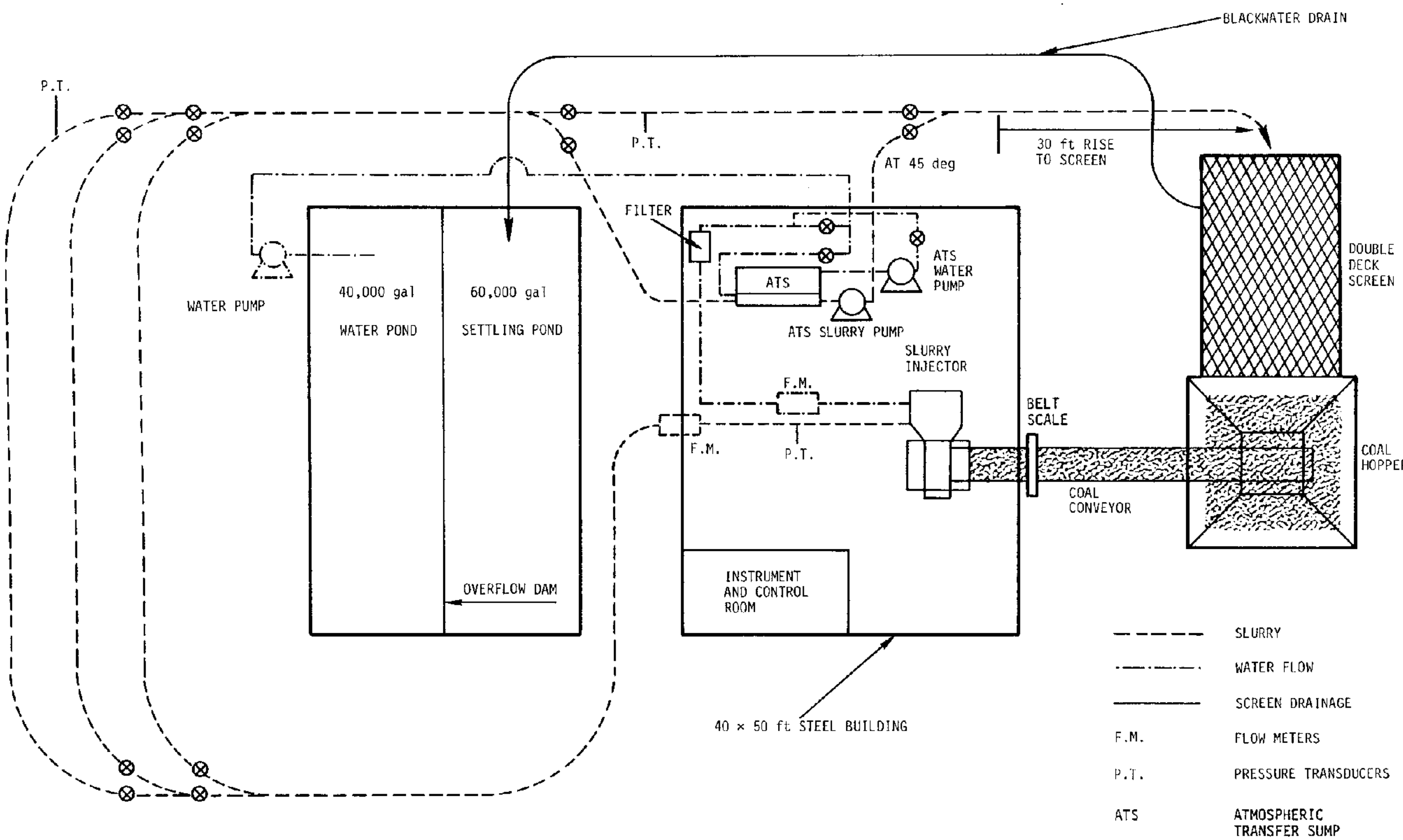


FIGURE 37. - Slurry test facility conceptual layout.

m. Feed conveyor
n. Controls and instrumentation
o. Front end loader
p. Two 6-in ball valves
q. One 8-in ball valve
r. Hydraulic power pack.

Constructed on a 60- by 60-ft concrete pad is a 40- by 50-ft pre-engineered clear span metal building which houses the helical injector skid, ATS skid and instrument/ control room. In operation, water stored in the 40,000 gal clear water pond is delivered to the injector by a centrifugal water pump located on the east side of the ponds. Prior to entering the injector, water is passed through a Haywood double basket filter and a Fischer-Porter magnetic flow meter to, respectively, remove plus 1/32 in trash and monitor flow. A series of Victaulic butterfly valves are used to regulate water flow to the injector.

Coal is delivered to the injector from a 25-ton coal storage hopper by a 50-ft long, 36-in wide traughing idler belt conveyor. The discharge portion of the 71^{o} rectangularly tapered coal hopper is equipped with:

a. A perpendicular-to-belt-travel, internal, A-shaped divider to minimize coal loading on the belt

b. A slide gate to regulate coal height on conveyor belt

c. An unloading gate to empty hopper.

The belt conveyor is equipped with a variable speed drive system that can be monitored and adjusted from the instrument/control room. A BLH load cell incorporated into the terminal (discharge end) support structure of the conveyor provides a reading proportional to coal weight on the belt. Load cell weight readings were determined to be 95% accurate based on samples cut from the belt.

By entering load cell reading, conveyor speed and desired coal flow into the STF "data generator," the operator can receive the required belt speed and make appropriate adjustments to maintain a relatively stable flow rate.

As slurry discharges from the injector, it passes through a second Fischer-Porter magnetic flow meter. The difference between this reading and the water flow meter provides coal flow rate by volume which can be verified by calculations from the belt weight data. During slurry operation both the injector power and feed screw power are monitored by analog AEMC watt meters which simultaneously measure voltage, amperage and phase angle.

On the water inlet side and slurry discharge side of the injector is a set of three KTM ball valves, one 8-in and two 6-in valves (see figure 35). Starting with the top horizontal valve and moving counterclockwise, they are: slurry discharge valve, bypass valve and water inlet valve. The bypass valve provides the means for continuing slurry flow should the injector fail. By sequentially opening the bypass valve, and closing the water inlet and slurry discharge valves, the pipe can be flushed of solids, eliminating potential pipeline plugging. Valve operation is accomplished with a 1000 lb/in^2 Sunstrand hydraulic power pack.

Slurry can be routed through any of three pipelines (300-, 700- and 1,000-ft). The slurry and water pipelines are constructed from ductile iron class 250 pipe. The bell and spigot mechanism for joining pipes provides a relatively smooth transition which eliminates flow disturbances that cause undesirable pressure drops. Pipe friction factor, as determined by water tests described in section 4, is 0.016.

Short sections of 8- and 6-in, ID schedule 40 steel pipe and hoses with Victaulic couplings were used to make final plumbing connections inside the injector building.

Line pressures, such as injector discharge pressure and intermediate pipeline pressures, were measured by recessed pressure transducers (figure 38). These pressure transducers were installed in the pipeline, 45^o from the top centerline, and never experienced solids plugging.

As slurry reaches the end of the pipeline it goes through a 40^o, 35-ft rise to an 8- by 20-ft incline double deck Allis-Chalmers dewatering screen with 3/4-in and 12-mesh cloth. Dewatered coal from this elevated screen discharges into the 25-ton coal storage hopper for reuse. Screen drainage (water plus coal fines) drains to the 60,000-gal settling pond where solids settle out and clear water overflows a north to south concrete dam into the clear water pond.

Prior to reaching the east wing of the injector building on its way to the dewatering screen, the slurry stream can be routed into the injector building where it discharges into the ATS. This sump's purpose was to provide the means for discharging a constant volumetric slurry flow rate while receiving a variable flow rate. Such a system would have been required if the helical

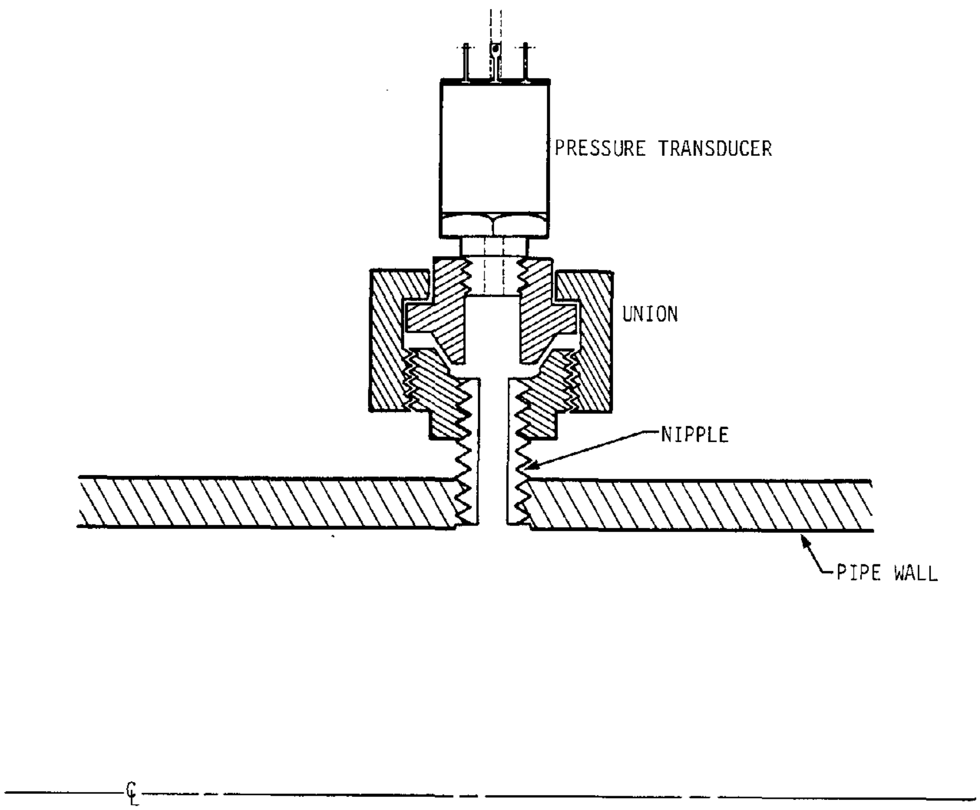


FIGURE 38. - Pipe mounted pressure transducer.

injector, which operates at a steady water flow and variable coal flow, was connected with a system of downstream boost pumps that requires a constant flow rate, such as centrifugal pumps. Details of this ATS system are discussed in appendix C. The adjusted slurry flow would be centrifugally pumped to the dewatering screen.

Adjacent to the dewatering screen on the southwest side is a flat storage area for holding the 300 tons of coal utilized during injector testing. Hopper loading was accomplished using the front end loader which discharged dry coal onto the operating dewatering screen. A 6-ft high chain link fence enclosed the STF from the coal storage area to the east side of the ponds.

4.3.2 Instrumentation, Controls and Power

Process data normally included:

a. Injector power

b. Feed screw power

c. Water flow

d. Slurry flow

e. Coal flow (by weight)

f. Injector discharge pressure

g. Pipeline pressure 1 } normally on a straight section of pipe

h. Pipeline pressure 2 } approximately 200-ft long.

These data were recorded on an 8-channel Hewlett Packard oscillographic recorder in the instrument room.

All units except the magnetic slurry flow meter worked well. This meter has received measured water accurately. When coal entered into the line the signal became meaningless. Eventually this problem was corrected by changing the flow tube source signal from an ac source to a dc source. This change produced a slight zero drift which was easily adjusted between runs.

Prior to making this flow meter change, the slurry flow meter was moved to the end of the slurry line just before the dewatering screen. At this location, any increase in flow at the injector due to coal addition would show up as a water only increase at the slurry flow meter. The flow difference between the water flow meter and the slurry flow meter was used to determine coal flow (by volume) until coal reached the slurry flow meter. Coal flow (by weight) was still measured by the conveyor speed and load cell readings.

Primary power for the entire facility is 13,000 V. This service was brought to the facility by underground cable. At the facility, the 13,000 V was reduced to 980 V and 460 V by two General Electric transformers rated at 500 and 750 kVA, respectively. Once in the control room, this dual service is fed to a distribution/breaker panel with further reductions of voltage to 210- and 110-Vac single-phase, and 100-Vac three-phase (for injector wattmeter) services.

The control panel monitors and controls all motor and valve operations. Remote controls are located on both the injector and ATS skids. Control circuitry includes ground fault protection. A detailed ladder-logic control circuitry diagram is included in appendix D.

4.4 HELICAL INJECTOR TESTING

The purpose of this test program was to determine the performance characteristics of the helical injector over a wide range of operating conditions. As a result of this test program, an extensive, coarse, coal slurry line loss data base was also established. A chronological description of this test program and details of the generated line loss data are presented in the following sections.

4.4.1 Chronological History of the Injector Test Program

Testing of the prototype helical injector began in June 1980 and ended in January 1983 with the installation of the injector into the slurry injector vehicle. During this period of time, numerous development and performance tests were conducted. Major program events are highlighted on a monthly basis in the following pages.

June 1980 through July 1980

The actual test program began on 11 June 1980 with the determination of the pipeline characteristics in terms of line lengths, elevations, k factors for bends and friction factors. The clean water pond pump and not the injector was used in this initial phase of the test program. The theory used in determining the k factors and friction factors, and the generated results are provided in appendix E.

The first actual injector tests were conducted with clear water in July. Minor vibrational problems with the injector/feed screw test bed, electrical problems with the injector breaker and ground fault system, and pipeline leaks were located, isolated and corrected. By the end of July, the pipeline had been characterized and the system shake-down completed.

August 1980

Initial injector performance tests were conducted with a clear water feed. A maximum discharge pressure of 95 to 100 lb/in^2 could be obtained at a pump speed of 1,000 rpm. Above this pressure, water began to discharge from the injector's ventilated inlet.

To gain operating experience with a totally operational facility, some short duration coal tests were conducted. During these initial coal tests, it was observed that the selected magnetic flow meters[1] gave very erratic readings when coal was passed through them. This virtually nullified the ability to determine coal feed rates. Since the flow meter manufacturer could not identify the cause of the erratic readings at this point

[1]Fischer & Porter Model 10D1435A.

in time, a weigh belt system was designed to measure the coal feed rate. Technical details relative to the weight belt design and its use are provided in appendix F. In the interim, before the feed belt system was installed, coal feed rates were estimated by taking belt samples.

On 21 August, the rotor bearing failed causing the injector to seize. When the injector was disassembled, the bronze labyrinth seal was found to be cracked and excessively worn. The rotor bearing was also cracked. A new shorter labyrinth seal was fabricated and a replacement bearing with a looser radial tolerance was ordered.

September 1980

The new labyrinth seal and rotor beaing were installed and testing resumed. Maximum pressure tests with clear water again indicated a maximum discharge pressure of 97 lb/in^2 at 278 motor input horsepower.

Coal tests were run using the 300-ft slurry loop. Coal degradation from a 3-in top size to a 1-1/2 in top size occurred in about 5 min of operation at coal feed rates of 3 tons/min. At this rate of degradation, constant system adjustments were required to maintain a steady state operation and more belt samples were required to estimate coal feed rates. There was no way of determining whether the coal degradation was caused primarily by the helical injector or by the slurry system network.

During testing, a rubber transition sleeve which coupled the injector outlet to the slurry discharge valve wore through. To protect the replacement sleeve from coal abrasion, a metal insert was fabricated out of 3/16 in sheet steel and installed with the new rubber transition sleeve.

<u>Summary of September 1980 Coal Tests</u>

Line length - 300 ft
Concentration by volume - 9 to 34%
Coal feed rate - 1.5 to 6.8 tons/min
Average coal feed rate - 3.1 tons/min
Total tonnage to date - 462.3 tons
Total slurry time to date - 150.0 min
Average total coal feed rate - 3.1 tons/min

October 1980

During October, the weigh belt system was installed and the slurry flow meter repositioned at the end of the pipeline near the dewatering screen. The slurry meter was moved far enough downstream so that it could detect the increased volumetric water flow rate due to the addition of coal at the injector. Once slurry reached the flow meter, the signal became useless. Both the weigh belt system and downstream flow meter gave reasonable results when compared to a weigh belt sample derived flow rate.

The six single belts driving the injector were replaced by two pairs of common-backed belts. This eliminated the belt twisting problem which began in September. Differential pressure gauges were installed on pipe bends to measure pressure losses. Screen analysis of coal samples was also initiated during this month.

The MSHA-approved permissible 350-hp Louis-Allis motor driving the injector caught fire. An inspection of the motor showed that the motor shaft was slightly bent causing the in-board bearing rubber seal to rub, overheat and catch fire. Since the shaft run-out was not sufficient to cause motor damage, the rubber seal was removed and testing resumed. Corrective action was postponed to a later date.

Summary of October 1980 Coal Tests

Line length - 300 ft
Concentration by volume - 21 to 45%
Coal feed rate - 3 to 10.2 tons/min
Average coal feed rate - 6.4 tons/min
Total tonnage to date - 729.3 tons
Total slurry time to date - 192.0 min
Average total coal feed rate - 3.8 tons/min

November 1980

Coal tests continued in November and after a total of 1713 tons of coal were passed through the injector, the maximum discharge pressure had dropped to 80 lb/in^2. Upon disassembling the injector, it was found that the rotor and cone, which were cast from 4,130 steel, ASTM-1248-90-60, showed excessive wear on the edges and face respectively. The rotor edges were not only worn down but also had very deep gouges. The rotor tip had lost several inches from its leading edge and the cone had ripples along its face. The gap between the rotor and the cone at the start of testing was less than 0.125 in. The gap after 1,713 tons of coal was in excess of 0.5 in.

Summary of November 1980 Coal Tests

Line lengths - 300 and 700 ft
Concentration by volume - 18 to 46%
Coal feed rate - 3.1 to 11.1 tons/min
Average coal feed rate - 5.4 tons/min
Total tonnage to date - 1713.5 tons
Total slurry time to date - 378.4 min
Average total coal feed rate - 4.6 tons/min

December 1980

In December, a 6-in-diam steel line was installed for hydrotransport testing under a second USBM contract. The helical injector was used to pump crushed gravel slurry through the 6-in line to obtain line loss data.

After pumping 40 tons of gravel, the gap between the rotor and cone increased from 1/2± to 7/8± in and the maximum developed discharge pressure dropped from 80 to 50 lb/in². The injector was disassembled and the rotor and cone were shipped to a machine shop for rebuild with a Stellite hardfacing.

Summary of December 1980 Gravel Tests

Line length - 300 ft 6 in ϕ
Concentration by volume - 10 to 19%
Stone feed rate - 1.2 to 4.0 tons
Total crushed stone tonnage - 39.6 tons
Total crushed stone slurry time - 22.5 min
Total average crushed stone feed rate - 1.8 tons/min

January through May 1981

The slurry test facility was shut down for the winter months of January, February and March. In March, testing was conducted to determine particle drag coefficients as a function of particle size and type (coal or stone). During April and May, the injector rotor and cone were rebuilt, hardfaced and matched to provide a small gap when reassembled.

June 1981

The Stellite faced rotor and cone were received from the fabricator and the injector was reassembled. The cone had been slightly warped during fabrication so that the gap between the rotor and cone varied from 0.297 to 0.094 in.

With this rebuilt injector, a maximum discharge pressure of only 80 lb/in² could be attained. In spite of this, a decision was made to proceed with coal tests. After 282.1 tons of coal, most of the Stellite hardfacing was worn away with the ripple patterns reappearing on the cone and the deep gauges reappearing on the rotor edges. The maximum pressure remained at 80 lb/in².

A clear plastic section of pipe was placed in-line in an effort to identify the different flow regimes for coal slurry. What was discovered instead was that the helical injector was entraining air. A new cone with chrome oxide hardfacing was ordered and the existing rotor was rehardfaced with Stellite. This change was made to see if the warped cone caused air entrainment.

Coal testing that was conducted indicated that -3/4 in material was transported in a heterogeneous flow regime with the material concentrated in the lower two-thirds of the pipeline; larger material was not run during this period of time.

Experimental zirconium electrodes were installed in the slurry flow meter just downstream of the injector. The meter, however, still put out an erratic signal when operating with a coal slurry. At this time, Fischer & Porter representatives suggested the slurry meters were seeing signal noise and that they should be converted to an ac excited meter, their standard Model 10D1430. The Model 10D1435A meters, which were presently in-line, were dc excited meters and although they require less power, they are susceptible to signal noise. The conversion units were ordered.

<u>Summary of June 1981 Coal Tests</u>

Line length - 700 ft
Concentration by volume - 16 to 33%
Coal feed rate - 1.9 to 6.2 tons/min
Average coal feed rate - 3.7 tons/min
Total tons on first Stellite rotor - 282.1 tons
Total slurry time on first Stellite rotor - 75.8 min
Total average coal feed rate on first rotor - 3.7 tons/m

September 1981

The Stellite refaced rotor and the new chrome oxide faced rotor were installed. The gap between cone and rotor ranged from 0.107 in at the back to 0.018 in at the tip. The maximum developed discharge pressure was 94 lb/in².

Air entrainment was still observed, and was not considered a characteristic of the injector. For test purposes, four air removal chambers were placed in-line and were used to bleed air from the pipeline during testing.

In an effort to better understand the air entrainment problem, tests were conducted on the 3-in helical injector. The results indicated that air entrainment was a function of discharge pressure. Entrainment was minimized at higher discharge pressure. It was also noted that when coal was fed to the injector, air entrainment was minimized. Additional study of these phenomenon are to be made under the Bureau contract No. J0133934.

After pumping 478 tons of coal, the rotor and cone showed high wear in the rear-high velocity area of the injector. Wear measurements varied from 0.119 in at the rear to 0.020 in at the tip. The next alternative, based on a survey of state-of-the-art materials, was to cast the rotor and cone from HC-250. However, due to long lead times for this material, it was decided to have the existing rotor hardfaced with a vanadium carbide material and the existing cone hardfaced with a tungsten carbide. This option would provide quick turnaround hardware for further testing. A new rotor and cone using HC-250 was also ordered for installation in the slurry injector vehicle.

Summary of September 1981 Coal Tests

Line length - 700 ft
Concentration by volume - 22 to 43%
Coal feed rate - 4.3 to 8.2 tons/min
Average coal feed rate - 5.3 tons/min
Total tonnage on new rotor - 478.5 tons
Total slurry time on new rotor - 90.4 min
Toatl average tonnage on new rotor - 5.3 tons/min

October 1981

In October, the vanadium carbide rotor and tungsten carbide cone were received and installed. Great care was taken to minimize the rotor-cone gap. The gap ranged from slightly touching to 0.017 in at the tip and the rear of the rotor respectively.

November 1981

Coal testing was resumed in November. After 500 tons, the rotor to cone gap measurements ranged from 0.017 in at the tip to 0.017 in at the rear and there was evidence of slight ripples and pitting on the cone. After 1,000 tons, the rotor-cone gap measurements ranged from 0.018 in at the tip to 0.040 in at the rear and the cone coating was worn in the high velocity zone. The point, which had been touching at the start of the testing, now had a 0.019 in gap.

A total of 2,391.6 tons were pumped using the vanadium carbide rotor and tungsten carbide cone. The maximum developed discharge pressure remained at 100 lb/in². Air entrainment was still present.

Several of the coal tests were continuous for 60 to 100 min. Slurry flow conditions were varied from 4.2 tons/min at $C_V = 22\%$ to 8.5 tons/min at $C_V = 43\%$ The helical injector adjusted to all of these flow conditions automatically.

A power outage during the $C_V = 30\%$ coal test provided an unexpected opportunity to observe the sediment transport phenomena as the material went from a stationary bed to a fully developed heterogenous slurry. When power was restored, various water flow rates were passed through this partially filled pipeline. The following observations were made through the in-line clear plastic pipes:

a. Approximately, one-half the pipe was filled with coal characterized by the gradation curves given in figure 39.

b. At 100 to 170 gal/min, the 8-mesh material started saltating.

c. At 640 gal/min, the -3/8 in material went into suspension and the +3/8 material started rolling.

d. At 480 gal/min, the -3/8 in material started coming out of suspension and the larger material was on the stationary bed.

e. At 920 gal/min, -3/8 in material went into a heterogeneous suspension, the larger material starting to saltate and the entire bed starting to slide.

f. At 1,200 gal/min, a fully developed heterogeneous flow was achieved.

<u>Summary of November 1981 Coal Tests</u>

Line lengths - 700 ft 8 in ϕ and 300 ft 6 in ϕ
Concentration by volume - 22 to 48%
Coal feed rate - 3 to 9.3 tons/min

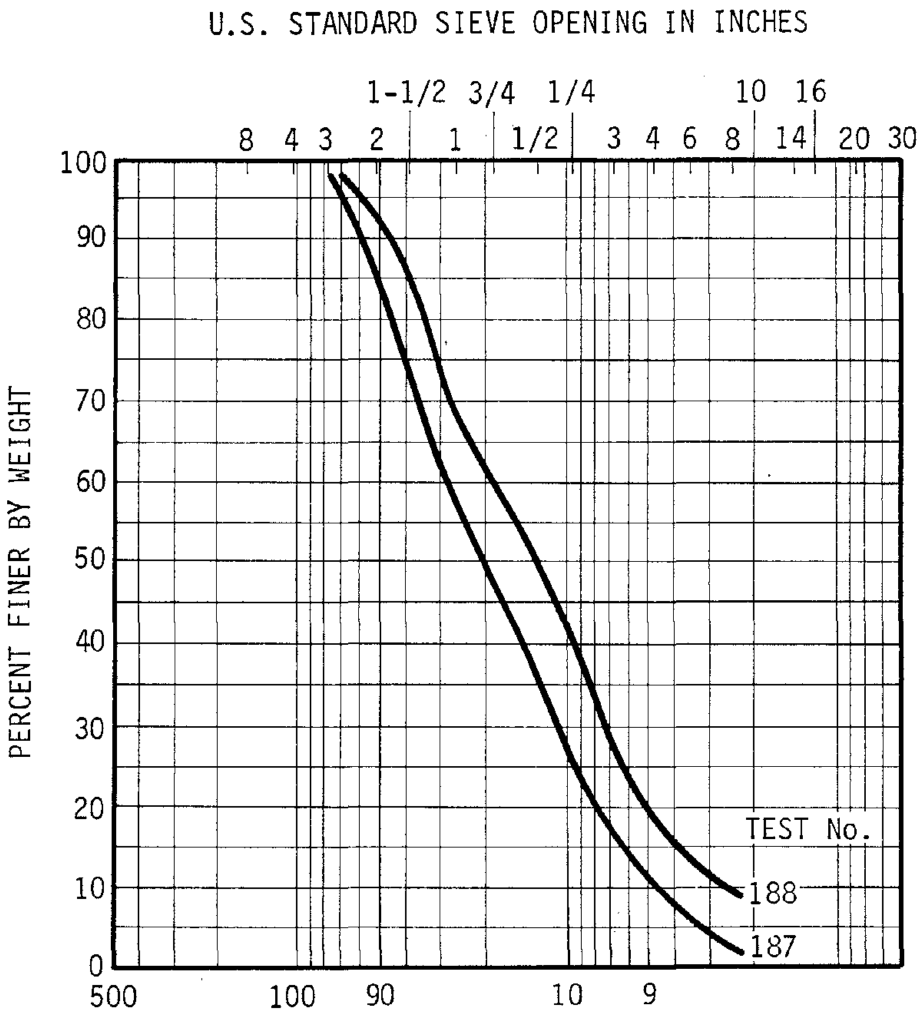


FIGURE 39. - Material gradation for sediment transport observations.

Total tons on new rotor - 2,391.6 tons
Total slurry time on new rotor - 406.8 min
Average total feed rate - 5.9 tons/min

December 1981 to December 1982

During this period, the slurry test facility was used to evaluate a jet pump injector under a separate government contract. Coal degradation rates were comparable to those produced during helical injector testing.

The slurry injector vehicle was assembled and debugged in the FMI shop during this period of time.

December 1982

The final helical injector tests were run as part of the slurry vehicle test program. Vehicle test results are presented in subsection 5.3. A couple of changes were made to the helical injector installed in the vehicle. The rotor and cone were fabricated from HC-250. A gear reducer was used in place of belts to drive the helical injector.

The maximum developed discharge pressure with water only was 108 lb/in^2. Air entrainment was still present. A total of 186 tons of coal were pumped through the vehicle-mounted injector. The limited quantity of coal was due to a bearing failure on the delivery conveyor to the injector feed screw. Further testing will be required to determine the wear characteristics of the HC-250 cone and rotor. A summary of rotor and cone materials, and results obtained with each is provided in table 11.

TABLE 11. - Summary of rotor and cone material

Material	Tons of material	Maximum pressure at start and finish ($lb/in.^2$)	Comments
Rotor - 4130 steel	Coal - 1713	100 at start	Deep gouges in rotor
Cone - 4130 steel	Crushed stone - 39.6	80 at finish	Deep ripples in cone 7/8± in. rotor-cone gap
Rotor - Stellite	Coal - 282	80 at start	Cone warped in fabrication
Cone - Stellite		80 at finish	Gouges reappeared on rotor and cone
Rotor - Stellite	Coal - 478.5	94 at start	High wear in the rear high velocity
Cone - Chrome oxide		94 at finish	Area of the inducer rotor cone gap in this area went from 0.017 in. at the start to 0.119 in. at the end. Ripples and gouges reappeared
Rotor - Vanadium carbide	Coal - 2391.6	100 at start	Wear pattern same as previous tests with rear rotor-cone gap going from 0.017 to 0.040 in. after 1000 tons.
Cone - Tungsten carbide		100 at finish	
Rotor - HC-250	Coal - 186.2	108 at start	Not enough coal pumped to determine any wear patterns
Cone - HC-250		108 at finish	

The Fischer & Porter magnetic flow meters were finally converted to the 60-cycle ac excited Model 10D1430 and worked very well in the coarse coal slurry application. Calculated concentrations from the water and slurry flow meters were within 2% of the weigh belt readings.

Summary of December 1982 Coal Tests

Line length - 8 in at 800 ft
Concentration by volume - 20 to 45%
Coal feed rate - 3.4 to 11.0 tons/min
Average coal feed rate - 6.40 tons/min
Total tonnage on HC-250 parts - 186.2 tons
Total slurry time on HC-250 parts - 29.1 min

4.4.2 Injector Performance Tests

A summary of all tests and test results are tabulated in Appendix G. Theory, techniques and programs used in generating this data are presented in Appendices H through M. These appendices include:

a. Specific gravity calculations

b. Line loss calculations

c. Injector efficiency calculations

d. Particle gradation calculations

e. Drag coefficient calculations

f. Data reduction programs for HP-95 and HP-34-C calculators.

Injector performance was evaluated at a fixed rotor speed of 1,000 rpm; pumping clear water over a range of injector discharge pressures; and pumping coarse coal slurries at various discharge pressures. The injector discharge pressure requirements were altered by varying the coal slurry concentrations; changing the injector water supply flow rate or the dry coal injection rate; and by changing the discharge pipeline length.

Hydraulic and mechanical performance of the injector, injector component wear characteristics, and coarse coal pipeline performance characteristics were documented throughout the injector development program. The results of these performance tests are presented in this section.

4.4.2.1 Hydraulic and Mechanical Performance of the Injector

The operating range of the full-scale helical injector, which was developed and evaluated in this program, was effectively limited by the maximum speed at which the rotor was operated and by hydraulic constraints imposed by the test apparatus and pipeline network. The injector was operated in two distinct modes:

a. Pumping water only; water was fed into the ventilated injector at various flow rates; the water entered through the annular ring at atmospheric pressure; back pressure was varied by a control valve on the discharge pipeline.

b. Dry coal feed - slurry injection; water was supplied to the injector at various flow rates as with the water only mode of operation; coarse dry coal was delivered at various rates to the injector annulus by the axial mounted feed screw; back pressure was varied by changing the discharge line length.

All performance tests were operated at a fixed rotor speed of 1,000 rpm. A summary of the performance test data for both modes of operation is presented in figure 40. The minimum output flow of the centrifugal pump which supplied water to the injector was 1,000 gal/min. No flows below this were tested. However, the helical injector appears to have no lower flow rate bound since the volume throughput is a function of the back pressure and the size of the resulting vortex core. It is significant to note that the hydraulic efficiency of the 1,000 rpm helical injector would drop below 15% at flow rates below 1,000 gal/min. This can be seen in figure 41. Efficiency increases at higher flow rates. The clear water performance plot in figure 42 clearly shows this characteristic. The maximum supply flow rate to the injector was 2,300 gal/min. This upper test bound was also limited by the centrifugal pump which supplied water to the injector.

The lower bounds, indicated on both figures 40 and 42, for the water only tests, were determined by the pressure required to move the given water flow through the shortest unrestricted pipeline in the hydraulic test loop. It is clear that the helical injector would operate within the "system limited range" of figure 40 if the discharge pipeline was shortened. This same limitation was experienced with the coarse coal slurry performance tests.

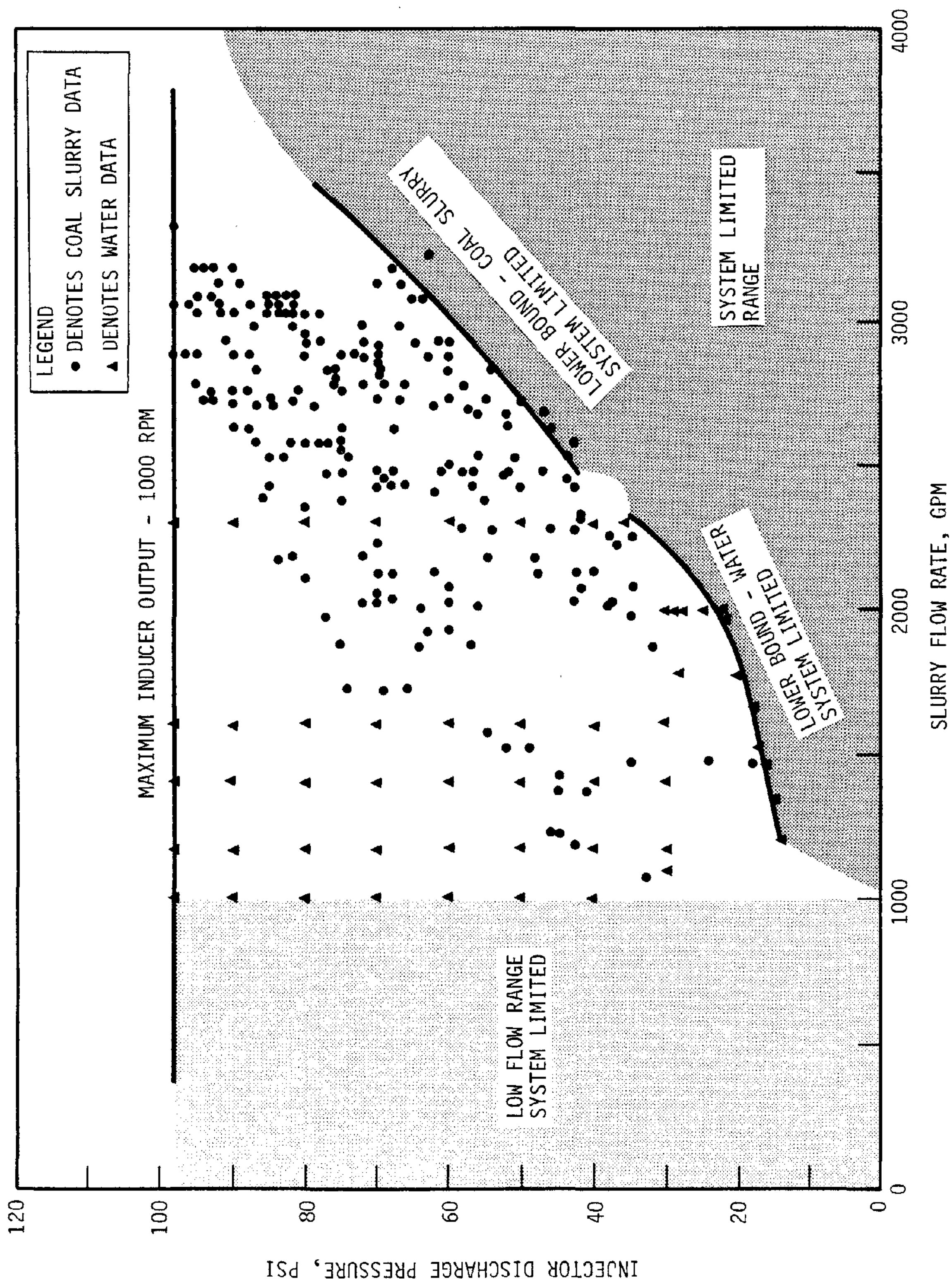


FIGURE 40. - Helical injector performance test data summary.

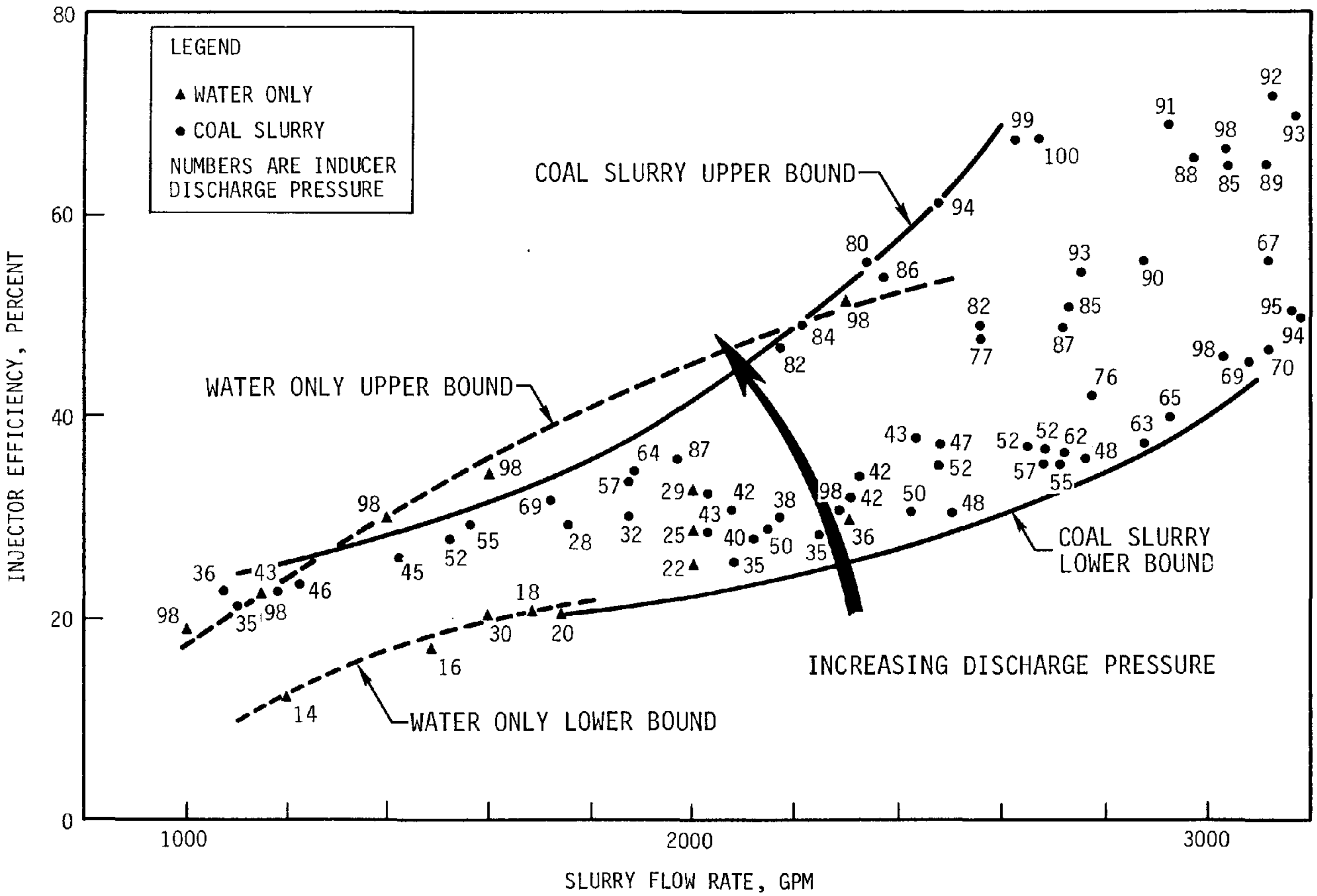


FIGURE 41. - Injector efficiency.

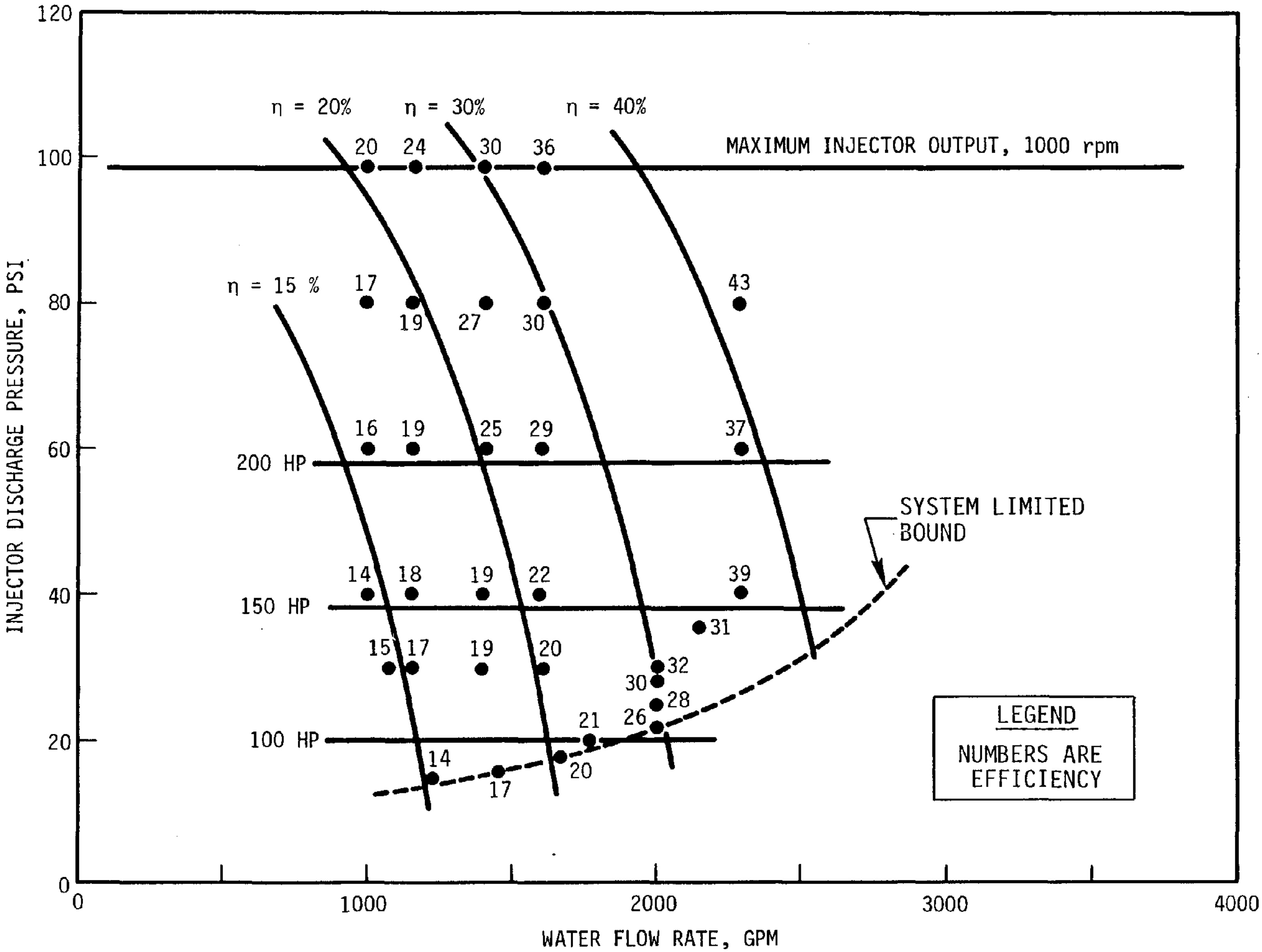


FIGURE 42. - Injector performance, water only.

The lower bound for the slurry tests, figure 40, lies in a range of higher discharge pressures than that of the water only tests. This is due to the higher pressures required to pump the coarse coal slurries. The maximum injector discharge pressure which was developed, pumping both a coarse coal slurry and water only at 1,000 rpm, was 98 lb/in^2. This is the maximum output pressure for the full-scale, 32-in-diam rotor, operating at 1,000 rpm and driven by a 350-hp electric motor.

These three figures show the injector's ability to operate at any combination of discharge head and flow rate without altering the pump's rotational speed. The injector's discharge head-flow rate capabilities are limited on the high end by the injector's operating speed and on the low end by the pipeline flow resistance.

The performance of the helical injector, while pumping various concentrations of coal slurries, is shown in figure 43. This plot identifies the shaft horsepower required to generate specific injector discharge pressures for a range of slurry concentrations. The coal slurries were composed of generally coarse coal with a maximum top size of 3 in. The composition of the slurries varied throughout the test program; the percentage of fines was not possible to control. This resulted in a significant amount of scatter in the data as evidenced by the broad range of test results plotted in figure 43. This is further noted in figure 44 where shaft horsepower is plotted as a function of the slurry flow rate. Clear water tests are shown in this figure as well as selected slurry tests. For a given slurry flow, it was determined that the required shaft horsepower increased as the required discharge pressure was increased. Discharge pressure was increased in proportion to the slurry concentration for a given total flow rate.

In summary, the operating range of the helical injector is limited by the system characteristics which include:

a. The supply water flow rate which is determined by the operating range of the pump used to supply water to the injector.

b. The speed of the rotor - For a fixed diameter rotor, the maximum discharge pressure of the injector is directly proportional to the tip speed of the rotor.

c. The discharge pipeline system characteristics - The injector will self-regulate to generate that discharge pressure which instantaneously matches

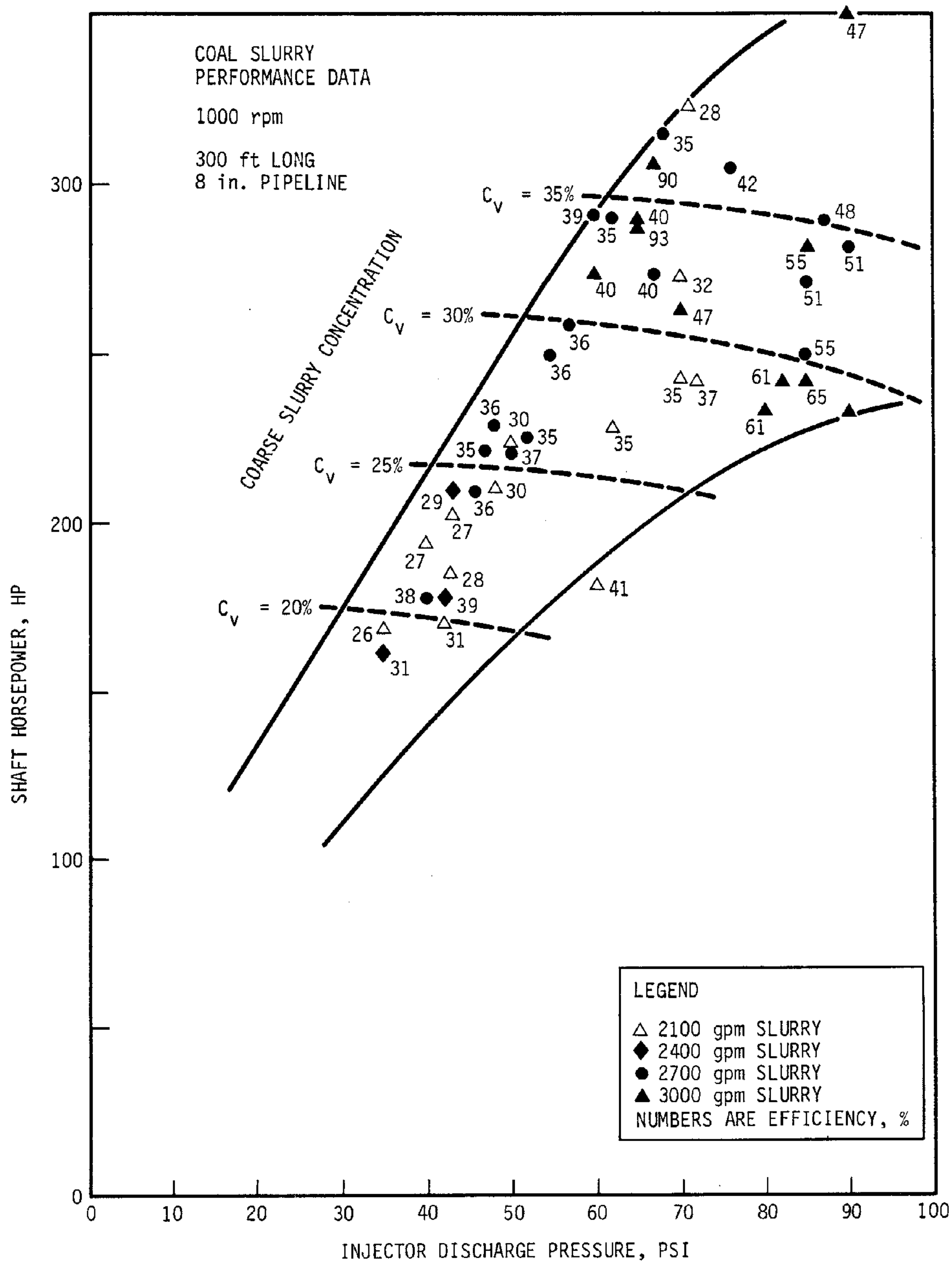


FIGURE 43. - Helical injector performance with coarse coal.

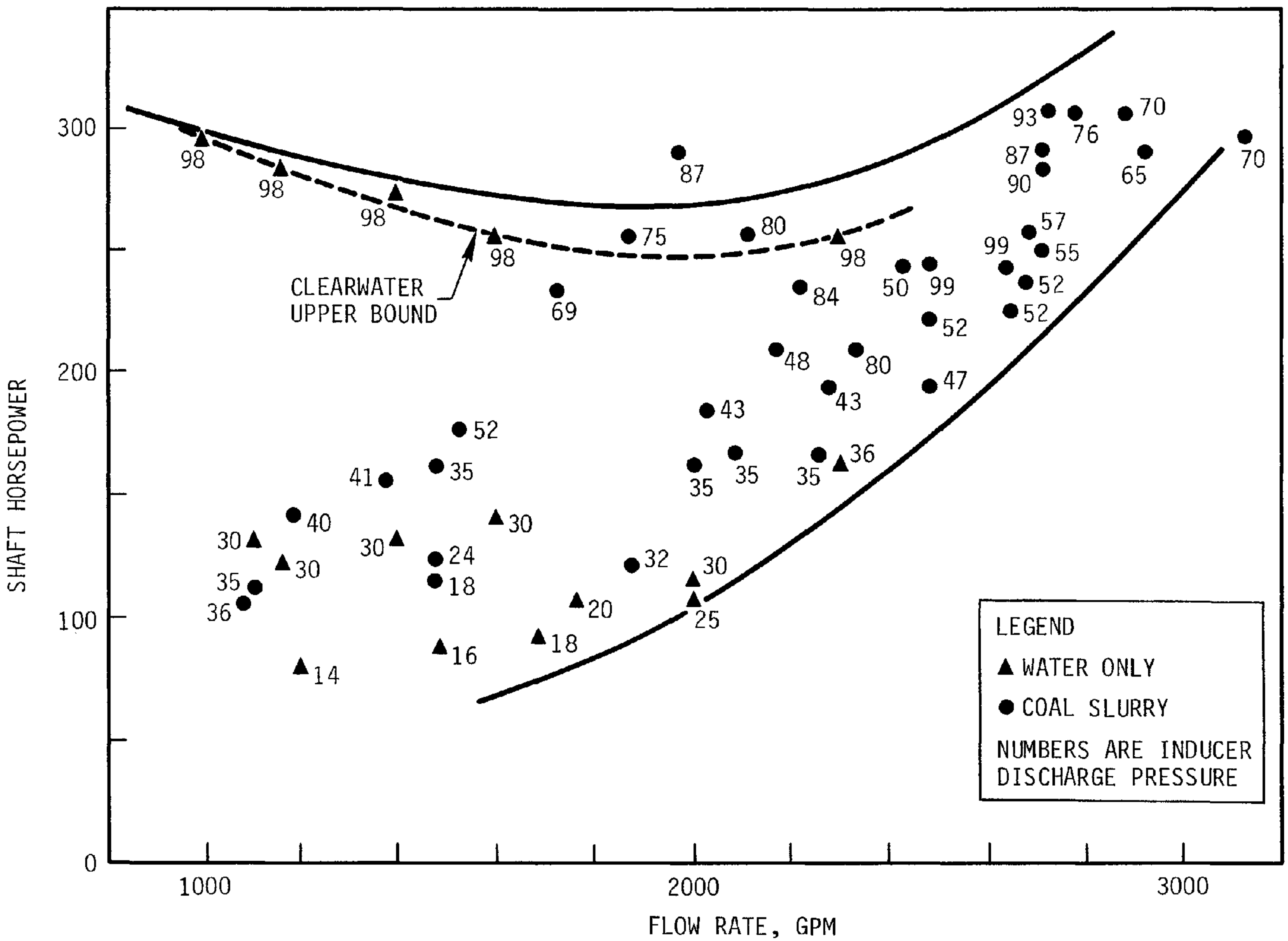


FIGURE 44. - Injector power use.

the pipeline system curve. The pipeline system curve is dependent upon the volume flow rate, coal concentration, coarse coal size distribution, pipeline layout and elevation, and pipeline length.

d. The shaft power input to the helical injector.

4.4.2.2 Injector Component Wear Characteristics

The principal wear components of the helical injector assembly were the helical rotor and the conical housing. In addition, the dry coal feed screw experienced wear on the edges of the helical blades. The performance of the injector was not affected by the deterioration of the feed screw. However, the maximum discharge pressure of the injector was reduced by up to 51% due to extreme abrasive wear of the rotor blade tip and gouging of the conical housing surface in some of the materials which were tested. This abrasive wear resulted in large gaps between the rotor tip and the cone. It is likely that some wear could be attributed to impact erosion. Design specifications required a gap of between 0.005 and 0.015 in.

The chronology of the wear performance evaluation program is detailed in subsection 4.4.1. Four distinct material combinations were evaluated:

a. The initial rotor and cone, which were tested, were made of mild steel. The rotor was cast from a 4,000 series carbon steel, designed for strength and toughness as opposed to surface abrasion resistance. The cone was fabricated from a 1,030 cold rolled mild steel. It was anticipated that these pieces would wear. The location and relative severity of material erosion were identified on these components.

b. The helical rotor tips were hardfaced with Stellite and ground to design specifications. The conical housing was coated with 0.020-in thick plasma application of chrome oxide. Stellite is easily applied in the field and is commonly used for "quick-fix" of mining and dredging equipment. The chrome oxide coating was chosen to observe the effect of an abrasion/corrosion resistant mechanical coating on the wear characteristics of the cone.

c. The helical rotor blade tips were coated with a vanadium carbide weld to a depth of approximately 0.75 in. The tips were relieved by machining (grinding) to provide a "point contact" area between the tip and the conical housing. The approximate surface hardness was 80 R_C. The cone was mechanically coated with a tungsten carbide composite with an approximate surface hardness of 75 R_C.

d. Both the rotor and the cone were cast in Abex HC-250. This material is a commercially available, abrasive resistant cast iron alloy. The material is machinable in the annealed condition. This allowed the rigorous design tolerances on both pieces to be maintained. After heat treating, the HC-250 maintains a high surface hardness, approximately 600 BHN. This material has been extensively used by commercial slurry pump manufacturers.

The tip speed of the tapered helical rotor ranged from approximately 3,900 to about 8,240 ft/min. Significant wear of the mild steel components was evident all along the rotor blade tip and across the entire cone face after running with coal. Each of the mechanically bonded plasma coatings experienced near total erosion of the coating in the tip speed zone corresponding to aproximately 6,000 ft/min and above. Gouging of the mild steel and hardfaced rotor tips was noted at the maximum diameter, corresponding to the tip speed range above 7,000 ft/min. These gouges ran across the blade face, normal to and concentric with the rotor axis, as depicted in the photograph of figure 45. The conical housing also experienced significant wear. Figure 46 shows the wear pattern of the plasma coating on the cone. The outer one-third of the cone, corresponding to the surface zone between 27- and 32-in-diam, was eroded by both impact and abrasive wear. This wear zone corresponded with that of the rotor; rotor tip speeds in excess of 6,000 ft/min severely eroded the rolled steel and plasma coated conical housings.

4.5 SLURRY PIPELINE PERFORMANCE

Throughout the duration of the injector development and performance evaluation program, coarse coal pipeline transport data was recorded. Two pipeline flow loops, as described in subsection 4.3, were used in the development of this pipeline transport information. The 8-in nominal pipeline was constructed of ductile iron pipe with an average inner diameter of 0.698 ft. The pipe friction factor, f, equivalent to Nikuradse's aritificial friction factor and Moody's classical roughness equivalent was determined to be 0.0169 for clear water passing through the 8-in nominal ductile iron pipe. The 6-in nominal pipeline was constructed of steel pipe with an average

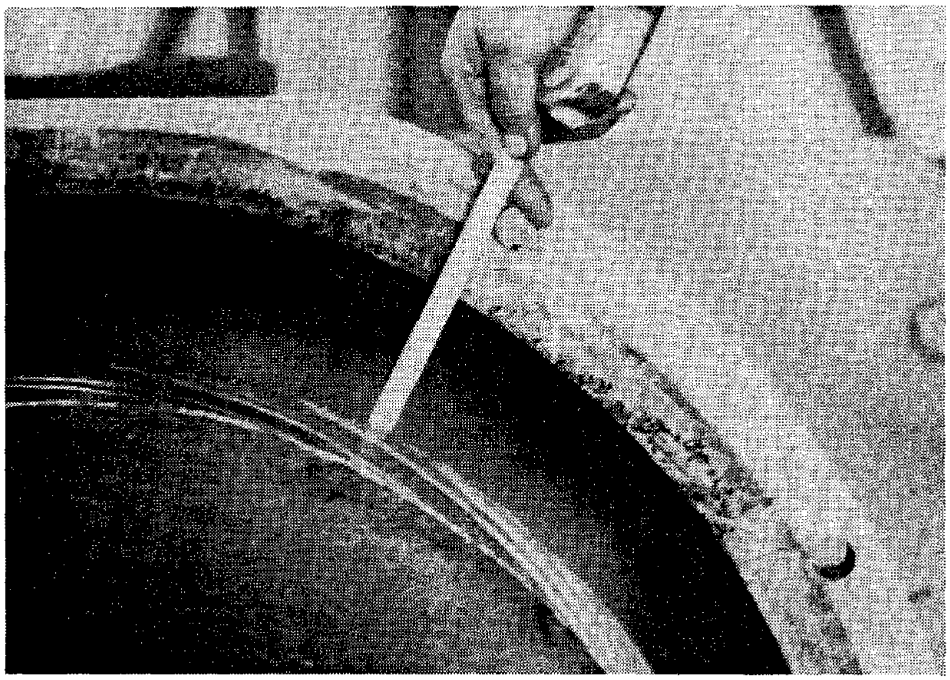


FIGURE 45. - Rotor tip wear pattern.

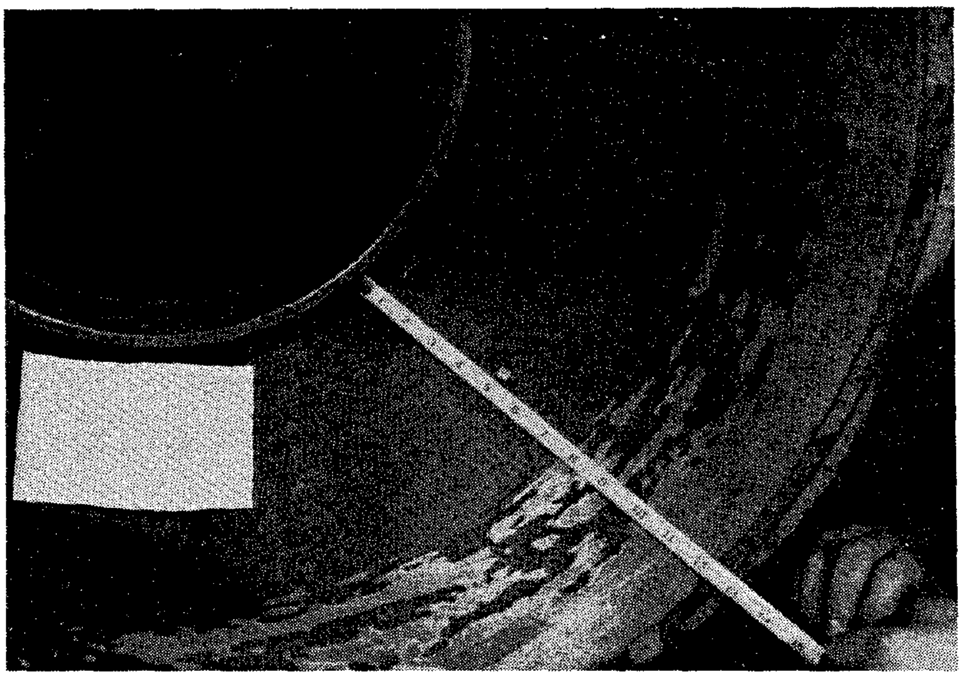


FIGURE 46. - Conical housing wear pattern.

inner diameter of 0.5054 ft. The pipe friction factor was 0.0145 for clear water.

The test results are summarized in table 12. The table lists the tests by test number and includes a summary of:

a. Volumetric concentration, C_V
b. Injector supply water flow rate
c. Coal flow rate
d. Area average slurry velocity
e. Head loss, ΔP/1000 ft of pipe
f. Nominal pipe size.

Data plots were prepared which include summary plots of the 6- and 7-in pipeline head loss data as well as selected plots of C_V = 30% and 40% for the 6-in line, and C_V = 20%, 30% and 40% for the 8-in line. The curves drawn through the data points on these plots are indicative of the trend of the data and do not represent any statistical curve fit.

A polynomial regression analysis of the 30% concentration, 8-in pipeline data indicated that the higher order fits take the form of the other plots as well as of other investigators data.

Eight-Inch Pipeline Tests

The range of pressure drop - velocity relationships for the coarse anthracite coal is shown in figure 47. Slurry concentrations ranged between 19% and 48% by volume. Specific test parameters are listed in table 12. Figures 48 through 50 show the pressure drop - velocity relationships for coal slurries of 20, 30 and 40%, respectively. Figure 51 presents the 30% concentration data curves generated by first, second and third order polynomial regression analysis.

Velocities below 10 ft/s were not run because of the presumed increased potential for pipeline blockage. Head losses through the 8-in ductile iron pipe for the coal slurry concentrations plotted were 2.2 to 3.6 times higher than for water at velocities of 10 ft/s. As the velocity was increased, the data trend indicated that the head losses of the slurry, asymptotically approached those of water. Head loss appeared to be nearly independent of slurry concentration at velocities greater than 30 ft/s.

The data scatter in figure 49 was attributed to degree of degradation of the coal run through the injector system. The upper bound is typical of fresh coal tests while the lower bound represents those tests which ran

TABLE 12. - Data summary slurry pipeline performance program

Test No.	C_v (%)	Water flow rate (gal/min)	Coal flow rate (tons/min)	Slurry V (ft/sec)	ΔP/ 1000 ft (lb/in.^{2})	Pipe diameter, nominal (in.)
1[1]	29.4	2160	6.38	17.82	61.47	8
2	26.8	2160	5.62	17.19	50.98	8
3	28.3	2160	6.06	17.55	64.94	8
5[1]	29.6	2160	6.43	17.86	96.34	8
6	28.4	2160	6.09	17.58	92.84	8
7	25.1	2160	5.14	16.80	89.35	8
8	24.5	2160	4.97	16.66	78.84	8
10	28.2	2160	6.02	17.52	64.94	8
11	26.8	2160	5.52	17.11	64.92	8
12	27.5	2160	5.80	17.34	68.42	8
13[1]	29.0	2160	6.25	17.71	64.95	8
14[1]	29.5	2160	6.40	17.83	64.96	8
15	28.1	2160	5.99	17.50	64.94	8
16[1]	29.2	2160	6.32	17.77	78.90	8
103	25.0	1520	3.60	11.81	49.80	8
104[2]	20.0	2000	3.50	14.52	48.20	8
105	24.0	2000	4.60	15.42	54.80	8
106[1]	29.0	1760	5.10	14.44	62.30	8
107	34.0	1440	5.20	12.65	64.20	8
108	28.0	1920	5.40	15.60	52.40	8
109[1]	31.0	1680	5.30	14.13	47.82	8
110	33.0	1440	5.00	12.49	67.11	8
111[1]	30.0	1440	4.50	12.08	45.24	8

TABLE 12. - Data summary slurry pipeline performance program--Continued

Test No.	C_v (%)	Water flow rate (gal/min)	Coal flow rate (tons/min)	Slurry V (ft/sec)	ΔP/ 1000 ft (lb/in.^{2})	Pipe diameter, nominal (in.)
112	25.0	1760	4.10	13.61	47.25	8
114	28.0	2000	5.40	16.08	64.40	8
115	33.0	1680	5.80	14.54	65.70	8
116	34.0	1520	5.50	13.37	71.49	8
117	27.0	1560	4.10	12.45	55.66	8
118	23.0	1800	3.80	13.60	53.72	8
119	21.0	2000	3.80	14.77	59.09	8
120	22.0	2000	4.00	14.93	59.85	8
121	22.0	1760	3.50	13.12	47.16	8
122	23.0	1560	3.30	11.79	51.54	8
123A	32.0	1560	5.10	13.27	62.72	8
123B	42.0	1560	8.10	15.73	73.93	8
124[3]	39.0	1760	8.00	16.82	79.28	8
125	36.0	2000	7.90	18.13	100.22	8
128[2]	20.0	1560	2.30	10.97	32.22	8
129	42.0	1520	7.90	15.34	79.38	8
130[3]	39.0	1760	7.90	16.73	79.28	8
131[1]	31.0	2000	6.30	16.82	70.67	8
132	35.0	2000	7.70	17.97	70.80	8
133[3]	41.0	1720	8.80	17.23	73.54	8
134	46.0	1480	8.80	15.84	61.73	8
135A	43.0	2040	11.10	20.99	108.79	8
135B	37.0	2040	8.40	18.78	75.95	8
136	37.0	2000	8.30	18.46	75.85	8
137[3]	40.0	1800	8.40	17.38	72.42	8
138	44.0	1520	8.50	15.83	76.90	8

TABLE 12. - Data summary slurry pipeline performance program--Continued

Test No.	C_V (%)	Water flow rate (gal/min)	Coal flow rate (tons/min)	Slurry V (ft/sec)	ΔP/ 1000 ft ($lb/in.^2$)	Pipe diameter, nominal (in.)
139A	37.0	1440	6.00	13.31	58.53	8
139B	32.0	1440	4.90	12.41	47.85	8
140	28.0	1760	4.90	14.27	45.90	8
141	26.0	2000	4.90	15.67	51.28	8
142[2]	19.0	2000	3.40	14.44	47.78	8
143[2]	21.0	1760	3.40	11.59	42.04	8
144	24.0	1520	3.40	11.64	39.60	8
145	24.0	1520	3.50	11.72	39.97	8
146A[1]	31.0	1760	5.71	14.94	69.02	8
146B	34.0	1760	6.67	15.72	83.67	8
146B	30.0	1760	5.50	14.55	70.30	8
147[1]	29.7	1360	4.07	11.26	57.99	8
148[1]	30.8	1480	4.67	12.45	44.06	8
149[1]	29.5	1720	5.10	14.20	47.53	8
151	28.0	1440	3.98	11.65	57.97	8
152[1]	29.3	1720	5.05	14.16	71.93	8
154[1]	31.0	1480	4.71	12.49	58.01	8
155	31.6	1680	5.51	14.31	85.90	8
156	32.0	1360	4.55	11.65	61.50	8
157	33.1	1480	5.20	12.89	68.49	8
158	33.5	1720	6.15	15.06	82.44	8
159	35.3	1360	5.25	12.23	72.01	8
160	31.2	1480	4.75	12.52	64.98	8
161[1]	29.2	1720	5.04	14.15	68.44	8
162	32.7	1680	5.78	14.53	75.46	8
163A	32.3	2040	6.88	17.53	71.97	8

TABLE 12. - Data summary slurry pipeline performance program--Continued

Test No.	C_v (%)	Water flow rate (gal/min)	Coal flow rate (tons/min)	Slurry V (ft/sec)	ΔP/ 1000 ft (lb/in.^{2})	Pipe diameter, nominal (in.)
163B[1]	29.1	2000	5.83	16.43	64.95	8
164[1]	31.3	1720	5.53	14.56	61.50	8
165[3]	39.9	1320	6.22	12.79	61.61	8
166	32.0	1720	5.74	14.73	51.05	8
167[1]	29.3	1480	4.33	12.17	51.01	8
168A[2]	21.8	2160	4.27	16.08	50.91	8
168B	28.9	2160	6.23	17.69	61.46	8
169[1]	30.0	2000	6.08	16.64	61.46	8
170	34.2	1680	6.20	14.87	61.53	8
171	35.2	1440	5.55	12.94	54.57	8
172[1]	31.1	2080	6.66	17.58	71.95	8
173	32.8	2000	6.91	17.32	71.97	8
174	35.9	1720	6.82	15.61	72.01	8
175	35.9	1480	5.89	13.45	61.55	8
176	32.0	2120	7.06	18.41	75.45	8
177[1]	29.8	1960	5.91	16.27	64.96	8
178	33.9	1720	6.25	15.15	65.01	8
179	36.1	1520	6.09	13.85	65.04	8
180[3]	40.0	1520	7.20	14.76	68.58	8
181A	38.9	1760	7.95	16.78	82.51	8
181B	37.7	1760	7.55	16.45	72.04	8
181C	34.8	1760	6.66	15.72	72.00	8
181D	36.0	1760	7.02	16.01	68.53	8
181E	35.0	1760	6.74	15.78	65.03	8

TABLE 12. - Data summary slurry pipeline performance program--Continued

Test No.	C_V (%)	Water flow rate (gal/min)	Coal flow rate (tons/min)	Slurry V (ft/sec)	ΔP/ 1000 ft (lb/in.^{2})	Pipe diameter, nominal (in.)
181F	33.2	1760	6.19	15.33	61.52	8
182[1]	30.1	1960	5.98	16.32	61.48	8
183A	24.3	2160	4.91	16.61	64.89	8
183B[1]	29.3	2160	6.35	17.79	68.44	8
183C[1]	30.3	2160	6.65	18.04	64.97	8
184	33.6	2000	7.17	17.53	68.50	8
185[3]	40.4	1640	7.89	16.03	75.56	8
186A[1]	31.8	2160	7.16	18.45	70.40	8
186B[1]	30.4	2160	6.70	18.07	70.94	8
186C	32.1	1160	3.89	9.95	40.59	8
189[3]	39.8	1040	4.88	10.06	54.63	8
190	37.7	1280	5.49	11.96	61.58	8
191	42.1	1440	7.42	14.48	58.15	8
192A[1]	30.7	1680	5.27	14.11	51.03	8
192B	43.0	1640	8.76	16.74	79.08	8
193[3]	40.7	1440	7.00	14.13	65.10	8
194	42.9	1200	6.38	12.23	65.13	8
195	48.3	960	6.35	10.80	58.23	8
196	22.6	1840	3.82	13.85	43.95	8
197	30.9	1640	5.22	26.43	144.80	6
198	24.9	1360	3.20	20.15	88.20	6
199	31.6	1040	3.41	16.92	73.80	6
200	33.4	920	3.28	15.38	66.70	6
201	27.5	1360	3.66	20.87	88.20	6
202	31.1	1040	3.33	16.79	73.80	6

TABLE 12. - Data summary slurry pipeline performance program--Continued

Test No.	C_V (%)	Water flow rate (gal/min)	Coal flow rate (tons/min)	Slurry V (ft/sec)	ΔP/ 1000 ft (lb/in.[2])	Pipe diameter, nominal (in.)
203	33.0	960	3.36	15.95	66.70	6
204	35.6	1600	6.26	23.07	173.20	6
205	44.6	1040	5.94	20.88	130.30	6
206	38.6	960	4.29	17.41	87.80	6
207	37.1	960	4.01	16.97	80.80	6
208[3]	42.2	1000	5.17	19.23	116.20	6
209	44.1	960	5.38	19.11	116.10	6
210	31.8	1600	5.29	26.09	144.80	6
211[3]	41.9	1560	7.99	29.88	187.20	6
212	42.2	1520	7.89	29.28	187.20	6
213[3]	40.4	1320	6.34	24.62	165.90	6
214[3]	39.7	1320	6.15	24.33	151.70	6
215[3]	41.4	720	3.61	13.67	73.40	6
216[3]	41.4	720	3.61	13.67	73.60	6

[1]Data plotted in 30 percent C_V.
[2]Data plotted in 20 percent C_V.
[3]Data plotted in 40 percent C_V.

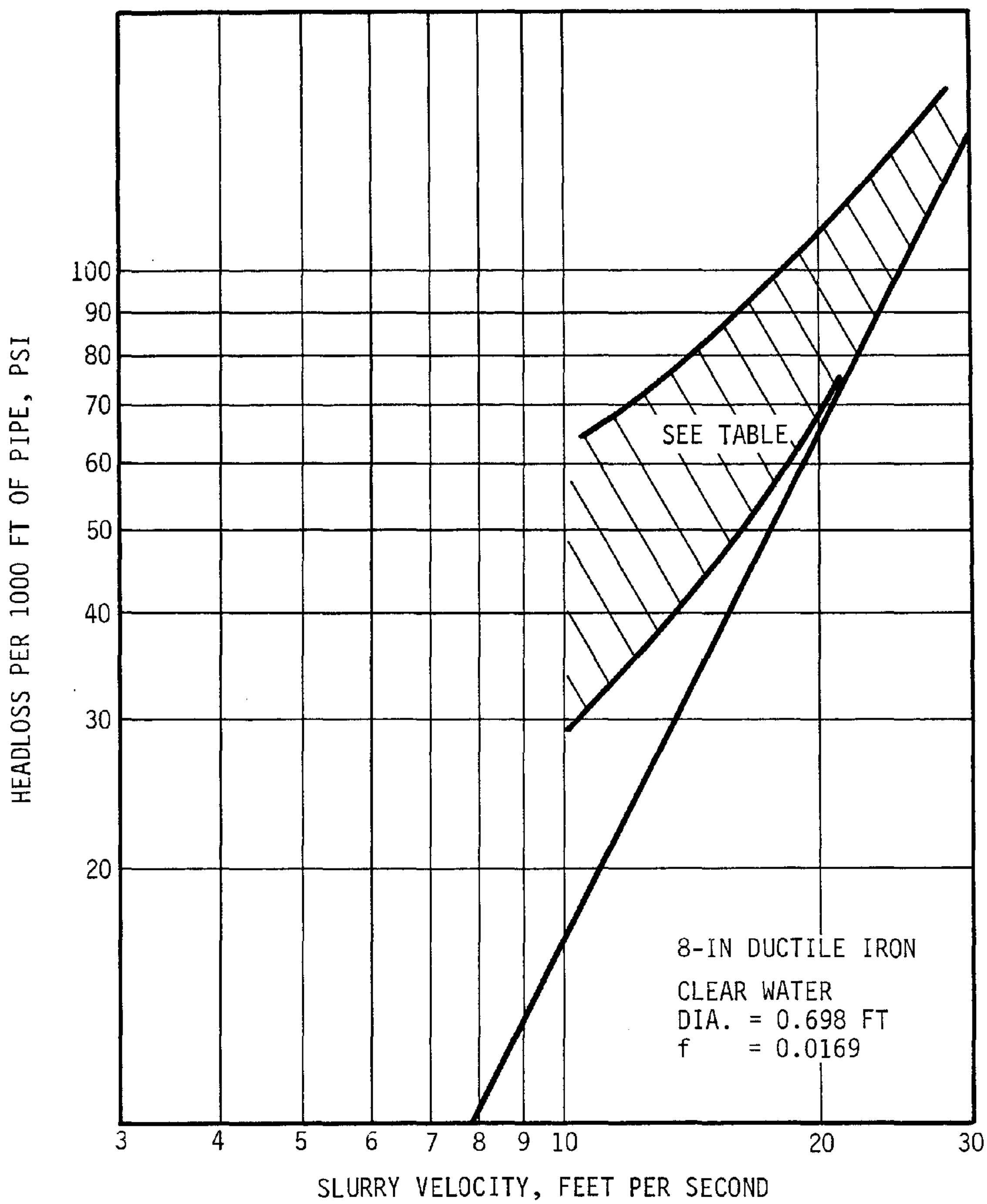


FIGURE 47. - Eight-inch ductile iron pipe, data summary plot, coarse coal transport.

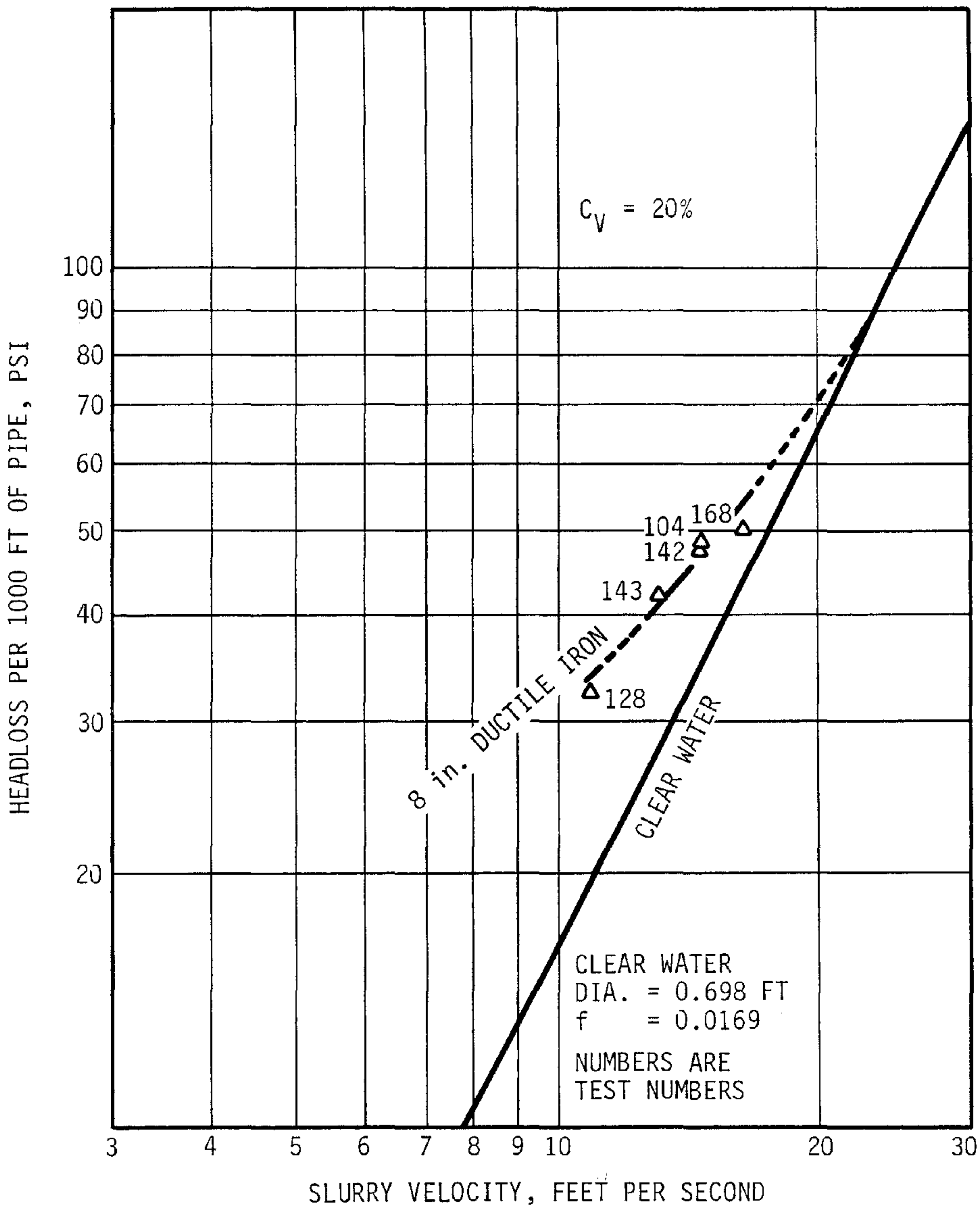


FIGURE 48. - Eight-inch nominal ductile iron pipe, coarse coal transport, 20% volumetric concentration.

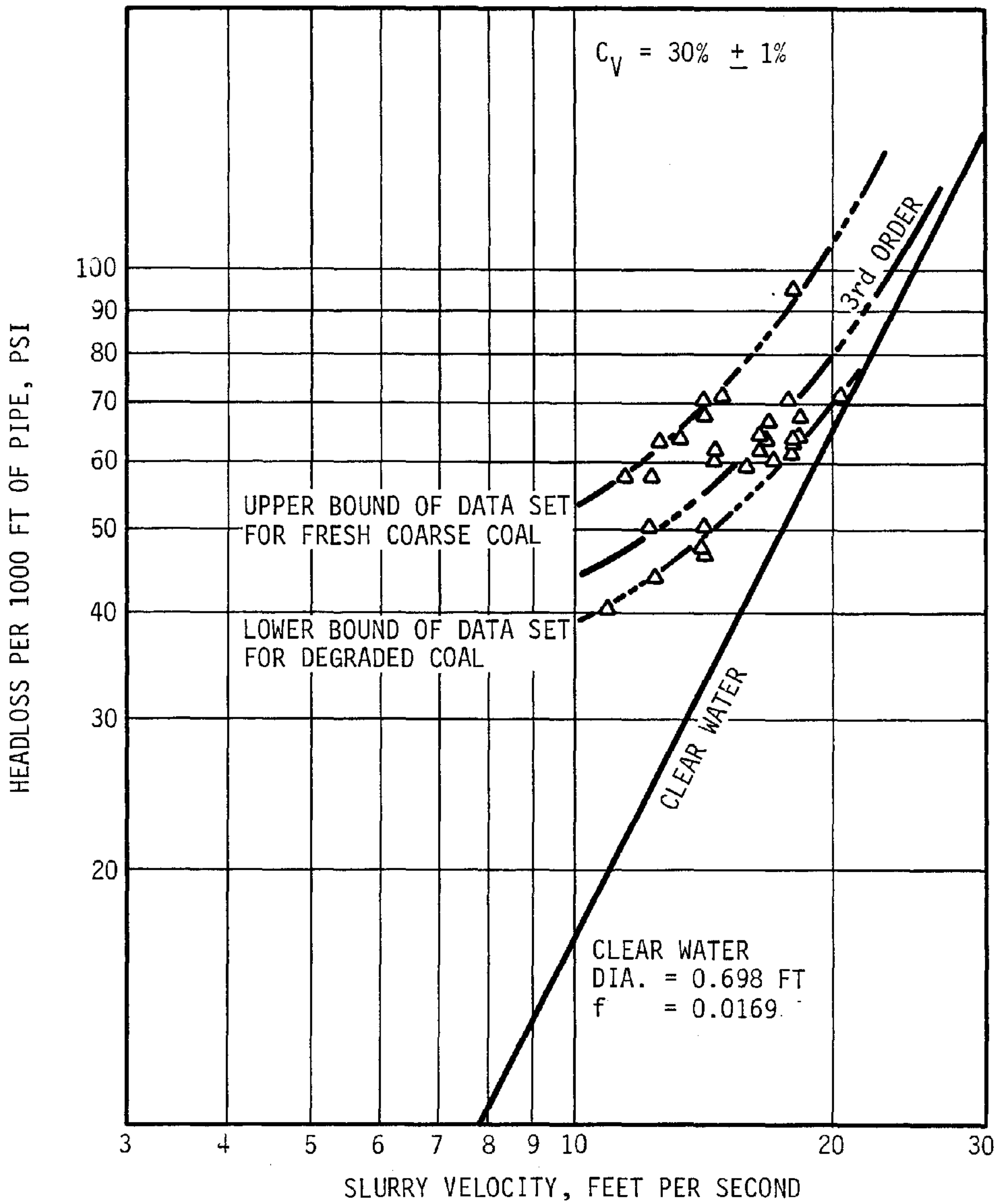


FIGURE 49. - Eight-inch nominal ductile iron pipe, coarse coal transport, 30% volumetric concentration.

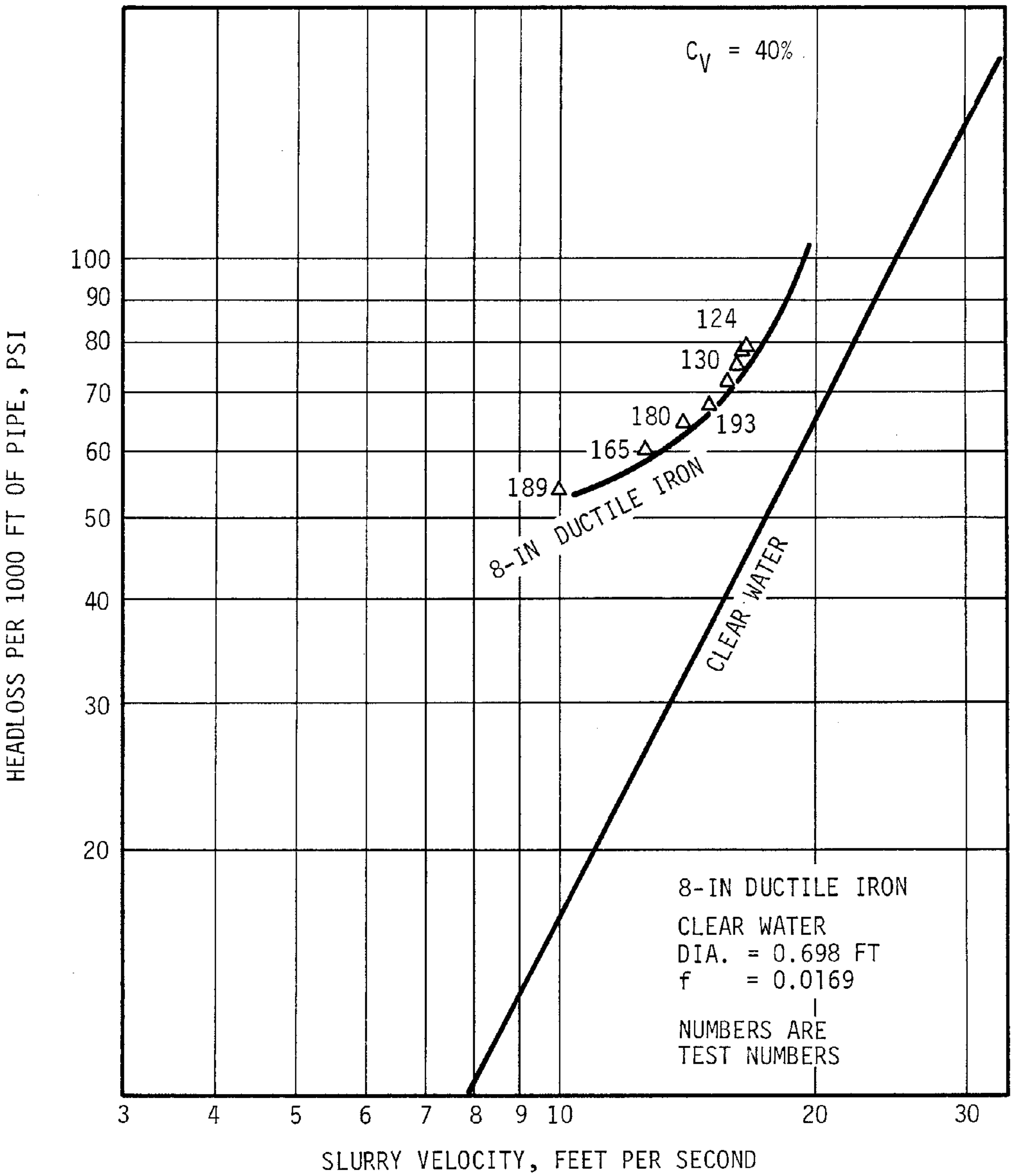


FIGURE 50. - Eight-inch nominal ductile iron pipe, coarse coal transport, 40% volumetric concentration.

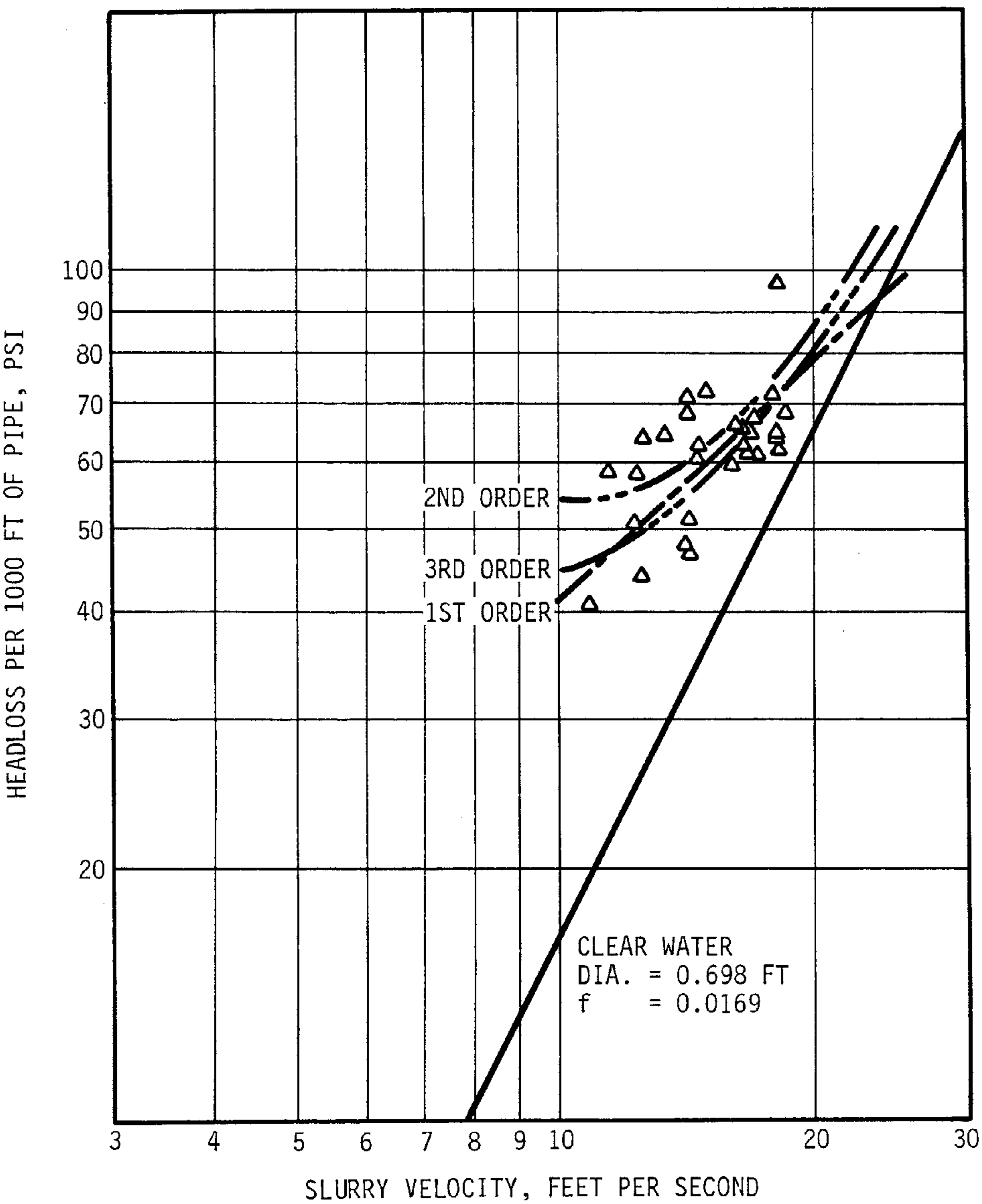


FIGURE 51. - Curve fit approximations, 8-in pipeline, C_v = 30%.

degraded coal. The third order curve fit represents the entire data set.

The curves plotted in figure 51 represent the first, second and third order approximations of the data by a polynomial regression analysis. The third order fit appears to best represent the scattered data of the 30% concentration tests. The third order plot would fall between the data trend curves drawn through the 20 and 40% concentration data in figures 48 and 50, respectively.

Six-Inch Pipeline Tests

The range of pressure drop - velocity relationships for the coarse anthracite coal in the 6-in steel pipeline is shown in figure 52. Slurry concentrations ranged between 25 and 45% by volume. Additional test parameters are listed in table 12. Figures 53 and 54 show the pressure drop - velocity relationships for coal slurries of 30 and 40%. Figure 55 is a plot comparing the pressure drop - velocity relationships for the 6-in steel and 8-in ductile iron pipelines transporting coarse coal at a concentration of 40% by volume.

The curves plotted through the data points for both the 30 and 40% tests were extrapolated to intersect the 10 ft/s velocity line. Velocities were maintained above this value to avoid possible line blockage. The pressure losses through the 6-in steel pipe for the coal slurries were 2.5 to 3.4 times higher than for water at velocities of 10 ft/s. Pressure losses for slurry approached those of water as the velocities increased.

A comparison of pressure losses of 40% concentration slurries in 6- and 8-in pipelines is shown in figure 55. The relative increase in pressure loss due to slurry over water is nearly the same over the velocity range between 10 and 30 ft/s. Head losses for slurry in the 6-in pipeline were approximately 16% higher than those in the 8-in line. Water through the same pipe systems experiences similar differences in pressure loss for the same range of velocities.

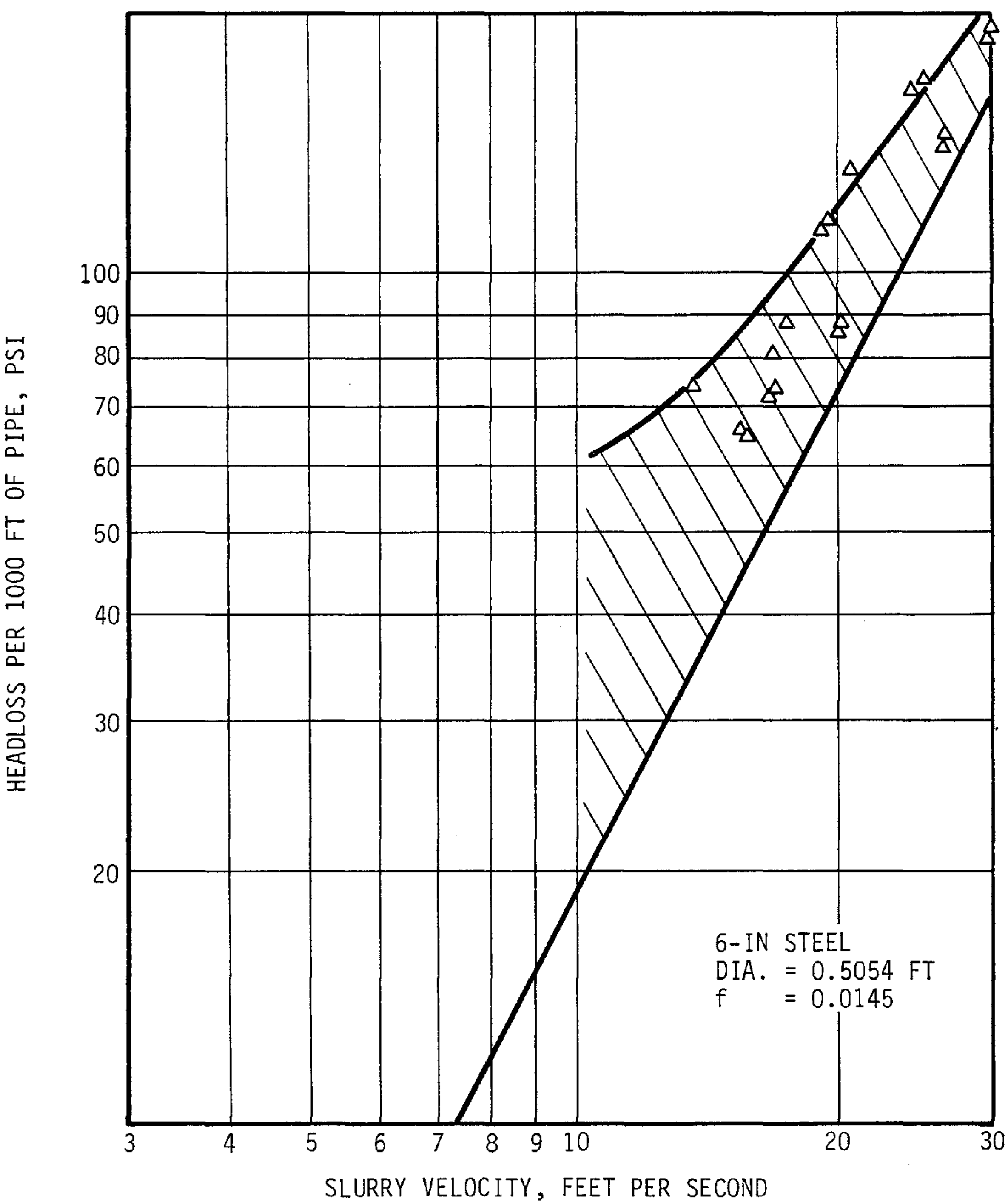


FIGURE 52. - Six-inch steel pipeline, data summary plot, coarse coal transport.

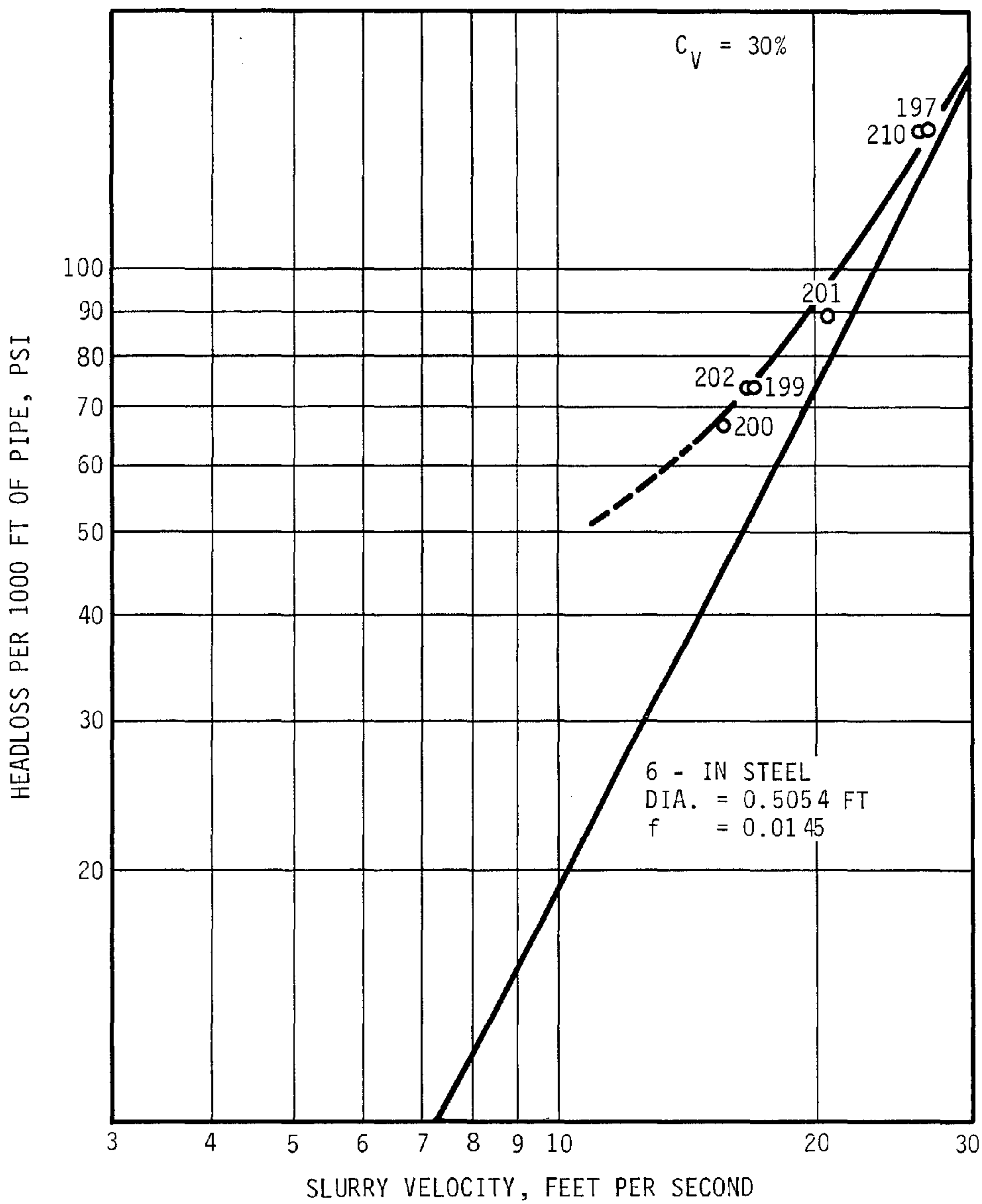


FIGURE 53. - Six-inch nominal steel pipeline, coarse coal transport, 30% volumetric concentration.

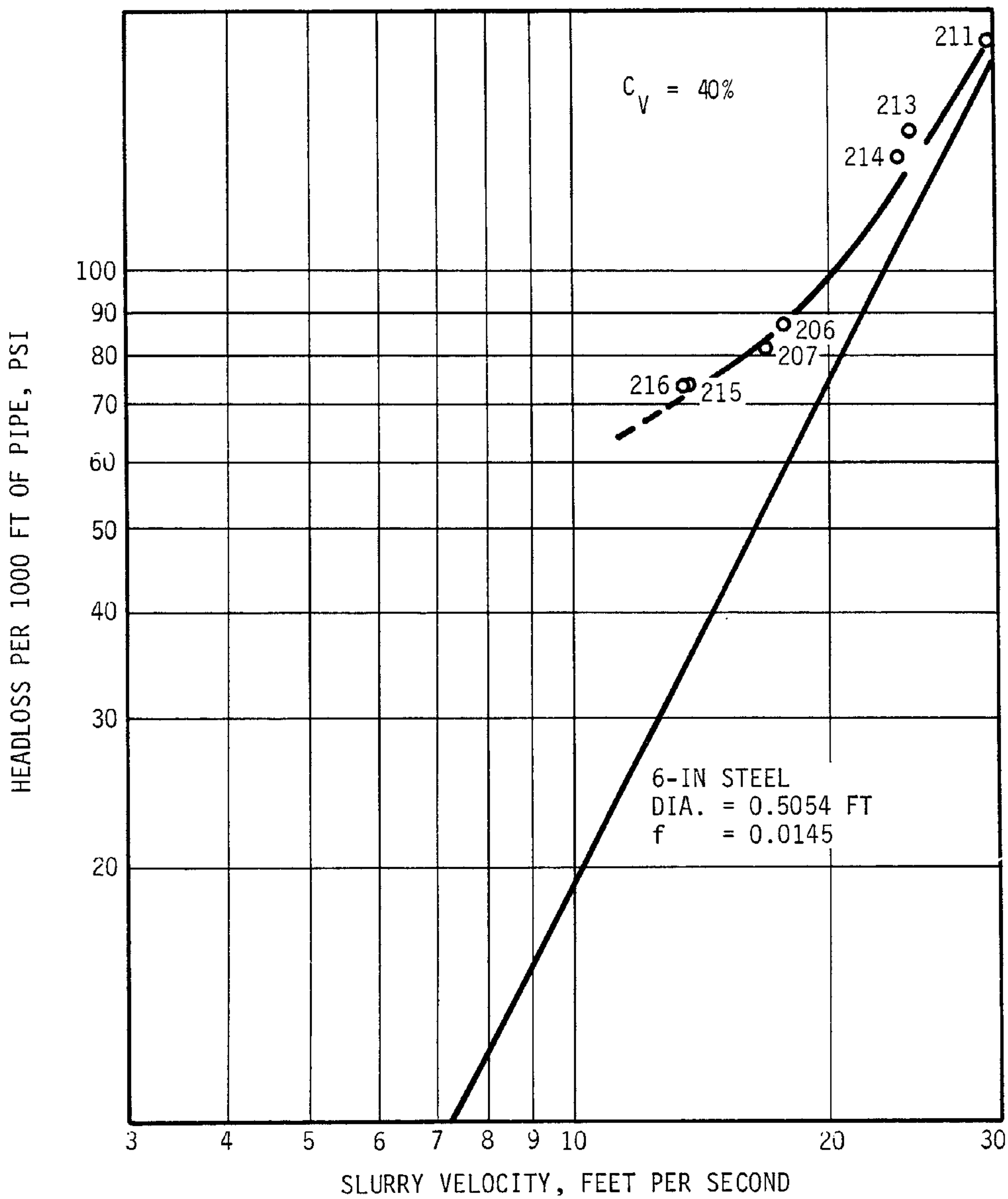


FIGURE 54. - Six-inch nominal steel pipeline, coarse coal transport, 40% volumetric concentration.

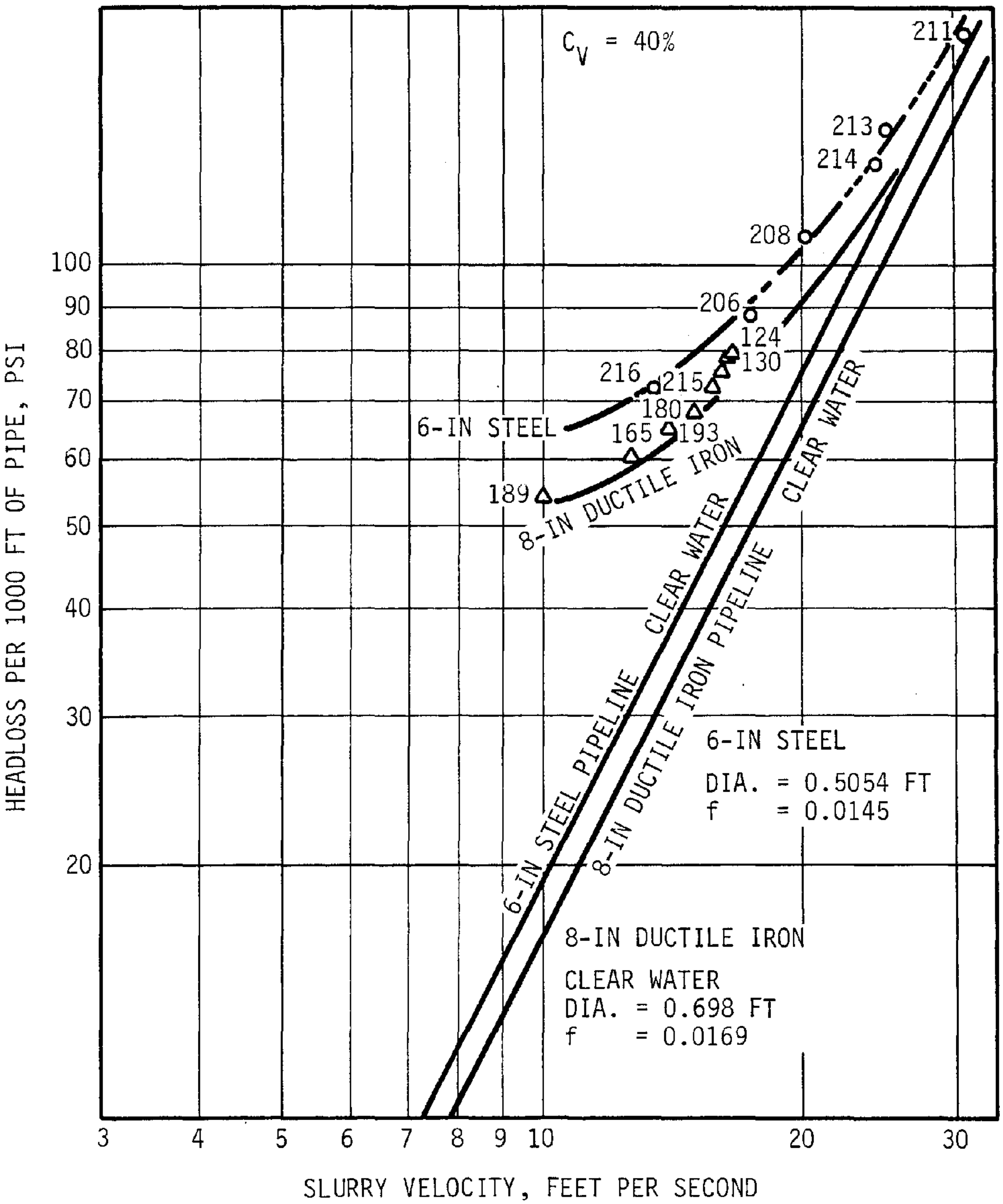


FIGURE 55. - Eight- and six-inch pipeline, coarse coal transport, 40% volumetric concentration.

5. PHASE III SLURRY INJECTOR VEHICLE (SIV) DEVELOPMENT AND TESTING

5.1 DESIGN GOALS

The basic functions and constraints which formulate the design goals of the injector vehicle can be summarized as follows:

a. Size limitation: 42 in, 27 ft and 9 ft (height, length and width)

b. Have sufficient mobility to follow and receive coal from a continuous miner - maximum tram speed 70 ft/min

c. 20,000-lb drawbar pull for slurry line handling

d. One ton ROM storage for miner surges

e. Receive ROM coal and reduce top size to one-third the pipeline diameter

f. Transport sized ROM to the injector feed screw and with proper water addition produce a slurry for discharge at pressures up to 95 lb/in² and coal flows of 6.4 tons/min

g. Fail safe operation - no water leakage.

5.2 SLURRY VEHICLE CONCEPT

An artist's concept of the SIV is illustrated in figure 56. Actual photographs of the completed vehicle are shown in figures 57, 58, and 59.

In operation, ROM coal from the continuous miner is delivered to the vehicle's inby end (fig. 40) where it is transported by chain conveyor to the scalper assembly for size sorting. Minus 3-in particles pass through the scalper while 3-in solids are delivered to a hooked-tooth, double-roll crusher. Crushed and scalped solids are then transported by chain conveyor to the coarse coal injector feed screw (fig. 57). The separately delivered water and solids are combined into a slurry within the injector pump and discharged at pressures up to 95 lb/in². The valving arrangement shown in figure 57 provides the capability of bypassing the injector and maintaining slurry

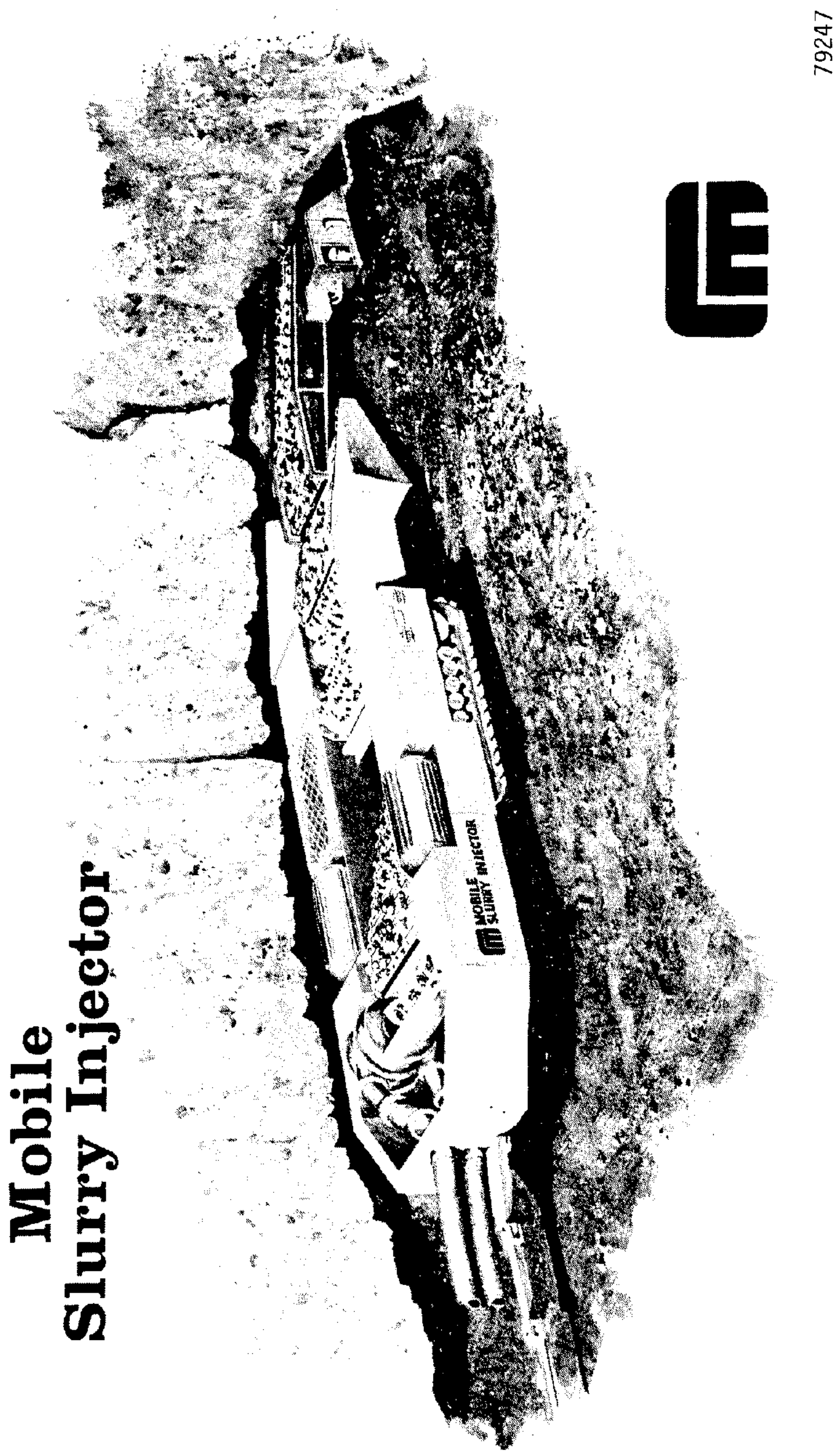


FIGURE 56. - Slurry injector vehicle (artist's concept).

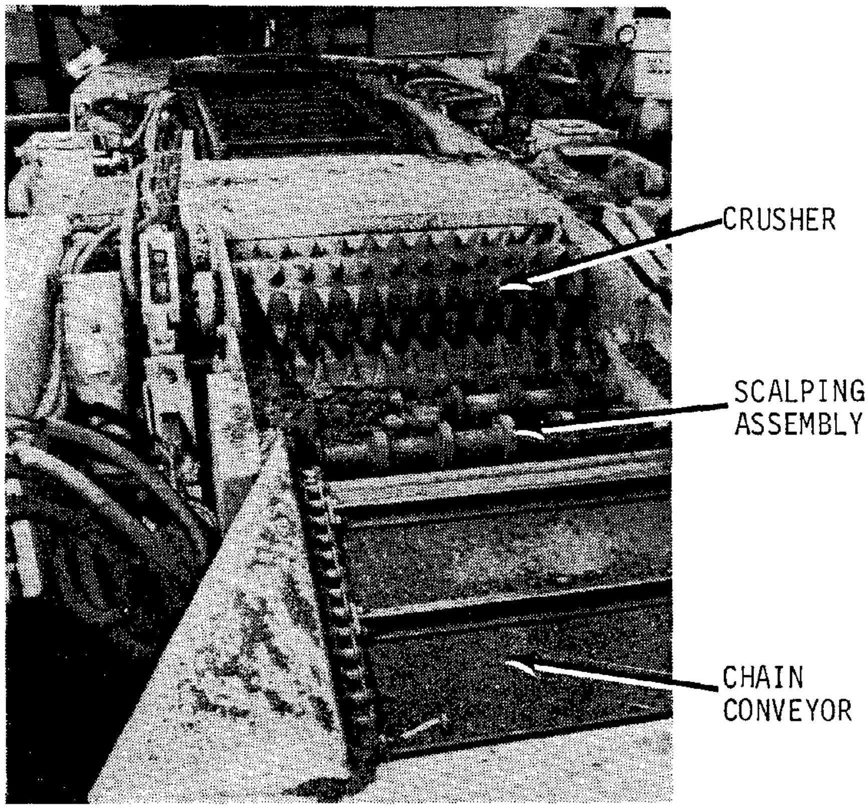


FIGURE 57. - Injector vehicle (inby end).

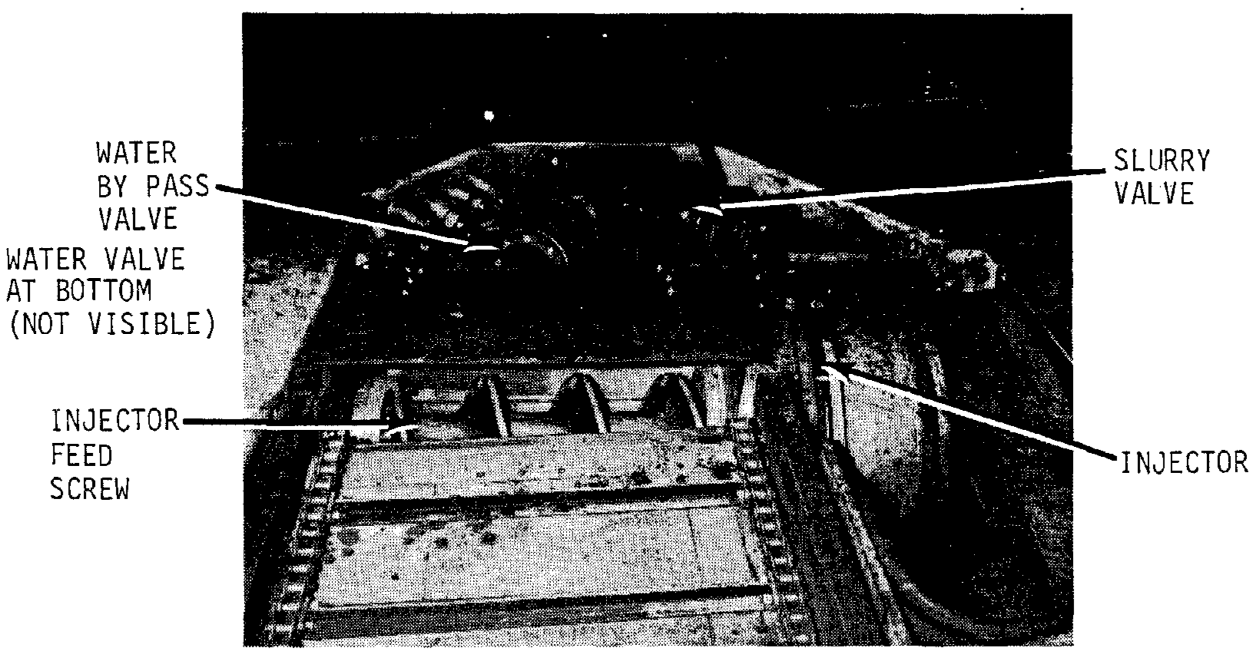


FIGURE 58. - Injector vehicle (discharge end).

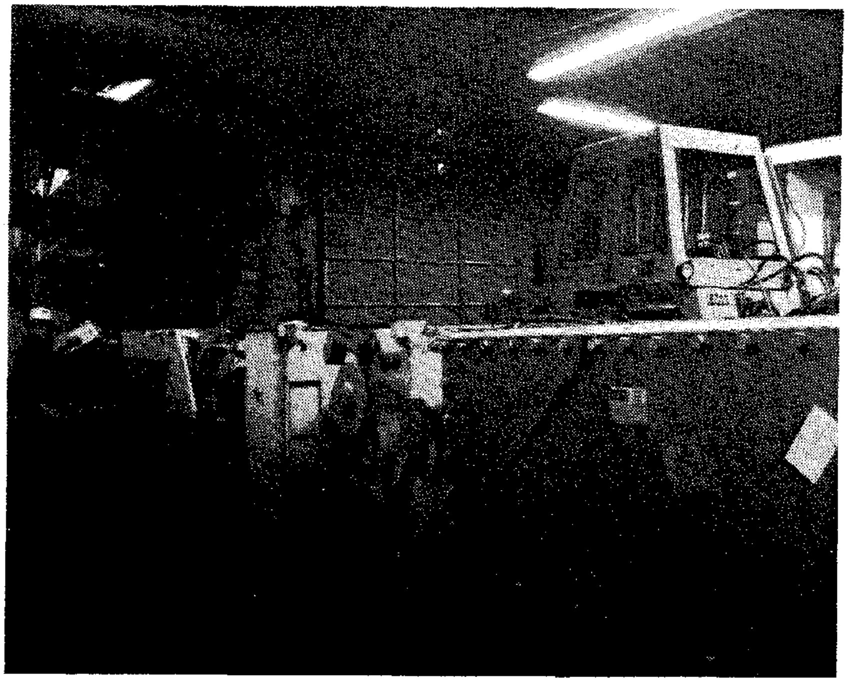


FIGURE 59. - Injector vehicle (side view).

flow through the hydrotransport system, in the event of injector failure. Figure 42 shows a side view of the injector with an operator at the vehicle control station.

Details of the SIV design development and testing are supplied in the remainder of this section.

5.2.1 Basic Vehicle Configuration

At the outset of the program, a number of vehicle configurations were investigated such as:

a. Single-axle, rubber tire equipped trailer

b. Single-axle, track equipped trailer

c. Dual-axle, self propelled, track equipped vehicle.

The single-axle, rubber tire equipped trailer offered, at first glance, a simple and cost-effective solution. In its basic concept, pump and discharge valves were to be mounted on a trailer which was to be hitched to the miner by means of a universal coupling. Such an arrangement offers a fixed relationship between the discharge end of the miner's conveyor and the inby end ofthe slurry vehicle. Beyond this, the complexity and expense for a separate propulsion and steering system is eliminated.

Closer examination of this scheme revealed serious flaws. Unless an articulating trailing system is employed, the trailer cannot follow in the miner's path, resulting in a collision with the rib when advancing or retreating. If one assumes an average length of approximately 25 ft for the unit and 29 ft (as determined from layouts) for the trailing slurry vehicle, it is readily evident that the combination of the two is unmanageable in the close confinements of a mine. Operator control from the front of the miner is also seriously impaired due to lack of visibility and depth perception.

The single-axle, track equipped trailer concept suffered from the same handicaps as the rubber tired vehicle, except that the footprint is more generous and allows operation in relatively wet and soft soils without getting mired.

The dual-axle, self propelled track equipped vehicle was the final choice. Numerous layout studies were made to determine the optimum location for all subassemblies and principal components, always keeping in mind that a length of 27 ft, a width of 9 ft and a height of 3 ft 6 in could not be exceeded.

5.2.2 Choice of Vehicle

The preliminary studies performed at the beginning of the program showed that the "ideal" vehicle chassis should conform to the following specifications:

a. Tracks should not exceed 24 in. in height and 96 in. in length

b. Clear distance between tracks must be at least 55 in

c. Load carrying capacity of at least 60,000 lb dead weight

d. Individual, infinitely variable, track drive system in forward as well as in reverse mode

e. Attachment points for the vehicle's upper structure must be located within each track bogie.

At the outset, it seemed that a pair of bogies with matching tracks meeting the above specifications could easily be purchased from a manufacturer of earth moving equipment, such as CATERPILLAR, JOHN DEERE, etc. It was quickly discovered, however, that tracks with a height of 24 in were not available on the market. Beyond this, they all were more sophisticated than what our application demanded since they are produced for exacting applications in the earth moving field. Deliveries at the time of our inquiries were between 12 to 16 months, a time span which was deemed unacceptable.

The next alternative was to investigate track supported mining equipment with the idea of possibly "pirating" the undercarriage. This turned out to be easier said than done since most of it is closely related to the specific machine and hence is difficult to separate.

After carefully scouring the used equipment market, an appropriate vehicle was located. This unit was a hardrock crawler loader made in the late sixties by Jeffrey for use by the White Pine Copper Company. While this machine had been used, it seemed that it was in reasonable mechanical condition and could be adapted to our needs. Cost of the equipment was reasonable, allowing for a comfortable margin for the contemplated mechanical alterations.

The vehicle was stripped of the gathering plate and drive assembly, the center pull flight conveyor, the articulated discharge boom at the rear of the machine, and all hydraulic and electric controls including the two traction motors. Replacement of the electrical equipment and components was necessary because operation of the modified vehicle at the candidate underground test site required compatibility with their 980-V power grid.

The empty chassis and track assembly with its associated planetary transmissions provided the foundation for the slurry injector vehicle.

5.2.3 Placement of Subassemblies

Placement of the various subsystems within the vehicle was largely determined by the flow relationship of the coal with respect to scalper, crusher, and redesigned chain flight conveyor. The offset and transverse placement of the screw feeder necessitated the placement of the conveyor axis slightly off center and skewed to the longitudinal vehicle axis. In this way, the full width of the conveyor could be maintained for maximum delivery of coal.

With the conveyor in place, space could now be identified where the rest of the drive motors and gear reducers, fresh water and slurry pipelines, motor control boxes, etc., could be located. While the drive motor for the tracks, crusher, and helical feed screw could conveniently be located on either side of the vehicle, the conveyor drive had to be placed underneath the conveyor and in between the tracks, a location which is hard to reach and difficult to service.

Space for the switch gear and associated controls is provided within the tapered box beam located at the very rear of the vehicle. Suitable structural adaptors are provided at the lower end of this box beam for attachment of the draglink to the hose-hauling assembly.

5.2.4 Power Budget

The level of electric power at the candidate underground test site that could be delivered to the vehicle was limited by the capacity of the available substation/load center and the matching 980-V supply cable.

It was determined that a maximum of 500 hp could be made available to all the electric motors of the slurry vehicle.

Based on calculations, scale model tests, and other available information, the following power budget (table 13) was established for various modes of operation.

TABLE 13. - Vehicle power budget

Mode of operation	Feed screw	Conveyer	Crusher	Winch	Slurry pump	Vehicle trans-mission	Total power, hp
"Inching"	50	20	50	0	350	20	490
Fast tramming	5	5	5	0	20	100	135
Corner turning	5	5	5	0	20	100	135
Pulling hose line	5	5	5	50	10	0	75

5.2.5 Modifications to Jeffrey's Crawler/Loader

As discussed earlier, the vehicle had been stripped of all electrical and mechanical components except the two planetary track drives. Beyond that, the front third of the original vehicle was also cut off to make room for the substructure supporting the crusher/scalper and feed hoppers. The rear of the vehicle was equipped with welded vertical flanges to which is bolted a reinforced, tapered support frame containing the slurry pump and its integral gearbox and drive motor, the helical feed screw with its prime mover, the fresh water and coal slurry plumbing stack with its control valves.

Within the external recesses of the support frame are placed the control boxes containing the switchgear, safety interlocks, overload monitors, etc.

5.2.6 Interface with the Hoseline System

The hoseline system attaches to the rearmost portion of the slurry vehicle by means of a draglink. Fresh water and slurry return lines are attached at the same point by means of standard, quick disconnect, hydraulic couplings.

The winch cable is fed out below the hoselines between two fairlead pulleys along the longitudinal centerline of the vehicle.

5.3 VEHICLE SUBSYSTEMS

5.3.1 Slurry Pump

5.3.1.1 General Considerations

Placement of the helical injector at the outby end of the vehicle was dictated by its relationship to the feed screw and coal conveyor. Transverse placement of the injector's shaft axis with respect to the vehicle's longitudinal axis was, therefore, a logical approach.[1]

Total available axial length for the pump, gear reducer and 350-hp electric motor amounted to approximately 48 in total. A "U"-shaped arrangement was chosen which permitted the placement of the electric motor side by side with the pump housing and placed the spur gearbox at the bottom of the "U".

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[1]A number of other configurations were also laid out, but were abandoned for various reasons.

Axial space limitations did not allow for separate pump shaft bearings; the pump impeller had to be directly supported by the gear reducer.

5.3.1.2 Mechanical Pump Details (Reference fig. 60)

The pump impeller, pump housing and conical inlet scroll are cast from a wear resistant alloy HC-250 by ABEX. Due to the extreme hardness of this material, only grinding and drilling operations can be performed. As a result, no key or spline connection could be employed between impeller hub and impeller shaft. Instead, torque is transmitted by means of two expandable, tapered bushings which are wedged between the shaft end and the inside bore of the impeller. Simple mechanical means allow for the removal of the tapered bushings and the impeller.

A labyrinth seal is employed to control the leakage flow of slurry fluid, and an elastomeric slinger ring is used to divert the fluid away from the pump bearing housing.

To still further protect the bearing from contaminants, a grease packed, double lip seal is employed.

Should the leakage through the labyrinth prove too excessive, a water cooled stuffing box can be installed in its place.

Process water to be mixed with the incoming coal is admitted to the pump inlet through a concentric annular opening. A plenum chamber surrounds the conical inlet scroll.

Process water to be mixed with the incoming coal first enters the toroidal plenum chamber which surrounds the conical inlet scroll. From there it is admitted to the pump inlet through an annular opening.

Piping connections to the pump housing for process water and slurry discharge are made with flexible, heavy duty, rubber bellows. This feature allows for easy removal of the pump housing and pump rotor should servicing be required.

All fluid lines leading to and from the pump are equipped with remotely operated, hydraulic valve actuators.

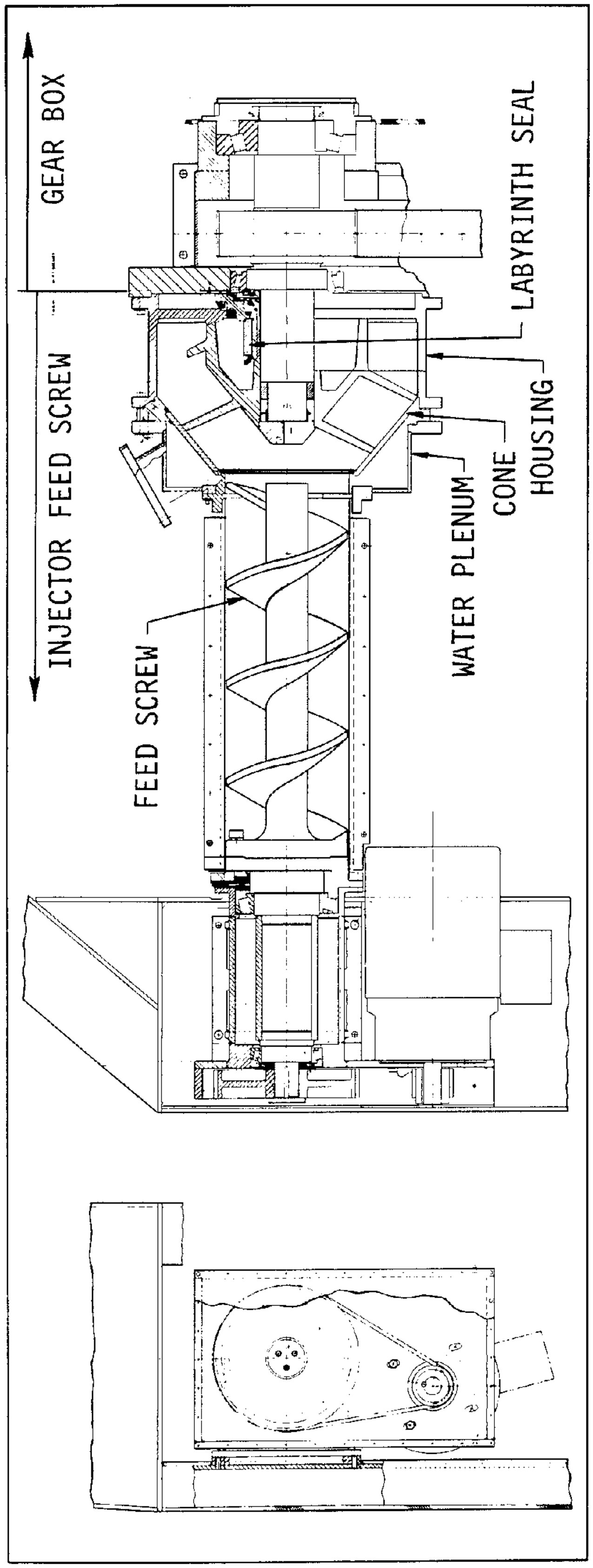

FIGURE 60. - Helical inducer injector.

5.3.1.3 Gear Reducer

The gear reducer is of special design to fit into the available space envelope in the vehicle and to provide the necessary support functions for the pump and its impeller.

The input and output shafts are coupled through three straight tooth spur gears. The principle specifications are as follows:

Shaft enter distance	34.125 in
Gear pitch	7 D.P
Pressure angle	20°
Gear classification	GMA 10
Gear reduction	1.452:1
Optional gear reduction	1.75:1
Service factor	3.42
Power rating	350 hp
Design rating	1200 hp

Mechanical provisions are made in the gearbox for the convenient exchange of the gears should operating conditions require a different gear reduction ratio.

An internally, directly driven positive displacement pump is employed to provide pressure lubrication of the gears and bearings. Oil is recirculated through a water cooled heat exchanger and passed through a 75-μ filter.

In addition, there are the usual oil drain plugs, oil level plugs, breather caps and special input shaft seals.

The power from the electric motor to the gear reducer is transmitted via a self-aligning gear coupling equipped with integral shearpins. Breakage of the pins will occur at 227% of running torque, but well below the torque level which would cause damage to the gears.

5.3.2 Helical Feed Screw Assembly

The helical feed screw is of cantilever design incorporating a forward tilting helix. Radial and axial support forces for the feed screw are provided by a set of Timken tapered roller bearings located in a short and stiff support structure.

Special care has been taken to prevent any dirt and coal from entering the outboard bearing. This is accomplished with an outside rotating slinger-type seal followed by a tandem set of grease filled lip seals.

Power from the 50-hp, 1800-rpm motor is transmitted to the feed screw by means of a splash lubricated high speed silent chain, enclosed by an oil tight housing. The feed screw can easily be removed for servicing by breaking a bolted flange connection near the bearing support end.

Torque overloads are monitored by a current sensor which will trip the motor contactor and shut the drive down. Feed screw jam-ups due to material pinched between the helical screw and the trough can be corrected by means of a forward/reverse jogging button and the master control station.

5.3.3 Chain Conveyor

The conveyor chosen for this application consists of two parallel chains interconnected with crosswise flights of welded design. Chain turnaround at various locations is accomplished either with matching sprickets or smooth idler wheels. At some locations, severe space limitations prohibited the use of the idlers of recommended minimum diameter; increased wear and service difficulties are to be expected as a result of this limitation.

Preloading and tensioning of the conveyor chain is accomplished by means of a swivel bar equipped with an idler at each end and supported in the center by a grease filled cylinder. A standard grease gun is used to establish the desired chain tension. Chain motion can be monitored by means of a speed sensitive, electrical switch.

5.3.4 Comminution Circuit

The ROM product must be reduced to 100% passing a 3-in^2 opening screen cloth at a minimum rate of 400 tons/hr. A review of investigative work down by USBM on the size consist of coal extracted conventionally and mechanically was done to determine the required capacity of the comminution circuit. Table 14 represents a comparative test sieve analysis.

No specific sizes over a 4-in material were given, but it can readily be seen that a mechanized mining system produces a small amount of material that has to be sized prior to being introduced to the helical inducer. Foster-Miller's observations combined with this study lead to the conclusion that no more than 30 percent of the ROM would have to be crushed to size.

TABLE 14. - Test sieve analysis - summary ROM coal

Type of mining	Retained on a percent square opening			Total percent retained
	+4 in	-4 to 1-1/2 in	-1-1/2 to +3/8 in	
Conventional	45	25	20	90
Borer	13	20	32	65
Ripper	7	43	15	65

5.3.4.1 Preliminary Concept

Several feeder-breaker manufacturers recommended the use of pick breakers to convey and reduce the ROM coal to -3-in at 400 tons/hr. A field investigation of this crushing principle showed that these goals could not be met.

A pick breaker produces an extremely coarse product because of the setting procedure of the picks to the flights of the conveyor. The flights of the conveyor can be as thick as 2 in and tend to ride up on the layers of coal that are formed under the conveyor chain. This problem makes the manufacturers hesitate to recommend close pick-to-flight settings, and 1 in is about a minimum setting. It can readily be seen that a slab material as high as the critical 3 in could easily pass under the picks and not be reduced at all.

Figure 61 schematically shows a typical feeder-breaker design in the pick area of a conventional 48-in wide conveyor deck. A typical flight speed is 700 ft/min. If we consider coal at a weight of 55 lb/ft^3, the following capacity calculation can be done:

$$\text{tons/hr} = \frac{4\ \text{ft}\ (2.5\ \text{in} + 3\ \text{in} + 2\ \text{in})}{12\ \text{in/ft}} \times 70\ \text{ft/min} \times$$

$$\frac{55\ \text{lb/ft}^3}{2000\ \text{lb/ton}} \times 60\ \text{min/h}$$

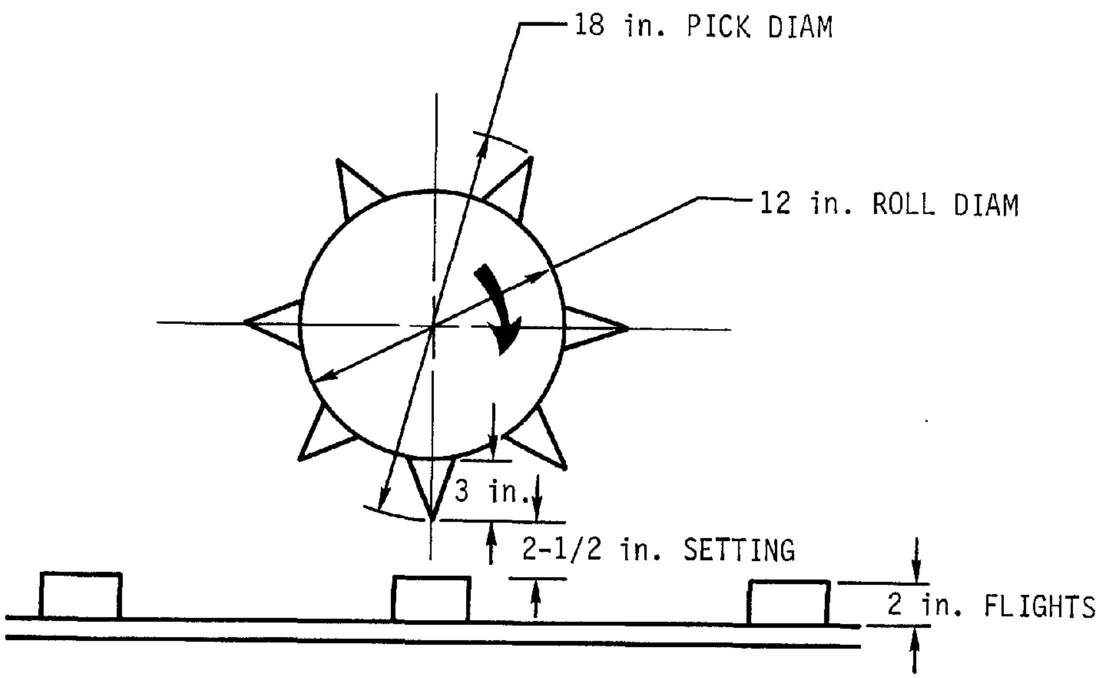

FIGURE 61. - Typical pick breaker.

At best, about 290 tons/h could be processed on the bed of a conventional 48-in feeder-breaker, and no guarantee of the required size consistent in at least two dimensions could be assured. An alternate would be to enlarge the pick tip to flight setting to increase capacity; but this, of course, aggravates the sizing problem. Flight speed and pick speed increases were unacceptable to the manufacturers contacted, and only a wider bed was considered a viable alternate.

5.3.4.2 Crusher Selection

A reasonable conclusion from the previous analysis of the ROM coal produced in a continuous type miner is that only secondary reduction, and a minimum of that, is necessary to assure the required 100% passing 3 in for feed to the helical injector. Any reduction device set to crush material to 3 in or less will act as a chute primarily, but should be capable of a 4:1 reduction ratio if necessary. This will allow for the possibility of say, 12-in ROM lump coal. The comminution device should produce a minimum of fines and be capable of crushing 15,000 lb/in² compressive strength material. The crusher must be small enough to fit into the vehicle confines and not require excessive power per ton of coal crushed.

The most commonly employed secondary crushers used in the coal industry are the hook tooth roll crushers marketed by the McLananhan Corporation and the T.J. Gundlach Machine Company. This type of crusher reduces the material through compression and shear and, therefore, generates the least amount of fines. All other requirements necessary for operating in low seam coal can be met if this type of crusher can produce the capacity required in a single reduction stage in open circuit (no recirculation load).

The Gundlach model 45 DA single-stage, hook tooth roll crusher was selected because of the following reasons:

a. Theoretical capacity favorable

b. Rugged components

c. Willingness of the Gundlach Company to cooperate in a special design modification to their machine.

The theoretical capacity through the selected crusher follows the formula:

$$\text{tons/h} = \frac{A \times B \times \pi \times C \times D \times E \times 60}{1{,}728 \times 2{,}000}$$

where: A = length of the hook roll, in

B = gap setting between hook teeth, in

C = hook roll diameter, root to root, in

D = roll speed, rpm

E = material weight per cubic foot, lb/ft^3

It follows then that:

$$\text{tons/h} = \frac{49 \times 2.5 \times \pi \times 10.75 \times 300 \times 55 \times 60}{3{,}456{,}000}$$

$$\text{tons/h} = 1{,}185 \text{ or } 19.7 \text{ tons/min}$$

Gundlach's theory is that nothing is perfect; therefore, they divide the theoretical capacity by two, subtract an arbitrary percent for extremely hard coal and an arbitrary percent for rock in the coal and moisture efforts. This results in a guaranteed capacity. It could readily be assumed that this crusher could easily pass all of the necessary tonnage, say 260 tons/h minimum, and that the remainder required to feed the injector at a rate of 400 tons/h must be scalped prior to entering the crusher rolls.

5.3.5 Scalper

The conservative estimates of the crusher capacity and the potential amount of sized/undersized material estimated in the USBM survey of ROM material led Foster-Miller to the conclusion that pre-crusher scalping was necessary. Pre-scalping of the project from a mechanized miner also:

a. Minimizes power consumption at the crusher

b. Reduces fines in the slurry system

c. Minimizes crusher roll wear

d. Allows for surges from a mechanized miner.

Scalping Capacity

A nominal 8 ft^2 of scalping area to remove the 3-in fraction from the ROM feed was available. It is an accepted fact in the sizing of coarse minerals, especially in primary scalping, that experience and testing must dictate the capacity of that system. The following variables collectively and individually will affect the scalper capacity:

a. Tons/h passing at ft^2 of area at a given bed depth

b. Percent oversize in the feed

c. Percent half-sized in the feed

d. Percent moisture

e. Material weight

f. Particle shape

g. Slope

h. Conveying characteristics of the live scalper rolls

i. Depth of bed

j. Whether or not the crusher is choke fed.

No theoretical approach to determine the capacity of the scalper is currently available to take all of these factors into consideration. Horizontal screen selection criteria suggests that 90 ft^2 is necessary. Pick breaker theory suggests that more than 220 tons/h can be conveyed and scalped in the available scalping area. It is suffice to say that the available area was utilized to its fullest to convey the ROM coal to the crusher and separate as much of the undersize in the feed to the crusher as possible.

5.3.6 Track Drive

As mentioned earlier in this report, the original vehicle came equipped with a fixed speed planetary track drive system, powered by two 50-hp ac motors. Braking one or the other of two control shafts on each transmission results in a fixed forward or reverse tran-velocity of approximately 80 ft/min.

The operation of the slurry vehicle in the mine requires that it be able to:

a. "Tram at 50:70 ft/min

b. "Inch" at 2:5 ft/min

c. Stop in either mode

d. Brake and hold on an incline of 25%.

The "inching" condition of 2:5 ft/min is impossible to meet with the existing transmission clutches since such a mode of operation implies continuous slip and energy dissipation at the brake pads, which at zero vehicle velocity could amount to 50 hp! Operation of the vehicle under such conditions would both waste energy and constitute a serious fire hazard. It was therefore concluded that the drive system had to be modified in order to meet the tram, inch stop and brake/hold condition.

5.3.6.1 Review of Drive Options

A number of diverse drive options were evaluated both in terms of electrical and mechanical complexities as well as cost. The following schemes were examined in detail:

a. Variable speed main motor ac drive

b. Auxiliary drive - variable speed ac motor

c. Auxiliary drive - three-speed power shift gearbox with fixed speed ac motor

d. 50-hp mechanical powershift transmission

e. Hydrostatic drive.

5.3.6.2 Variable Speed Main Motor ac Drive (fig 62)

This configuration makes use of the principal power transmission elements of the vehicle. A variable frequency solid state power supply has been added to drive and control the 50-hp ac motor - a straightforward and relatively simple solution for stationary applications, but not very practical for mobile use where space is at a premium. Size of the power supply module is excessive. It also lacks in ruggedness. Furthermore, motor torque available at the low speed end is inadequate. Total cost of the drive system for both tracks is estimated at approximately $9,000.

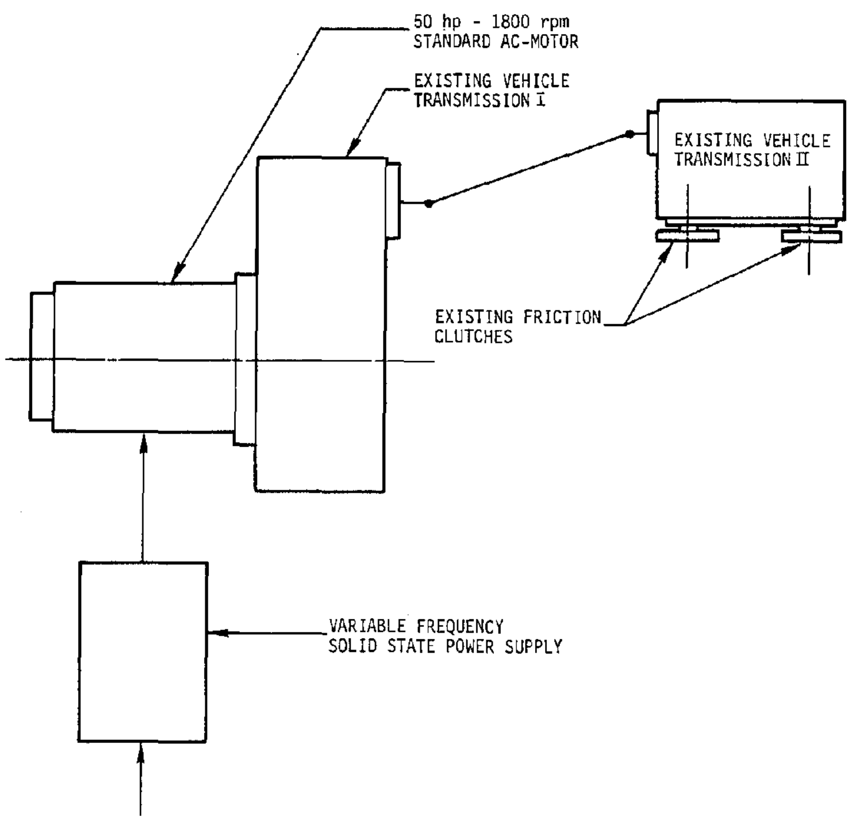

FIGURE 62. - Variable speed ac motor drive.

5.3.6.3 Auxiliary Drive - Variable Speed ac Motor (fig. 63)

Next an attempt was made to separate the tramming function from the "inching" function. In this way, the 50-hp motor would be reserved for "tramming" only, and a separate and much smaller variable speed ac motor employed for the "inching" function.

As can be observed from figure 63, the small ac motor is directly coupled to an intermediate gear reducer of 15:1 which in turn is coupled to the existing vehicle transmission. The variable frequency power pack which drives the ac motor is of manageable proportions. The physical placement of the small ac motor, however, and its gear reducer and electromagnetic clutch proved to be difficult to accomplish due to severe envelope constraints. Mechanical or electric interlocks are mandatory so as to prevent the accidental overdriving of the "inching" motor by the "tramming" motor. Beyond this, a relatively large investment in money has to be made in the "tramming" motor in relation to the total duty cycle.

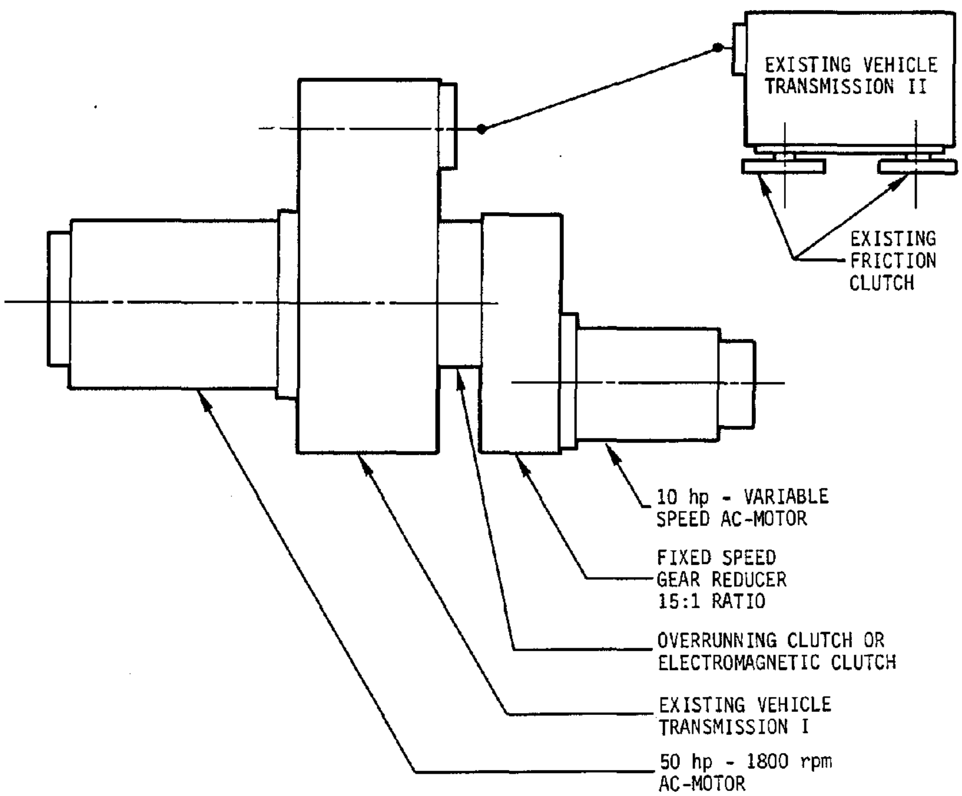

FIGURE 63. - Auxiliary drive - variable speed ac motor.

5.3.6.4 Auxiliary Drive: Three-speed Powershift Gearbox with Fixed Speed ac Motor (fig. 64)

This system is similar to the one depicted in figure 63 with the exception that the speed variation is accomplished with a three-speed powershift gearbox. Tramming function is again separated from the "inching" function. As was the case in figure 63, the motor-gearbox combination is difficult to place within the available vehicle outline, and motor overhang is excessive. The various speed ratios can be engaged by remote control either hydraulically or electrically. Mechanical separation between the 50-hp main drive motor and the auxiliary ac motor is provided by a mechanical one-way clutch.

Reverse "inching" is accomplished by selectively applying one or the other of the main transmission brakes.

As in the case in figure 63, the cost-effectiveness of this proposition is rather low, since the 50-hp tramming motor is used only a very small portion of the total time.

Also, valuable space is lost which is needed for the placement of a hydraulic power supply for the operation of the cable winch and the slurry discharge valves.

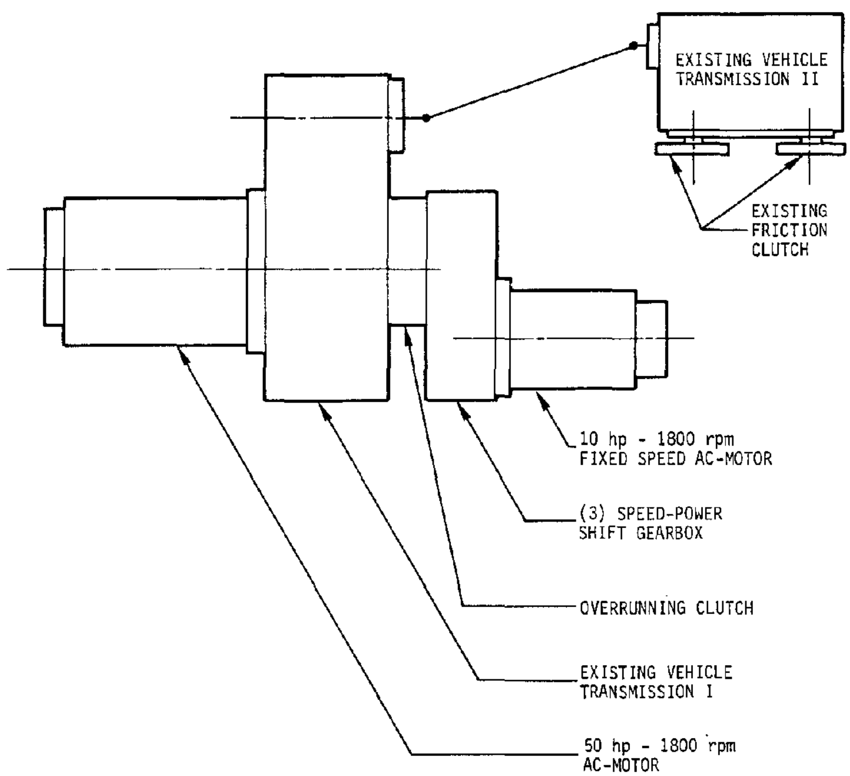

FIGURE 64. - Auxiliary drive - fixed speed ac motor with three-speed powershift gearbox.

5.3.6.5 50-hp Mechanical Powershift Transmission

The multiple speed powershift transmission which was considered next, at first glance offered the simplicity of design and the ruggedness desired for this application.

Closer examination revealed, however, that available transmissions could not be fitted into the available space envelope, and generally lacked the low speed output necessary to meet the desired "inching" velocity. A special output gear ratio was also considered, but had to be rejected because of high cost and long delivery.

5.3.6.6 Hydrostatic Drive (figure 65)

After reviewing the various electrical and mechanical drive options, it was reasoned that the time had arrived to consider a radically different drive concept; for example, one which would make full and continuous use of the 50-hp motors. Beyond this, it was desirable to have an onboard hydraulic supply capable of:

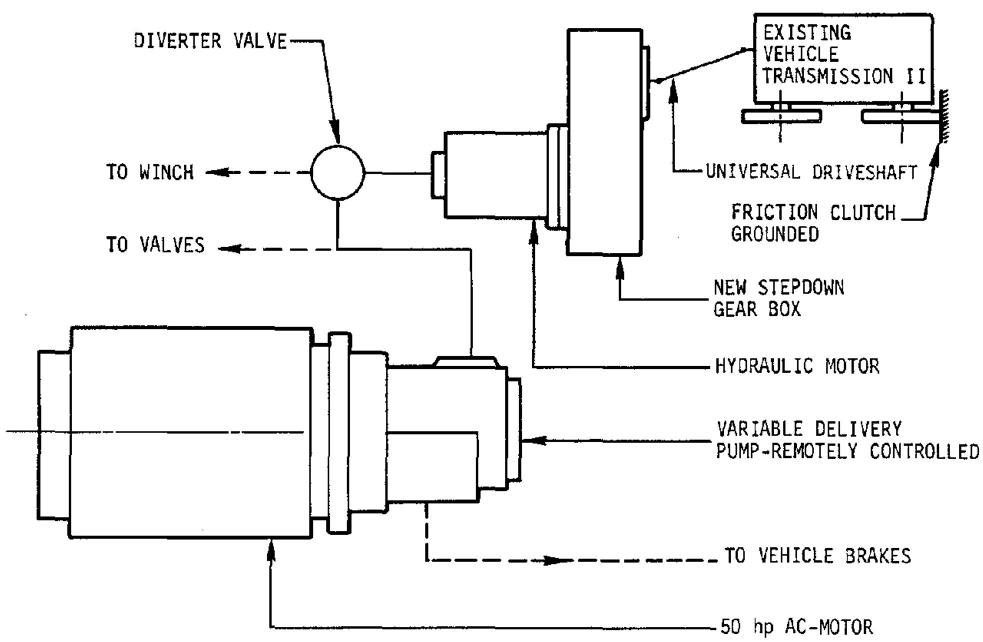

FIGURE 65. - Hydrostatic drive

a. Providing regulated hydraulic power to each track for the "inching" and tramming" functions

b. Infinitely variable speeds over a wide range, forward and backwards, and be remotely controllable

c. Providing hydraulic power to the winch

d. Providing hydraulic power to the three slurry pump control valves.

e. Providing hydraulic power to the slurry vehicle's fail-safe transmission brakes.

5.3.7 The Hydraulic Power-Supply System

The hydraulic system capable of meeting the design conditions consisted of an electric motor driving a variable delivery pump equipped with a remotely operated flow control. The output from each pump is piped to a fixed displacement hydraulic motor which drives one of the vehicle's tracks through the existing planetary transmission system. It should be noted that each of the vehicle's tracks is separately controlled and separated, powered by its own 50-hp hydraulic power supply. It certainly would have been more cost-effective to concentrate all power needs in one 100-hp centralized hydraulic supply instead of two 50-hp systems; vertical space limitations, however, did not permit the installation of a single, but larger diameter electric motor.

Motion reversal and stepless velocity changes are accomplished by a servo positioner which changes the tilt angle of the pump's swashplate. Position commands to the servo originate at two side by side joysticks located on an auxiliary console at the rear of the vehicle and are transmitted to the remotely located servo through low voltage circuits.

Oil circulates in a closed loop between each variable delivery pump and its associated load centers. From there it flows directly back to the main pump via pressurized lines. A charge pump built into the main pump provides a positive pressure of approximately 40 lb/in^2 on the suction side and also compensates for the leakage in the hydraulic system.

This flow path was chosen over an open loop gravity to tank return system because it suppresses the creation of oil foam and, hence, drastically reduces the required oil tank storage volume from approximately 50 gal to 8 gal.

Beyond this, the charge pressure is also used to relieve the vehicle's fail-safe transmission brakes[1]. Means for adequate cooling and filtration are also made. In addition, these are also the usual control element indicators which guard against system over-pressure, loss of oil, electric motor overload, etc.

5.3.7.1 Hydraulic Winch

The hydraulic winch is located in the center of the vehicle and underneath the conveyor. It shares the output from one of the variable delivery pumps with a traction drive motor through a manually engaged selector valve.

Winch speed control from full forward to zero, to full reverse, is accomplished through the same circuits and control stick which control the vehicle propulsion mode.

5.3.7.2 Valve Actuators

Hydraulic power for the operation of the three valve actuators is provided by a small integral gear pump attached to one of the variable delivery pumps. Oil under pressure is admitted either to the frontside or rearside of a rotary vane actuator by means of a remotely controlled solenoid valve.

Fail-safe operation under emergency conditions, such as loss of hydraulic pump power, is assured by means of three nitrogen charged hydraulic accumulators.

5.4 OPERATIONAL CONSIDERATIONS

5.4.1 Placement of Operator

There is no space available for an enclosed operator station within the vehicle outline. It was therefore decided to place him to the rear and to the left or right of the vehicle and let him follow the vehicle on foot.

A control station mounted on a swinging boom can, depending on circumstances, be placed on the rear left or right-hand side. No electrical disconnects are required since the mobile station is permanently wired to the vehicle by means of a flexible control cable.

---

[1]Remote application and release of the brakes is controlled from the auxiliary console at the rear of the vehicle.

5.4.2 Control Functions

The control station is designed to provide the following functions:

a. Motion control of the vehicle by means of two joysticks from full forward to full backwards at any velocity from 0 to 70 ft/min

b. Control of hydraulic winch by means of the right-hand joystick and a selector valve

c. On/off control of the feed screw, hydraulic pumps, slurry pump, crusher, conveyor, freshwater slurry bypass valve and vehicle brake control

d. Emergency motion stop of the vehicle and total emergency shutdown of all prime movers.

5.5 VEHICLE ELECTRICS

The electrical system of the SIV, as like any face operating underground coal mining vehicle, was designed and fabricated to meet all permissibility requirements.

The three main electrical enclosures which house all contactors and controls were manufactured by Ensign Electric Division of Hubbell and have received Mine Safety and Health Administration (MSHA) certification for meeting the explosion proof requirements of 30 CFR Part 18.

All six vehicle motors (conveyor, feed screw, injector, crusher and two-tram) were manufactured by Louis Allis and are certified as mine motors.

All gland fittings except those supplied on the motors are an approved type, supplied by Ensign Hubbell. All wiring is within MSHA approved conduit. The 980-V wiring is of type G-GC, SHD and has MSHA approval.

The lighting system is a Crouse-Hinds Mine Guard packaged system and has MSHA approval. This lighting system to date has not been installed on the vehicle.

The operator control station does not have MSHA certification. All of the switch functions are accomplished through MSHA approved Safe-Pack units. The tram controls on the operator station operate from intrinsically safe power supplies (J-Tee model PS-201, PN DAA0154-0001).

Foster-Miller has all of the required forms to begin the final certification process. Time and money expired before the total package could be submitted to MSHA.

A logic diagram of the SIV's electrical system is presented in figures 66, 67, 68 and 69.

5.6 VEHICLE TESTING

In December 1982, the Injector Vehicle was physically tested at the slurry test facility. Tests were conducted to determine the vehicle's coal handling capabilities and mobility. Although testing was rather limited due to cost and time constraints, sufficient testing was conducted to demonstrate the vehicle's ability to:

a. Process in excess of 6.4 tons/min of coal. With a 3-in top size anthracite coal (SG 1.75), the vehicle was capable of delivering 11 tons/min to the helical injector. A total of 186 tons of coal were processed before a failure in the chain conveyor assembly terminated testing

b. Discharge a slurry at 95 lb/in^2

c. Effectively scalp minus three inch coal and deliver plus three coal to the crusher

d. Have sufficient mobility to follow a continuous miner.

In addition, this testing also identified a number of vehicle assemblies requiring further development/ debugging. These systems include:

a. Chain conveyor

b. Crusher

c. Hydraulic circuitry

d. Slurry valve.

5.6.1 Chain Conveyor

The first problem with the chain conveyor occurred when one of the crosswise flights caught on one of the conveyor bed support members. The bent flight was subsequently removed and the protruding support was trimmed. A second problem occurred when one side of the conveyor drive chain jumped a link on the right front idler sprocket. This was corrected by disconnecting the conveyor chain and manually resetting the chain on the

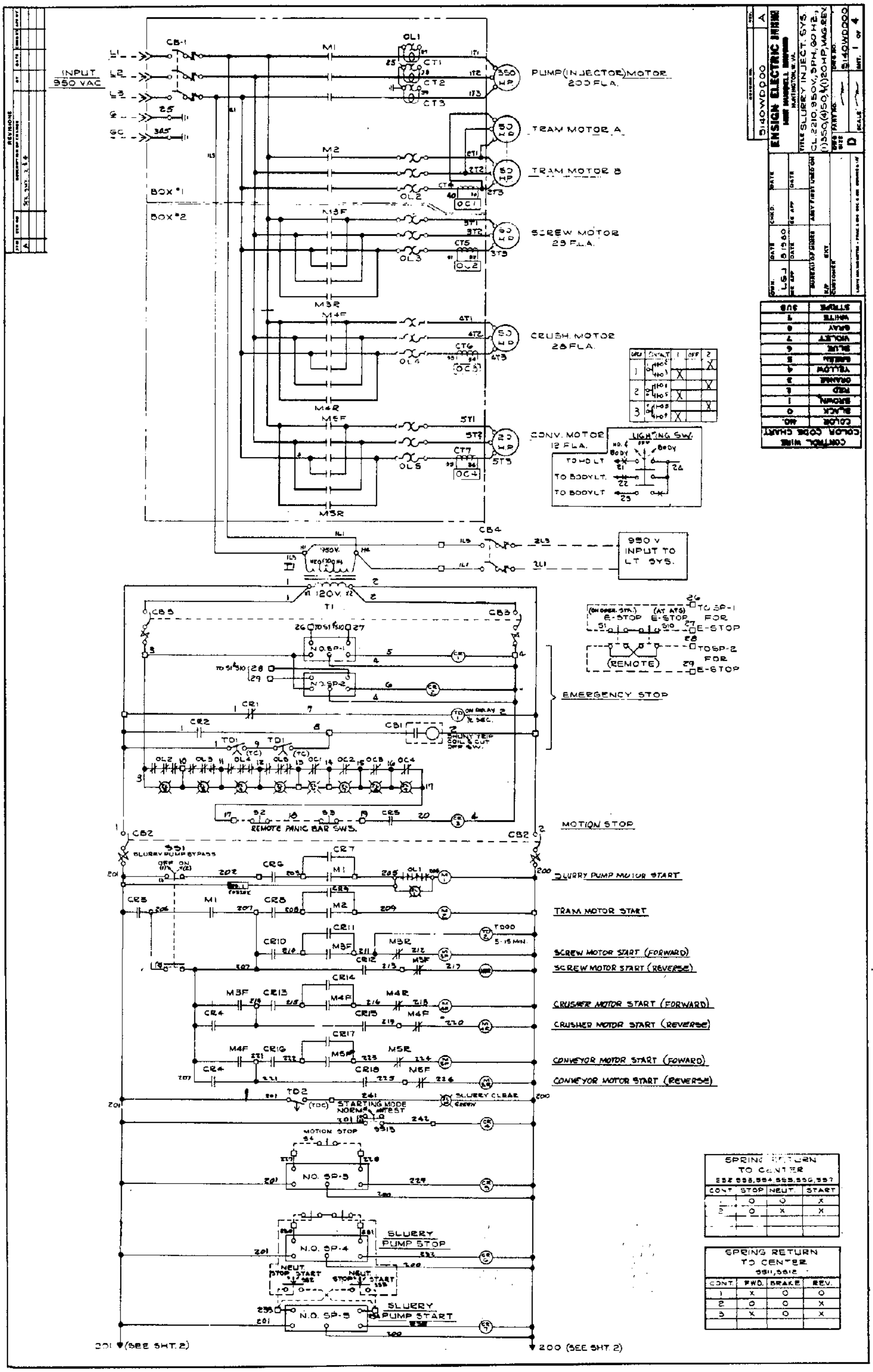

FIGURE 66. - SIV electrics.

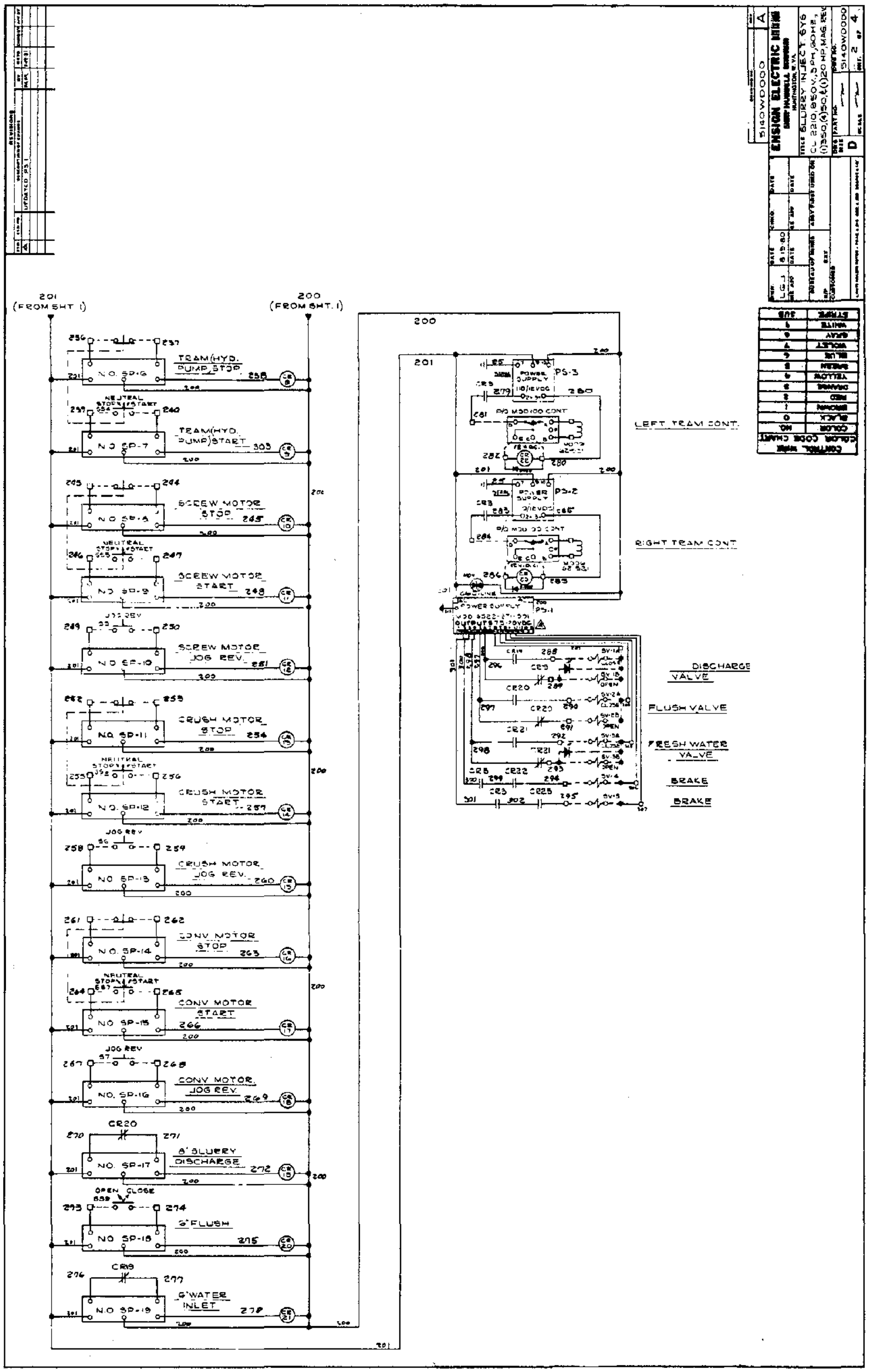

FIGURE 67. - SIV electrics.--Continued

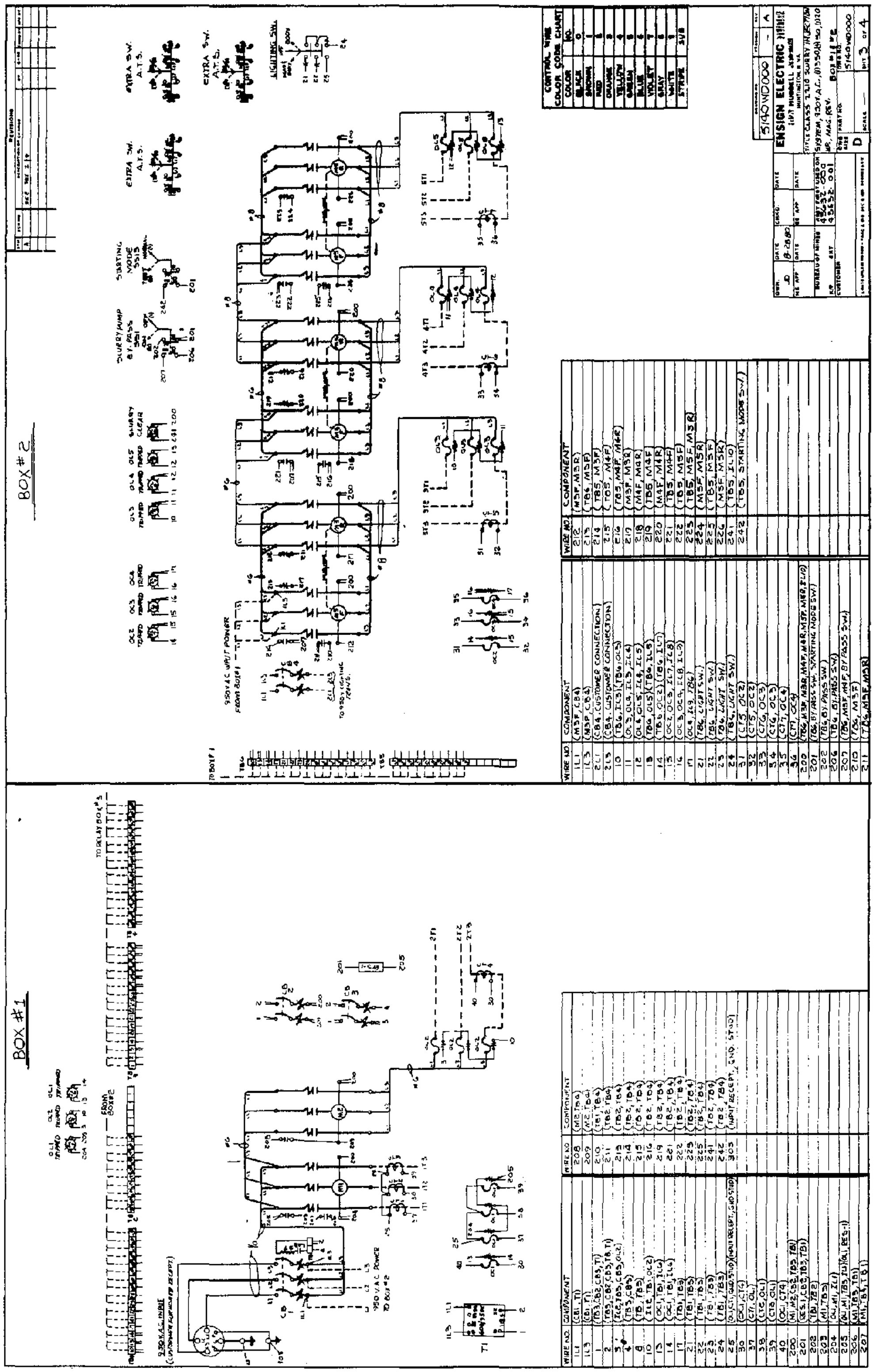

FIGURE 68. - SIV electrics.--Continued

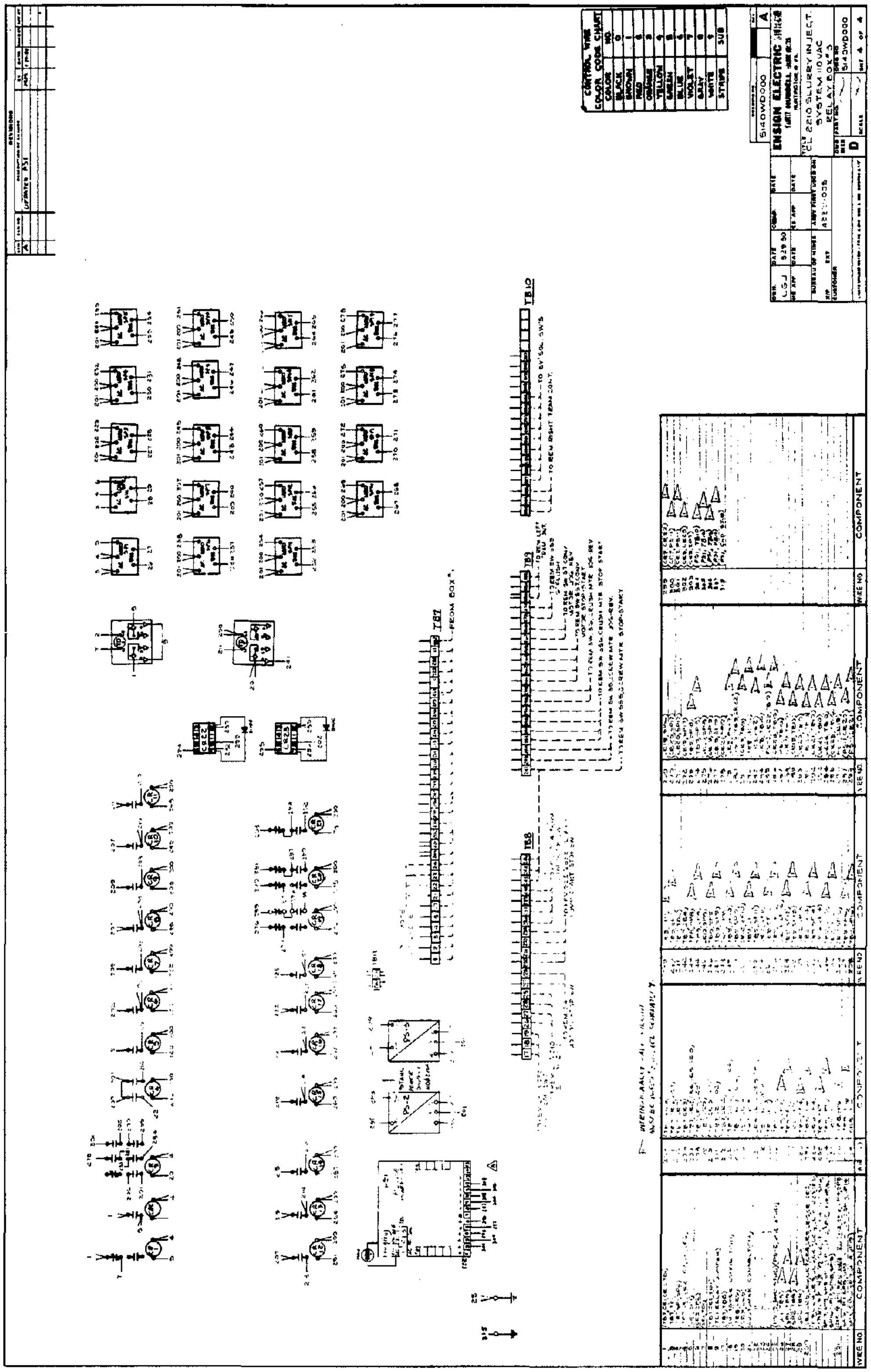

FIGURE 69. - SIV electrics.--Continued

sprocket. It was the failure of this same sprocket which eventually terminated vehicle coal handling tests. This occurred when the sprocket broke loose from its hub.

Two features should be added to the conveyor system to aid operation and maintenance. These include:

a. A mechanism for conveniently loosening or tightening the conveyor

b. A reversing switch on the control panel.

5.6.2 Crusher

The crusher's rotational speed is too high for this application. Arriving plus three inch coal was frequently thrown clear of the vehicle and never passed through the crusher. The crusher's speed should be substantially reduced, and an enclosure should cover the crusher rolls for better safety and operation.

5.6.3 Hydraulics

The hydraulic accumulators would not hold pressure and could not provide the necessary backup for valve cycling in the event of a power failure. Further debugging is required.

The vehicle's slurry valve and actuator assembly was the same unit used for all skid mounted injector testing. Through the years of operation, the actuator has developed some leakage. This has resulted in sluggish valve operation. The actuator seals should be replaced.

5.6.4 Vehicle Mobility

Vehicle mobility tests were conducted after completing the coal handling tests. Mobility tests were limited to a one-day demonstration. The vehicle trammed and maneuvered well once the operator became accustomed to the control levers. The control levers were relatively sensitive and, when combined with the short track base, the vehicle pivoted on its tracks with very little control lever motion. Once the operator attained a "feel" for the controls, maneuverability was easily controlled.

During the one-day demonstration, the right-hand track began to drag or would not pull on startup. Once the vehicle started moving, this track would start and keep pace with the left track. Because of this track problem, vehicle draw bar tests were abandoned.

Later conversations with the hydraulic manufacturer/ supplier indicated that the cause of this problem can be traced to the presence of air in the hydraulic circuitry. A complete bleeding of the right-side circuitry should correct this problem.

6. CONCLUSIONS - RECOMMENDATIONS

6.1 CONCLUSIONS

An injector pumping device/vehicle suitable for use in a 4-ft seam and sized for an 8- to 10-in-diam pipeline was designed, developed and tested by Foster-Miller. This program was executed under USBM Contract No. JO333914. A contract performance summary is included in table 15. Key results of this program, relative to performance of the prototype helical injector and vehicle, are highlighted as follows:

a. Coal[1] handling capability in excess of 11 tons/min at concentrations up to 50% by volume

b. Maximum injector discharge pressure is proportional to the square of the pump speed. At 1,000 rpm, the prototype injector developed a maximum of 98 lb/in^2

c. Pump efficiency at normal operating levels is in the 45 to 60% range

d. The injector is capable of handling any combination of flow and pressure below a discharge pressure of 98 lb/in^2 and is totally self-regulating to changes in volumetric flow and discharge pressure requirements

e. Some injector air ingestion was experienced at low discharge pressures. This parameter was not quantified

f. Acceptable parts life for the rotor and cone has not been demonstrated to date

g. All vehicle components except the roll tooth crusher, chain conveyor, hydraulic circuitry and the slurry valve performed satisfactorily. The nonenclosed crusher's speed was too high causing coal spit-back. The chain conveyor turning sprockets failed. Hydraulic hose failures were caused by improperly installed end-fittings. The slurry valve experienced recurring seal failures.

h. Based upon these conclusions and the contract performance summary, it is estimated that this program was 97% successful.

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[1]Anthracite at 1.7 SG.

TABLE 15. - Contract performance summary

Item	Contract specification	Prototype achievement
I. Concept development	Patent survey Literature search Manufacturer's review Concept development Concept analyses Injector recommendation	Phase I Report Accepted by USBM December 1975
II. Design and engineering • Coarse coal injector Helical Inducer Feed screw • Injector vehicle Coal feed system Inducer Tram operations ATS Chassis Coal top sizing Control systems • Boost pump evaluation	Engineering design Drawings Parts list Specifications Test program Test facility Model tests	Phase II Report Accepted by USBM June 1977
III. Prototype fabrication • Test facility	8-in pipeline loops; 300 ft, 600 ft and 1,000 ft long with associated operating equipment and instrumentation	Three 8-in pipeline loops; 350 ft, 750 ft, 1,000 ft long One 6-in pipeline loop; 350 ft long Equipment and instrumentation
• Injector vehicle Receiving hopper Coal sizing	2-ton storage Acceptable to hydraulic system	2-ton storage 3-in topsize Crusher/scalper
Pumping capacity	6.4 tons/min maximum	6.4 tons/min average 11.2 tons/min maximum

TABLE 15. - Contract performance summary--Continued

Item	Contract specification	Prototype achievement
Drawbar pull	20,000 lb	[1]Estimate exceeds 20,000 lb
Dimensions	Length: 27 ft Width: 9 ft Height: Operate in 4-ft seam	27 ft [2]10.4 ft 42 in
Weight	Adequate to allow maneuverability	25 lb/in^2 ground pressure loading
Mobility	Travel on difficult terrain Self-powered or towable tramming speed; 2 to 68 ft/min	Tested successfully in sand; not evaluated on incline Self-powered; dual hydrostatic track drive, 50 hp per side 2 to 80 ft/min; infinitely variable
Wearing parts	"Shall be abrasion resistant"	[3]Injector rotor and cone fabricated of HC-250; state-of-the-art abrasion resistant material
Controls	"Simple controls and read-out devices shall be specified for the operator"	Controls for the injector and vehicle are entirely contained in a 14- by 11- by 5-in enclosed box which is cantilevered off the vehicle at a convenient height for the operator. All functions and startup/shutdown operations are automatically sequenced for fail-safe operation

TABLE 15. - Contract performance summary--Continued

Item	Contract specification	Prototype achievement
	"Injector shall control the rate of injection of coal;" 0 to 50% volumetric concentration	C_V = 0 to 52%
• Safety Features	"...shall meet all applicable federal legislation, and design for permissibility."	• [4]All permissibility documentation was prepared
	"...injector shall fail in a safe manner."	• All motors equipped with electric current sensors which take the motor off-line if overloaded • Fail-safe brakes on each track • Slurry pump and feed screw drive systems include shear pin fuse links • Conveyor motor system includes a velocity switch for cut-off in case of break • Hydraulic accumulator system provides for autotic valve closure and bypass operation in case of power loss.

[1]Not measured; estimate based on static pull of weighted sled.
[2]Specification relaxed by government.
[3]Not fully tested due to lack of funding.
[4]MSHA approval process was not completed and lighting was not placed on the vehicle after underground tests were cancelled.

6.2 RECOMMENDATIONS

a. Endurance test the injector with the HC-250 cone and rotor to determine wear characteristics and overall reliability

b. Reduce speed of the crusher or provide an enclosure to prevent coal spit-back

c. Quantify air ingestion-discharge pressure relationship and evaluate effects of three-phase flow.

APPENDIX A.--MAXIMUM INJECTOR PRESSURE

Figure A-1 is a front view of the Annular Water Vortex showing the rotor swept circumference, ventilated core and inlet. An element of the vortex is shown enlarged. For the element with unit axial length, a balance of surface and body forces yields:

$$\Delta F = \rho X(\Delta r) r w^2$$

$$F = P \times \text{area} \qquad \text{(A-1)}$$

Since

$$\Delta P = \frac{\rho X(\Delta r) r w^2}{X \times 1}$$

$$\Delta P = \rho(\Delta r) r w^2 \qquad \text{(A-2)}$$

The pressure at a radius r, Pr, is determined by summation of the ΔP contributions of all the elements in a radial direction.

$$\Delta r \rightarrow 0$$

$$\eta \rightarrow \infty$$

In the limit with Δr and ΔP approaching 0, equation A-2 produces the integral:

$$Pr = \int_{r_a}^{r} \rho(\Delta r) r w^2$$

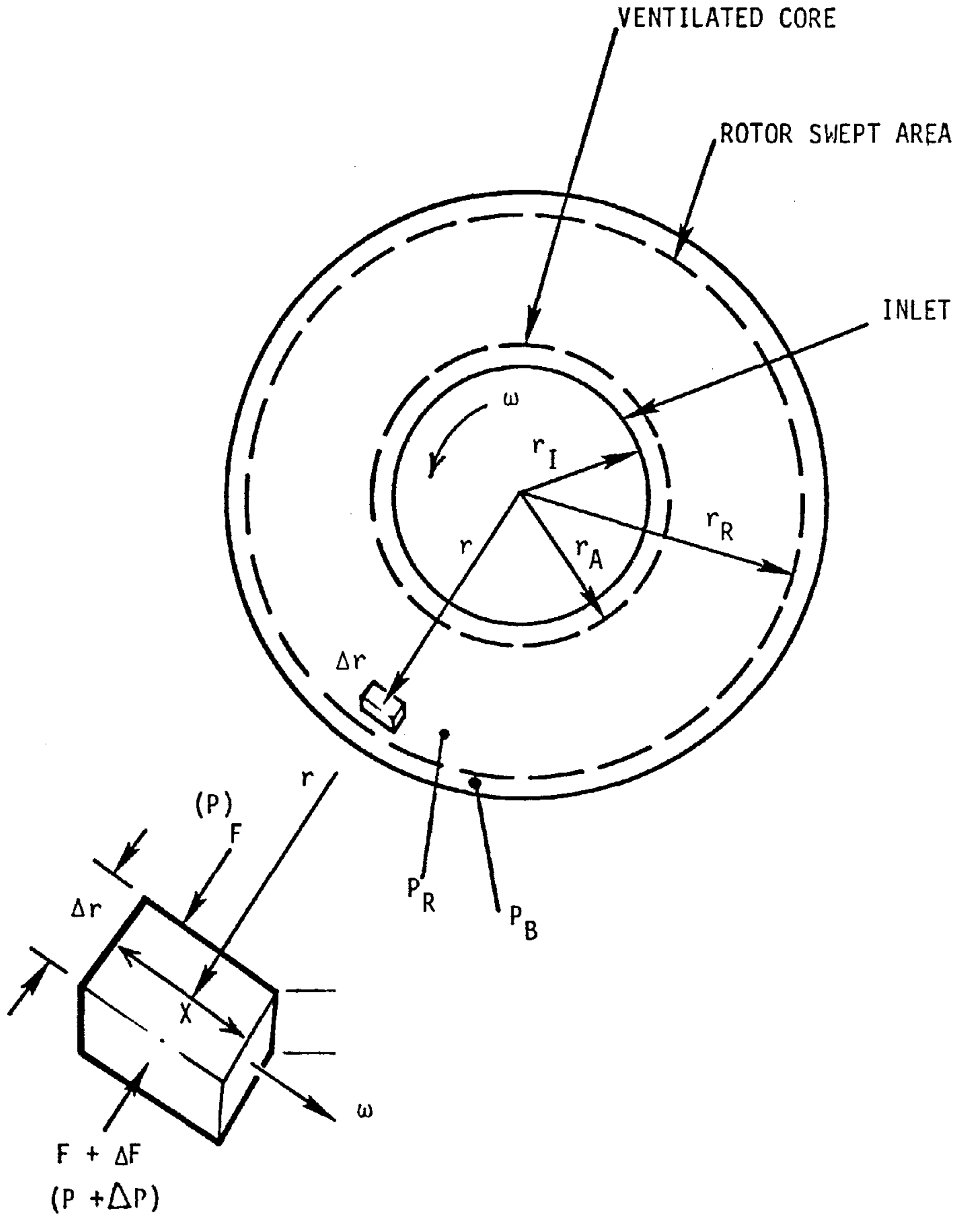


FIGURE A-1. - Front view of the annular water vortex.

Moving constants out of the integral gives:

$$Pr = \rho w^2 \int_{r_a}^{r} r(\Delta r)$$

Solving this gives:

$$Pr = \rho w^2 \int_{r_a}^{r} \frac{r^2}{2} \qquad \text{(A-3)}$$

The back pressure, P_B, occurs at the rotor outside diameter (at $r = r_R$). Thus:

$$P_B = \frac{\rho w^2}{2} \left(r_R{}^2 - r_A{}^2\right) \qquad \text{(A-4)}$$

The ventilated core cannot be smaller than the atmospheric pressure coal inlet without overflow of water or restriction of coal flow. Thus, the limiting theoretical maximum back pressure on the inducer is given by:

$$P_{B(max)} = \frac{\rho w^2}{2} \left(r_R{}^2 - r_I{}^2\right) \qquad \text{(A-5)}$$

One consistent set of units for use with this equation is:

$P \rightarrow lb/in^2$

$w \rightarrow rad/s$

$r \rightarrow in$

$\rho \rightarrow 9.3123 \times 10^{-5}\ lbf\text{-}s^2/in^4$ (water)

Factors which cause $P_{B(max)}$ to deviate from this theoretical value are:

a. Flow has some velocity with respect to the rotor and therefore swirl velocity is somewhat less than rotor speed.

b. Coal density is greater than that of water.

c. There may be a pressure recovery in the scroll.

d. There may be some free vortex outside the rotor outside diameter.

Equation A-5 proved to be accurate for a wide range of inducer configurations, speeds, and flows. Introducing a factor, K, in equation A-5 permits adjustment for the factors listed above.

$$P_{B(max),\ K} = K \frac{\rho w^2}{2} \left(r_R^{\ 2} - r_I^{\ 2} \right) \qquad \text{(A-6)}$$

For all combinations tested, K fell between 0.85 and 1.1.

APPENDIX B.--ROTOR BLADE LAYOUT

As part of the helical inducer design, a computer program has been written which provides the templates for the vane of the helical rotor. Figure B-1 shows the design variables which are used as input to the program, given in figure B-2.

A sample output of this program is given in table B-1. (This is the data which was used to lay out the vane of the 40° prototype rotor.) Plotting the two sets of points, NX, NY and OX, OY, provides a planar drawing which is the vane shape. Figure B-3 is a sample section plotted of the prototype rotor. This may be transferred directly to the vane material which is then cut, bent and welded to the hub. For a cast rotor, the vane would be cut out of a soft material and wrapped on a hub shape to provide a mold form. Rotor leading and trailing edges and final tip finishing are obtained by turning and grinding the rotor.

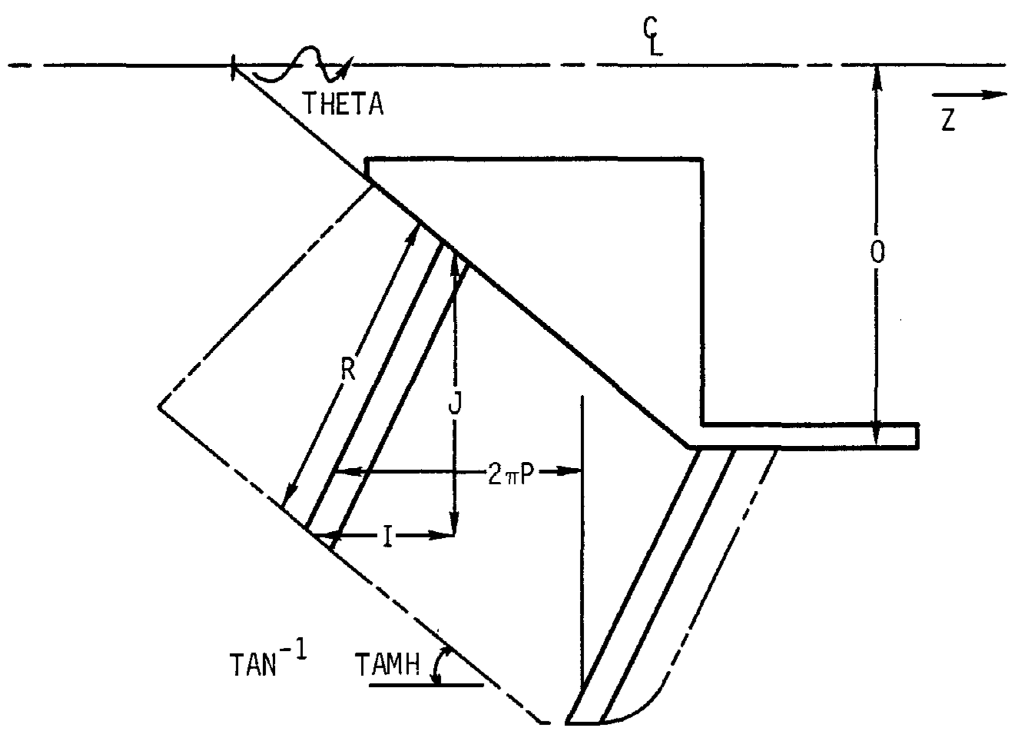


FIGURE B-1. - Rotor design program variables.

```
*GO
./list bhp1
/LOAD PLC
 ROTOR:PROC OPTIONS(MAIN);
 /* AUGER PROFILE PROGRAM */
                                                NOVEMBER 13, 1977 */
 PUT EDIT (' THETA       NX        NY        OX        OY')(A);
 DCL R FLOAT DEC INIT(2.6);
 DCL LLM FLOAT DEC INIT (0.0);
 DCL TANTH FLOAT DEC INIT(.8390996312);
 DCL I FLOAT INIT(1.139764982);
 DCL J FLOAT INIT (2.336452);
 DCL P FLOAT DEC INIT(.3169913754);
 DCL (X,Y,Z,X1,Y1,Z1,X1P,Y1P,Z1P,NX,NY,OX,OY,MX,MY,A,B,C,
                    D,LON,THE,THETA,PHEE) FLOAT DEC INIT(0);
 XP=TANTH*P*SIN(1);
 YP=TANTH*P*COS(1);
 ZP=P;
 X1P=(P*TANTH+J)*SIN(1);
 Y1P=(P*TANTH+J)*COS(1);
 Z1P=(P-I);
 PUT SKIP(2);
 DO TH=1.1 TO 20 BY .1;
 D=TANTH*P*TH;
 IF D>=3.05 THEN D=3.05;
 X=D*SIN(TH);
 Y=D*COS(TH);
 Z=P*TH;
 X1=(D+J)*SIN(TH);
 Y1=(D+J)*COS(TH);
 Z1=P*TH-I;
 A=SQRT(((X-XP)**2)+((Y-YP)**2)+((Z-ZP)**2));
 B=SQRT(((X1-X1P)**2)+((Y1-Y1P)**2)+((Z1-Z1P)**2));
 C=SQRT(((X1-XP)**2)+((Y1-YP)**2)+((Z1-ZP)**2));
 THE=((R**2+(C**2)-(A**2))/(2*R*C));
 THETA=ATAN((SQRT(1-THE**2))/THE);
 OX=-C*SIN(LLM-THETA)+MX;
 OY=C*COS(LLM-THETA)+MY;
 THE=((B**2-(R**2)-(C**2))/(-2*R*C));
 PHEE=ATAN((SQRT(1-THE**2))/THE);
 LON=LLM-THETA+PHEE;
 NX=OX+(R*SIN(LON));
 NY=OY-(R*COS(LON));
 PUT SKIP EDIT (TH,NX,NY,OX,OY)(F(7,4),X(3));
 XP=X; YP=Y; ZP=Z; Z1P=Z1; X1P=X1; Y1P=Y1;
 LLM=LON; MX=NX; MY=NY;
 END;
 END ROTOR;
*GO
.bhp1
```

FIGURE B-2. - Rotor design program.

TABLE B-1. - Sample program output

THETA	NX	NY	OX	OY
1.1000	0.2640	0.0198	0.0487	2.6108
1.2000	0.5280	0.0617	0.0979	2.6259
1.3000	0.7902	0.1262	0.1472	2.6454
1.4000	1.0485	0.2131	0.1964	2.6695
1.5000	1.3011	0.3225	0.2450	2.6983
1.6000	1.5459	0.4540	0.2927	2.7320
1.7000	1.7809	0.6071	0.3390	2.7707
1.8000	2.0041	0.7812	0.3834	2.8143
1.9000	2.2137	0.9755	0.4255	2.8629
2.0000	2.4077	1.1888	0.4646	2.9164
2.1000	2.5844	1.4200	0.5004	2.9747
2.2000	2.7420	1.6677	0.5322	3.0377
2.3000	2.8791	1.9304	0.5596	3.1051
2.4000	2.9941	2.2063	0.5819	3.1766
2.5000	3.0857	2.4936	0.5987	3.2519
2.6000	3.1528	2.7903	0.6095	3.3306
2.7000	3.1943	3.0943	0.6139	3.4122
2.8000	3.2096	3.4033	0.6112	3.4961
2.9000	3.1979	3.7152	0.6013	3.5819
3.0000	3.1589	4.0275	0.5837	3.6688
3.1000	3.0923	4.3378	0.5582	3.7562
3.2000	2.9983	4.6436	0.5245	3.8433
3.3000	2.8770	4.9426	0.4825	3.9294
3.4000	2.7290	5.2322	0.4321	4.0137
3.5000	2.5548	5.5100	0.3733	4.0953
3.6000	2.3555	5.7736	0.3062	4.1735
3.7000	2.1321	6.0208	0.2308	4.2473
3.8000	1.8860	6.2493	0.1476	4.3160
3.9000	1.6188	6.4571	0.0567	4.3787
4.0000	1.3322	6.6420	-0.0414	4.4345
4.1000	1.0282	6.8024	-0.1462	4.4828
4.2000	0.7089	6.9365	-0.2571	4.5226
4.3000	0.3765	7.0429	-0.3734	4.5534
4.4000	0.0334	7.1203	-0.4944	4.5744
4.5000	-0.3177	7.1675	-0.6193	4.5851
4.6000	-0.6742	7.1839	-0.7472	4.5849
4.7000	-1.0335	7.1687	-0.8770	4.5734
4.8000	-1.3926	7.1216	-1.0079	4.5503
4.9000	-1.7488	7.0426	-1.1387	4.5152
5.0000	-2.0991	6.9316	-1.2683	4.4679
5.1000	-2.4408	6.7892	-1.3956	4.4085
5.2000	-2.7709	6.6160	-1.5195	4.3370
5.3000	-3.0866	6.4130	-1.6387	4.2534
5.4000	-3.3853	6.1812	-1.7521	4.1582
5.5000	-3.6643	5.9222	-1.8586	4.0515
5.6000	-3.9212	5.6376	-1.9570	3.9340
5.7000	-4.1534	5.3293	-2.0462	3.8063
5.8000	-4.3589	4.9994	-2.1251	3.6690
5.9000	-4.5357	4.6504	-2.1928	3.5230
6.0000	-4.6818	4.2846	-2.2483	3.3692

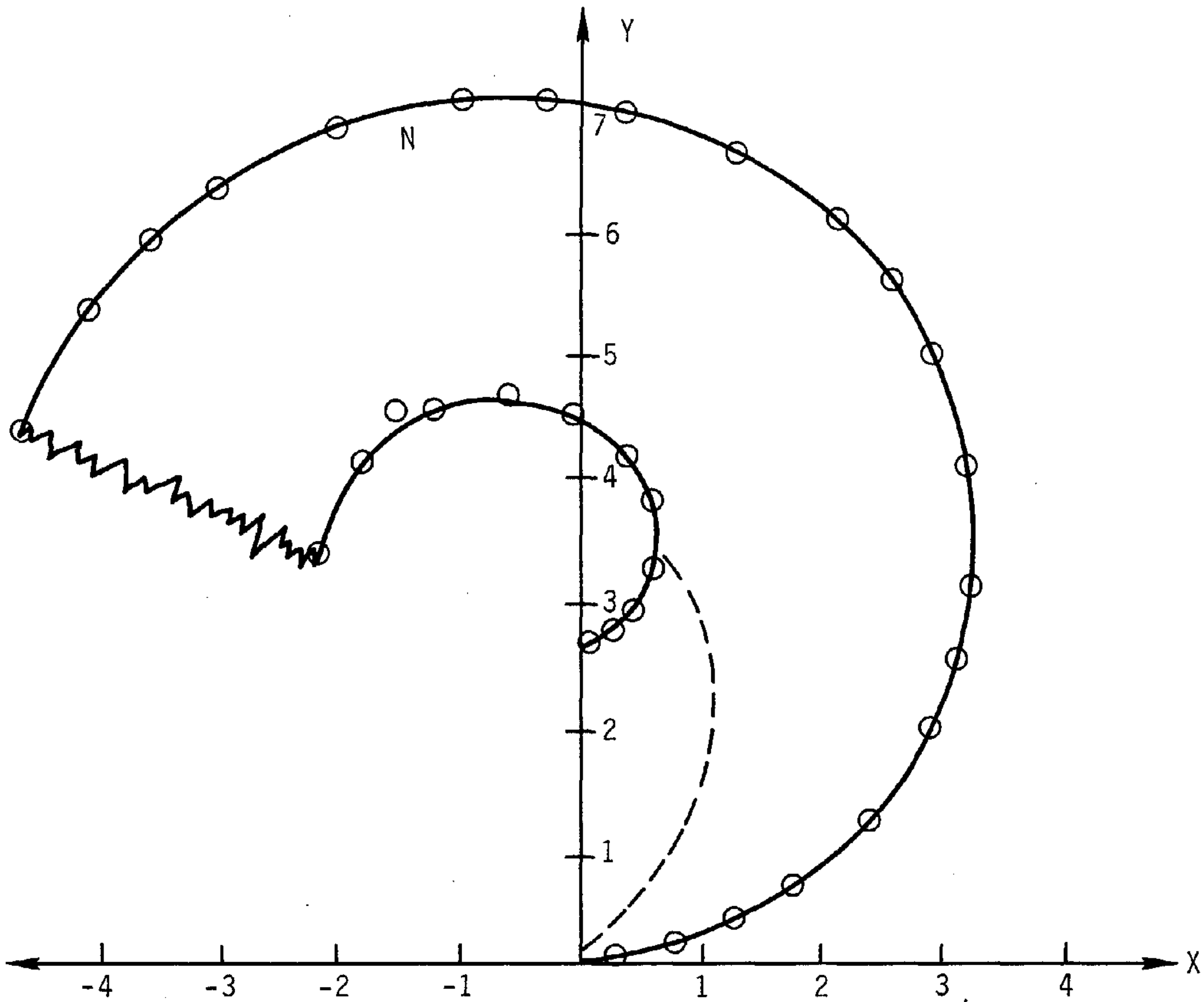


FIGURE B-3. - Plotted section of prototype rotor.

APPENDIX C.--ATMOSPHERIC TRANSFER SUMP (ATS)

C.1 SYSTEM DESCRIPTION

The ATS was to be used as the junction between the face haulage subsystem and the demonstration mine's main hydraulic system. Because the use of the demonstration mine was withdrawn, this unit was never tested. However, the design is included in this report as documentation for work done in the contract and for any potential application in the future. As schematically shown in figure C-1, this unit provides the hydraulic functions of stabilizing slurry volumetric flow, removing air and initiating boost pumping by centrifugal pumps.

The 3-ft high ATS has a level control which maintains the level by controlling mainline water flow. Water is pumped from the ATS to the face haulage injector at a constant volume flow rate. Because water enters the injector at atmospheric pressure, the flow is unaffected by pressure changes in the slurry system. Coal from the face is injected into the face haulage slurry line by the helical injector at any rate up to the maximum design flow of the system, the slurry velocity in the line being allowed to vary with coal flow rate.

The slurry from the face injector enters the ATS and, with proper baffling, is pumped from the ATS by the ATS slurry pump. The pumping volume of this pump is designed to handle the maximum potential flow. Boost pump flow above that provided from the faces is automatically drawn from the water supply in the ATS. Sufficient volume exists in the sump to permit ample response time for a control valve to correct the incoming flow rate. Any trapped air in the face slurry line surfaces in the ATS and is not transferred to the long main haulage system, which could be damaged by water hammer due to air passing through boost pumps. Additionally, the main haulage slurry line control is simplified by a constant head at the first boost pump.

C.2 DESIGN AND ANALYSIS OF ATS LEVEL CONTROL

The requirement of the level control is to maintain the water level in the ATS within ±6 in as the coal flow varies from nil to a volume equivalent to 1,000 gal/min and the water flow is constant at 1,500 gal/min. Figure C-2 shows a diagram of the system as presently conceived at 2,500 and 1,500 gal/min respectively. The slurry flow from the mine face is assumed to vary from 1,500 to 2,500 gal/min. The flow out of the control valve must, therefore, vary from 1,500 to 2,500 gal/min.

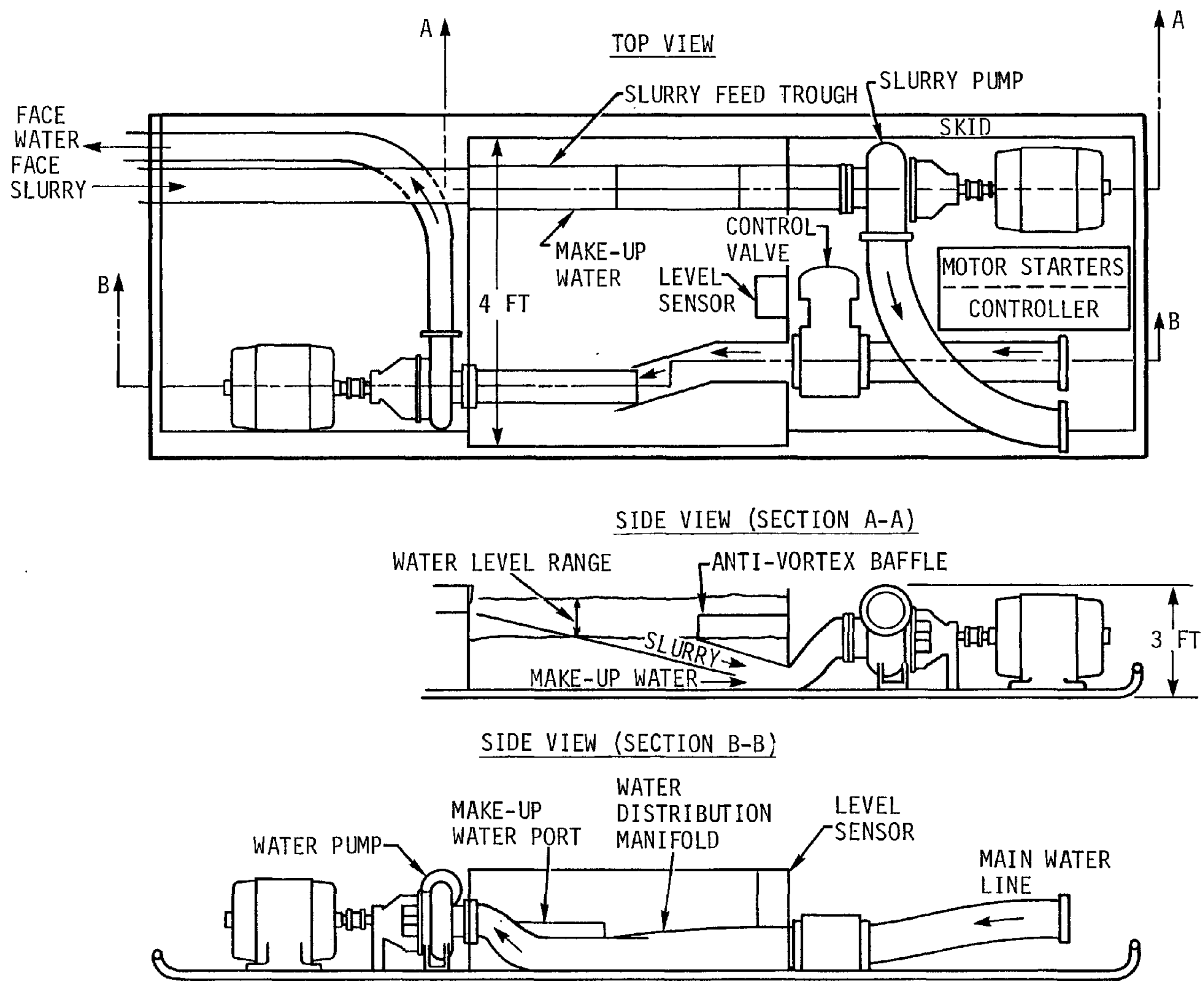


FIGURE C-1. - Atmospheric transfer sump.

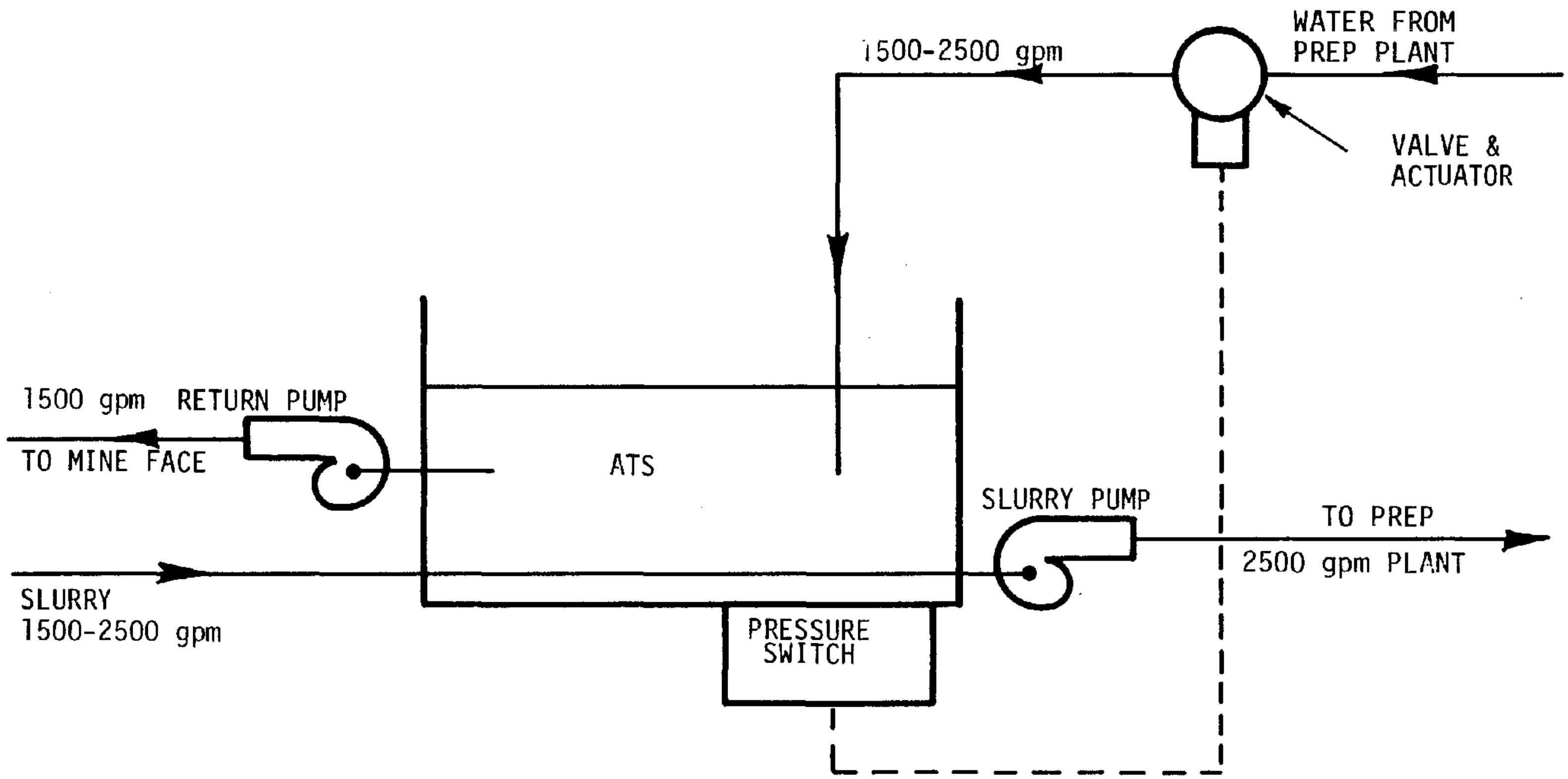


FIGURE C-2. - Level control schematic.

The control valve is operated by a reversible electric motor through a gear train. Figure C-3 shows an electrical schematic of the system. Switches 1 and 2 are cam actuated limit switches. As shown in the diagram, the valve is in the "closed" position and switch 1 is in the normal condition.

Since the flow out of the valve is low, the level in the tank is dropping and eventually the pressure switch will be actuated so that the "low" contact is energized. The motor starts to open the valve and switch 1 is actuated by the cam closing the normally open contacts. Switch 2 remains actuated until the valve "open" position is reached at which point switch 2 reverts to the normal position opening the normally open contacts. Current flow to the motor is interrupted and the valve stops.

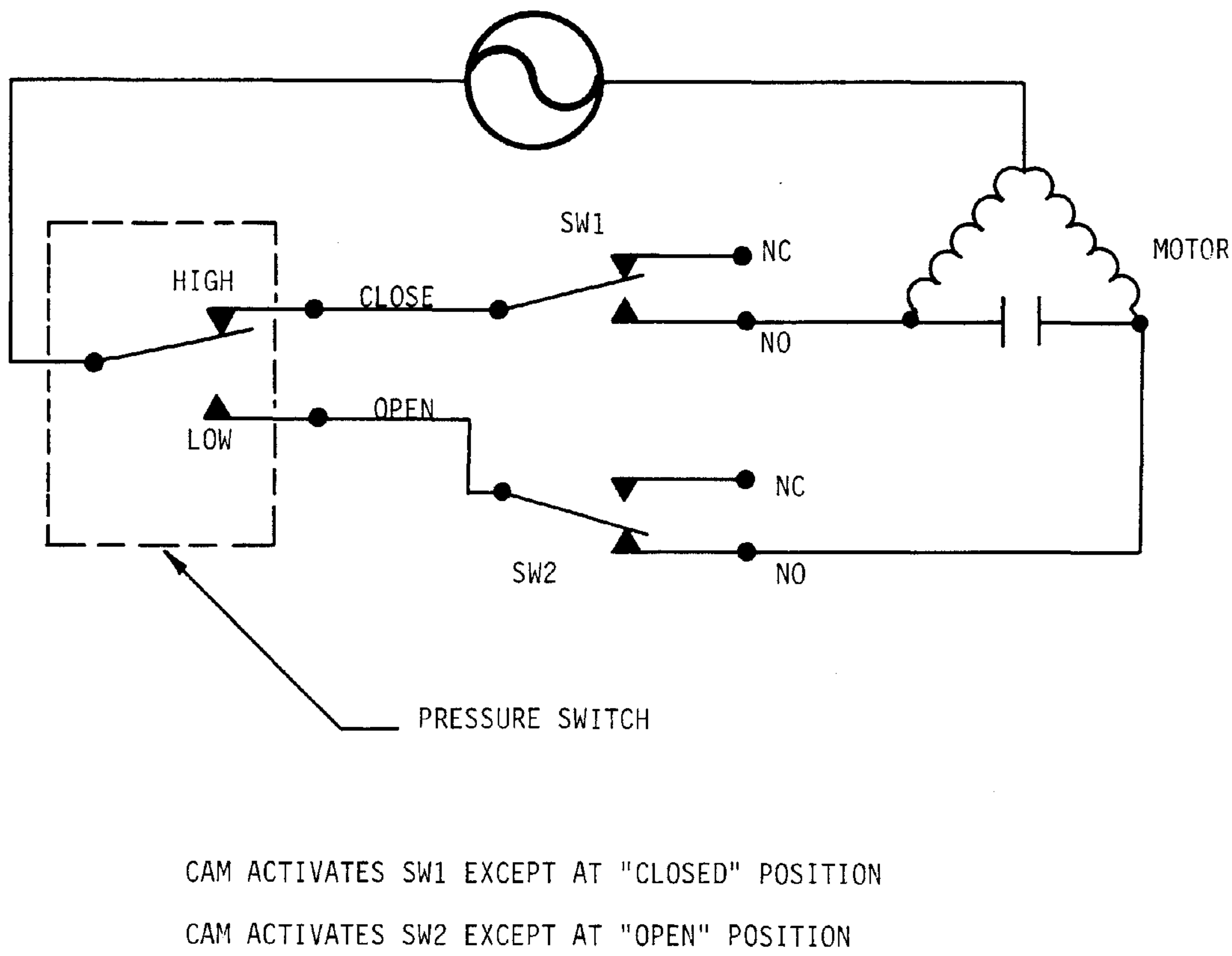


FIGURE C-3. - Pressure switch wiring.

The level in the tank is now rising and soon the pressure switch will return to the "high" position as shown in the diagram. The normally open contacts of switch 1 are closed so current flows through the motor to close the valve. When the valve reaches the "closed" position switch 1 reverts to normal, the normally open contacts open and the motor stops. In this way, one complete cycle has taken place. The "closed" and "open" positions of the valve are independently selected by adjusting the cam operated limit switches 1 and 2. Flow out of the control valve is 1,500 gal/min in the "closed" position and 2,500 gal/min in the "open" position.

Figure C-4 shows how the flow out of the control valve and the water level in the tank vary with time while the coal flow is steady at 500 gal/min. When the control valve is in the "closed" position, flow rate into the tank is 500 gal/min less than the flow out and the level drops at a rate:

$$R = \frac{500}{60 \times 7.48} \frac{1}{A_T} = \frac{1.11}{A_T} \text{ ft/s}$$

where

A_T = cross sectional area of tank in square feet

At zero time, the level has dropped to the point where the low setting of the pressure switch is reached and the control valve opens at a constant rate until switch 2 reverts to the normal condition at the valve "open" position. To simplify the analysis, flow out of the control valve is assumed to respond instantaneously to valve position.

The level in the ATS is the integral of the net flow and so the level continues to drop after the valve starts to open until the net flow out of the tank is equal to zero. Beyond this point, the level rises with an increasing slope until the valve reaches the "open" position and stops. The level continues to increase linearly until the high level is reached and the valve starts to close. The level continues to rise until the net flow into the tank is zero and starts to fall with an increasing slope until the valve reaches the "closed" position and stops, continuing to fall at a constant rate until the low level is reached completing one cycle. At a steady coal flow of 500 gal/min, the curves are symmetrical with equal periods of time at 1,500 and 2,500 gal/min, averaging 2,000 gal/min. At lower coal flow, the proportion of the time during which the valve is "open" and delivering

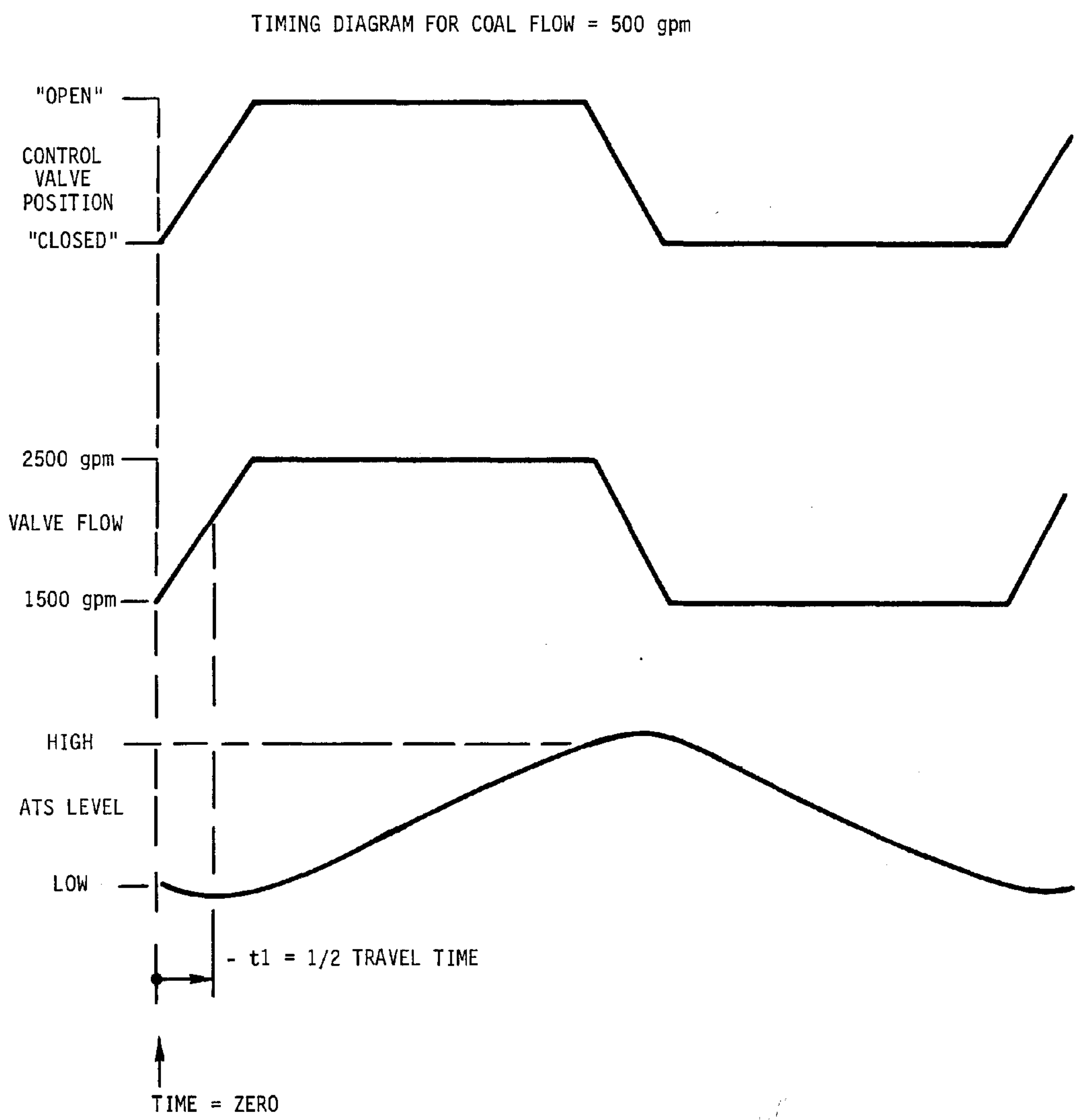


FIGURE C-4. - Timing diagram for coal flow = 500 gal/min.

2,500 gal/min would be longer than the time during which it is "closed" and delivering 1,500 gal/min. The valve would remain "open" all the time, delivering 2,500 gal/min when the coal flow ceases altogether. The reverse happens as the coal flow increases, the valve remaining "closed" all the time delivering 1,500 gal/min when the coal flow is 1,000 gal/min (maximum).

Note that the lowest tank level is below the low pressure switch setting and the highest is higher so the high to low differential must be less than the 12 in allowed. The lowest level is reached at t_1 when the valve has completed half of its travel and the flow equals 2,000 gal/min.

The change in level from $t = 0$ to $t = t_1$ is:

$$\Delta h = \frac{1}{60} \times \frac{1}{7.48} \times \frac{1}{A_T} \int_0^{t_i} (Q - 2{,}000)dt$$

$$Q = 1{,}500 + \frac{500\ t}{t_1}$$

$$\Delta h = \frac{500}{2} \left(\frac{1}{60 \times 7.48} \right) \frac{t_1}{A_T}$$

The change is proportional to the average flow deficiency for the time t_1 and inversely proportional to the tank cross sectional area. The minus sign signifies a drop in level. The most severe case for a given t_1 and A_T is when the flow deficiency is maximum. If the coal flow is near maximum and is suddenly reduced to nearly zero at the instant the low level is reached

$$\Delta h = -\frac{1{,}000}{2} \left(\frac{1}{60 \times 7.48} \right) \frac{2\ t_1}{A_T}$$

The drop in the level is four times as great as the nominal condition because the flow deficiency is doubled and it takes twice as long to establish the new equilibrium valve flow. An equal overshoot in level will occur if the coal flow is near zero and is suddenly increased to 1,000 gal/min at the instant the high level is reached.

The requirement to keep the level at ±6 in or ±0.5 ft leads to:

$$2\ \Delta h + h_{high} - h_{low} = 1\ \text{ft}$$

By definition:

$$h_{high} - h_{low} = \Delta P_s$$

which is the pressure switch differential

$$\Delta h = \frac{1 - \Delta P_s}{2} = \frac{2.33\ t_1}{A_T}$$

Given the pressure switch differential and the valve actuation time the above equation defines the minimum tank cross sectional area required to meet the level tolerance.

The proper actuator for the valve will have a speed of 10°/s, or a travel time from "closed" to "open" of 2.5 s and t_1 = 1.5 s (see fig. C-4).

$$A_T = \frac{2.33\ (1.25)\ 2}{1 - \Delta P_s} = \frac{5.58}{1 - \Delta P_s}$$

The electric motor for an actuator should have its duty cycle limited to 25%.

$$\frac{\text{on time}}{\text{on time + off time}} = 0.25$$

$$\text{on time} = \frac{\text{off time}}{3} = 2.5\ \text{s}$$

$$\text{off time} = 7.5\ \text{s} = \frac{\Delta P_s}{R} \quad \frac{\Delta P_s (A_T)}{1.11}$$

$$A_T = \frac{8.33}{\Delta P_s}$$

which is a second criterion for the cross sectional area of the tank, both a function of ΔP_S only. Since $(A_T)_1$ for the first criterion increases with increasing ΔP_S and $(A_T)_2$ for the second criterion decreases with increasing ΔP_S, the minimum A_T to satisfy both is when they are equal:

$$\frac{5.58}{1 - \Delta P_s} = \frac{8.33}{\Delta P_s}$$

$$\Delta P_s = 0.60 \text{ ft}$$

for which $A_T = 13.9 \text{ ft}^2$.

Since this was a much smaller tank than anticipated, it seemed reasonable to increase the size to gain design margin and so a 4 x 8 (32 ft^2) tank cross sectional area was recommended. With a 0.58-ft (0.25-lb/in^2) differential pressure switch:

Maximum overshoot =

$$\Delta h = \frac{2.79}{32} = 0.087 \text{ ft}$$

Maximum swing of the tank level =

$$2\,\Delta h + \Delta P_s = 0.75 \text{ ft}$$

$$\text{Off time} = \frac{0.58(32)}{1.11} = 16.7 \text{ s}$$

Duty cycle = 13%

With a differential pressure switch of 0.29 ft:

Maximum swing of the tank level = 0.46 ft

Duty cycle = 23%

APPENDIX D.--STF CONTROL ROOM CIRCUITRY

The STF ladder logic diagram for the control room is presented in figure D-1.

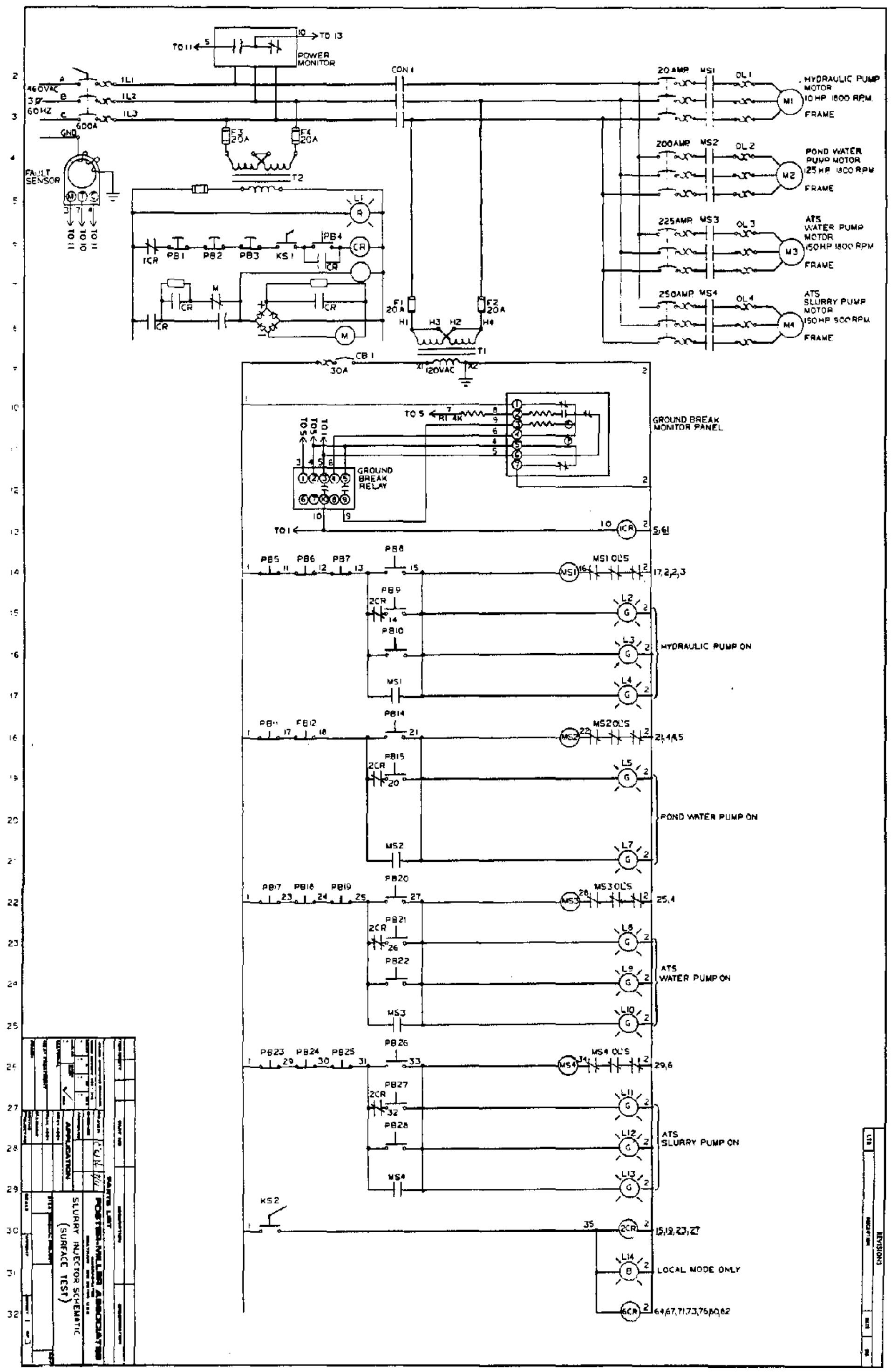


FIGURE D-1. - Control room ladder logic.

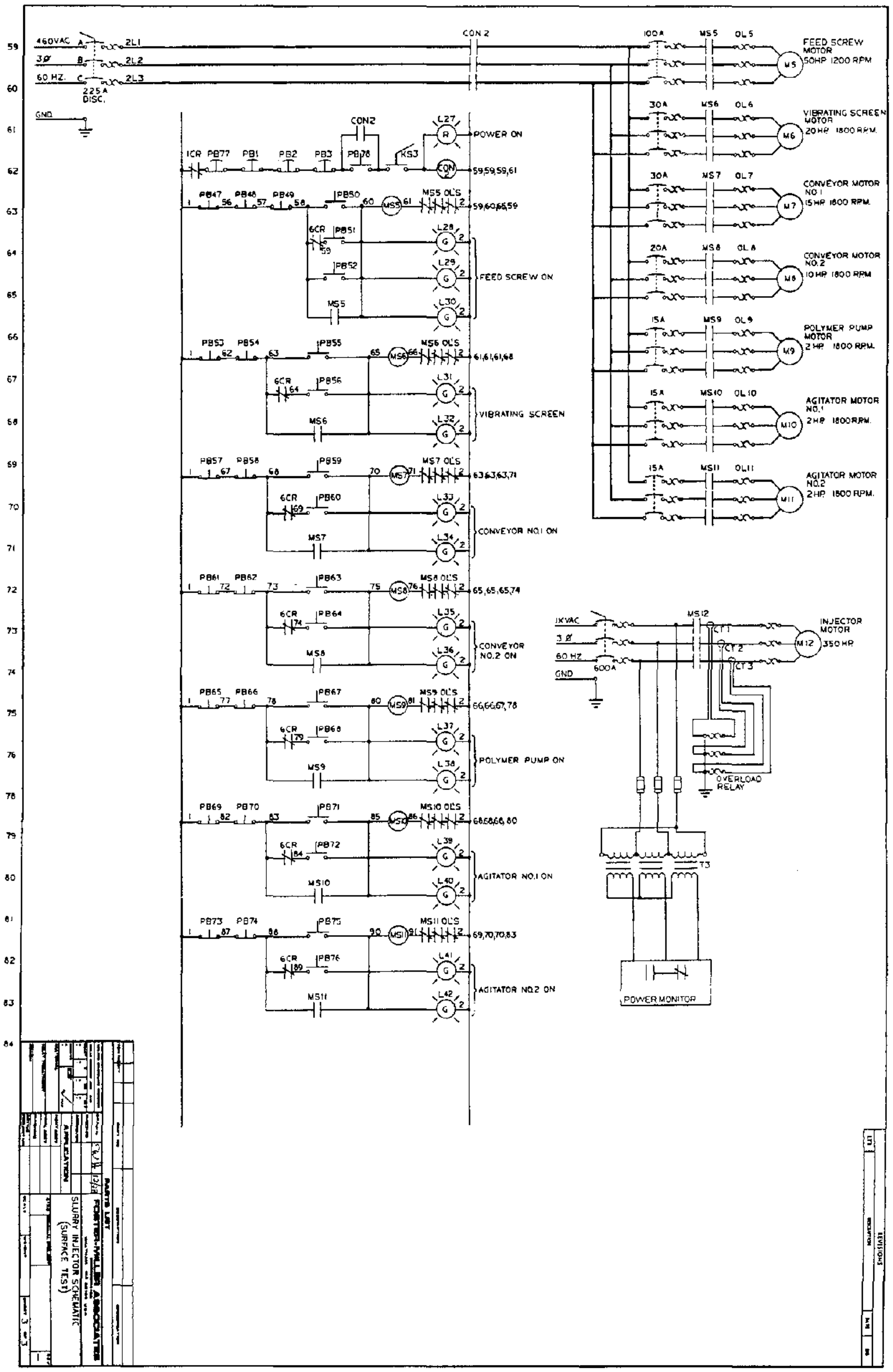


FIGURE D-1. - Control room ladder logic.--Continued

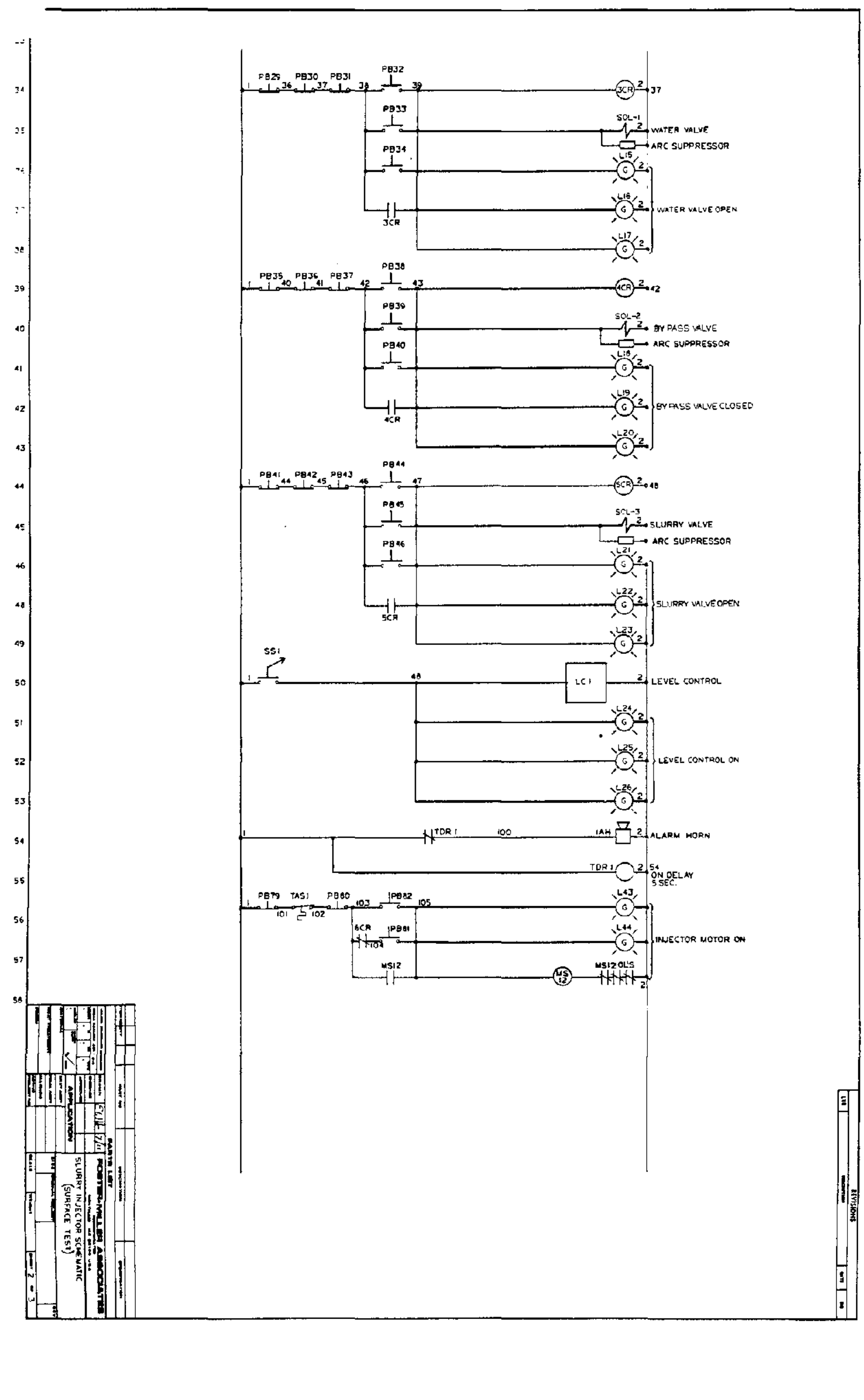


FIGURE D-1. - Control room ladder logic.--Continued

APPENDIX E.--FRICTION FACTOR CALCULATIONS AND MINOR LOSS CALCULATIONS

E.1 GENERAL

Apply Bernoulli equation between any two points in a pipeline;

$$\frac{V_1^2}{2g} + \frac{P_1}{\gamma} + Z_1 = \frac{V_2^2}{2g} + \frac{P_2}{\gamma} + Z_2 + H_f$$

$$+ H_k \frac{\text{ft-lb}}{\text{lb}} = \text{ft} \qquad \text{(E-1)}$$

where all terms are in feet of fluid.

Assume pipe diameter at point 1 and point 2 are equal, then

$$V_1 = V_2$$

and

$$H_f = \frac{P_1 - P_2}{\gamma} + Z_1 - Z_2 - H_k$$

For water, $\gamma = 62.4 \text{ lb/ft}^3$ and P_1 and P_2 in pounds per square inch.

$$H_f = \frac{P_1 - P_2}{(62.4)} (144) + Z_1 - Z_2 - H_k$$

$$H_f = (P_1 - P_2)(2.308) + (Z_1 - Z_2) - H_k \qquad \text{(E-2)}$$

$$H_f = \frac{fL}{D} \frac{V^2}{2g} \quad \text{(Darcy friction factor)}$$

letting

$$a = \frac{fL}{D}\frac{1}{2g}$$

and

$$b = 2$$

then

$$H_f = aV^b \tag{E-3}$$

and

$$f = \frac{2\ gDa}{L} \tag{E-4}$$

$$\text{Velocity:}\quad V = \frac{Q}{A_p}$$

For water and Q measured in gallons per minute

$$V = \frac{Q\ \text{gal/min} \times 2.228 \times 10^{-3}}{A_p} = \frac{Q\ \text{gal/min}}{448.833 \times A_p}$$

For 8-in ductile iron pipe

$$V_{8\ \text{in}} = \frac{Q\ \text{gal/min}}{448.833 \times 0.383} = \frac{Q\ \text{gal/min}}{171.89}\ \text{ft/s}$$

For 6-in schedule 40 steel pipe

$$V_{6\ \text{in}} = \frac{Q\ \text{gal/min}}{448.833 \times 0.200} = \frac{Q\ \text{gal/min}}{89.89}$$

Equation E-3 is a power curve which by a regression analysis can be transformed into a general linear equation

$$Y = A + bX$$

where

$Y = \ln H_f$

$A = \ln a$

$X = \ln V$

The general linear equation becomes

$$\ln H_f = \ln a + b \ln V$$

a and b are regression coefficients which are found by solving the following system of linear equations

$$\begin{bmatrix} n & \Sigma \ln V_i \\ \Sigma \ln V_i & \Sigma(\ln V_i)^2 \end{bmatrix} \begin{bmatrix} \ln a \\ b \end{bmatrix} = \begin{bmatrix} \Sigma \ln H_{f_i} \\ \Sigma(\ln H_{f_i} \ln V_i) \end{bmatrix}$$

$$n \ln a + b\Sigma \ln V_i = \Sigma \ln H_{f_i}$$

$$\ln a\Sigma \ln V_i + b\Sigma(\ln V_i)^2 = \Sigma(\ln H_{f_i} \ln V_i)$$

$$\ln a = \frac{\Sigma \ln H_{f_i} - b\Sigma \ln V_i}{n}$$

$$b = \frac{\Sigma(\ln H_{f_i} \ln V_i) - \ln a\Sigma \ln V_i}{\Sigma(\ln V_i)^2}$$

Substituting for $\ln a$ and solving for b gives:

$$b = \frac{\Sigma \ln H_{f_i} \ln V_i - \frac{\Sigma \ln V_i \Sigma \ln H_{f_i}}{n}}{\Sigma(\ln V_i)^2 - \frac{(\Sigma \ln V)^2}{n}}$$

Then substituting for b and solving for

$$a = \exp\left(\frac{\Sigma \ln H_{f_i}}{n} - \frac{b\Sigma \ln V_i}{n}\right)$$

E.2 FRICTION FACTOR

Substituting a and b into equation E-3 will give the best fit curve for a given set of data.

The friction factor data was first plotted on a Moody diagram (fig. E-1) to eliminate any obviously erroneous data. The remaining data was used to determine a best fit curve for the data assuming $b = 2$. It should be noted that actual values for b were between 1.8 and 1.9. Once a was established, the determination was a straightforward calculation. The results are summarized below:

6-in pipeline gravel runs

$f = 0.0145$ at VD" = 76.9 — 19 data points

6-in pipeline coal runs

$f = 0.0137$ at VD" = 100.0 — 13 data points

8-in pipeline 300-ft coal runs

$f = 0.0169$ at VD" = 83.55 — 6 data points

8-in pipeline 700-ft coal runs

$f = 0.0133$ at VD" = 89.4 — 22 data points

8-in pipeline 1000-ft loops

$f = 0.0212$ at VD" = 88.6 — 12 data points

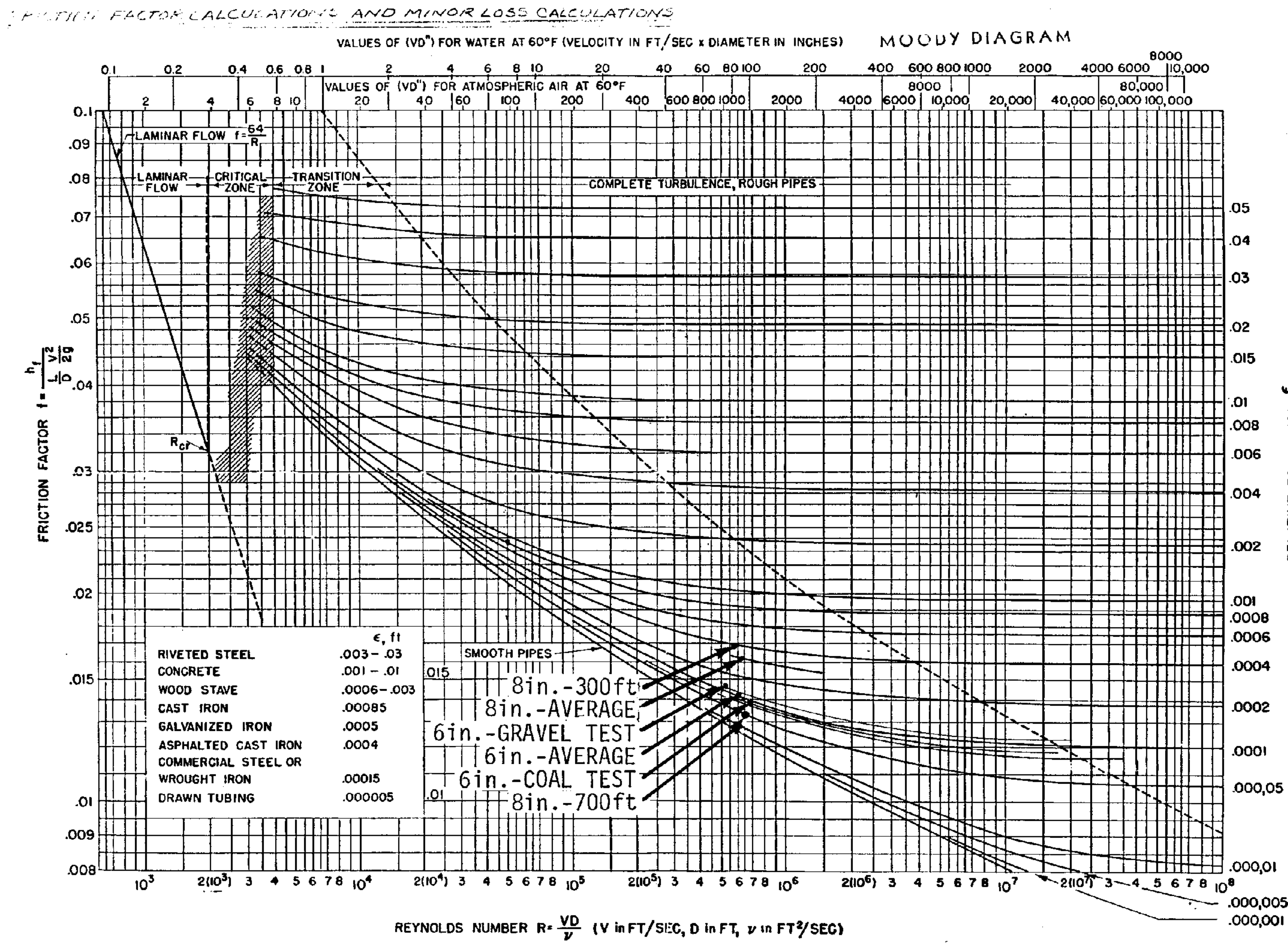


FIGURE E-1. - Moody diagram.

6-in pipeline averages

$$f_{AVE} = 0.0145 \left(\frac{19}{19 + 13}\right) + 0.0137 \left(\frac{13}{19 + 13}\right) = 0.0142$$

$$VD''_{AVE} = 76.9 \left(\frac{19}{19 + 13}\right) + 100.0 \left(\frac{13}{19 + 13}\right) = 86.3$$

8-in pipeline averages

$$f_{AVE} = 0.0133 \left(\frac{22}{22 + 12 + 6}\right) + 0.0212 \left(\frac{6}{22 + 12 + 6}\right)$$

$$+ 0.0169 \left(\frac{6}{22 + 12 + 6}\right) = 0.0162$$

$$VD''_{AVE} = 89.4 \left(\frac{22}{40}\right) + 88.6 \left(\frac{12}{40}\right) + 83.6 \left(\frac{6}{40}\right) = 88.3$$

E.3 MINOR LOSS

From equation E-1, the term H_K is the minor loss term where

$$H_k = K \frac{V^2}{2g} = \frac{K}{2g} V^2$$

or letting

$$\frac{K}{2g} = a \quad \text{and} \quad 2 = b$$

then again

$$H_k = aV^b \tag{E-3a}$$

where

$$a = \exp\left(\frac{\Sigma\ \ell n\ H_{k_i}}{n} - \frac{2\Sigma\ \ell n\ V_i}{n}\right)$$

The minor loss configurations are shown in figure E-2.

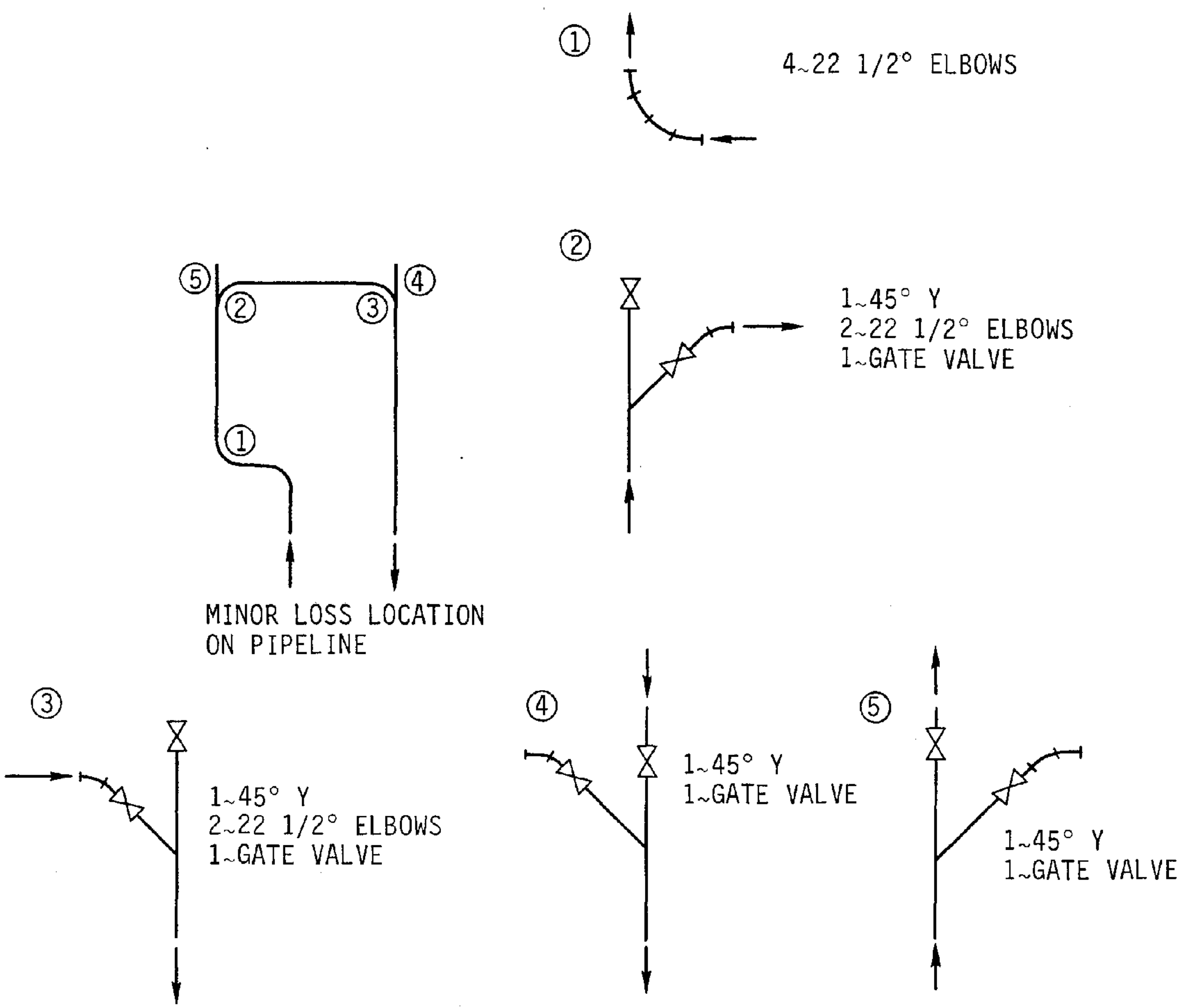
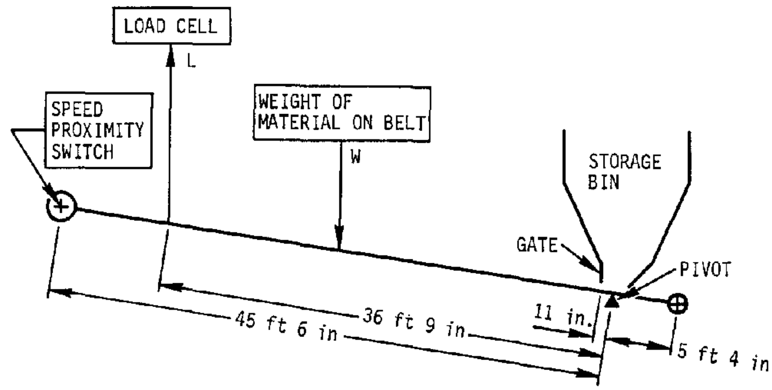


FIGURE E-2. - Minor loss location on pipeline.

For each of the minor loss configurations shown head loss for different flow rates was measured using a differential water-air manometer. The data was reduced using the analysis outline above and the following k factors for each configurations were calculated based on b = 2.0. The actual value of b ranged from 2.15 to 1.93 and averaged 2.03.

Configuration	a at b = 2	K
1	0.011	0.717
2	0.016	1.050
3	0.013	0.810
4	0.005	0.309
5	0.010	0.646

APPENDIX F.--WEIGH BELT CALCULATIONS



W = WEIGHT OF MATERIAL ON THE BELT FROM THE GATE TO CENTERLINE OF THE HEAD SHEEVE

L = LOAD CELL LOAD DUE TO W

NOTE THAT LOAD CELL DID NOT READ THE WEIGHT OF MATERIAL WITHIN THE HOPPER SINCE THIS LOAD WAS APPLIED DIRECTLY OVER THE PIVOT SUPPORT. THIS WAS VERIFIED BY ACTUAL LOAD CELL READINGS.

FIGURE F-1. - Free body diagram of belt.

From statics:

$$\Sigma^{+}M_{PIVOT} = 0 \text{ lb-ft}$$

$$-W\,\frac{[(45 \text{ ft } 6 \text{ in}) - (0 \text{ ft } 11 \text{ in})]}{2}$$

$$+ L[36 \text{ ft } 9 \text{ in}) - (0 \text{ ft } 11 \text{ in})] = 0 \text{ lb-ft}$$

$$-W(22.29) + L(35.83) = 0 \text{ lb-ft}$$

$$W = L\,\frac{35.83}{22.29} = 1.61\,L$$

$$L = (mV_{TOTAL} - mV_{TARE})\ (\text{load cell factor})$$

LCF = load cell factor is determined from load cell calibration

$$\text{Output at } 0\# = 0.021\ \frac{mV}{V}$$

$$\text{Output at } 7{,}500\# = 2.198\ \frac{mV}{V}$$

Input voltage = 13.5 V dc

Therefore:

Load cell output at 0# = 0.021 x 13.5 = 0.284 mV

Load cell output at 7,500# = 2.198 x 13.5 = 29.673 mV

Therefore:

$$LCF = \frac{7{,}500\# - 0\#}{29.673 - 0.284} = \frac{7{,}500\#}{29.389} = 255.198\ \frac{lb}{mV}$$

From page number 1

$$W = 1.61\ L = (mV_{TOTAL} - mV_{TARE})\ (LCF)\ 1.61$$

$$= (mV_{TOTAL} - mV_{TARE})\ (255.198)\ (1.61)$$

$$= (mV_{TOTAL} - mV_{TARE})\ 410.869\ lb$$

Letting tons per minute equals tons per minute of material delivered by the belt, then:

$$\text{tons/min} = \frac{W}{44.58\ ft} \times \frac{1}{2{,}000}\ \frac{tons}{lb} \times \text{belt speed}\ \frac{ft}{min}$$

$$= \frac{W}{89{,}160} \times (\text{belt speed})$$

$$\text{tons/min} = \frac{410.869}{89.160} (mV_{TOTAL} - mV_{TARE}) \text{ (belt speed)}$$

Note that belt speed was read from proximity switch calibrated to read in feet per minute. These readings were periodically verified by timing given belt sections with a hand-held stop-watch.

APPENDIX G.--TEST DATA

Date	Test No.	Q_w (gpm)	Coal (tpm)	C_v (%)	Inj. (psi)	Inj. (hp_m)	Shaft (hp)	Slurry (hp)	Inj. eff.	Pipe $\phi \sim L$ (in.-ft)	Q_s (gpm)	V_s (fps)	Δpsi/ 1000 ft	3 in./ 75 mm
8/21/80	-	1440	3.4	25	48	250				8-300				
9/10/80	0 min	Fresh coal		-	-	-	-	-	-	-	-	-	-	
	0-5	1600	4.3	27	49	291				8-300				
	5 min	-	-	-	-	-	-	-	-	-	-	-	-	
	5-15	1600	4.4	28	42	243				8-300				
	15 min	-	-	-	-	-	-	-	-	-	-	-	-	
	15-24	1600	3.0	21						8-300				
	24-25	1600	1.6	12	33	200				8-300				
	25 min	Degraded coal		-	-	-	-	-	-	-	-	-	-	
	0-5	1960	3.1	18	45	254				8-300				Fresh coal
9/17/80	5-10	1960	3.5	20	35	200				8-300				
	10-20	1960	1.5	10	32	180				8-300				
	20-30	1960	1.4	9	22	169				8-300				Degraded coal
	0-15	1400	2.2	18	-	222				8-300				Fresh coal
9/18/80	0-5	1400	2.9	23	30	169				8-300				Fresh coal
	0-15	1400	1.6	14	27	132				8-300				
	15-26	1400	1.5	13	24	127				8-300				Degreaded coal
	0-5	1600	2.5	18	56	296				8-700				Fresh coal
	5-15	1600	3.7	24	62	317				8-700				
	15-21	1600	4.3	27	62	317				8-700				Degraded coal
	0-3	2100	4.4	23	75	291				8-700				Fresh coal
	3-9	2100	3.1	17	70	275				8-700				
	9-11	2100	3.9	21	70	275				8-700				Degraded coal
9/22/80	0-2	1400	3.4	25	58	315				8-700				Fresh coal
	2-5	1400	3.8	28	58	315				8-700				
	5-7	1800	3.8	23	63	338				8-700				
	7-8	1800	4.0	24	63	338				8-700				
	8-10	1600	3.8	30	82	370				8-1000				
	10-12	1800	6.8	34	65	344				8-300				
	12-14	1800	6.7	34	66	370				8-300				
9/23/80	14-16	1800	5.9	32	83	359				8-700				
	16-17	1800	5.7	31	79	354				8-700				
9/24/80	17-22	1800	5.7	31	78	359				8-700				

Δpsi/ 1000 ft	Material gradation ∿ % passing								C_{DM}	D_M (mm)	D_{95} (mm)	D_{84} (mm)	D_{60} (mm)	D_{50} (mm)	D_{30} (mm)	D_{16} (mm)	D_{10} (mm)	D_5 (mm)
	3 in./ 75 mm	2 in./ 50 mm	1 in./ 25 mm	3/4 in./ 19 mm	1/2 in./ 12.5 mm	1/4 in./ 6.3 mm	#8/ 2.36 mm	Pan										
-																		
-																		
-																		
-																		
	Fresh coal																	
	Degraded coal																	
	Fresh coal																	
	Fresh coal																	
	Degraded coal																	
	Fresh coal																	
	Degraded coal																	
	Fresh coal																	
	Degreaded coal																	
	Fresh coal																	

Date	Test No.	Q_w (gpm)	Coal (cpm)	C_v (%)	Inj. (psi)	Inj. (hp_m)	Shaft (hp)	Slurry (hp)	Inj. eff.	Pipe $\phi \sim L$ (in.-ft)	Q_S (gpm)	V_s (fps)	Δpsi/ 1000 ft	3 in./ 75 mm
10/9/80	0-10	1800	6.3	33	57	288	258.6	94.1	0.364	8-300	2688			Fresh coal
10/10/80	10-12	1800	7.0	35	76	340	305.3	128.8	0.422	8-300	2786			
10/11/80	12-17	1750	9.1	42	90	396	355.6	166.0	0.467	8-300	3032			
	17-20	1800	9.7	43	95	396	355.6	183.2	0.515	8-300	3167			
10/16/80	20-23	2100	5.8	28	65	324	290.9	116.3	0.400	8-300	2917			
10/22/80	100	1750	10.2	45	94	405	363.7	182.7	0.502	8-300	3187			Fresh coal
10/29/80	3-4	1660	3.0	21	35	189	169.7	44.8	0.264	8-300	2083			
	4-5	1660	4.2	26	35	189	169.7	48.8	0.288	8-300	2252			
10/31/80	S102	Fresh coal		-	-	-	-	-	-	-	-			98
	T102	1760	3.7	23	43	216	194.0	60.1	0.310	8-300	2281	13.29	86.7	
	T103	1520	3.6	25	43	207	185.9	53.0	0.285	8-300	2027	11.81	49.8	
	S104	-	-	-	-	-	-	-	-	-	-	-	-	97
	T104	2000	3.5	20	47	216	194.0	71.9	0.371	8-300	2493	14.52	48.2	
	T105	2000	4.6	24	52	252	226.3	84.9	0.374	8-300	2648	15.42	54.8	
	S105	-	-	-	-	-	-	-	-	-	-	-	-	97
	T106	1760	5.1	29	52	248	222.7	78.9	0.354	8-300	2479	14.44	62.3	
	S106	-	-	-	-	-	-	-	-	-	-	-	-	98
	T107	1440	5.2	34	48	234	210.1	63.6	0.303	8-300	2173	12.65	64.2	
	S107	-	-	-	-	-	-	-	-	-	-	-	-	98
11/1/80	T108	1920	5.4	28	52	261	234.4	85.9	0.366	8-300	2681	15.60	52.4	
		1920	5.6	29	55	279	250.5	91.6	0.366	8-300	2709	15.78	59.7	
	S108	-	-	-	-	-	-	-	-	-	-	-	-	99
	T109	1680	5.3	31	50	270	242.5	74.4	0.307	8-300	2427	14.13	47.82	
	T110	1440	5.0	33	50	252	226.3	65.2	0.288	8-300	2145	12.49	67.11	
	S110	-	-	-	-	-	-	-	-	-	-	-	-	100
	29-45	1440	~5.0	~33	~51	~261				8-300				Degraded coal
	T111	1440	4.5	30	42	189	169.7	53.2	0.313	8-300	2074	12.08	45.24	
	T112A	1760	3.9	24	42	207	185.9	59.6	0.321	8-300	2309	13.45	47.22	
	T112B	1760	4.1	25	42	198	177.8	60.4	0.340	8-300	2338	13.61	47.25	
	T113	2000	3.1	18	43	189	169.7	64.4	0.380	8-300	2437	14.19	84.03	
	S113	Degraded coal		-	-	-	-	-	-	-	-	-	-	100

Δpsi/ 1000 ft	Material gradation ~ % passing								C_{UM}	D_M (mm)	D_{95} (mm)	D_{84} (mm)	D_{60} (mm)	D_{50} (mm)	D_{30} (mm)	D_{16} (mm)	D_{10} (mm)	D_5 (mm)
	3 in./ 75 mm	2 in./ 50 mm	1 in./ 25 mm	3/4 in./ 19 mm	1/2 in./ 12.5 mm	1/4 in./ 6.3 mm	#8/ 2.36 mm	Pan										
	Fresh coal																	
	Fresh coal																	
-	98	52	31	26	20	13	4	0	2.14	41.44	76	72	60	50	25	9.0	6.3	4.0
86.7																		
49.8																		
-	97	63	33	28	21	12	4	0	2.49	38.45	72	62	50	44	24	8.8	5.6	3.2
48.2																		
54.8																		
-	97	72	35	30	22	11	2	0	2.50	35.80	71	60	48	41	19	9.0	6.4	4.0
62.3																		
-	98	73	33	29	21	11	1	0	2.54	35.81	70	59	48	41	21	9.4	6.9	4.2
64.2																		
-	98	76	37	33	23	11	2	0	2.48	34.04	68	56	44	36	10.8	8.4	6.4	4.0
52.4																		
59.7																		
-	99	86	53	46	32	10	0	0	2.33	27.57	66	50	32	24	11	8.0	6.3	4.1
47.82																		
67.11																		
-	100	91	61	50	33	14	2	0	2.30	24.32	59	40	25	18	11	7.2	5.4	3.8
	Degraded coal																	
45.24																		
47.22																		
47.25																		
84.03																		
-	100	100	94	86	72	15	2	0	1.94	12.17	30	17	9.8	8.9	8.0	7.0	5.8	3.6

Date	Test No.	Q_w (gpm)	Coal (tpm)	C_v (%)	Inj. (psi)	Inj. (hp_m)	Shaft (hp)	Slurry (hp)	Inj. eff.	Pipe $\phi \sim L$ (in.-ft)	Q_s (gpm)	V_s (fps)	Δpsi/ 1000 ft	3 in./ 75 mm
11/3/80	S114	Fresh coal		-	-	-	-	-	-	-	-	-	-	90
	T114	2000	5.4	28	48	252	226.3	82.2	0.363	8-300	2761	16.08	64.4	
	T115	1680	5.8	33	48	261	234.4	73.8	0.315	8-300	2497	14.54	65.7	
	T116	1520	5.5	34	48	279	250.5	67.4	0.269	8-300	2295	13.37	71.49	
	S116	-	-	-	-	-	-	-	-	-	-	-	-	99
	T117	1560	4.1	27	40	216	194.0	52.4	0.270	8-300	2138	12.45	55.66	
	S117	-	-	-	-	-	-	-	-	-	-	-	-	100
	T118	1800	3.8	23	43	234	210.1	61.7	0.293	8-300	2336	13.60	53.72	
	S118	-	-	-	-	-	-	-	-	-	-	-	-	98
	T119	2000	3.8	21	44	234	210.1	68.8	0.328	8-300	2536	14.77	59.09	
	S119	-	-	-	-	-	-	-	-	-	-	-	-	99
11/3/80	T120	2000	4.0	22	43	234	210.1	68.2	0.325	8-300	2564	14.93	59.85	
	S120	-	-	-	-	-	-	-	-	-	-	-	-	99
11/4/80	T121	1760	3.5	22	38	198	177.8	52.7	0.297	8-300	2253	13.12	47.16	
	T122	1560	3.3	23	38	189	169.7	56.6	0.334	8-300	2025	11.79	51.54	
	T123A	1560	5.1	32	46	253	227.2	64.2	0.283	8-300	2279	13.27	62.72	
	T123B	1560	8.1	42	62	324	291.4	102.7	0.353	8-300	2702	15.73	73.93	
	T124	1760	8.0	39	63	333	299.0	112.0	0.375	8-300	2887	16.82	79.28	
	T125	2000	7.9	36	70	333	299.0	134.2	0.449	8-300	3113	18.13	100.22	
	S125	-	-	-	-	-	-	-	-	-	-	-	-	97
		2000	7.8	35	64	315	282.9	122.7	0.434	8-300	3099			Degraded coal
		2000	6.6	32	60	306	274.8	108.4	0.395	8-300	2930			
		2000	6.0	30	54	274	246.0	95.0	0.386	8-300	2846			
		2000	5.2	27	50	252	226.3	84.5	0.373	8-300	2733			
		2000	4.6	24	46	234	210.1	75.4	0.359	8-300	2648			Degraded coal
	S126	-	-	-	-	-	-	-	-	-	-	-	-	100
	T126	2000	5.2	27	40	198	177.8	68.5	0.385	8-300	2733	15.92	33.9	
	T127	1720	4.4	27	35	180	161.6	50.9	0.315	8-300	2340	13.63	19.39	
	T128	1560	2.3	20	32	135	121.2	36.9	0.305	8-300	1884	10.97	32.22	
	S128	Degraded coal		-	-	-	-	-	-	-	-	-	-	100
11/5/80	S129	Fresh coal		-	-	-	-	-	-	-	-	-	-	89
	T129	1520	7.9	42	68	351	315.2	109.1	0.346	8-300	2633	15.34	79.38	
	T130	1760	7.9	39	70	338	303.5	123.1	0.406	8-300	2873	16.73	79.28	

Δpsi/ 1000 ft	Material gradation ~ % passing								C_{DM}	D_M (mm)	D_{95} (mm)	D_{84} (mm)	D_{60} (mm)	D_{50} (mm)	D_{30} (mm)	D_{16} (mm)	D_{10} (mm)	D_5 (mm)
	3 in./ 75 mm	2 in./ 50 mm	1 in./ 25 mm	3/4 in./ 19 mm	1/2 in./ 12.5 mm	1/4 in./ 6.3 mm	#8/ 2.36 mm	Pan										
-	90	21	12	11	9	7	2	0	2.54	55.73	75	75	74	73	70	40	19	5.0
64.4																		
65.7																		
71.49																		
-	99	57	25	21	16	10	3	0	2.55	41.67	70	60	50	49	32	12.5	6.3	3.2
55.66																		
-	100	79	39	33	19	10	2	0	2.46	32.88	68	58	46	38	18	10	6.3	3.6
53.72																		
-	98	80	49	41	32	17	2	0	2.45	29.80	68	54	36	28	12	6.8	4.8	3,2
59.09																		
-	99	83	46	38	27	14	2	0	2.45	29.97	71	56	36	28	13	7.6	5.4	3.8
59.85																		
-	99	88	55	45	32	15	2	0	2.38	26.52	66	48	30	22	12	7.2	5.1	3.2
47.16																		
51.54																		
62.72																		
73.93																		
79.28																		
100.22																		
-	97	79	57	47	33	19	3	0	2.37	28.45	68	52	30	21	12	6	4.1	2.8
	Degraded coal																	
	Degraded coal																	
-	100	99	74	61	45	24	3	0	2.31	18.32	46	32	18	13	8.2	4.8	3.8	2.8
33.9																		
19.39																		
32.22																		
-	100	100	83	72	57	26	4	0	2.23	15.10	42	28	13	10	7.1	4.6	3.6	2.6
-	89	20	8	7	6	4	1	0	2.55	57.41	80	75	75	75	70	42	29	12
79.38																		
79.28																		

Date	Test No.	Q_w (gpm)	Coal (tpm)	C_v (%)	Inj. (psi)	Inj. (hp_m)	Shaft (hp)	Slurry (hp)	Inj. eff.	Pipe $\phi \sim L$ (in.-ft)	Q_s (gpm)	V_s (fps)	Δpsi/ 1000 ft	3 in./ 75 mm
11/5/80	T131	2000	6.1	30	60	315	282.9	105.5	0.373	8-300	2860	16.65	63.38	
		2000	6.3	31	61	302	271.2	108.4	0.400	8-300	2888	16.82	70.67	
	S131	-	-	-	-	-	-	-	-	-	-	-	-	99
	T132	2000	7.7	35	65	320	287.4	123.9	0.431	8-300	3085	17.97	70.80	
	S132	-	-	-	-	-	-	-	-	-	-	-	-	100
	T133	1720	8.8	41	67	320	287.4	122.1	0.425	8-300	2960	15.84	73.54	
	S133	-	-	-	-	-	-	-	-	-	-	-	-	100
	T134	1480	8.8	46	60	324	291.0	100.4	0.345	8-300	2720	14.56	61.73	
	S134	-	-	-	-	-	-	-	-	-	-	-	-	100
	T135	2040	11.1	43	80	369	331.4	179.2	0.541	8-300	3604	20.99	108.79	
		2040	8.4	37	63	333	299.0	126.3	0.422	8-300	3224	18.78	79.95	
	S135	-	-	-	-	-	-	-	-	-	-	-	-	100
11/6/80	18-21	2000	8.3	37	68	324	291.0	133.2	0.458	8-300	3170	18.46	83.93	Degraded coal
	S136	-	-	-	-	-	-	-	-	-	-	-	-	100
	T136	2000	8.3	37	68	297	266.7	133.2	0.500	8-300	3170	18.46	75.58	
	T137	1800	8.4	40	67	342	307.1	123.1	0.401	8-300	2984	17.38	72.42	
	T138	1520	8.5	44	67	306	274.8	111.4	0.405	8-300	2718	15.83	76.90	
	S139	-	-	-	-	-	-	-	-	-	-	-	-	100
	T139	1440	6.0	37	54	284	255.0	75.2	0.295	8-300	2286	13.31	58.53	
		1440	4.9	32	43	225	202.0	56.0	0.277	8-300	2130	12.41	47.85	
	T140	1760	4.9	28	44	225	202.0	66.5	0.327	8-300	2450	14.27	45.90	
	T141	2000	4.9	26	47	248	222.7	78.3	0.352	8-300	2690	15.67	51.28	
	S141	-	-	-	-	-	-	-	-	-	-	-	-	100
	T142	2000	3.4	19	40	193	173.3	61.3	0.354	8-300	2479	14.44	47.78	
	T143	1760	3.4	21	37	189	169.7	51.1	0.301	8-300	2239	13.04	42.04	
	T144	1520	3.4	24	35	180	161.6	42.9	0.266	8-300	1999	11.64	39.60	
	T145	1520	3.5	24	38	198	177.8	46.8	0.263	8-300	2013	11.72	39.97	
	S145	Degraded coal		-	-	-	-	-	-	-	-	-	-	100
11/7/80	S146-1	Fresh coal		-	-	-	-	-	-	-	-	-	-	99
	T146	1760	0	0	28	112	100.6	30.1	0.299	8-700	1760			
		1760	6.9	36	85	302	271.2	140.5	0.518	8-700	2732			
		1760	6.7	35	87	324	290.9	142.0	0.488	8-700	2704			
		1760	7.9	39	90	315	282.9	156.7	0.554	8-700	2873			

Δpsi/ 1000 ft	Material gradation ~ % passing								C_{DM}	D_M (mm)	D_{95} (mm)	D_{84} (mm)	D_{60} (mm)	D_{50} (mm)	D_{30} (mm)	D_{16} (mm)	D_{10} (mm)	D_5 (mm)
	3 in./ 75 mm	2 in./ 50 mm	1 in./ 25 mm	3/4 in./ 19 mm	1/2 in./ 12.5 mm	1/4 in./ 6.3 mm	#8/ 2.36 mm	Pan										
63.38																		
70.67																		
-	99	82	46	39	29	17	3	0	2.48	29.86	70	56	36	28	14	6.8	4.2	2.8
70.80																		
-	100	87	54	44	34	21	5	0	2.45	26.28	66	49	30	22	11	5.0	3.6	2.4
73.54																		
-	100	86	51	44	34	21	6	0	2.47	26.97	65	49	32	25	11	5.0	3.6	2.4
61.73																		
-	100	95	63	52	39	23	7	0	2.37	21.92	50	36	23	18	9	4.2	3.0	2.0
108.79																		
75.95																		
-	100	97	72	60	48	30	4	0	2.42	18.67	48	30	19	13	6.3	4.6	3.6	2.8
83.93	Degraded coal																	
-	100	95	71	62	52	30	6	0	2.40	18.90	50	37	18	12	6.3	4.0	3.0	2.1
75.58																		
72.42																		
76.90																		
-	100	99	82	72	58	38	9	0	2.37	14.71	41	28	13	9.8	5.9	3.1	2.6	2.0
58.53																		
47.85																		
45.90																		
51.28																		
-	100	99	84	75	62	41	9	0	2.39	13.81	40	25	11	8.9	5.0	3.1	2.6	1.9
47.78																		
42.04																		
39.60																		
39.97																		
-	100	99	86	78	65	43	10	0	2.38	12.99	40	24	10	8.0	5.0	3.1	2.4	2.0
-	99	40	12	10	8	6	2	0	2.61	49.35	75	71	60	54	48	32	20	7.0

Date	Test No.	Q_w (gpm)	Coal (tpm)	C_v (%)	Inj. (psi)	Inj. (hp_m)	Shaft (hp)	Slurry (hp)	Inj. eff.	Pipe $\phi \sim L$ (in.-ft)	Q_s (gpm)	V_s (fps)	Δpsi/1000 ft	3 in./75 mm
11/7/80	T146	1760	5.7	31	77	279	250.5	119.2	0.476	8-700	2563	14.94	69.02	
		1760	5.7	31	82	292	262.2	126.7	0.483	7-700	2563	14.94	76.68	
	S146-2	-	-	-	-	-	-	-	-	-	-	-	-	98
		1760	6.7	35	85	279	250.5	138.9	0.555	8-700	2704	15.72	68.34	
		1760	6.7	35	90	315	282.9	146.8	0.519	8-700	2704	15.72	83.67	
		1760	5.7	31	78	288	258.6	120.7	0.467	8-700	2563	14.93	76.30	
		1760	5.1	29	68	270	242.5	102.0	0.421	8-700	2479	14.44	65.13	
	S146-3	-	-	-	-	-	-	-	-	-	-	-	-	100
6/16/81	S147	Fresh coal		-	-	-	-	-	-	-	-	-	-	88
	T147	1360	4.1	30	63	270	242.5	73.2	0.300	8-700	1938	11.26	57.99	
	S148	-	-	-	-	-	-	-	-	-	-	-	-	100
	T148	1480	4.7	31	62	256	229.9	80.1	0.348	8-700	2142	12.45	44.06	
	S149	-	-	-	-	-	-	-	-	-	-	-	-	100
	T149	1720	5.1	29	68	261	234.4	100.3	0.428	80700	2439	14.20	47.53	
6/18/81	S150	-	-	-	-	-	-	-	-	-	-	-	-	100
	T150	1360	4.8	33	70	270	242.5	85.5	0.353	8-700	2036	11.84	75.47	
	S151	-	-	-	-	-	-	-	-	-	-	-	-	100
	T151	1440	4.0	28	56	234	210.1	67.6	0.322	8-700	2003	11.65	57.97	
	S152	-	-	-	-	-	-	-	-	-	-	-	-	90
	T152	1720	5.0	29	66	247	221.8	96.9	0.437	8-700	2425	14.16	71.93	
	S153	-	-	-	-	-	-	-	-	-	-	-	-	100
	T153	1360	3.6	27	64	228	204.7	71.5	0.349	8-700	1867	10.88	71.90	
6/19/81	S154	-	-	-	-	-	-	-	-	-	-	-	-	100
	T154	1480	4.7	31	70	306	274.8	90.1	0.328	8-700	2142	12.49	.58.01	
	S155	-	-	-	-	-	-	-	-	-	-	-	-	100
	T155	1680	5.5	32	69	315	282.9	102.5	0.362	8-700	2455	14.31	85.90	
	S156	-	-	-	-	-	-	-	-	-	-	-	-	100
	T156	1360	4.6	32	60	256	229.9	72.5	0.315	8-700	2008	11.65	61.50	
	S157	-	-	-	-	-	-	-	-	-	-	-	-	100
	T157	1480	5.2	33	71	360	323.9	91.3	0.282	8-700	2141	12.89	68.49	
	S158	-	-	-	-	-	-	-	-	-	-	-	-	100
	T158	1720	6.2	34	80	301	270.3	125.3	0.464	8-700	2594	15.06	82.44	

Δpsi/1000 ft	Material gradation ~ % passing								C_{DM}	D_M (mm)	D_{95} (mm)	D_{84} (mm)	D_{60} (mm)	D_{50} (mm)	D_{30} (mm)	D_{16} (mm)	D_{10} (mm)	D_5 (mm)
	3 in./75 mm	2 in./50 mm	1 in./25 mm	3/4 in./19 mm	1/2 in./12.5 mm	1/4 in./6.3 mm	#8/2.36 mm	Pan										
69.02																		
76.68																		
-	98	80	42	34	26	15	3	0	2.50	31.78	71	56	39	32	17	7.1	4.5	3.0
68.34																		
83.67																		
76.30																		
65.13																		
-	100	96	66	51	38	21	5	0	2.34	21.48	49	31	22	19	9.6	5.0	4.6	2.3
-	88	30	14	12	10	7	5	0	2.51	53.37	80	75	71	69	50	29	12.5	2.4
57.99																		
-	100	61	41	37	34	29	12	0	2.51	34.75	75	72	56	40	8.0	3.9	2.2	1.9
44.06																		
-	100	85	45	38	32	17	3	0	2.51	28.93	69	49	34	30	12	6.0	4.0	2.6
47.53																		
-	100	87	55	49	42	25	3	0	2.52	25.15	70	49	31	20	7.9	4.8	3.8	2.9
75.47																		
-	100	78	31	24	17	10	3	0	2.55	35.04	69	58	49	41	18	12	6.3	4.0
57.97																		
-	90	71	24	17	13	8	2	0	2.57	40.68	80	68	48	40	30	16	9.6	4.6
71.93																		
-	100	92	65	56	49	23	3	0	2.39	21.57	60	41	20	14	8.2	5.0	3.9	3.0
71.90																		
-	100	82	35	28	24	15	8	0	2.53	32.34	70	58	39	32	22	7.2	4.0	2.0
58.01																		
-	100	90	59	54	50	22	2	0	2.45	23.14	58	49	28	12.5	8.9	5.2	4.0	3.0
85.90																		
-	100	100	40	34	30	20	2	0	2.68	26.21	50	50	40	34	12.5	5.0	3.8	2.8
61.50																		
-	100	96	75	70	65	42	5	0	2.55	16.12	49	39	12	8.1	4.8	3.2	2.8	2.4
68.49																		
-	100	96	70	65	60	38	7	0	2.52	17.67	49	40	12.5	10.0	5.1	3.3	2.8	2.1
82.44																		

Date	Test No.	Q_w (gpm)	Coal (tpm)	C_v (%)	Inj. (psi)	Inj. (hp_m)	Shaft (hp)	Slurry (hp)	Inj. eff.	Pipe $\phi \sim L$ (in.-ft)	Q_s (gpm)	V_s (fps)	Δpsi/ 1000 ft	3 in./ 75 mm
6/19/81	S159	-	-	-	-	-	-	-	-	-	-	-	-	100
	T159	1360	5.2	35	72	270	242.5	90.4	0.373	8-700	2093	12.23	72.01	
	S160	-	-	-	-	-	-	-	-	-	-	-	-	100
	T160	1480	4.8	31	68	288	258.6	88.2	0.341	8-700	2156	12.52	64.98	
	S161	-	-	-	-	-	-	-	-	-	-	-	-	100
	T161	1720	5.0	29	70	292	262.2	102.5	0.391	8-700	2425	14.15	68.44	
	T162	1680	5.8	33	82					8-700		14.53	75.46	
	T162	1600	5.0	30	70					8-700		13.38	64.97	
7/6/81	-	1440	1.9	16						8-700				
9/18/81	-	Degraded coal		-	-	-	-	-	-	-	-	-	-	100
	-	2080	4.3	22	56	288	156.6	92.2	0.356	8-700	2686	15.62	51.04	
	-	2080	5.4	27	70	293	263.1	121.3	0.461	8-700	2841			
	-	2080	5.9	28	70	293	263.1	124.5	0.473	8-700	2911			
	-	2080	5.4	27	60	288	258.6	104.7	0.405	8-700	2841			
9/19/81	-	2160	4.5	23	66	279	250.2	112.5	0.449	8-700	2794	16.62	65.01	
	-	-	-	-	-	-	-	-	-	-	-	-	-	100
	-	2160	4.9	24	70	279	250.5	121.6	0.485	8-700	2850	16.63	68.38	
9/22/81	-	-	-	-	-	-	-	-	-	-	-	-	-	100
	-	2080	5.0	25	72	279	250.5	121.9	0.487	8-700	2785	16.24	71.88	
	-	2160	5.0	25	72	288	258.6	125.6	0.486	8-700	2865	16.46	71.87	
	-	2160	5.0	24	73	297	266.7	127.3	0.477	8-700	2865	16.64	71.87	
	-	2160	5.1	25	73	297	266.7	128.0	0.480	8-700	2879	16.74	71.87	
	-	-	-	-	-	-	-	-	-	-	-	-	-	100
9/23/81	-	2120	5.2	26	72	270	242.5	125.1	0.516	8-700	2853	16.66	71.89	
9/24/81	S163	-	-	-	-	-	-	-	-	-	-	-	-	100
	T163A	2040	6.9	32	85	315	281.9	155.7	0.551	8-700	3012	17.53	71.97	
	T163B	2000	5.8	29	70	288	258.6	120.3	0.465	8-700	2817	16.67	64.95	
	S164	-	-	-	-	-	-	-	-	-	-	-	-	100
	T164	1720	5.5	31	70	270	242.5	105.7	0.436	8-700	2495	14.56	61.50	
	S165	-	-	-	-	-	-	-	-	-	-	-	-	100
	T165	1320	6.2	40	72	288	258.6	95.0	0.367	8-700	2194	12.79	61.61	
	S166	-	-	-	-	-	-	-	-	-	-	-	-	100
	T166A	1720	8.2	41	88	342	307.1	153.5	0.500	8-700	2876	16.72	75.56	
	T166B	1720	7.4	38	95	306	274.8	158.3	0.576	8-700	2763	16.08	127.82	

Δpsi/ 1000 ft	Material gradation ~ % passing																	
	3 in./ 75 mm	2 in./ 50 mm	1 in./ 25 mm	3/4 in./ 19 mm	1/2 in./ 12.5 mm	1/4 in./ 6.3 mm	#8/ 2.36 mm	Pan	C_{DM}	D_M (mm)	D_{95} (mm)	D_{84} (mm)	D_{60} (mm)	D_{50} (mm)	D_{30} (mm)	D_{16} (mm)	D_{10} (mm)	D_5 (mm)
-	100	99	84	79	74	27	3	0	2.25	13.65	46	25	9.8	9.0	7.1	'3.4	3.6	2.8
72.01																		
-	100	95	82	79	72	31	3	0	2.31	14.89	50	26	9.4	7.9	6.8	4.6	3.8	2.9
64.98																		
-	100	97	78	74	70	26	3	0	2.29	15.70	42	30	10.0	9.6	7.2	5.1	4.0	3.0
68.44																		
75.46																		
64.97																		
-	100	100	91	85	78	58	1	0	2.67	10.16	32	18	7.9	6.8	4.8	3.6	3.0	2.6
51.04																		
65.01																		
-	100	100	92	87	78	52	3	0	2.54	10.14	32	17	7.1	6.0	4.0	3.2	2.9	2.6
68.38																		
-	100	100	90	85	75	51	3	0	2.54	10.82	34	18	8.1	6.9	4.8	3.6	3.0	2.6
71.88																		
71,87																		
71.87																		
71.87																		
-	100	100	91	85	78	58	1	0	2.67	10.16	32	18	7.9	6.8	4.8	3.6	3.0	2.6
71.89																		
-	100	100	87	80	68	50	6	0	2.53	12.02	40	24	9.0	6.3	4.0	3.1	2.6	2.4
71.97																		
64.95																		
-	100	100	85	73	58	38	5	0	2.38	14.16	41	24	13.0	10.0	5.6	3.8	3.0	2.4
61.50																		
-	100	100	89	78	69	46	4	0	2.48	12.02	40	22	10	8.9	4.8	3.2	2.8	2.6
61.61																		
-	100	100	95	93	83	62	4	0	2.64	8.45	25	13	6.5	5.4	3.8	3.0	2.8	2.4
75.56																		
127.82																		

Date	Test No.	Q_w (gpm)	Coal (tpm)	C_v (%)	Inj. (psi)	Inj. (hp_m)	Shaft (hp)	Slurry (hp)	Inj. eff.	Pipe $\phi \sim L$ (in.-ft)	Q_s (gpm)	V_s (fps)	Δpsi/ 1000 ft	3 in./ 75 mm
9/24/81	S167	-	-	-	-	-	-	-	-	-	-	-	-	100
	T167	1480	7.8	43										
11/6/81	Degraded coal		-	-	-	-	-	-	-	-	-	-	-	100
	1	2160	6.4	29	85	270	242.5	158.4	0.653	8-700	3062	17.82	61.47	
	2	2160	5.6	27	80	261	234.4	143.4	0.612	8-700	2949	17.19	50.98	
	3	2160	6.1	28	90	261	234.4	164.8	0.703	8-700	3020	17.55	64.94	
	4	2160	6.6	30	95	270	242.5	178.0	0.734	8-700	3090	17.99	120.75	
	5	2160	6.4	30	93	261	234.4	172.7	0.737	8-700	3062	17.86	96.34	
	6A	2160	6.1	28	92	261	234.4	168.3	0.718	8-700	3020	17.58	92.84	
	6B	2160	6.1	28	95	261	234.4	173.6	0.741	8-700	3020	17.58	103.29	
	6C	2160	6.1	28	85	261	234.4	156.0	0.665	8-700	3020	17.58	96.32	
	6D	2160	6.1	28	80	261	234.4	147.2	0.628	8-700	3020	17.58	89.35	
	7	2160	5.1	25	80	243	218.2	139.7	0.640	8-700	2879	16.80	89.31	
	8	2160	5.0	24	75	234	210.1	130.6	0.622	8-700	2865	16.66	78.84	
	9	2160	7.4											
	10	2160	6.0	28	82	248	223.6	149.9	0.671	8-700	3006	17.52	66.68	
	11	2160	5.5	27	78	238	213.7	139.3	0.652	8-700	2935	17.11	64.92	
	12A	2160	5.8	27	72	238	213.7	131.0	0.613	8-700	2977	17.34	57.96	
	12B	2160	5.8	27	87	265	238.0	157.1	0.660	8-700	2977	17.34	71.91	
	12C	2160	5.8	27	82	256	229.9	148.4	0.646	8-700	2977	17.34	68.42	
	12D	2160	5.8	27	82	261	234.4	148.4	0.633	8-700	2977	17.34	64.92	
	13A	2160	6.3	29	84	252	226.3	155.8	0.688	8-700	3048	17.71	68.44	
	13B	2160	6.3	29	82	252	226.3	152.2	0.673	8-700	3048	17.71	61.47	
	13C	2160	6.3	29	82	252	226.3	152.2	0.673	8-700	3048	17.71	64.95	
	14A	2160	6.4	29	82	270	242.5	153.0	0.631	8-700	3062	17.83	64.96	
	14B	2160	6.4	29	85	279	250.5	158.3	0.632	8-700	3062	17.83	64.96	
	14C	2160	6.4	29	85	279	250.5	158.4	0.632	8-700	3062	17.83	64.96	
	14D	2160	6.4	29	85	279	250.5	158.4	0.632	8-700	3062	17.83	64.96	
	14E	2160	6.4	29	85	279	250.5	158.4	0.632	8-700	3062	17.73	64.96	
	15A	2160	6.0	28	78	270	242.5	142.9	0.589	8-700	3006	17.50	64.94	
	15B	2160	6.0	28	80	270	242.5	146.4	0.604	8-700	3006	17.50	64.94	
	15C	2160	6.0	28	82	270	242.5	149.9	0.618	8-700	3006	17.50	69.94	

Δpsi/ 1000 ft	Material gradation % passing								C_{DN}	D_M (mm)	D_{95} (mm)	D_{84} (mm)	D_{60} (mm)	D_{50} (mm)	D_{30} (mm)	D_{16} (mm)	D_{10} (mm)	D_5 (mm)
	3 in./ 75 mm	2 in./ 50 mm	1 in./ 25 mm	3/4 in./ 19 mm	1/2 in./ 12.5 mm	1/4 in./ 6.3 mm	#8/ 2.36 mm	Pan										
-	100	100	89	83	72	54	9	0	2.53	10.99	39	20	8.4	6.4	3.8	2.8	2.4	2.2
-	100	100	91	85	78	58	1	0	2.67	10.16	32	18	7.9	6.8	4.8	3.6	3.0	2.6
61.47																		
50.98																		
64.94																		
120.75																		
96.34																		
92.84																		
103.29																		
96.32																		
89.35																		
89.31																		
78.84																		
66.68																		
64.92																		
57.96																		
71.91																		
68.42																		
64.92																		
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64.94																		

Date	Test No.	Q_w (gpm)	Coal (tpm)	C_v (%)	Inj. (psi)	Inj. (hp_m)	Shaft (hp)	Slurry (hp)	Inj. eff.	Pipe ϕ ∿ L (in.-ft)	Q_s (gpm)	V_s (fps)	Δpsi/ 1000 ft	3 in./ 75 mm
11/7/81	Degraded coal		-	-	-	-	-	-	-	-	-	-	-	100
	16A	2160	6.3	29	85	270	242.5	157.6	0.650	8-700	3048	17.77	71.93	
	16B	2160	6.3	29	83	261	234.4	154.0	0.657	8-700	3048	17.77	68.44	
	16C	2160	6.3	29	85	252	226.3	157.6	0.696	8-700	3048	17.77	71.93	
	16D	2160	6.3	29	84	261	234.4	155.8	0.665	8-700	3048	17.77	71.93	
	16E	2160	6.3	29	88	261	234.4	162.9	0.695	8-700	3048	17.77	75.41	
	16F	2160	6.3	29	88	261	234.4	162.9	0.695	8-700	3048	17.77	78.90	
	16G	2160	6.3	29	92	261	234.4	170.0	0.725	8-700	3048	17.77	78.90	
	16H	2160	6.3	29	96	270	242.5	177.1	0.731	8-700	3048	17.77	85.87	
	16I	2160	6.3	29	88	261	234.4	162.9	0.695	8-700	3048	17.77	85.87	
	16J	2160	6.3	29	85	256	229.9	157.5	0.685	8-700	3048	17.77	78.90	
	-	Degraded coal			-	-	-	-	-	-	-	-	-	100
11/9/81	S163	-	-	-	-	-	-	-	-	-	-	-	-	85
	T163	2160	4.2	22	75	270	242.5	125.1	0.516	8-700	2752	16.07	50.91	
	S164	-	-	-	-	-	-	-	-	-	-	-	-	
	T164	2160	6.3	29	98	306	274.8	180.7	0.658	8-700	3048	17.73	71.93	
	S165	-	-	-	-	-	-	-	-	-	-	-	-	93
	T165	2000	6.4	31		270								
	T165	2000	6.6	32	91	279	250.5	161.4	0.644	8-700	2930	17.03	70.22	
	S166	-	-	-	-	-	-	-	-	-	-	-	-	97
	T166	1720	5.6											
	T166	1720	5.7	32	74	252	226.3	112.9	0.499	8-700	2523	14.73	51.05	
	S167	-	-	-	-	-	-	-	-	-	-	-	-	98
	T167	1480	4.3	29	60	202	181.4	75.4	0.416	8-700	2086	12.17	51.01	
	S168	-	-	-	-	-	-	-	-	-	-	-	-	100
	T168	2160	4.3	22	69	248	222.7	116.1	0.521	8-700	2766	16.08	50.91	
	T168	2160	6.2	29	84	270	242.5	155.0	0.639	8-700	3034	17.69	61.46	
	S169	-	-	-	-	-	-	-	-	-	-	-	-	100
	T169	2000	6.1	30	80	261	234.4	138.9	0.593	8-700	2860	16.64	61.46	
	S170	-	-	-	-	-	-	-	-	-	-	-	-	100
	T170	1680	6.2	34	75	270	242.5	108.4	0.447	8-700	2554	14.87	61.53	
	S171	-	-	-	-	-	-	-	-	-	-	-	-	100
	T171	1440	5.6	35	70	270	242.5	94.0	0.388	8-700	2229	12.94	54.57	

Δpsi/ 1000 ft	Material gradation ∿ % passing								C_{DM}	D_M (mm)	D_{95} (mm)	D_{84} (mm)	D_{60} (mm)	D_{50} (mm)	D_{30} (mm)	D_{16} (mm)	D_{10} (mm)	D_5 (mm)
	3 in./ 75 mm	2 in./ 50 mm	1 in./ 25 mm	3/4 in./ 19 mm	1/2 in./ 12.5 mm	1/4 in./ 6.3 mm	#8/ 2.36 mm	Pan										
-	100	100	91	85	78	58	1	0	2.67	10.16	32	18	7.9	6.8	4.8	3.6	3.0	2.6
71.93																		
68.44																		
71.93																		
71.93																		
75.41																		
78.90																		
78.90																		
85.87																		
85.87																		
78.90																		
-	100	100	91	85	78	58	1	0	2.67	10.16	32	18	7.9	6.8	4.8	3.6	3.0	2.6
-	85	30	21	9	7	5	1	0	2.45	53.46	79	74	73	73	50	22	18	6.3
50.91																		
-																		
71.93																		
-	93	55	36	31	25	16	4	0	2.48	40.14	79	71	58	48	18	6.3	4.2	3.0
70.22																		
-	97	62	32	26	20	11	2	0	2.51	39.14	74	70	48	41	22	10	6.8	3.9
51.05																		
-	98	75	37	29	21	11	1	0	2.50	34.69	72	60	42	38	19	9.6	6.9	4.4
51.01																		
-	100	79	41	34	25	13	2	0	2.49	31.97	74	60	39	32	16	8.0	5.6	3.8
50.91																		
61.46																		
-	100	87	56	47	35	16	1	0	2.39	26.07	68	49	28	21	11	6.3	5.2	4.0
61.46																		
-	100	81	48	39	29	14	1	0	2.44	29.80	71	58	32	28	13	7.6	5.8	4.0
61.53																		
-	100	93	69	60	48	24	2	0	2.37	20.49	58	39	19	13	7.8	5.8	4.6	3.4
54.57																		

Date	Test No.	Q_w (gpm)	Coal (tpm)	C_v (%)	Inj. (psi)	Inj. (hp_m)	Shaft (hp)	Slurry (hp)	Inj. eff.	Pipe $\phi \sim L$ (in.-ft)	Q_s (gpm)	V_s (fps)	Δpsi/ 1000 ft	3 in./ 75 mm
11/9/81	S172	-	-	-	-	-	-	-	-	-	-	-	-	100
	T172	2080	6.7	31	85	261	234.4	156.3	0.667	8-700	3024	17.58	71.95	
	S173	-	-	-	-	-	-	-	-	-	-	-	-	100
	T173	2000	6.9	33	88	270	242.5	158.7	0.655	8-700	2972	17.32	71.97	
	S174	-	-	-	-	-	-	-	-	-	-	-	-	100
	T174	1720	6.8	36	85	279	250.5	137.5	0.549	8-700	2678	15.61	72.01	
	S175	-	-	-	-	-	-	-	-	-	-	-	-	100
	T175	1480	5.9	36	70	270	242.5	97.6	0.403	8-700	2311	13.45	61.55	
	S176	-	-	-	-	-	-	-	-	-	-	-	-	100
	T176	2120	7.5	33	93	279	250.5	179.7	0.717	8-700	3177	18.49	113.82	
	T176	2120	7.1	32	92	270	242.5	174.5	0.720	8-700	3121	18.41	75.45	
	S177	-	-	-	-	-	-	-	-	-	-	-	-	91
	T177	1960	5.9	30	76	266	238.9	128.9	0.540	8-700	2791	16.27	64.96	
	S178	-	-	-	-	-	-	-	-	-	-	-	-	98
	T178	1720	6.2	34	75	270	242.5	117.8	0.486	8-700	2594	15.15	65.01	
	S179	-	-	-	-	-	-	-	-	-	-	-	-	100
	T179	1520	6.1	36	75	270	242.5	107.6	0.444	8-700	2380	13.85	65.04	
	S180	-	-	-	-	-	-	-	-	-	-	-	-	98
	180A	1520	6.2	36	75	270	242.5	108.3	0.447	8-700	2394	13.95	61.56	
	180B	1520	7.2	40	85	302	271.2	129.9	0.479	8-700	2535	14.76	68.58	
	180C	1520	7.0	39	83	292	262.2	125.4	0.478	8-700	2506	14.56	65.08	
	180D	1520	7.6	41	87	306	274.8	136.0	0.495	8-700	2591	15.07	72.08	
	180E	1520	7.8	41	88	306	274.8	139.0	0.506	8-700	2691	15.25	75.58	
	180F	1520	8.5	43	94	310	278.4	154.2	0.554	8-700	2718	15.80	82.58	
	180G	1520	7.2	40	83	302	271.2	126.9	0.468	8-700	2535	14.75	68.58	
	180H	1520	7.9	42	90	315	282.9	142.9	0.505	8-700	2633	15.35	79.07	
	181A	1760	8.0	39	95	297	266.7	165.9	0.622	8-700	2887	16.78	82.51	
	181B	1760	7.6	38	87	293	263.1	149.2	0.567	8-700	2831	16.45	72.04	
	181C	1760	6.7	35	82	284	255.0	134.2	0.526	8-700	2704	15.72	72.00	
	181D	1760	7.0	36	81	279	250.5	134.8	0.538	8-700	2746	16.01	68.53	
	181E	1760	6.7	35	79	270	242.5	129.5	0.534	8-700	2704	15.78	65.03	
	181F	1760	6.2	33	75	270	242.5	119.7	0.494	8-700	2634	15.33	61.52	

Δpsi/ 1000 ft	Material gradation ~ % passing																	
	3 in./ 75 mm	2 in./ 50 mm	1 in./ 25 mm	3/4 in./ 19 mm	1/2 in./ 12.5 mm	1/4 in./ 6.3 mm	#8/ 2.36 mm	Pan	C_{DM}	D_M (mm)	D_{95} (mm)	D_{84} (mm)	D_{60} (mm)	D_{50} (mm)	D_{30} (mm)	D_{16} (mm)	D_{10} (mm)	D_5 (mm)
-	100	89	59	50	41	22	2	0	2.45	24.21	62	48	28	19	8.9	6.0	4.8	3.4
71.95																		
-	100	91	61	50	38	20	1	0	2.44	24.09	60	46	24	19	9.0	6.4	5.0	3.9
71.97																		
-	100	93	63	53	40	21	2	0	2.39	22.52	59	39	23	18	9.1	6.0	4.6	3.2
72.01																		
-	100	96	67	55	41	22	2	0	2.37	20.91	49	38	21	17	9.8	5.6	4.1	3.1
61.55																		
-	100	98	80	64	48	20	2	0	2.23	17.49	44	30	18	12	9.0	6.0	4.2	3.0
113.82																		
75.45																		
-	91	63	47	43	35	10	3	0	2.28	35.77	80	71	48	30	11	8.1	6.3	4.8
64.96																		
-	98	80	52	46	36	16	3	0	2.39	28.79	70	51	36	23	10	6.3	5.8	3.9
65.01																		
-	100	85	64	56	45	23	2	0	2.39	23.76	68	49	22	16	8.0	5.8	4.6	3.2
65.04																		
-	98	89	67	60	49	29	9	0	2.37	21.71	66	42	19	13	7.0	3.9	2.6	1.9
61.56																		
68.58																		
65.08																		
72.08																		
75.58																		
82.58																		
68.58																		
79.07																		
82.51																		
72.04																		
72.00																		
68.53																		
65.03																		
61.72																		

Date	Test No.	Q_w (gpm)	Coal (tpm)	C_V (%)	Inj. (psi)	Inj. (hp_m)	Shaft (hp)	Slurry (hp)	Inj. eff.	Pipe $\phi \sim L$ (in.-ft)	Q_s (gpm)	V_s (fps)	Δpsi/ 1000 ft	3 in./ 75 mm
11/9/81	182A	1960	6.0	30	77	261	234.4	131.2	0.560	8-700	2806	16.38	64.97	
	182B	1960	5.9	30	72	252	226.3	122.4	0.541	8-700	2791	16.32	61.48	
	182C	1960	5.3	27	71	251	225.4	116.7	0.518	8-700	2707	15.74	64.93	
	183A	2160	4.9	24	70	238	213.7	121.6	0.569	8-700	2850	16.61	64.89	
	S183	-	-	-	-	-	-	-	-	-	-	-	-	100
	183B	2160	6.4	29	85	288	258.6	158.4	0.612	8-700	3062	17.79	68.44	
	183C	2160	6.6	30	83	279	250.5	156.3	0.624	8-700	3090	18.04	64.97	
	184A	2000	7.2	34	84	292	262.2	154.2	0.588	8-700	3015	17.53	68.50	
	184B	2000	7.2	34	84	292	262.2	154.2	0.588	8-700	3015	17.53	68.50	
	184C	2000	7.2	34	90	292	262.2	164.7	0.628	8-700	3015	17.53	71.98	
	185A	1640	7.9	40	88	315	282.9	146.6	0.518	8-700	2753	16.03	75.26	
	185B	1640	7.9	40	90	310	278.4	149.8	0.538	8-700	2753	16.03	75.56	
	185C	1640	7.9	40	93	315	282.9	154.6	0.547	8-700	2753	16.03	79.04	
	186A	2160	7.2	32	90	284	255.0	174.0	0.682	8-700	3175	18.46	70.22	
	186B	2160	6.7	30	89	288	256.8	168.0	0.650	8-700	3104	18.08	70.20	
11/10/81	S186	-	-	-	-	-	-	-	-	-	-	-	-	100
	186A	1160	3.9	32	67	252	226.3	68.3	0.232	8-700	1710	9.95	40.59	
	186B	1160	3.9	32	56	207	185.9	57.4	0.236	8-700	1710	9.95	33.61	
	186C	1160	3.9	32	56	216	194.0	57.4	0.228	8-700	1710	9.95	44.07	
	186D	1160	3.9	32	51	198	177.8	52.4	0.228	8-700	1710	9.95	30.13	
	S187	-	-	-	-	-	-	-	-	-	-	-	-	100
	187A	1040	3.0	29	18	130	116.7	16.4	0.141	8-700	1463	8.49	5.68	
	187B	1040	3.0	29	24	135	121.2	21.5	0.178	8-700	1463	8.49	2.19	
	187C	1040	3.0	29	35	180	161.6	30.9	0.191	8-700	1463	8.49	16.14	
11/12/81	S188	-	-	-	-	-	-	-	-	-	-	-	-	100
	T188	960	3.0	30	41	171	153.5	34.0	0.222	8-700	1383	8.00	19.64	
	S189	-	-	-	-	-	-	-	-	-	-	-	-	95
	T189	1040	4.9	40	74	279	250.5	76.3	0.305	8-700	1730	10.06	54.63	
	S190	-	-	-	-	-	-	-	-	-	-	-	-	100
	T190	1280	5.5	38	70	261	234.4	86.4	0.368	8-700	2055	11.96	61.58	
	191A	1440	7.4	42	77	306	274.8	115.5	0.420	8-700	2483	14.48	58.15	
	191B	1440	7.4	42	75	302	271.2	112.7	0.415	8-700	2483	14.48	54.66	

Δpsi/ 1000 ft	Material gradation ~ % passing								C_{DM}	D_M (mm)	D_{95} (mm)	D_{84} (mm)	D_{60} (mm)	D_{50} (mm)	D_{30} (mm)	D_{16} (mm)	D_{10} (mm)	D_5 (mm)
	3 in./ 75 mm	2 in./ 50 mm	1 in./ 25 mm	3/4 in./ 19 mm	1/2 in./ 12.5 mm	1/4 in./ 6.3 mm	#8/ 2.36 mm	Pan										
64.97																		
61.48																		
64.93																		
64.89																		
-	100	100	95	88	71	22	2	0	2.04	11.60	25	16	11	10	8.0	6.1	4.8	4.6
68.44																		
64.97																		
68.50																		
68.50																		
71.98																		
75.56																		
75.56																		
79.04																		
70.22																		
70.20																		
-	100	92	70	61	49	26	3	0	2.37	21.32	59	38	18	13	7.6	5.2	4.1	3.0
40.59																		
33.61																		
44.07																		
30.13																		
-	100	83	59	49	36	17	2	0	2.35	26.35	69	51	29	20	11	6.4	5.0	2.6
5.68																		
2.19																		
16.14																		
-	100	93	69	61	49	28	9	0	2.34	19.99	58	39	18	13	7.1	4.4	2.9	1.8
19.64																		
-	95	62	49	46	40	23	2	0	2.48	33.97	75	69	46	28	8.2	6.0	4.9	3.6
54.63																		
-	100	85	57	49	39	20	2	0	2.46	25.81	68	49	30	20	9.0	6.4	5.0	3.6
61.58																		
58.15																		
54.66																		

Date	Test No.	Q_w (gpm)	Coal (psi)	C_v (%)	Inj. (psi)	Inj. (hp_m)	Shaft (hp)	Slurry (hp)	Inj. eff.	Pipe $\phi \sim L$ (in.-ft)	Q_s (gpm)	V_s (fps)	Δpsi/ 1000 ft	3 in./ 75 mm
11/12/81	191C	2000	9.3	40	98	306	274.8	197.8	0.720	8-700	3311	19.27	68.57	
	191D	2000	5.8	29	76	216	194.0	130.1	0.671	8-700	2817	16.39	54.49	
	S192A	-	-	-	-	-	-	-	-	-	-	-	-	100
	192A1	1680	5.3	31	70	256	229.9	102.7	0.444	8-700	2427	14.11	51.03	
	192A2	1680	8.5	42	96	315	282.9	167.1	0.591	8-700	2878	16.75	75.07	
	192A3	1640	7.8	40	93	338	303.5	153.8	0.507	8-700	2739	15.94	68.58	
	S192B	-	-	-	-	-	-	-	-	-	-	-	-	100
	T192B	1640	8.8	43	100	324	290.9	174.0	0.598	8-700	2880	16.74	79.08	
	S193	-	-	-	-	-	-	-	-	-	-	-	-	94
	T193	1440	7.0	41	85	351	315.2	124.1	0.384	8-700	2426	14.13	65.10	
	S194	-	-	-	-	-	-	-	-	-	-	-	-	98
	T194	1200	6.4	43	80	288	258.6	100.7	0.390	8-700	2102	12.23	65.13	
	S195	-	-	-	-	-	-	-	-	-	-	-	-	100
	T195	960	6.4	48	75	284	255.0	83.5	0.327	8-700	1862	10.80	58.23	
	S196	-	-	-	-	-	-	-	-	-	-	-	-	100
	196A	1840	3.8	23	55	198	177.8	79.4	0.447	8-700	2376	13.85	43.95	
	196B	1840	4.2	24	57	207	185.9	84.3	0.454	8-700	2432	14.12	43.97	
	196C	1840	4.6	26	57	207	185.9	86.4	0.465	8-700	2488	14.51	40.51	
	196D	1840	4.0	24	57	207	185.9	83.3	0.448	8-700	2404	14.03	36.99	
	196E	1840	4.5	26	61	212	190.4	91.7	0.482	8-700	2474	14.43	47.48	
	196F	1920	4.1	23	60	216	194.0	91.1	0.470	8-700	2498	14.55	43.96	
	196G	1840	4.0	23	62	230	206.0	90.2	0.437	8-700	2404	13.96	43.96	
	196H	1360	4.1	30	60	216	194.0	69.8	0.360	8-700	1938	11.30	40.56	
	196I	1360	4.6	32	64	234	210.1	77.2	0.367	8-700	2008	11.71	44.08	
	196J	1360	4.8	33	68	194	174.2	83.1	0.477	8-700	2036	11.87	51.06	
11/13/81	S197	-	-	-	-	-	-	-	-	-	-	-	-	100
	T197	1640	5.2	31	86	261	234.4	127.6	0.545	6-300	2373	26.43	144.8	
	S198	-	-	-	-	-	-	-	-	-	-	-	-	100
	T198	1360	5.8	25	55	189	169.7	76.4	0.450	6-300	2177	20.15	88.2	
	S199	-	-	-	-	-	-	-	-	-	-	-	-	100
	T199	1040	3.4	32	49	180	161.6	46.0	0.284	6-300	1519	16.92	73.8	
	S200	-	-	-	-	-	-	-	-	-	-	-	-	100
	T200	920	3.3	33	45	162	145.5	38.4	0.264	6-300	1385	15.38	66.7	

Δpsi/ 1000 ft	Material gradation ~ % passing								C_{DM}	D_M (mm)	D_{95} (mm)	D_{84} (mm)	D_{60} (mm)	D_{50} (mm)	D_{30} (mm)	D_{16} (mm)	D_{10} (mm)	D_5 (mm)
	3 in./ 75 mm	2 in./ 50 mm	1 in./ 25 mm	3/4 in./ 19 mm	1/2 in./ 12.5 mm	1/4 in./ 6.3 mm	#8/ 2.36 mm	Pan										
68.57																		
54.49																		
-	100	97	79	68	51	22	2	0	2.22	17.35	49	30	16	12	8.6	5.9	4.9	3.9
51.03																		
75.07																		
68.58																		
-	100	94	67	57	44	21	2	0	2.35	21.15	54	39	20	16	9.0	5.8	4.2	3.2
79.08																		
-	94	62	47	42	34	18	3	0	2.41	35.14	76	71	49	31	10	6.2	4.8	3.4
65.10																		
-	98	70	46	40	34	19	2	0	2.49	32.60	71	61	43	30	10	5.1	4.8	2.5
65.13																		
-	100	89	57	46	34	14	1	0	2.36	25.65	61	42	28	20	11	8.0	6.0	4.0
58.23																		
-	100	88	64	57	43	20	3	0	2.33	23.20	70	49	21	16	9.8	5.2	4.0	2.9
43.95																		
43.97																		
40.51																		
36.99																		
47.48																		
43.96																		
43.96																		
40.56																		
44.08																		
51.06																		
-	100	91	69	62	48	22	2	0	2.32	20.97	59	40	18	13	8.8	5.9	4.4	3.2
144.8																		
-	100	98	85	80	68	29	2	0	2.26	13.98	41	24	11	9.8	6.9	4.2	3.6	2.8
88.2																		
-	100	97	86	79	62	28	3	0	2.22	14.55	44	24	12.5	12.0	7.1	4.0	3.1	2.4
73.8																		
-	100	99	86	79	66	29	3	0	2.24	13.75	40	24	12	11	6.4	4.0	3.0	2.6
66.7																		

Date	Test No.	Q_w (gpm)	Coal (tpm)	C_v (%)	Inj. (psi)	Inj. (hp_m)	Shaft (hp)	Slurry (hp)	Inj. eff.	Pipe $\phi \sim L$ (in.-ft)	Q_s (gpm)	V_s (fps)	Δpsi/ 1000 ft	3 in./ 75 mm
11/13/81	S201	-	-	-	-	-	-	-	-	-	-	-	-	100
	T201	1360	3.7	27	57	225	202.0	67.0	0.331	6-300	1881	20.87	88.2	
	S202	-	-	-	-	-	-	-	-	-	-	-	-	100
	T202	1040	3.3	31	49	180	161.6	45.5	0.281	6-300	1505	16.79	73.8	
	T203	960	3.4	33	45	171	153.6	40.0	0.261	6-300	1439	15.95	66.7	
	S204	-	-	-	-	-	-	-	-	-	-	-	-	100
	T204	1600	6.3	36	94	270	242.5	146.6	0.604	6-300	2488	23.07	173.2	
	S205	-	-	-	-	-	-	-	-	-	-	-	-	100
	T205A	1040	6.0	45	75	297	266.7	87.4	0.328	6-300	1886	20.88	130.3	
	T205B	1040	6.7	48	87	324	290.9	106.4	0.366	6-300	1984	15.07	158.7	
	S206	-	-	-	-	-	-	-	-	-	-	-	-	100
	T206	960	4.3	39	55	202	181.4	53.1	0.294	6-300	1566	17.41	87.8	
	S207	-	-	-	-	-	-	-	-	-	-	-	-	100
	T207	960	4.0	37	52	198	177.8	48.9	0.275	6-300	1524	16.97	80.8	
	S208	-	-	-	-	-	-	-	-	-	-	-	-	100
	T208	1000	5.2	42	66	252	226.3	70.5	0.312	6-300	1733	19.23	116.2	
	S209	-	-	-	-	-	-	-	-	-	-	-	-	100
	T209	960	5.4	44	69	261	234.4	73.1	0.312	6-300	1721	19.11	116.1	
	S210	-	-	-	-	-	-	-	-	-	-	-	-	100
	T210	1600	5.3	32	80	234	210.1	117.9	0.561	6-300	2347	26.09	144.8	
	S211	-	-	-	-	-	-	-	-	-	-	-	-	100
	T211	1560	8.0	42	100	279	250.5	169.8	0.678	6-300	2687	29.88	187.2	
	S212	-	-	-	-	-	-	-	-	-	-	-	-	100
	T212	1520	7.9	42	99	270	242.5	164.4	0.678	6-300	2633	29.28	187.2	
	S213	-	-	-	-	-	-	-	-	-	-	-	-	100
	T213	1320	6.3	40	84	266	238.9	115.6	0.484	6-300	2208	24.62	165.9	
	S214	-	-	-	-	-	-	-	-	-	-	-	-	100
	T214	1320	6.2	40	82	261	234.4	112.2	0.479	6-300	2194	24.33	151.7	
	S215	-	-	-	-	-	-	-	-	-	-	-	-	100
	T215	720	3.6	41	45	162	145.5	33.8	0.232	6-300	1227	13.67	73.6	
	S216	-	-	-	-	-	-	-	-	-	-	-	-	100
	T216	720	3.6	41	46	162	145.5	34.5	0.237	6-300	1227	13.67	73.6	

Δpsi/ 1000 ft	Material gradation ~ % passing								C_{DM}	D_M (mm)	D_{95} (mm)	D_{84} (mm)	D_{60} (mm)	D_{50} (mm)	D_{30} (mm)	D_{16} (mm)	D_{10} (mm)	D_5 (mm)
	3 in./ 75 mm	2 in./ 50 mm	1 in./ 25 mm	3/4 in./ 19 mm	1/2 in./ 12.5 mm	1/4 in./ 6.3 mm	#8/ 2.36 mm	Pan										
-	100	96	85	78	63	31	3	0	2.28	14.30	42	24	12	10	6.8	4.2	3.2	2.6
88.2																		
-	100	96	88	83	67	31	2	0	2.25	13.80	46	21	12	10	7.0	4.4	3.6	2.6
73.8																		
66.7																		
-	100	98	88	80	64	28	3	0	2.20	13.80	36	21	12	10	7.0	4.4	3.6	2.6
173.2																		
-	100	98	86	78	63	29	3	0	2.24	14.25	38	26	11	9.0	7.0	4.8	3.8	2.6
130.3																		
158.7																		
-	100	97	82	76	61	29	2	0	2.29	15.40	49	29	12	9.0	6.0	4.0	3.2	2.6
87.8																		
-	100	98	88	81	64	30	2	0	2.24	13.66	38	22	12	9.1	6.9	4.6	3.8	2.8
80.8																		
-	100	99	89	83	69	35	3	0	2.30	12.54	38	21	10	8.1	6.2	4.6	2.6	2.8
116.2																		
-	100	98	83	74	60	30	4	0	2.28	15.08	39	26	12.5	12.0	6.3	4.6	3.8	2.9
116.1																		
-	100	99	83	73	57	27	2	0	2.26	15.29	40	26	15.0	12.0	7.8	4.8	2.8	1.9
144.8																		
-	100	100	91	86	72	24	2	0	2.11	12.18	32	18	11	9.9	6.9	5.1	4.0	3.1
187.2																		
-	100	97	81	72	58	29	2	0	2.31	15.99	41	29	15	11	7.0	4.6	3.8	1.9
187.2																		
-	100	99	83	75	62	32	10	0	2.42	14.39	39	24	12	11	6.6	3.4	2.36	1.9
165.9																		
-	100	99	86	78	62	29	2	0	2.25	13.46	37	24	12	11	7.0	4.2	3.2	2.6
151.7																		
-	100	98	84	75	61	27	3	0	2.23	14.98	43	25	12	9.1	6.9	5.2	4.0	2.8
73.6																		
-	100	99	91	84	66	27	2	0	2.16	12.79	34	19	12	11	8.9	4.9	3.9	2.6
73.6																		

Date	Test No.	Q_w (gpm)	Coal (tpm)	C_v (%)	Inj. (psi)	Inj. (hp_m)	Shaft (hp)	Slurry (hp)	Inj. eff.	Pipe $\phi \sim L$ (in.-ft)	Q_s (gpm)	V_s (fps)	Δpsi/ 1000 ft	3 in./ 75 mm
11/13/81	S217	-	-	-	-	-	-	-	-	-	-	-	-	100
	T217	720	3.4	40	43	158	141.9	31.5	0.222	6-300	1199	13.34	59.4	
	T217	720	2.5	33	36	117	105.1	23.6	0.225	6-300	1072	11.94	52.5	
	S218	-	-	-	-	-	-	-	-	-	-	-	-	100
	T218	720	2.7	34	35	126	113.1	23.6	0.209	6-300	1100	12.20	52.4	
	S219	-	-	-	-	-	-	-	-	-	-	-	-	100
	Coal - SG = 1.7 for all tests													
	Crushed stone - SG = 2.8													
	Start	-	-	-	-	-	-	-	-	-	-	-	-	100
	101	1000	2.3	16						6-300		13.3	80.3	
	102	1000	1.2	10						6-300		12.3	59.7	
	103	720	1.2	13						6-300		9.2	59.7	
	104	800	1.3	12						6-300		10.1	59.7	
	105	800	1.6	15						6-300		10.4	80.3	
	106	800	1.8	16						6-300		10.6	80.3	
	107	1040	2.0	14						6-300		13.5	88.1	
	108	1080	1.9	13						6-300		13.8	59.7	
	109	720	1.7	17						6-300		9.7	80.3	
	110	720	1.9	19						6-300		9.9	87.4	
	111	800	2.0	18						6-300		10.8	80.3	
	112	1040	2.0	14						6-300		13.5	73.9	
	113	1360	2.0	11						6-300		17.1	73.9	
	Stop	-	-	-	-	-	-	-	-	-	-	-	-	100

Inj. = injector

Δpsi/ 1000 ft	Material gradation ~ % passing								C_{DM}	D_M (mm)	D_{95} (mm)	D_{84} (mm)	D_{60} (mm)	D_{50} (mm)	D_{30} (mm)	D_{16} (mm)	D_{10} (mm)	D_5 (mm)
	3 in./ 75 mm	2 in./ 50 mm	1 in./ 25 mm	3/4 in./ 19 mm	1/2 in./ 12.5 mm	1/4 in./ 6.3 mm	#8/ 2.36 mm	Pan										
-	100	98	88	81	65	33	4	0	2.26	13.40	39	20	12	9.9	6.2	4.2	3.6	2.9
59.4																		
52.5																		
-	100	98	93	89	75	33	2	0	2.22	11.54	32	16	10	9.0	5.8	3.6	3.0	2.4
52.4																		
-	100	100	90	84	70	34	3	0	2.72	12.06	32	19	12	9.2	6.1	4.2	3.4	2.8
-	100	100	90	60	29	17	8	0	3.19	16.86	38	24	19	18	13	6.1	2.8	0.9
80.3																		
59.7																		
59.7																		
59.7																		
80.3																		
80.3																		
88.1																		
59.7																		
80.3																		
87.4																		
80.3																		
73.9																		
73.9																		
-	100	100	89	60	28	18	7	0	3.25	17.05	30	24	19	17	12	5	2.8	1.2

APPENDIX H.--SPECIFIC GRAVITY CALCULATIONS

H.1 TEST PROCEDURES

1. Determine volume of container

Volume, gal =

$$\frac{\text{(wt of water) + (wt of container) - (wt of container)}}{\text{8.347 lb/gal}}$$

2. Determine volume of dry material in filled container

wt of material = (wt of material + wt of container) - (wt of container)

wt of material + water = (wt of material + wt of water + wt of container) - (wt of container)

Volume of water, gal =

$$\frac{\text{(wt of material + water) - (wt of material)}}{\text{8.347 lb/gal}}$$

Volume of material = volume of container - volume of water

3. Determine weight of an equal volume of water

wt of equal volume of water = volume of material x 8.347 lb/gal

4. Determine specific gravity of material

$$SG = \frac{\text{wt of material}}{\text{wt of equal volume of water}}$$

H.2 DETERMINATION OF COAL SPECIFIC GRAVITY - SAMPLE CALCULATIONS

1. Volume of container

$$V_c = \frac{42.43 \text{ lb}}{8.347 \text{ lb/gal}} = 5.083 \text{ gal}$$

2. Volume of coal in filled container

wt of material = 36.25 lb

wt of material + water = 57.25 lb

$$\text{Volume of water} = V_w = \frac{57.25 - 36.25}{8.347} = 2.516 \text{ gal}$$

$$V_{coal} = 5.083 \text{ gal} - 2.516 \text{ gal} = 2.567 \text{ gal}$$

3. Weight of equal volume of water

$$wt_w = 2.567 \times 8.347 = 21.429 = 21.43 \text{ lb}$$

4. Specific gravity of coal

$$SG = \frac{36.25}{21.43} = 1.69 = 1.7$$

H.3 CONCENTRATION CALCULATIONS

1. Concentration by weight

$$C_{wt} = \frac{\text{wt flow rate of solids}}{\text{wt flow rate of slurry}}$$

$$\text{wt flow rate of solids} = \frac{\text{tons}}{\text{min}} \times 2{,}000 \frac{\text{lb}}{\text{min}}$$

where tons per minute comes from weigh belt calculations.

wt flow rate of slurry = wt flow rate of water

$$+ \text{ wt flow rate of solids} = \frac{\text{gal}}{\text{min}} \times 8.347 + \frac{\text{tons}}{\text{min}} \times 2{,}000 \ \frac{\text{lb}}{\text{min}}$$

where gallons per minute comes from clear water flow meter calibrated to read in gallons per minute.

$$C_{wt}\% = \frac{\text{tons/min} \times 2{,}000}{\text{gal/min} \times 8.347 + \text{tons/min} \times 2{,}000} \times 100$$

2. Concentration by volume

$$C_v = \frac{\text{volumetric flow rate of solids}}{\text{volumetric flow rate of slurry}}$$

Volumetric flow rate of solids =

$$\frac{\text{tons/min} \times 2{,}000}{\text{SG} \times 62.4} \ \frac{\text{ft}^3}{\text{min}}$$

where tons per minute comes from weigh belt, and SG is determined from SG test.

Volumetric flow rate of slurry = volumetric flow rate of solids + volumetric flow rate of water

$$= \left(\frac{\text{gal/min}}{7.48 \text{ gal/ft}^3} + \frac{\text{tons/min} \times 2{,}000}{\text{SG} \times 62.4}\right) \frac{\text{ft}^3}{\text{min}}$$

$$C_v\% = \frac{\dfrac{\text{tons/min} \times 2{,}000}{\text{SG} \times 62.4}}{\dfrac{\text{gal/min}}{7.48} + \dfrac{\text{tons/min}}{\text{SG} \times 62.4} \times 2{,}000} \times 100$$

For coal with SG = 1.7

$$C_v\% = \frac{\text{tons/min} \times 18.854}{\frac{\text{gal/min}}{7.48} + \text{tons/min} \times 18.854} \times 100$$

or for use of clear water and slurry flow meter

$$C_v\% = \frac{\text{gal/min}_{\text{slurry}} - \text{gal/min}_{\text{water}}}{\text{gal/min}_{\text{slurry}}} \times 100$$

APPENDIX I.--LINE LOSS CALCULATIONS

GENERAL

Apply Bernoulli equation between any two points in a pipeline

$$\frac{V_1^2}{2g} + \frac{P_1}{\gamma_s} + Z_1 = \frac{V_2^2}{2g} + \frac{P_2}{\gamma_s} + Z_2 + H_f + H_k \qquad \text{(I-1)}$$

where all terms are in feet of slurry and

H_f = lines loses due "friction"

H_k = loses due to fittings, bends, etc.

Assume pipe diameter at point 1 and point 2 are equal, then

$$V_1 = V_2$$

and

$$H_f = \left(\frac{P_1 - P_2}{\gamma_s}\right) + (Z_1 - Z_1) - H_k \quad \text{feet of slurry}$$

or

$$\frac{\gamma_s H_f}{144} = (P_1 - P_2) + (Z_1 - Z_2)\frac{\gamma_s}{144} - H_k\frac{\gamma_s}{144}\ \text{lb/in}^2 \qquad \text{(I-2)}$$

where P_1 and P_2 are in pounds per square inch.

Letting

$$\frac{\gamma_s H_f}{144} = H_L \quad \text{line loss in lb/in}^2$$

and

$$\frac{\gamma_s H_k}{144} = \Sigma\Delta P_k \quad \text{minor losses in lb/in}^2$$

then equation I-2 becomes

$$H_L = (P_1 - P_2) - \Sigma\Delta P_k + (Z_1 - Z_2)\,\frac{\gamma_s}{144}$$

where

$$\begin{aligned}\gamma_s &= C_v\,\gamma_{solids} + (1 - C_v)\gamma_{water} \\ &= C_v(SG)62.4 + (1 - C_v)62.4 \\ &= C_v(SG)62.4 + 62.4 - C_v\,62.4 \\ &= C_v\,62.4(SG - 1) + 62.4\end{aligned}$$

Therefore

$$H_L = \Delta P_{1-2} - \Sigma\Delta P_k + \Delta Z_{1-2}\,\frac{(C_v\,62.4(SG - 1) + 62.4)}{144} \qquad \text{(I-3)}$$

For coal SG = 1.7

$$H_L = \Delta P_{1-2} - \Sigma\Delta P_k + \Delta Z_{1-2}\,\frac{(C_v\,62.4(SG - 1) + 62.4)}{144}\ \text{lb/in}^2$$

$$H_L = \Delta P_{1-2} - \Sigma\Delta P_k + \Delta Z_{1-2}\,\frac{(43.68\,C_v + 62.4)}{144}\ \text{lb/in}^3$$

Line loss data is presented as a plot on log-log paper of

$$\frac{H_L}{1{,}000 \text{ ft}} \text{ versus } V_s$$

where

V_s = slurry velocity in feet per second

$H_L/1{,}000$ = pressure drop/1,000 ft in lb/in^2/1,000 ft

$$\frac{H_L}{1{,}000} = H_L \frac{(1{,}000)}{\Delta L}$$

where ΔL is the measured pipeline distance between points 1 and 2 less any distance between pressure taps used to measure the minor loss in the term $\Sigma \Delta P_K$ in equation I-3.

$$V_s = \frac{\text{slurry volumetric flow rate}}{\text{pipeline cross sectional area}}$$

when determined from a slurry flow meter

$$V_s = \frac{Q_s}{A_p} \text{ ft/s} = \frac{Q \text{ gal/min}}{448.8 \times A_p} \text{ ft/s}$$

where

$$A_p = \frac{\pi(0.698)^2}{4} = 0.383 \text{ ft}^2 \text{ for 8-in ductile iron}$$

and

$$A_p = \frac{\pi(0.505)^2}{4} = 0.200 \text{ ft}^2 \text{ for 6-in schedule 40 steel}$$

When determined using weigh belt data

$$V_s = \frac{Q_s}{A_p} \text{ ft/s} = \frac{Q_{water} + Q_{solids}}{A_p}$$

where

$$Q_{water} = \frac{\text{gal/min}_{water}}{448.8} \text{ ft}^3\text{/s}$$

$$Q_{solids} = \frac{\text{tons/min}_{solids} \text{ x } 2{,}000}{\text{SG x } 62.4 \text{ x } 60} \text{ ft}^3\text{/s}$$

Therefore, for coal at SG = 1.7

V_s in 8-in line =

$$\frac{(\text{gal/min}_{water}/448.8) + (\text{tons/min}_{coal} \text{ x } 0.314)}{0.383}$$

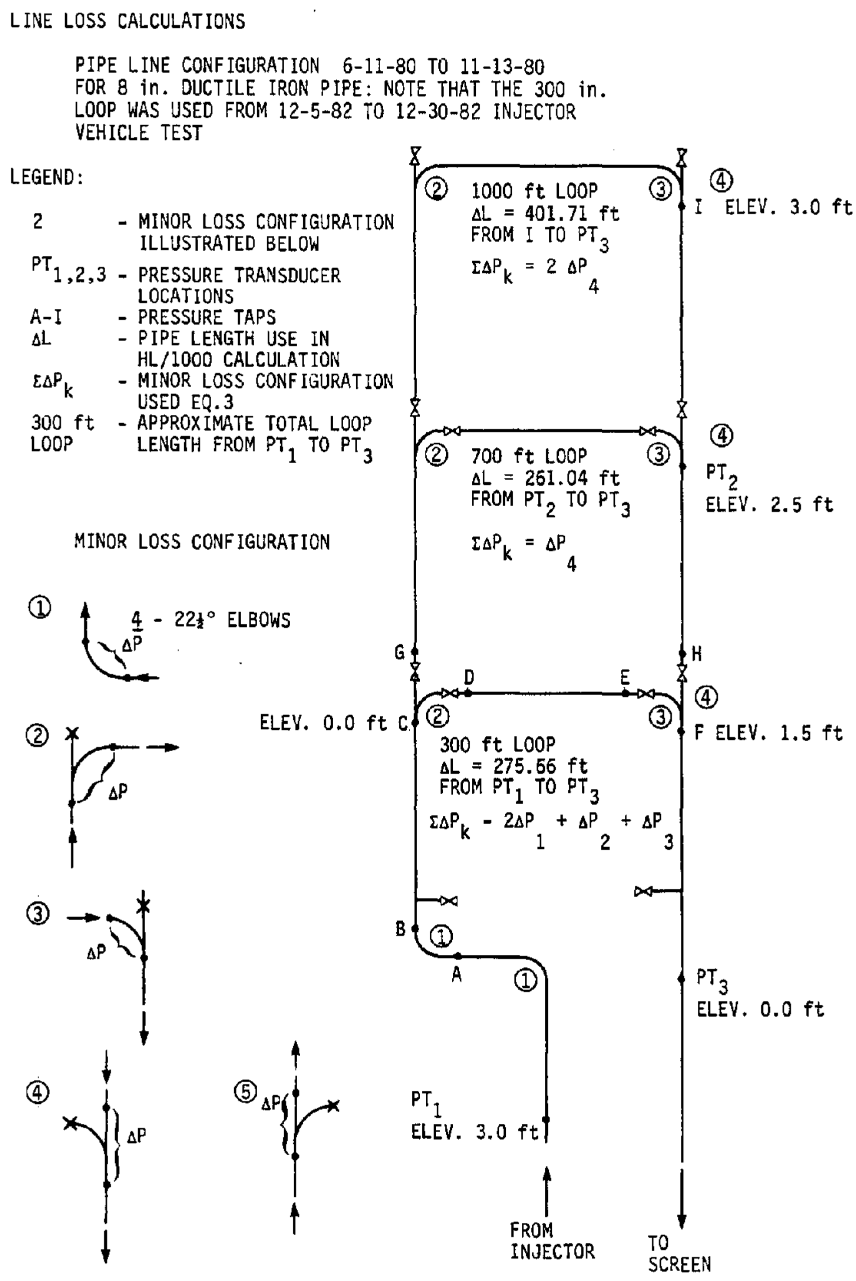
$$V_s = \frac{\text{gal/min}}{171.89} + (\text{tons/min x } 0.820)$$

V_s in 6-in line =

$$\frac{(\text{gal/min}_{water}/448.8) + (\text{tons/min}_{coal} \text{ x } 0.314)}{0.200}$$

$$V_s = \frac{\text{gal/min}_{water}}{89.89} + (\text{tons/min x } 1.568)$$

The minor loss configurations are shown in figures I-1 through I-3.



NOTE THAT THE 300 ft LOOP WAS USED FROM 5 DECEMBER 1982 TO 30 DECEMBER 1982 INJECTOR VEHICLE TEST

FIGURE I-1. - Pipeline configuration for 8-in ductile iron pipe (11 June 1980 to 13 November 1980).

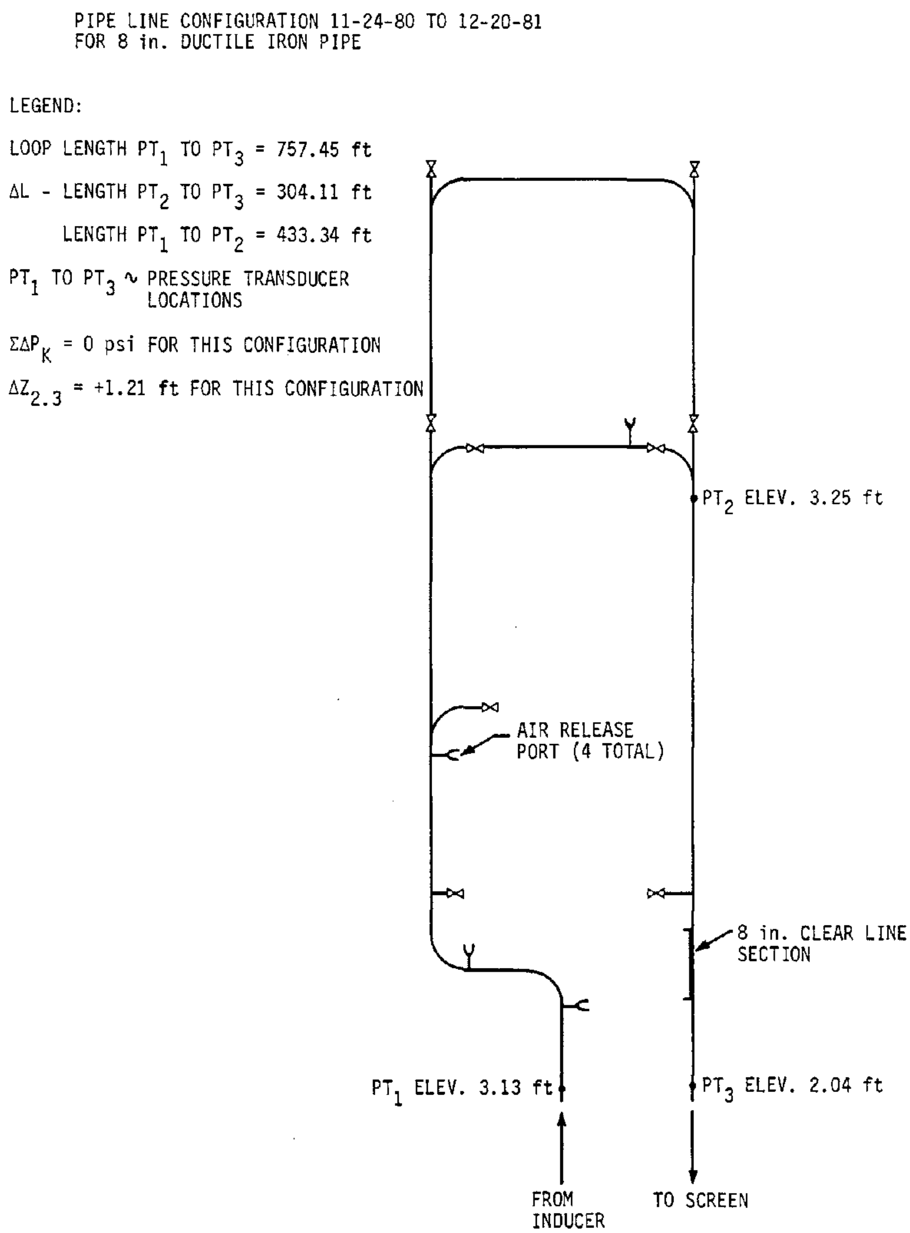


FIGURE I-2. - Pipeline configuration for 8-in ductile iron pipe (24 November 1980 to 20 December 1981).

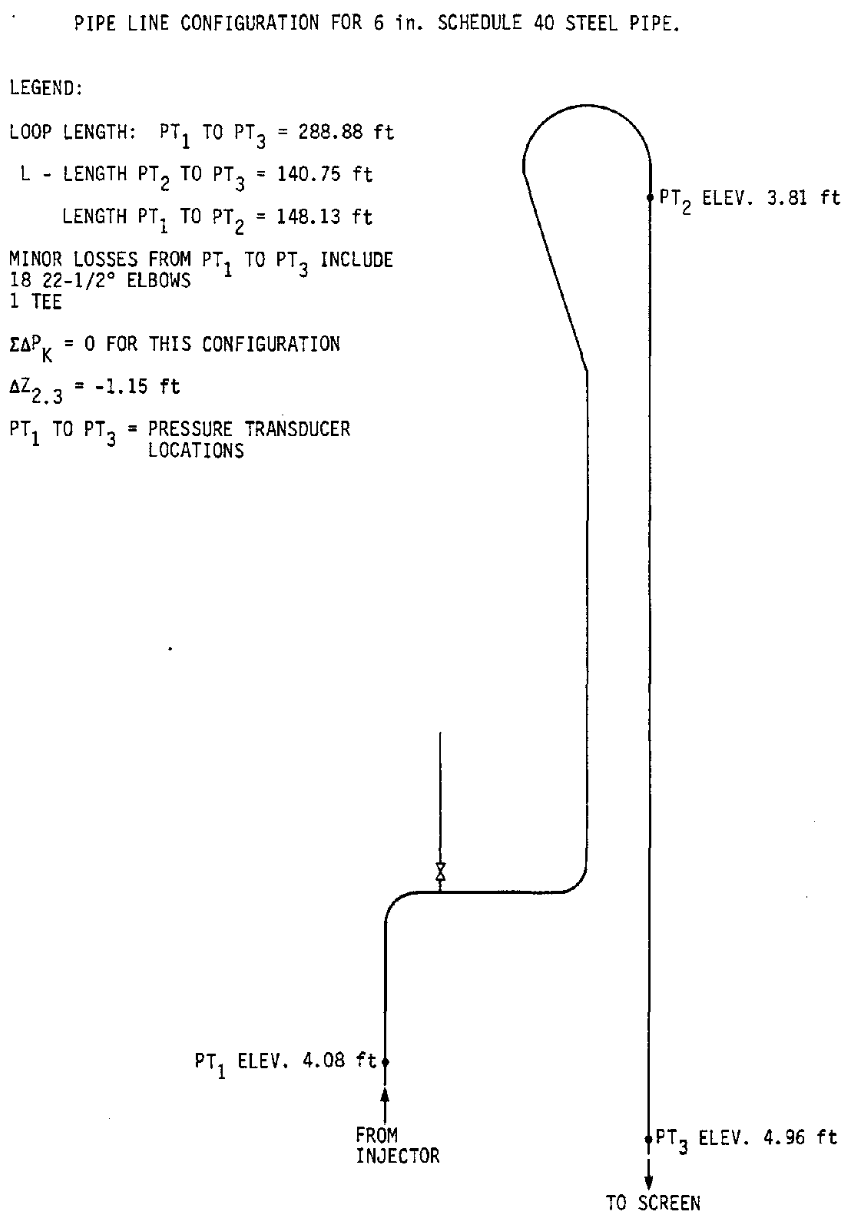


FIGURE I-3. - Pipeline configuration for 6-in schedule 40 steel pipe.

APPENDIX J.--INDUCER EFFICIENCY

Assumptions

For the testing of the helical inducer (fig. J-1), the solids were fed via a feed screw and water was introduced through a 6-in inlet. The feed screw was belt-driven by an electric motor which was instrumented to determine the input power to the drive motor. The inducer inlet was not instrumented. However, the pond pump flow rate changed very little from the case of pumping water through the bypass or pumping water into the injector indicating approximately the same back pressure for each case. This was for the 300-ft line.

Assuming that the pressure read by the inducer pressure transducer during the bypass case is approximately the pressure seen at the water inlet when water is being pumped to the inducer, then the horsepower required to put water into the inducer can be estimated. The pressure in the bypass case was 26 and 16 lb/in^2 for 2,200 and 1,240 gal/min, respectively. Therefore, the power required for the water input can be estimated to be 38 hp at 2,200 gal/min and 12 hp at 1,240 gal/min.

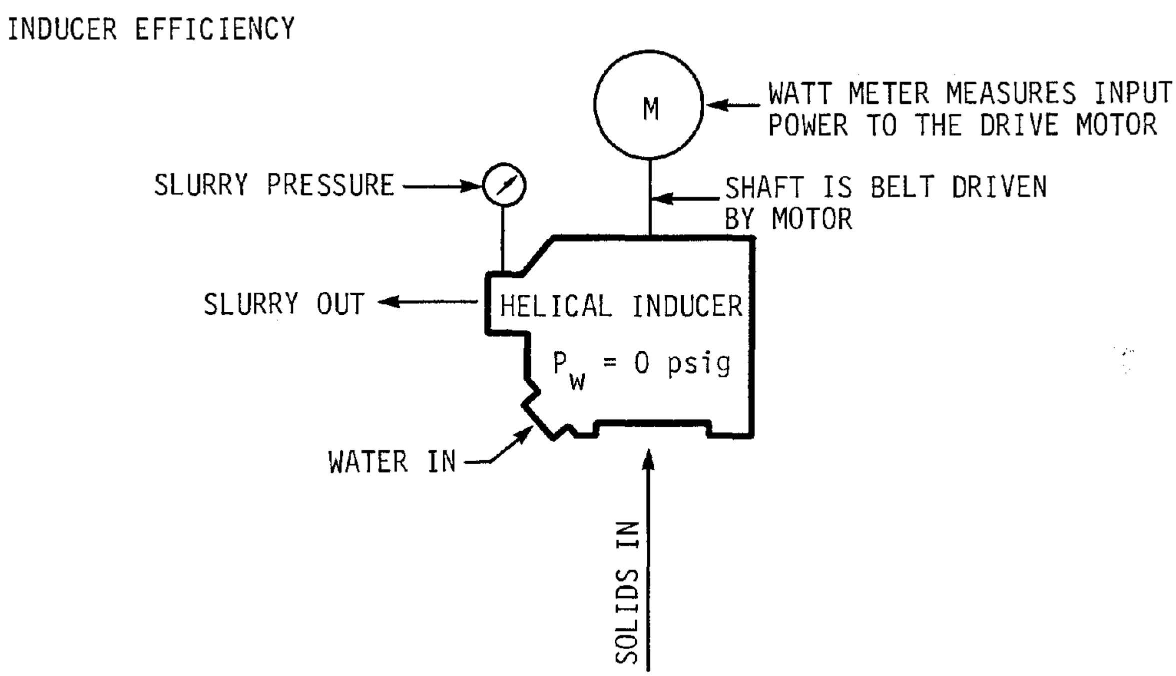


FIGURE J-1. - Definition sketch.

The feed screw horsepower ranged from 8 to 10 hp and assuming a motor-belt efficiency of 0.89, then the required feed screw power ranged from 7 to 9 hp. Therefore, the total material feed power ranges from 19 to 47 hp. However, the material feed power is a function of the coal feed system and the water inlet, and could be higher or lower depending on a particular feed system. Therefore, in this analysis, the required material feed power and the water supply power requirements were left out of the efficiency equation.

The basic assumption in this analysis is that the slurry is introduced into the inducer at 0 psig and with the addition of shaft horsepower, the slurry is discharged at some pressure. Based on this assumption, the efficiency equation is;

$$\eta = \frac{hp_{OUT}}{hp_{IN}}$$

or

$$\eta = \frac{\left(\frac{Q_s \gamma_s H_s}{550}\right)_{OUT}}{\left(\frac{Q_s \gamma_s H_s}{550}\right)_{IN} + (\text{shaft hp})_{IN}}$$

where

$$H_{s_{IN}} = \frac{V^2}{2g} + 0$$

assuming

$$\frac{V^2}{2g} \approx 0$$

then

$$\eta = \frac{\left(\frac{Q_s \gamma_s H_s}{550}\right)_{OUT}}{0 + (\text{motor hp} \times \eta_{motor} \times \eta_{belts})_{IN}}$$

$$\eta_{motor} = 0.945$$

$$\eta_{belt} = 0.95$$

then

$$\eta = \frac{(Q_s \gamma_s H_s)_{OUT}}{(\text{motor hp} \times 0.898)_{IN}}$$

APPENDIX K.--GRADATION CALCULATIONS

Sieve analysis was done for material samples taken during testing. For each sample taken, a gradation curve was plotted in terms of percent passing versus screen size. From the gradation curves, the following particle sizes, which are commonly used gradation parameters, were determined. These sizes are d_{60}; d_{10}; d_{30}; d_{84}; d_{16}; d_{95}; d_5; and d_{50}. A mean sample particle size was also calculated using the following equation:

$$d_m = \frac{\Sigma d_{AVE}\ \Delta P}{100}$$

where

d_{AVE} = average particle retained on a sieve screen

ΔP = percentage of a sample by weight retained on a sieve screen

Before sieving, the material samples were dried and weighed. The calculations for determining percent passing for each sieve size is outlined on the following data sheet.

Samples were not taken for each test run for tests 100 through 146, or in some cases the sample storage bag was damaged and the sample was destroyed. Therefore, a material balance scheme was devised to estimate the gradation curves for tests which were missing material samples. For most other tests, samples were taken at the start and finish, thus giving a more accurate estimate of the actual material gradation for each test. No samples were taken when fully degraded material was used since the gradation of this material did not change.

The actual material for each test can be estimated by taking an average of before and after samples for each test.

The material balance scheme used for filling in missing data is a method by which a weighted average correction factor is applied to the starting sample gradation curve. This correction factor is a function of the total amount of material passed through the slurry circuit in the time between samples and the accumulative incremental amount of material passed through the slurry circuit at the end of the test. This scheme is illustrated in the following sample.

Sample No.________			Dry Sample Weight________		
Sieve size	wt retain	Corrected wt retain	% retain by wt	% passing by wt	Notes
3 in.					
2 in.					
1 in.					
3/4 in.					
1/2 in.					
#3					
#8					
Σ					

$$C = \text{correction factor} = \frac{\text{dry sample wt}}{\Sigma \text{ wt retained}}$$

$$\text{corrected wt retained} = C \times \text{wt retained}$$

$$\% \text{ retained} = \frac{\text{corrected wt}}{\text{dry sample wt}} \times 100$$

$$\% \text{ passing} = 100 - \% \text{ retained}$$

Assumptions

For short periods of time, the coal volume in the circuit remains constant and the coal degrades at a uniform rate.

Letting

P_s = percentage passing for starting sample

P_f = percentage passing for ending sample

P_i = percentage passing for missing sample

T_s = total tonnage at starting sample

T_f = total tonnage at end sample

T_i = total tonnage at missing sample

$$\frac{P_s - P_i}{P_s - P_f} = \frac{T_s - T_i}{T_s - T_f}$$

Letting

$$K = \frac{T_s - T_i}{T_s - T_f}$$

then

$$\frac{P_s - P_i}{P_s - P_f} = K$$

Therefore

$$P_s - P_i = KP_s - KP_f$$

$$P_i = P_s(1 - K) + KP_f$$

where P_i is determined for each sieve size.

The following data is given for tests 102 to 105, K is to be determined.

Test No.	Sample No.	Running time (min)	tons/ min	Total (tons)	Correction factor K
	102				0.0
102		2	3.7	7.4	
					0.216
103		3	3.6	10.8	
	104[1]			} 34.4	0.529
104		2	3.5	7.0	
105		2	4.6	9.2	
	105				1.000/0.0[2]
106		2	5.1	10.2	
	106				1.000/0.0
107		3	5.2	15.6	
	107[1]			} 70.6	0.221
108		10	5.5	55.0	
	108				1.000/0.0

[1]Part of sample was destroyed, therefore only part of the gradation curve was available.

$$K_{102} = \frac{0 - 0}{0 - 34.4} = 0 \text{ at start}$$

$$K_{103} = \frac{0 - 7.4}{0 - 34.4} = 0.216 \text{ at start}$$

$$K_{104} = \frac{0 - (7.4 + 10.8)}{0 - 34.4} = 0.529 \text{ at start}$$

$$K_{105} = \frac{0 - (7.4 + 10.8 + 7.0 + 9.2)}{0 - 34.4} = 1.0 \text{ at end of 102/105 group}$$

$$K_{105} = \frac{0 - 0}{0 - 10.2} = 0.0 \text{ at start of 105/106 group}$$

[2]1.000/00 indicates factors at the end and the start of two groups which have a common sample.

For 102, $P_i = P_s(11 - 0) + 0\ P_f = P_s$

For 103, $P_i = P_s(1 - 0.216) + (0.216)P_f$

$= 0.784\ P_s + 0.216\ P_f$

For 104, $P_i = P_s(1 - 0.529) + 0.529\ P_f$

$= 0.471\ P_s + 0.529\ P_f$

For 105 at end, $P_i = P_s(1 - 1) + 1.0\ P_f = P_f$

For 105 at start, $P_i = P_s(1 - 0) + 0\ P_f = P_s$

APPENDIX L.--DETERMINATION OF DRAG COEFFICIENTS

Tests were conducted to determine the drag coefficients for two materials, coal specific gravity of 1.7 and crushed stone specific gravity of 2.8.

The materials were first sized into the following particle sizes using sieve analysis screens.

Note	Screen size	Screen D, mm	Particle size D_{AVE}, mm
Hand measure	3-1/2 in	90	82.5
Sieve	3 in	75	62.5
Sieve	2 in	50	37.5
Sieve	1 in	25	22.0
Sieve	3/4 in	19	15.75
Sieve	1/2 in	12.5	9.40
Sieve	#3	6.3	4.33
Sieve	#8	2.36	1.88
Dewater screen	#12	1.40	

Each particle size was dropped in a 4- by 4- by 2-ft tank and their fall velocities were determined as both individual and grouped particles. No difference in fall velocities were observed during the individual and grouped tests. For test, the particles were allowed to achieve a stable fall condition before measurements were taken.

Fall velocity and drag coefficient are related by the following equation:

$$W^2 = \frac{4}{3}\frac{gd}{C_D}\left(\frac{\gamma_S - \gamma_F}{\gamma_F}\right)$$

where

W = fall velocity

g = acceleration of gravity

d = particle diameter

C_D = drag coefficient

γ_S = unit weight of particle

γ_f = unit weight of fluid

Solving for drag coefficient in water

$$C_D = \frac{4}{3}\frac{gd}{W^2}\left(\frac{\gamma_S}{\gamma_F} - \frac{\gamma_F}{\gamma_F}\right) = \frac{4}{3}\frac{gd}{W^2}\,(SG - 1)$$

In metric units with d in millimeters

$$C_D = \frac{4}{3}\,\frac{9.81}{W^2}\left(\frac{d}{1,000}\right)(SG - 1)$$

Or for coal SG = 1.7

$$C_D = 0.092\left(\frac{d_{AVE}}{W^2}\right)$$

And for gravel SG = 2.8

$$C_D = 0.0235\left(\frac{d_{AVE}}{W^2}\right)$$

The results of the above calculations were plotted as C_D versus $\mathbb{R}_W$ on a log-log scale where $\mathbb{R}_W$ is the particle Reynolds number defined by the following equation:

$$\mathbb{R}_W = \frac{Wd}{\nu}$$

where

W = fall velocity

d = particle size, d_{AVE}

ν = kinematic viscosity

$\mathbb{R}_W$ = particle Reynolds number

or

$$\mathbb{R}_W = \frac{W(\text{m/sec})\ d(\text{mm})}{\nu\ \text{ft}^2/\text{s}} \times \frac{3.281 \times 10^{-3}\ (\text{ft/mm})}{3.048 \times 10^{-1}\ (\text{m/ft})}$$

for $\nu = 1.217 \times 10^{-5}\ \text{ft}^2/\text{s}$

$\mathbb{R}_W = 884.506\ WD_S$

W in millisecond

D_S in millimeter

The C_D versus $\mathbb{R}_W$ values were first plotted on existing similar curves to determine if the values were reasonable and to see if they were in the turbulent, transition or laminar range. The data points all fell in the turbulent range and were then fitted to a power linear regression type line. This second plot indicated that the data point for d_{AVE} between #8 and #12 screen size was low. So this data point was dropped, a new regression line plotted and a C_D value for the #8 to #12 material size was estimated from this line. The results of this analysis are given below.

Screen size	Particle size	Coal		Gravel	
		C_D	R_W	C_D	R_W
3-1/2 in.	82.5	2.32	4.16×10^4	-	-
3 in.	62.5	2.56	2.61×10^4	3.96	3.37×10^4
2 in.	37.5	2.86	1.15×10^4	3.71	1.62×10^4
1 in.	22.0	1.83	6.44×10^3	2.97	8.12×10^3
3/4 in.	15.75	1.55	4.24×10^3	2.81	5.05×10^3
1/2 in.	9.40	1.66	1.89×10^3	2.63	2.41×10^3
#3	4.33	3.23	4.24×10^2	5.31	5.31×10^2
#8	1.88	1.40 [1]	1.84×10^2	2.30	2.30×10^2
#12		2.05 [2]	1.84×10^2	3.30	2.30×10^2

[1] Measured C_D
[2] Estimate from best fit line C_D

A sample drag coefficient C_{DM} for each line-loss test sample was determined by the following equation.

$$C_{DM} = \sum_{i=1}^{N} \frac{(\Delta P_i)\left(C_{D_{AVE\ i}}\right)}{100} = \left[\frac{\left(P_{i+1} - P_i\right) \dfrac{\left(C_{D_i} \quad C_{D_{i+1}}\right)}{2}}{100} \right]$$

where

$C_{D_i} + C_{D_{i+1}}$ = drag coefficients for ith and i+1 screen size

$P_i + P_{i+1}$ = percentage of sample passing on the ith and i+1 screen size

C_{DM} = sample drag coefficient

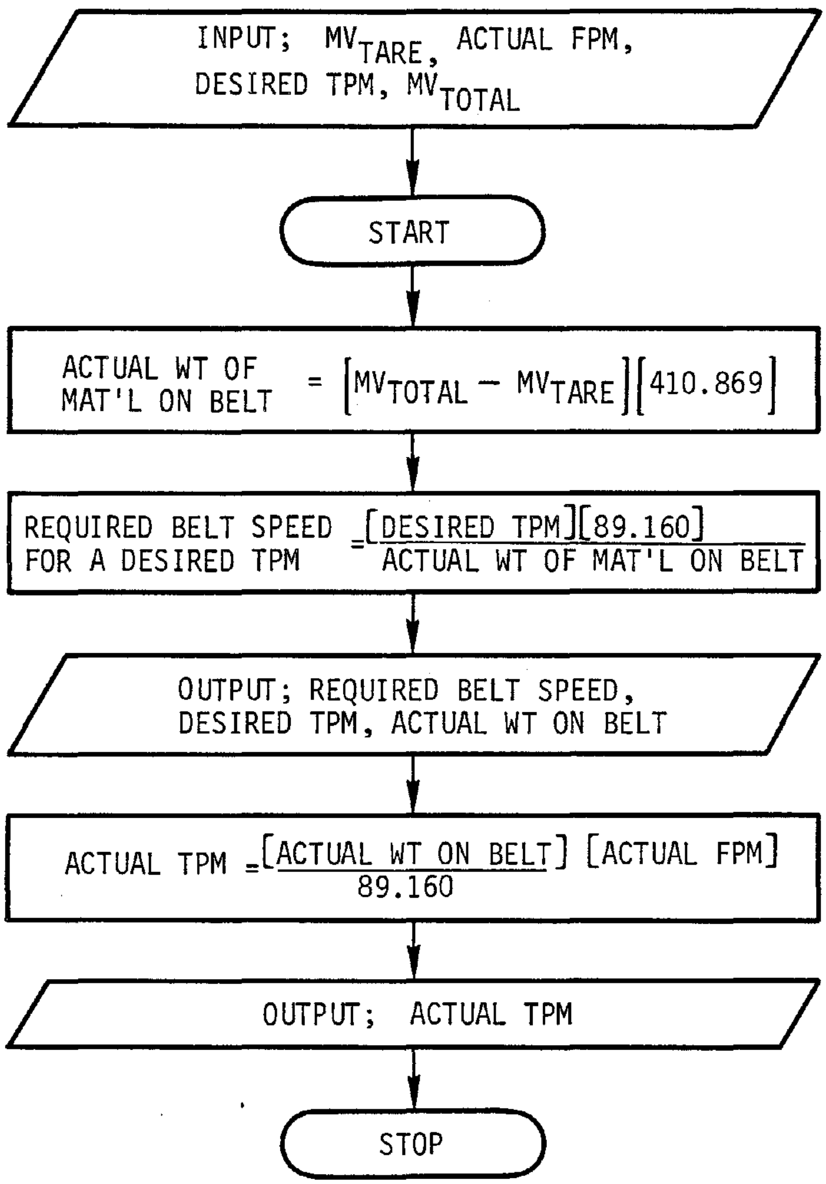
APPENDIX M

DATA REDUCTION PROGRAMS

1. Weigh belt calculations for HP-95 calculator basic equation:

$$\frac{\text{tons}}{\text{min}} = \frac{410.869}{89{,}160}\left(MV_{TOTAL} - MV_{TARE}\right)\left(\text{BELT SPEED}\right)$$

Flow chart:



Program:

```
LBL A
19.5        - M.V. TARE; This constant had to be updated as
              changes were made to the conveyor belt

 -
410.869     - LCF × 1.61
 ×
STOR 3      - Actual weight on belt in pounds
RCL 4       - Desired tons/min
89,160      - 44.58 × 2000
 ×
RCL 3
 ÷
PRT X       - (1) Print required belt speed for a desired tons/min;
              note for an HP-34C this is a R/S statement

RCL 4
PRT X       - (2) Print desired tons/min; note for an HP-34C this
              is a R/S statement

RCL 3
PRT X       - (3) Print weight on belt; note for an HP-34C
              this is a R/S statement

89,160
 ÷
RCL 2       - Actual belt speed
 ×
PRT X       - (4) Print actual tons/min; note for an HP-34C this
              is a R/S statement

STOR 4
RTN
```

Program instruction:

1. Change M.V. TARE if required
2. Actual ft/min (STOR) (2)
3. Desired tons/min (STOR) (4)
4. M.V. total (A)

Program output:

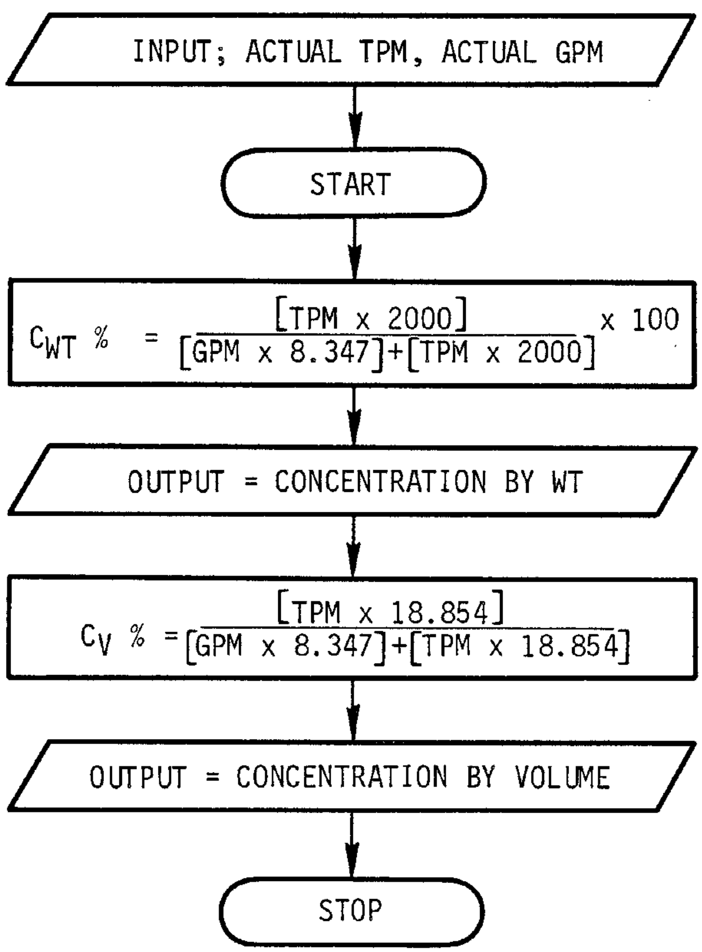
1. Required beltspeed for a desired tons/min
2. Desired tons/min
3. Actual weight on belt
4. Actual tons/min

2. Concentration calculations for HP-95 calculator basic equation:

$$C_{wt}\% = \frac{\left(\frac{\text{tons}}{\text{min}} \times 2000\right)}{\left(\frac{\text{gal}}{\text{min}} \times 8.347\right) + \left(\frac{\text{tons}}{\text{min}} \times 2000\right)} \times 100$$

$$C_{v}\% = \frac{\left(\frac{\text{tons}}{\text{min}} \times 18.854\right)}{\left(\frac{\text{gal/min}}{7.48}\right) + \left(\frac{\text{tons}}{\text{min}} \times 18.854\right)} \times 100 \quad ; \text{ for SG} = 1.7$$

Flow chart:



Program:

```
LBL B
RCL 4       -  Recall actual tons/min
2000
 ×          -  Convert tons/min to lb/min
ENTER ↑
ENTER ↑
RCL 5       -  Recall actual gal/min
8.347
 ×          -  Convert gal/min to lb/min
 +
 ÷          -  C_wt  as a decimal
100
 ×
PRINT X     -  (1) Print  C_wt%, note for HP-34C this is a
               R/S statement
RCL 4       -  Recall actual tons/min
18.854
 ×          -  Convert tons/min to cfm at SG = 1.7
ENTER ↑
ENTER ↑
RCL 5       -  Recall actual gal/min
7.48
 ÷          -  Convert gal/min to cfm
 +
 ÷          -  C_v  as a decimal
STORE 9     -  Store  C_v  decimal for future use
100
 ×
PRINT X     -  (2) Print  C_v%, note for HP-34C this is a
               R/S statement
RTN
```

Program instructions:

1. Actual tons/min (STOR) (4) or from weigh belt program
2. Actual gal/min (STOR) (5)
3. (B)

Program output:

1. Concentration by weight, percent
2. Concentration by volume, percent

3. Line loss calculations for HP-95 calculator basic equation:

$$HL = \Delta P_{1-2} - \Sigma \Delta P_k + \Delta Z_{1-2} \frac{(43.68C_v + 62.4)}{144} \text{ lb/in.}^2$$

at SG = 1.7

$$\frac{HL}{1000} = HL \frac{(1000)}{\Delta L} \quad \frac{\text{lb/in.}^2}{1000 \text{ ft}}$$

$$V_s = \left(\frac{\text{gal/min}}{171.89}\right) + \left(\frac{\text{ton}}{\text{min}} \times 0.820\right) \frac{\text{ft}}{\text{sec}} \quad \text{at 8 in. } \phi \text{ line}$$

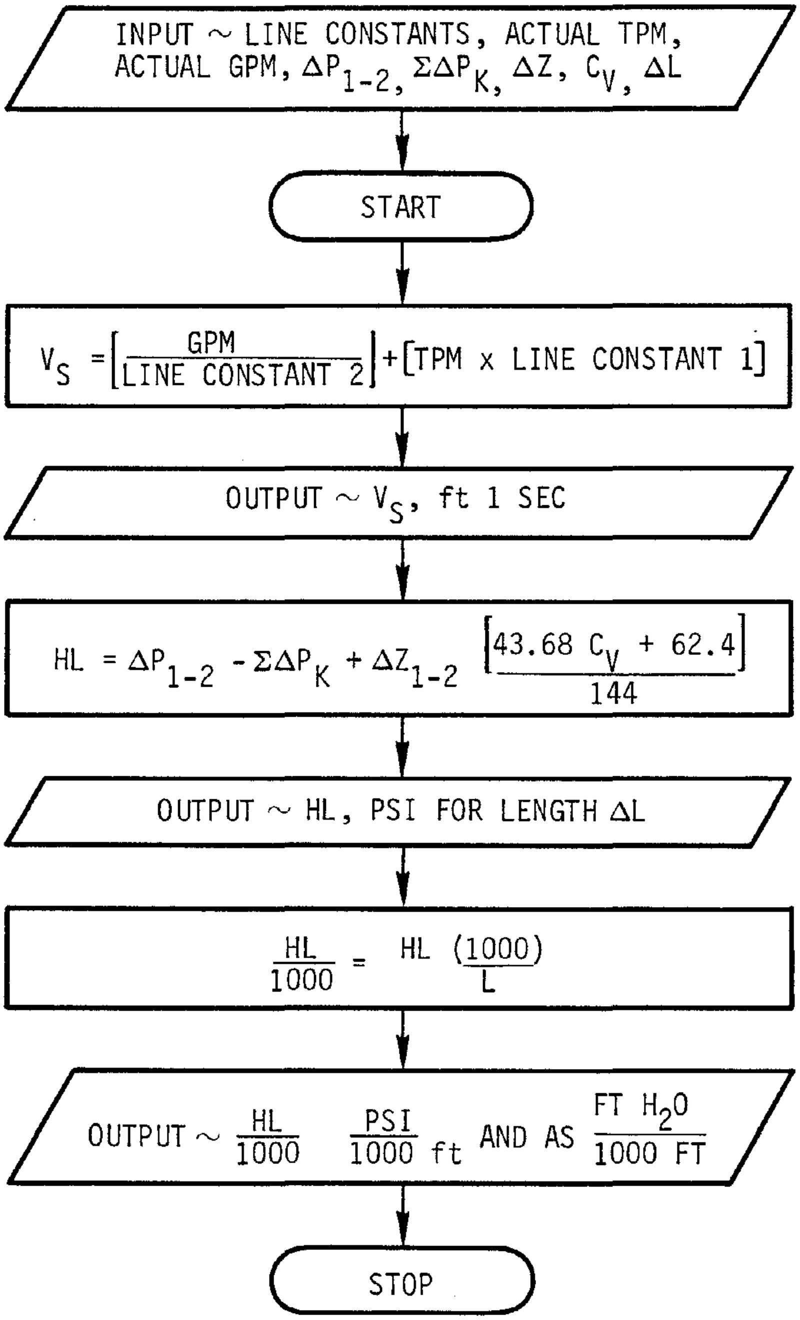
and SG = 1.7

or

$$V_s = \left(\frac{\text{gal/min}}{89.89}\right) + \left(\frac{\text{ton}}{\text{min}} \times 1.568\right) \frac{\text{ft}}{\text{sec}} \quad \text{at 6 in. } \phi \text{ line}$$

and SG = 1.7

Flow Chart:


INPUT ~ LINE CONSTANTS, ACTUAL TPM, ACTUAL GPM, ΔP_{1-2}, $\Sigma \Delta P_K$, ΔZ, C_V, ΔL
START
$V_S = \left[\frac{GPM}{LINE\ CONSTANT\ 2}\right] + [TPM \times LINE\ CONSTANT\ 1]$
OUTPUT ~ V_S, ft 1 SEC
$HL = \Delta P_{1-2} - \Sigma \Delta P_K + \Delta Z_{1-2} \left[\frac{43.68\ C_V + 62.4}{144}\right]$
OUTPUT ~ HL, PSI FOR LENGTH ΔL
$\frac{HL}{1000} = HL\ \left(\frac{1000}{L}\right)$
OUTPUT ~ $\frac{HL}{1000}$ $\frac{PSI}{1000\ ft}$ AND AS $\frac{FT\ H_2O}{1000\ FT}$
STOP

Program:

LBL C	-	For HP-34C must use LBLA or LBLB
RCL 4	-	Recall actual ton/min
Line Constant No. 1	-	0.820 at 8 in. ϕ and SG = 1.7; 1.568 at 6 in. ϕ and SG = 1.7
×	-	Convert ton/min into ft/sec
RCL 5	-	Recall actual gal/min
Line Constant No. 2	-	171.89 at 8 in. ϕ and SG = 1.7; 89.89 at 6 in. ϕ and SG = 1.7
÷	-	Convert gal/min into ft/sec
+	-	
Print ×	-	(1) Print versus in. ft/sec; note for HP-34C this is a R/S statement
RCL 6	-	Recall ΔP_{1-2}
RCL 7	-	Recall $\varepsilon \Delta P_K$
-		
RCL 9	-	Recall C_V (decimal)
0.303	-	(106.08-62.4)/144
×		
0.433	-	62.4/144
+		
RCL 8	-	Recall ΔZ
×		
+		
Print ×	-	(2) Print HL, lb/in.^{2} for length ΔL; note for HP-34C This is a R/S statement
1000	-	
×		
RCL 0	-	Recall ΔL
÷		
Print ×	-	(3) Print HL/1000, lb/in.^{2}/1000 ft; note for HP-34C This is a R/S statement
2.308	-	
×		
Print ×	-	(4) Print HL/1000, ft-H_2O/1000 ft; note for HP-34C This is a R/S statement
RTN		

Program instruction:

1. Enter line constants 1 and 2
2. Actual ton/min, store 4 or from weigh belt program
3. Actual gal/min, store 5 or from concentration program

4. ΔP_{1-2} , store 6
5. $\varepsilon \Delta P_K$, store 7
6. ΔZ , store 8
7. C_V (decimal), store 9 or from concentration program
8. ΔL , store 0
9. [C]

Program Output:

1. Slurry velocity, ft/sec
2. Line loss, $lbs/in.^2$ for length ΔL
3. Line loss, $lbs/in.^2/1000$ ft
4. Line loss, ft-H_2O/1000 ft

4. Inducer efficiency for HP-34C Calculator

Basic equation:

$$\eta = \frac{[Q_s \ \gamma_s \ H_s]_{out}}{[\text{Motor hp} \times 0.898]_{in}}$$

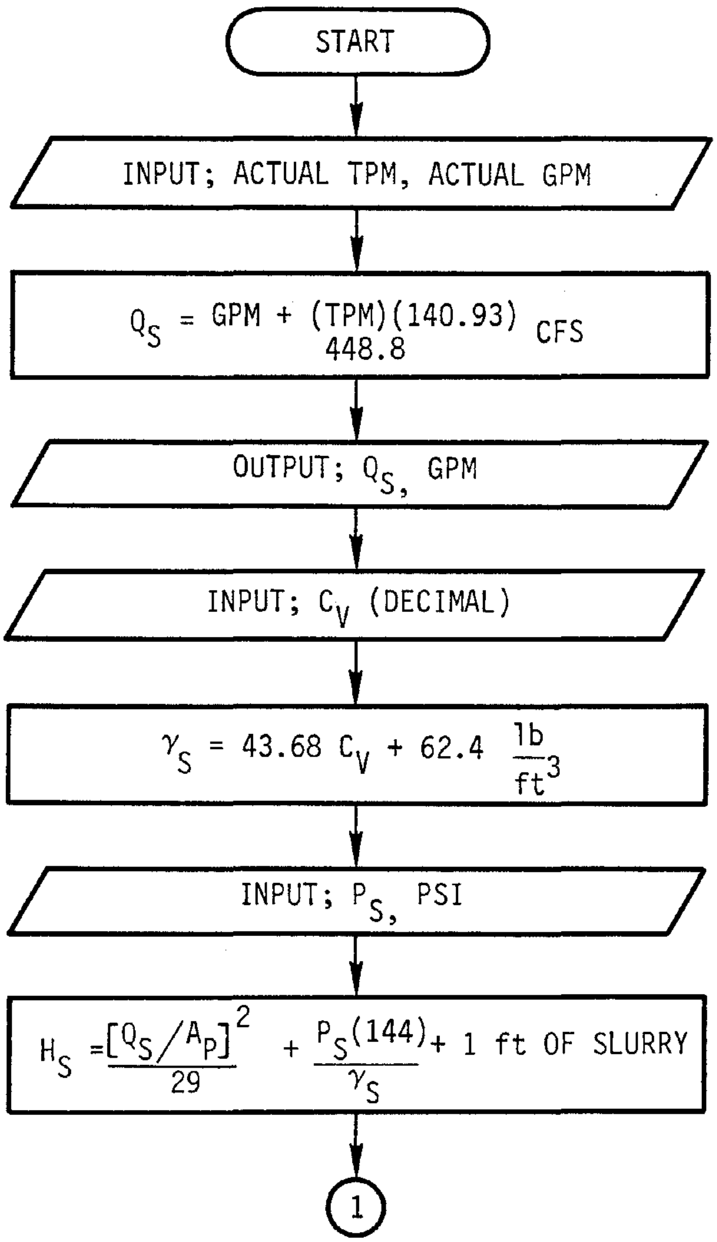
$$Q_s = \left[\frac{\text{gal/min}}{448.8}\right] + \text{ton/min}[0.314] \ \text{ft}^3\text{/sec at SG} = 1.7$$

$$\gamma_s = 43.68 \ C_V + 62.4 \ \frac{\text{lb}}{\text{ft}^3} \ \text{at SG} = 1.7$$

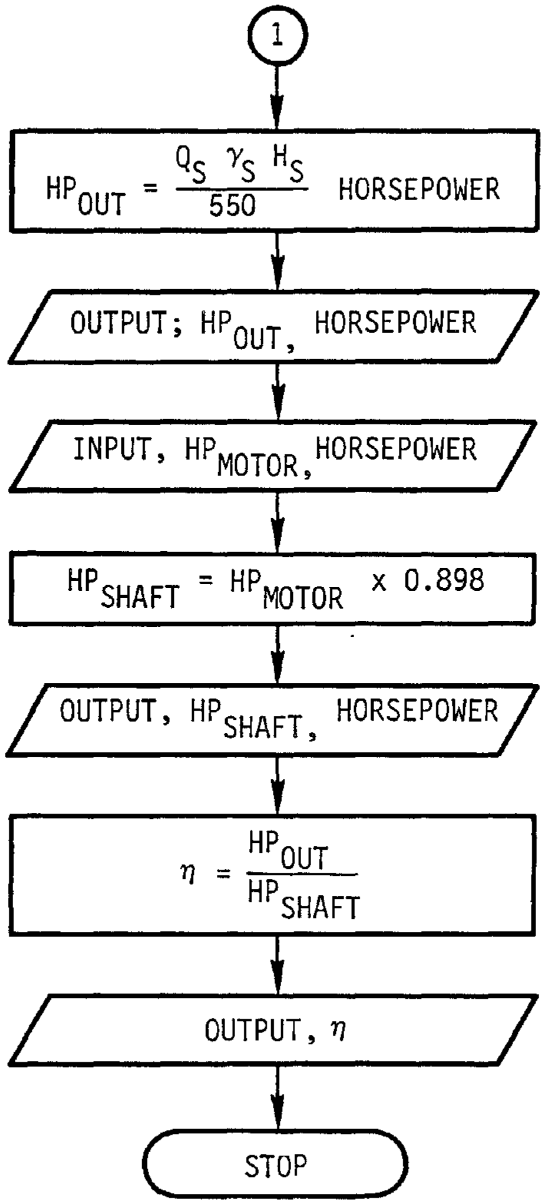
$$C_V = \frac{[\text{ton/min} \times 18.854]}{\left[\frac{\text{gal/min}}{7.48}\right] + [\text{ton/min} \times 18.854]} \quad \text{at SG} = 1.7$$

$$H_s = \frac{\left[\frac{Q_s}{A_p}\right]^2}{2g} + \frac{P_s(144)}{\gamma_s} + 1 \ \frac{\text{ft-lb}}{\text{lb}}, \text{ in ft of slurry}$$

Flow Chart:


START
INPUT; ACTUAL TPM, ACTUAL GPM
QS = GPM + (TPM)(140.93) / 448.8 CFS
OUTPUT; QS, GPM
INPUT; CV (DECIMAL)
γS = 43.68 CV + 62.4 lb/ft3
INPUT; PS, PSI
HS = [QS/AP]2 / 29 + PS(144) / γS + 1 ft OF SLURRY
1

Flow chart:


1
HP OUT = Qs γs Hs / 550 HORSEPOWER
OUTPUT; HP OUT, HORSEPOWER
INPUT, HP MOTOR, HORSEPOWER
HP SHAFT = HP MOTOR x 0.898
OUTPUT, HP SHAFT, HORSEPOWER
η = HP OUT / HP SHAFT
OUTPUT, η
STOP

Program:

Note that this program may be used in both a HP-34C and HP-95 Calculator

```
LBL A
1
R/S         -  Input gal/min, [enter↑] ton/min; the number 1
               will be displayed before input
140.93
 ×
 +
448.8
 ÷
Stor 1      -  Store Qs, f³/sec, in Register 1
448.8
 ×
R/S         -  (1) Output Qs, gal/min
CLX
2
R/S         -  Input Cv; the number 2 will be displayed
               before input
43.68
 ×
62.4
 +
Stor 2      -  Store γs, lb/ft³, in Register 2
3
R/S         -  Input Ps, lb/min; the number 3 will be
               displayed before input
144
 ×
RCL 2       -  Recall γs
 ÷
1
 +
RCL 1       -  Recall Qs
Area        -  Ap = 0.3477 ft² at 8 in. SCH 40
               Ap = 0.2003 ft² at 6 in. SCH 40
 ÷
x²
64.4
 ÷
 +
Stor 3      -  Store Hs, ft of slurry, in Register 3
RCL 1       -  RCL Qs, ft³/sec
 ×
RCL 2       -  RCL γs, lb/ft³
 ×
```

```
500
Stor 4          Store hp_out, horsepower, in Register 4
R/S             (2) Output, hp_out, horsepower
4
R/S             Input, hp_motor, horsepower, the number 4
                will be displayed before input
0.898
 ×
R/S             (3) Output, hp_shaft, horsepower
RCL 4           Recall hp_out, horsepower
x<>y
 ÷
RTN             (4) Output, η, decimal
```

Program instructions:

1. Change area constant in program if required
2. [A]
3. Program will stop and display "1"
   Input gal/min
   [enter 4]
   ton/min
   [R/S]
4. Program will stop and (1) output Q_s, gal/min
   [R/S]
5. Program will stop and display "2"
   Input C_v
   [R/S]
6. Program will stop and display "3"
   Input P_s
   [R/S]
7. Program will stop and (2) Output hp_{out}, horsepower
   [R/S]
8. Program will stop and display "4"
   Input hp_{motor}
   [R/S]
9. Program will stop and (3) Output hp_{shaft}, horsepower
   [R/S]
10. Program will stop and (4) Output η

Summary: Input	Output
1. gal/min	1. Q_s, gal/min
2. ton/min	2. hp_{out}, horsepower
3. C_v	3. hp_{shaft}, horsepower
4. lb/in.^{2}	4. η
5. hp_{motor}	
6. A_p	

5. Gradation calculation for HP-34C

5A. Sieve weights

This program simply converts lbs-oz weights to lb weights; therefore no flow chart is given.

```
LBL B
Enter ↑
Int
x<>y
Frac
0.16
x<>y
 +           lb + oz to lb decimal
RCL 1        Old running total
 +
Stor 1       New running total
RTN          Output new running total, lbs decimal
```

Instruction:

1. For new sample set, zero Register 1
2. Input lbs oz i.e. 16.1475 => 16#14-3/4 oz
3. [B]
4. Output lbs i.e. 16.9219#

5B. Drag coefficients

This program simply solves the following equation; therefore no flow chart is given.

$$C_D = \left[\frac{4}{3}\right] \frac{[9.81]}{W^2} \left[\frac{d_s}{1000}\right] [SG-1]$$

Program:

```
LBL A
4
Enter ↑
3
 ÷
1000
 ÷
9.81
 ×
RCL 1        - Recall d_s, mm
 ×
```

```
RCL 2     -  Recall W, m/sec
 x²
 ÷
RCL 3     -  Recall SG
 1
 -
 x
RTN       -  Output Cd
```

Instructions:

1. Input d_s, mm, Store Register 1
   W, m/sec, Store Register 2
   SG, Store Register 3
2. (A)
3. Output C_d

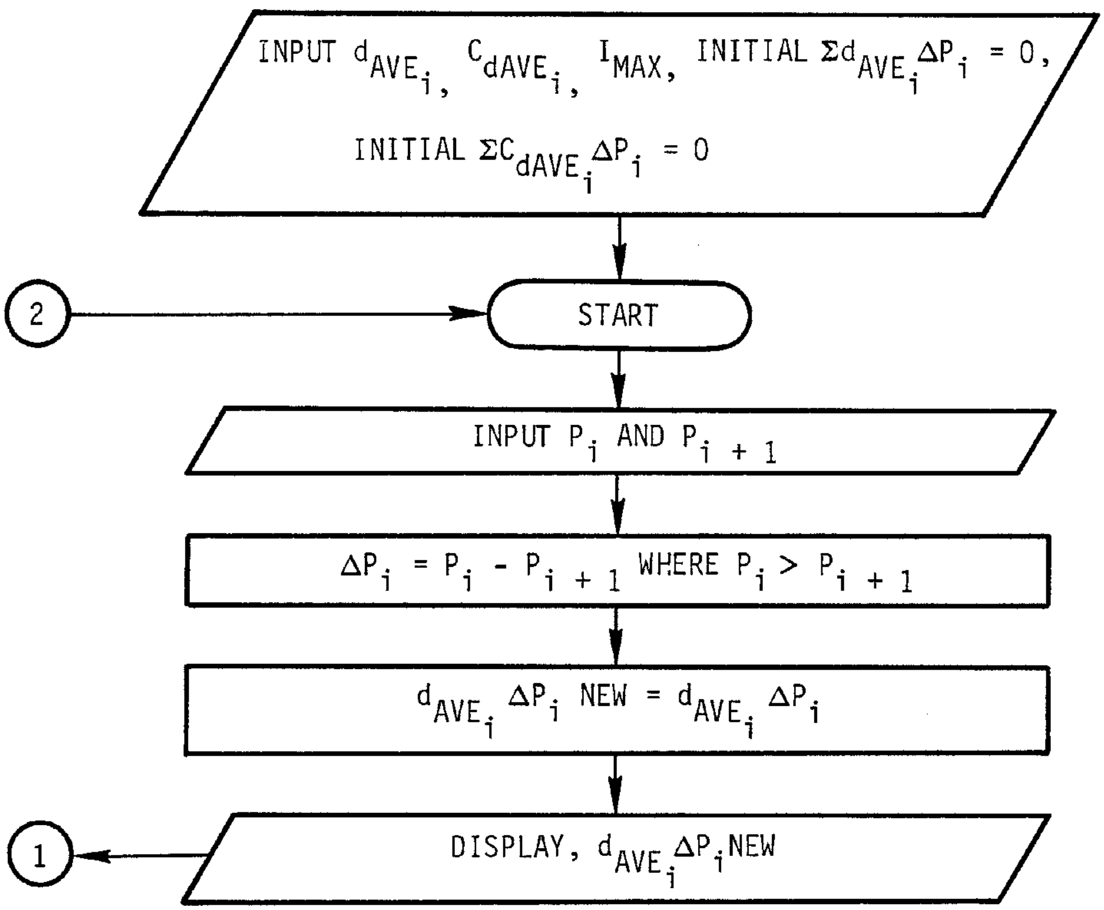
5C. Mean particle size and mean drag coefficient for a given material sample

Basic equation:

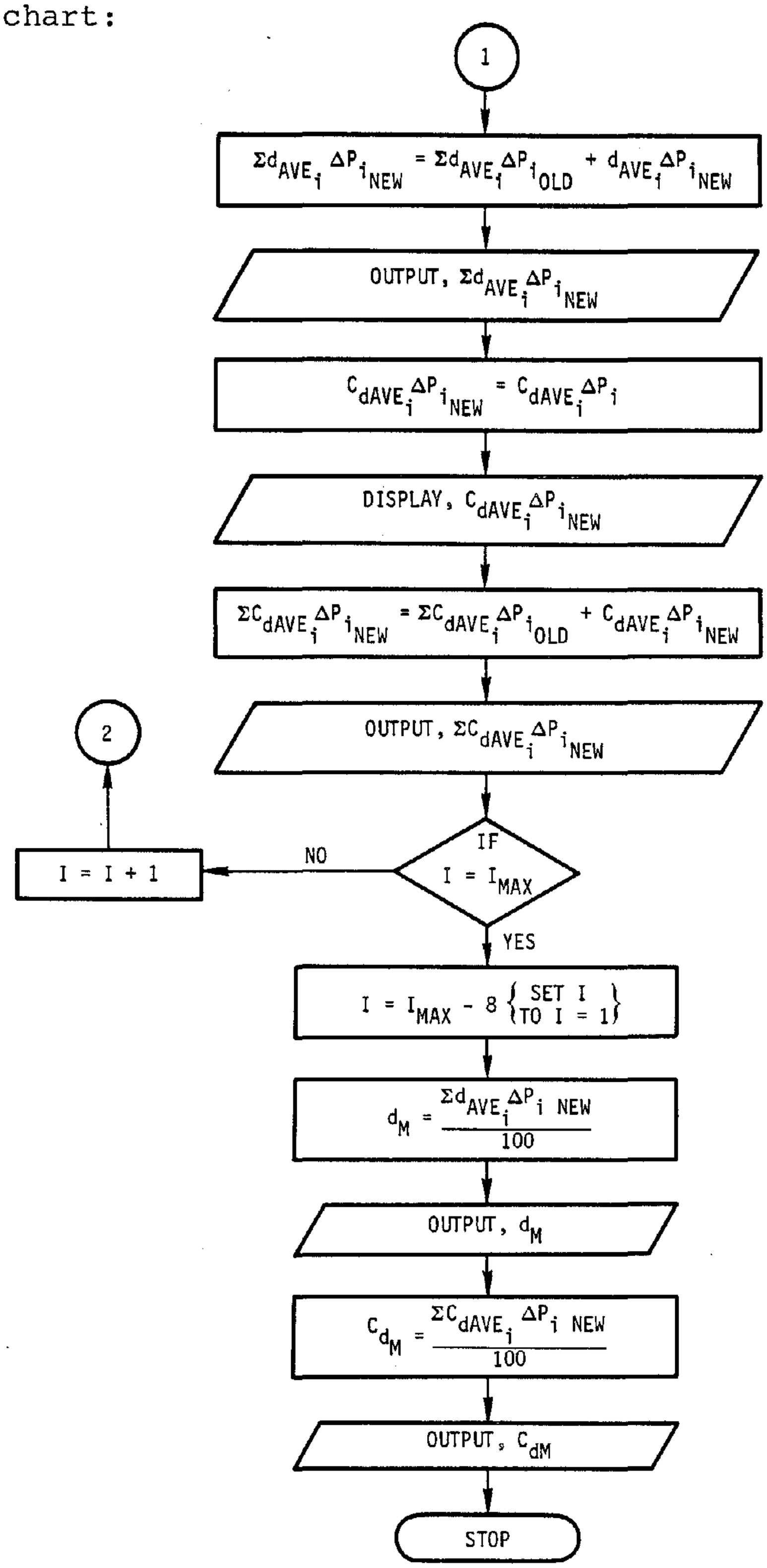
$$d_m = \sum_{i=1}^{N} \frac{d_{AVE_i} \, \Delta P_i}{100}$$

$$C_{dm} = \sum_{i=1}^{N} \frac{C_{dm_i} \, \Delta P_i}{100}$$

Flow Chart:


INPUT d_{AVE_i}, C_{dAVE_i}, I_{MAX}, INITIAL $\Sigma d_{AVE_i} \Delta P_i = 0$,
INITIAL $\Sigma C_{dAVE_i} \Delta P_i = 0$
2
START
INPUT P_i AND P_{i+1}
$\Delta P_i = P_i - P_{i+1}$ WHERE $P_i > P_{i+1}$
$d_{AVE_i} \Delta P_i$ NEW = $d_{AVE_i} \Delta P_i$
DISPLAY, $d_{AVE_i} \Delta P_i$NEW
1

Flow chart:


1
ΣdAVEi ΔPi NEW = ΣdAVEi ΔPi OLD + dAVEi ΔPi NEW
OUTPUT, ΣdAVEi ΔPi NEW
CdAVEi ΔPi NEW = CdAVEi ΔPi
DISPLAY, CdAVEi ΔPi NEW
ΣCdAVEi ΔPi NEW = ΣCdAVEi ΔPi OLD + CdAVEi ΔPi NEW
OUTPUT, ΣCdAVEi ΔPi NEW
IF I = IMAX
NO
I = I + 1
2
YES
I = IMAX - 8 {SET I TO I = 1}
dM = ΣdAVEi ΔPi NEW / 100
OUTPUT, dM
CdM = ΣCdAVEi ΔPi NEW / 100
OUTPUT, CdM
STOP

5C. Mean particle size and mean drag coefficient for a given material sample.

Program

LBL A		
R/S	-	Input P_i
ENTER		
R/S	-	Input P_{i+1}
-		
STOR 9	-	Store ΔP_i in Register 9
RCL(i)	-	d_{AVE_i}
×		
(3) PAUSE	-	(1) Display d_{AVE_i} ΔP_i new; program pauses 3 sec
RCLO	-	Recall Σd_{AVE_i} ΔP_i old
+		
STOR 0	-	Store Σd_{AVE_i} ΔP_i new
RIS	-	(2) Output $\Sigma d_{AVE_i} \Delta P_1$ new
RCL I		
10		Increment I to call up Σ_d values
+		
STOR I		
RCL 9	-	Recall ΔP_i
RCL(i)	-	Recall $C_{d_{AVE_i}}$
×		
(3) PAUSE	-	(3) Display $C_{d_{AVE_i}}$ ΔP_i new, program pauses 3 sec
RCL.0	-	Recall $\Sigma C_{d_{AVE_i}}$ ΔP_i old
+		
STOR.0	-	Store $\Sigma C_{d_{AVE_i}}$ ΔP_i new
R/S	-	(4) Output $\Sigma C_{d_{AVE_i}}$ ΔP_i new

```
RCL I
10
 -
1                  Increment I by 1 to call up d_AVE values
 +
STOR I
RCL.9
X = y
Yes GO TO B  No    Check loop increment limit
RCLI
RTN                (5) Output I
LBL B
RCL I
8                  Set I to 1
 -
STOR I
RCL 0
100
 ÷
R/S              - (5) Output d_m

RCL.0
100
 ÷
RTN              - (6) Output C_dm
```

Instructions:

1. Input data

Data		Storage Register
d_{AVE_i}	-	1, 2, 3, 4, 5, 6, 7, 8
$C_{d_{AVE_i}}$		.1, .2, .3, .4, .5, .6, .7, .8

$I_{MAX}=9$ .9

$\Sigma C_d \Delta P=0$ to start - .0

$\Sigma d_{AVE} \Delta P=0$ to start - 0

2. (A)
3. Input P_i at first program stop
4. (R/S)
5. Input P_{i+1} at second program stop
6. (R/S)
7. (1) Display $d_{AVE} \Delta P_i$ at first pause
8. (2) Output $\Sigma d_{AVE} \Delta P_i$ at third program stop
9. (R/S)
10. (3) Display $C_{dAVE} \Delta P_i$ at second pause
11. (4) Output $\Sigma C_{dAVE} \Delta P_i$ at fourth program stop
12. (R/S)

13.A (5) Output I. If $I < I_{max}$ at fifth program slop

14.A (A) start a step to and continue as before

13.B (5) Output d_m If $I=I_{max}$ at fifth program stop

14.B (R/S)

15. (6) Output C_{dm} at sixth program stop
16. Start at step 1 for new data set

5. Inducer efficiency

For coal SG=1.7

$$Q_s = Q_w + Q_{coal}$$

or $$Q_s = \frac{gal/min}{448.8} + ton/min\ (0.314)\ CFS$$

$$\gamma_s = \gamma_\omega(C_v[SG - 1] + 1)$$

or $$\gamma_s = 43.68C_v + 62.4 \quad 16/ft^3$$

$$H_s = \frac{V_s^2}{2g} + \frac{P_s}{\gamma_s} + Z \quad \frac{ft\text{-}lb}{lb}$$

For $Z = 1.0$ ft

P_s = psig

V_s = in ft/sec

Also $C_v = Q_{coal}/Q_{slurry}$

$$C_v = \frac{\text{ton/min} \times 18.854}{\frac{\text{gal/min}}{7.48} + \text{ton/min} \times 18.854}$$

Then

$$H_s = \frac{Q_s/A_p^{\ 2}}{2g} + \frac{P_s(144)}{\gamma_s} + 1$$

A_p - for 8 in. schedule 40 = 0.3474 ft^2

A_p - for 6 in. schedule 40 = 0.2003 ft^2

The measured quantities

Q_{water} in gal/min

Q_{coal} in ton/min

P_s in psig

HP_{motor} in watts but calibrated to output HP

Assumed Quantities

η motor

η belt

Determined Quantities

C_v

η inducer

Results are summarized on data sheets.