PB86159464

A mining research contract report
MAY 1985

A STUDY OF DESIGN AND DEVELOPMENT OF A COAL INJECTOR FOR COARSE SLURRY TRANSPORT

Contract J0333920
Ingersoll-Rand Research Inc.
Princeton, N.J.

Bureau of Mines Open File Report 133-85

BUREAU OF MINES
UNITED STATES DEPARTMENT OF THE INTERIOR

NOTICE

This report was completed under a contract to the U. S. Department of Energy (DOE). Responsibility for the program to which the report relates has recently been transferred to the Bureau of Mines, U. S. Department of the Interior, and the report is made public under a Bureau cover. Inquiries concerning the report should be directed to the Bureau.

50272-101

REPORT DOCUMENTATION PAGE	1. REPORT NO. BuMines OFR 133-85	2.	3. Recipient's Accession No. PB86 159464/AS
4. Title and Subtitle A Study of Design and Development of a Coal Injector for Coarse Slurry Transport			5. Report Date May 1985
			6.
7. Author(s) Boyd Wise, Sam Bell, and Dennis Thompson			8. Performing Organization Rept. No.
9. Performing Organization Name and Address Ingersoll-Rand Research, Inc. P.O. Box 301 Princeton, NJ 08542			10. Project/Task/Work Unit No.
			11. Contract(C) or Grant(G) No. (C) J0333920 (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, 6-30-75--3-15-85
			14.

15. Supplementary Notes

Approved for release December 19, 1985.

16. Abstract (Limit: 200 words)

A jet pump injector was developed and successfully tested, thereby demonstrating the feasibility of a jet injector for introducing coarse coal into an operating slurry face haulage system. In phase I, a variety of injection concepts were evaluated, and the jet pump was identified as best, owing to its simplicity, compactness, and performance potential. In phase IIA, subscale jet pump injector tests were conducted, and a computer model of a jet pump injector was developed whose performance predictions correlated well with subscale test results. In phase IIB, various system configurations were defined for integrating a jet pump injector into a fully operational underground coarse slurry face haulage system, further highlighting the feasibility and practicality of such a system. In phase III, a full-scale mobile haulage vehicle embodying a jet pump injector and coal sizing equipment was detail designed into a practical vehicle with attractive performance characteristics. More importantly, fully operational full-scale jet pump injector and feeder-breaker subassemblies were tested and proven successful in their ability to down-size coal to 3 inches in all three dimensions and to consistantly provide slurry of 42% weight concentration at 6.4 t/min throughout under a wide range of operating conditions including zero to maximum feed rate within 5 s.

17. Document Analysis a. Descriptors

Mining research
- Hydraulic transport
- Slurry pipelining
- Coal slurries
- Pipeline injection
- Solids injection
- Slurry transport
- Jet pump
- Face haulage

b. Identifiers/Open-Ended Terms

c. COSATI Field/Group

18. Availability Statement	19. Security Class (This Report)	21. No. of Pages
Release unlimited by NTIS.	Unclassified	359
	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 Ingersoll-Rand Research, Inc. Princeton, N.J. as an account of the work performed under USBM and DOE Contracts, Numbers H0155097, DE-AC-01-75ET12197, and J0333920. The work was initiated under the Coal Mine Health and Safety Program and administered under the technical direction of the Bureau with Mr. A. J. Miscoe serving as the Technical Project Officer throughout the program. Mr. H. Eveland was the initial Contract Administrator for the Bureau of Mines, Mr. E. F. Callaghan was the Contract Administrator for the Department of Energy and Mr. W. R. Mundorf was the final Contract Administrator for the Bureau of Mines.

This report is a summary of the work performed under Phases I, IIA, IIB and III of the program, as prescribed by the aforementioned contracts, during the period from June 30, 1975 to March 15, 1985.

Phases I and II activities related to the literature search, concept development, subscale model testing and analytical modelling of the jet injector were performed by D. Mistry, K. D. Paul, A. N. Saad, R. G. Malsbury and J. L. Dussourd. The Phase II work, involving the design and subassembly testing of a full scale mobile jet pump injector vehicle, was performed by B. Wise, Program Manager, supported by C. Reed, R. Dickol, B. Clouse, D. Hentosh and R. O'Brien. Significant additional Phase II contributions to the program were made by outside individuals. Mr. R. Cardenas of Foster Miller Inc. operated that company's surface test facility where the full scale jet pump subassembly was performance tested. Mr. D. Clonch of the S&S Corporation, a subsidiary of Ingersoll-Rand Company, was responsbile for the design and test evaluation of the Feeder-Breaker subassembly at that corporation's facility.

The material presented in this final report is a composite summary of the writings of these individuals which contains no patentable material.

TABLE OF CONTENTS

Page No.

Page No

APPENDICES:

A. Bibliography

B. Subscale Concentrator Test Data, Phase IIB

C. Full-Scale Jet Pump Subassembly Test Data, Phase III

D. Full Scale Breaker Subassembly Test Data, Phase III

E. Subscale Centrifugal Concentrator Test Data, Phase III

F. Coal Degradation Test, Phase III

LIST OF FIGURES

List of Figures - Continued

List of Figures - Continued

List of Figures - Continued

List of Figures - Continued

List of Figures - Continued

List of Figures - Continued

LIST OF TABLES

1.0 EXECUTIVE SUMMARY

1.1 Phase I - Conceptual Studies

The hydraulic transport of coal was studied as a potential means for safer haulage and higher productivity. One important development required to assure the system's usefulness in underground coal mining was the availability of a coal feeder (or injection mechanism) that could inject run-of-mine coal directly into an operating hydraulic pipeline. To be truly effective and to provide a necessary breakthrough, the feeder had to be low in profile, high in capacity, minimize breakage during feeding and accept coal feed directly from the continuous miner.

Ingersoll-Rand Research, Inc. (IRRI) received a contract from the Bureau of Mines whose objective was to conceive and develop such an injection device. Phase I - Conceptual Studies, was for the purpose of selecting and defining a suitable concept. The specification involved an injector capable of operating in a 4-foot coal seam and delivering up to 12 tons per minute of coal comprised of 3" maximum lump size coal.

A survey was conducted that was comprised of a patent search, a literature review and a literature screening of selected manufacturers to establish the state-of-the-art with respect to solid material feeding devices. The complete results of this activity were reported in the Phase-ending Report for Phase I.

The concepts arising from the survey, together with concepts conceived by IRRI, were screened resulting in the selection of four injector concepts for detailed evaluation. Only the four selected concepts are described in detail in the present report. These concepts were objectively compared with the design criteria specified by contract. The concepts were evaluated by Group Delphi and Point Scoring Method. Of the several candidate injection schemes screened, the jet pump injector concept was selected for further study on the basis of performance, size, weight, simplicity, durability and ease of maintenance.

In addition, face haulage systems incorporating the various injector concepts were defined and analyzed. A preferred system was identified and recommended for future development work. This system was considered to offer significant improvements in productivity and safety.

1.2 Phase IIA - Design and Engineering

The objective of Phase II was to demonstrate the feasibility and operability of a subscale model of the injector. The specification for this model was to inject 1 TPM of coarse coal into a 25 PSIG haulage line using 250 GPM of 125 PSIG water.

Both physical and analytical models were studied during this phase. Tests were run on gravity fed and screw fed injectors using both coal and gravel. The coal consisted of 3/4" top size with a median size of 0.36". The gravel had a 7/16" top size and 0.270" median size. All the basic jet pump parameters were experimentally varied permitting definition of the optimum jet pump for a broad variety of solids pumping applications. Good correlation was achieved between the analytical model results and the actual test results from the subscale jet pump injector work.

The overall performance and behavior offered by this type of injector was considered very attractive and practical for face haulage applications. At a static discharge pressure of 25 psig the maximum rates were 0.70 and 0.55 TPM for gravel and coal respectively. However, at 23 psig, these rates increased to 0.85 for gravel and 0.6 TPM for coal. The injector involved no rotating parts, was found to be simple, compact, rugged, and to have low maintenance costs coupled with high reliability and long life. It could be controlled over the complete flow range and was easily adapted to the various operational modes required.

A control system was demonstrated which allowed for easy slurry line velocity control under all loading conditions, including step load changes from full to zero. Under unloaded conditions the injector's pressure-developing capability and flow capacity was such as to cause an increase in line velocity which produced a flushing action in the haulage line. This feature added stability to the system and would be a significant benefit if one or more injectors were to share the same booster pump.

A complete hydraulic haulage system will normally involve slurry booster pumps -- usually of the centrifugal type for a coarse feed slurry. The performance of this injector can be matched with a centrifugal booster without the complications of speed control and the associated line velocity sensing control.

The simplicity and compactness of the jet pump injector concept would permit many configuration possibilities in a face-hydraulic haulage application. It is small enough that it could be mounted directly on the back of a continuous miner or a shuttle car. It has a low profile, making it suitable for low roof coal. It is even possible to mount the injector on a hose carrying vehicle. The natural limitations of the haulage system, such as the lack of flexibility or maneuverability of the hose, produce a far greater impact on how this injector can be used than the constraints of the injector itself.

Consequently, it was recommended that a study should be made to determine the most cost effective application of this injector including how the hose system should be handled, where the injector should be mounted, and what form, if any, the surge capacity should take.

Considering the initial development and introduction of this face haulage system, a recommendation was made that a feasibility study be conducted for dewatering the slurry underground. The objective would be to reduce the risk and capital cost of the initial demonstrations by limiting hydraulic haulage to only the distance between the face and the conventional panel belt.

1.3 Phase IIB - Hydraulic Face Haulage System and Concentrator Development

The objectives of the Phase IIB program were to:

- Develop a hydraulic face haulage system concept utilizing a jet pump injector.

- Obtain experimental frictional head loss data for coiled slurry hose, a feature envisioned for some haulage concepts.

- Design, build and test a coarse coal slurry concentrator that could be placed in the slurry discharge line downstream of the jet pump injector to increase the jet pump discharge slurry coal weight concentration from 32% to 50%.

A monorail suspended hose system with a small, compact breaker injector chassis was chosen as the most promising concept. The main components of this hydraulic haulage system concept, depicted in Figure EX-1, are:

- Injector and Feeder Breaker Vehicle

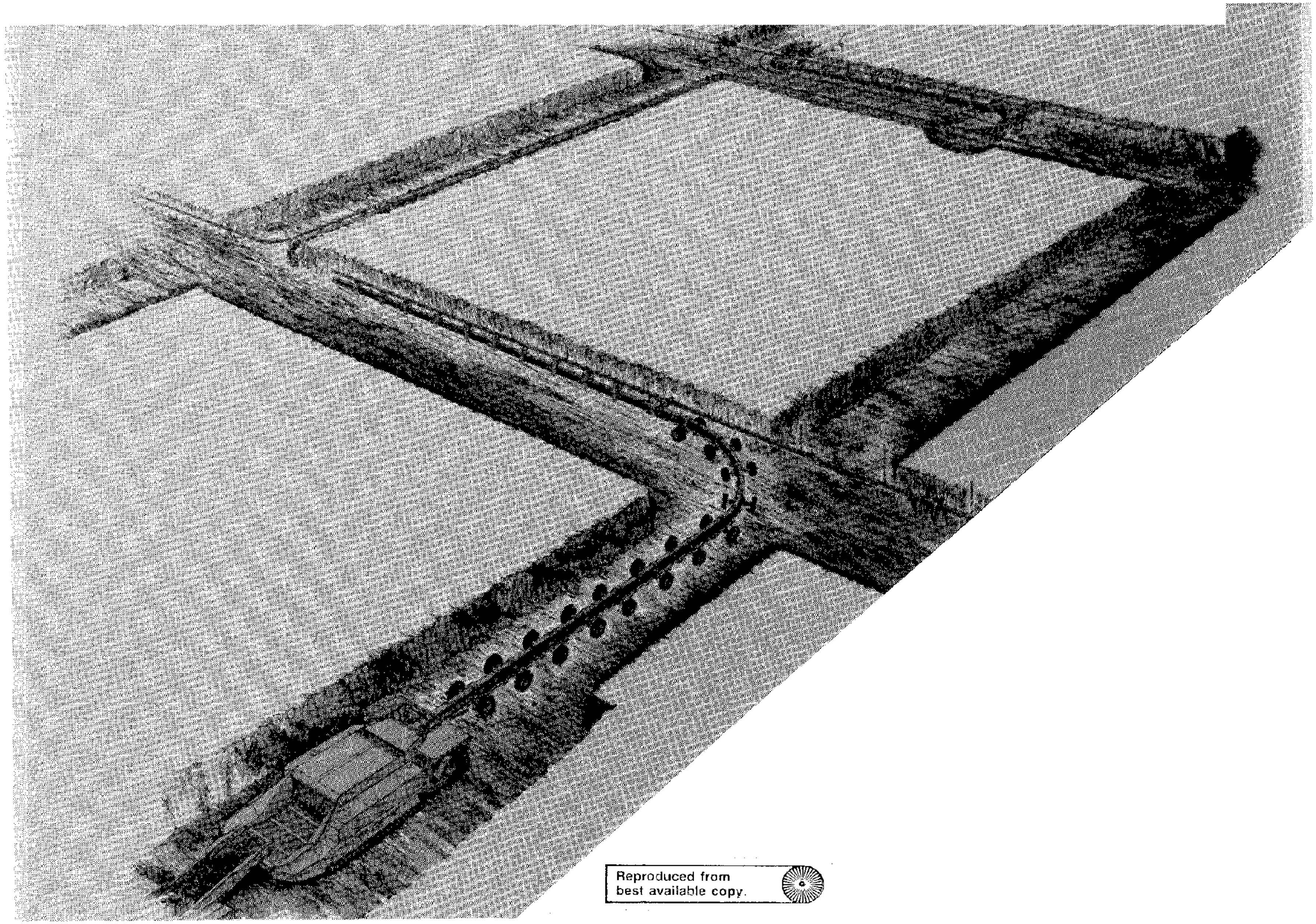


Figure EX-1: Artist's Rendition of Coarse Coal
Injector Hydraulic Haulage System

-Ground Transition Hose Train
-Monorail Suspended Hose Train
-Hose Loop Carriage

This system has the capability of mining a five-entry advance system, working in a 4-foot seam height, and tracking a continuous miner without hindering its movements. An option of such a system would be to increase the manueverability of the miner by supporting the continuous miner's power cables from both the injector chassis and the suspended hose train.

The hose train is retracted or extended by means of a looped length of hose in a loop forming carriage which is mounted in a back crosscut, one entry removed from the working cross cut. As the injector vehicle advances or retracts, the loop carriage traverses back and forth in the crosscut playing out or retracting the hose. The jet pump injector will generate sufficient pressure to convey slurry approximately 800 feet in a straight line. The pressure losses associated with the multiple bends and turns reduces this capacity to 640 feet. A hydraulic hose train suitable for a 5 entry system with 80 foot centers on entries and boxcuts requires only 320 feet of total hose length. There is, therefore, a substantial cushion in the pressure capability of the jet pump injector in this application. In Figure EX-2, a booster pump is shown at the discharge end of the flexible hose system to increase the slurry pressure and convey it out of the mine.

The design of a Face Hydraulic Haulage System is greatly determined by the choice of hose handling equipment behind the injector. Two concepts were investigated which differed substantially in their method of handling the flexible hose portion of the Haulage System. The Monorail System is described above and is the preferred system. The other concept utilized a large pair of coiled slurry water hoses mounted on the injector chassis and is not described in this report.

The third objective of this program was to develop a concentrating device for the coarse coal slurry as received from a jet pump injector. The linear concentrator that was developed for this purpose demonstrated a capability for receiving a coarse coal slurry at 32% weight concentration and removing sufficient water such as to create a 40% to 60% coal concentration by weight at the discharge of the concentrator. Fines carried out with the removed water were approximately 2-3% of the total solids flow through the slurry, insufficient to create an operational problem. Discussions with some coal preparation plant operators revealed a higher probability of clogging at slurry weight concentrations greater than 30%.

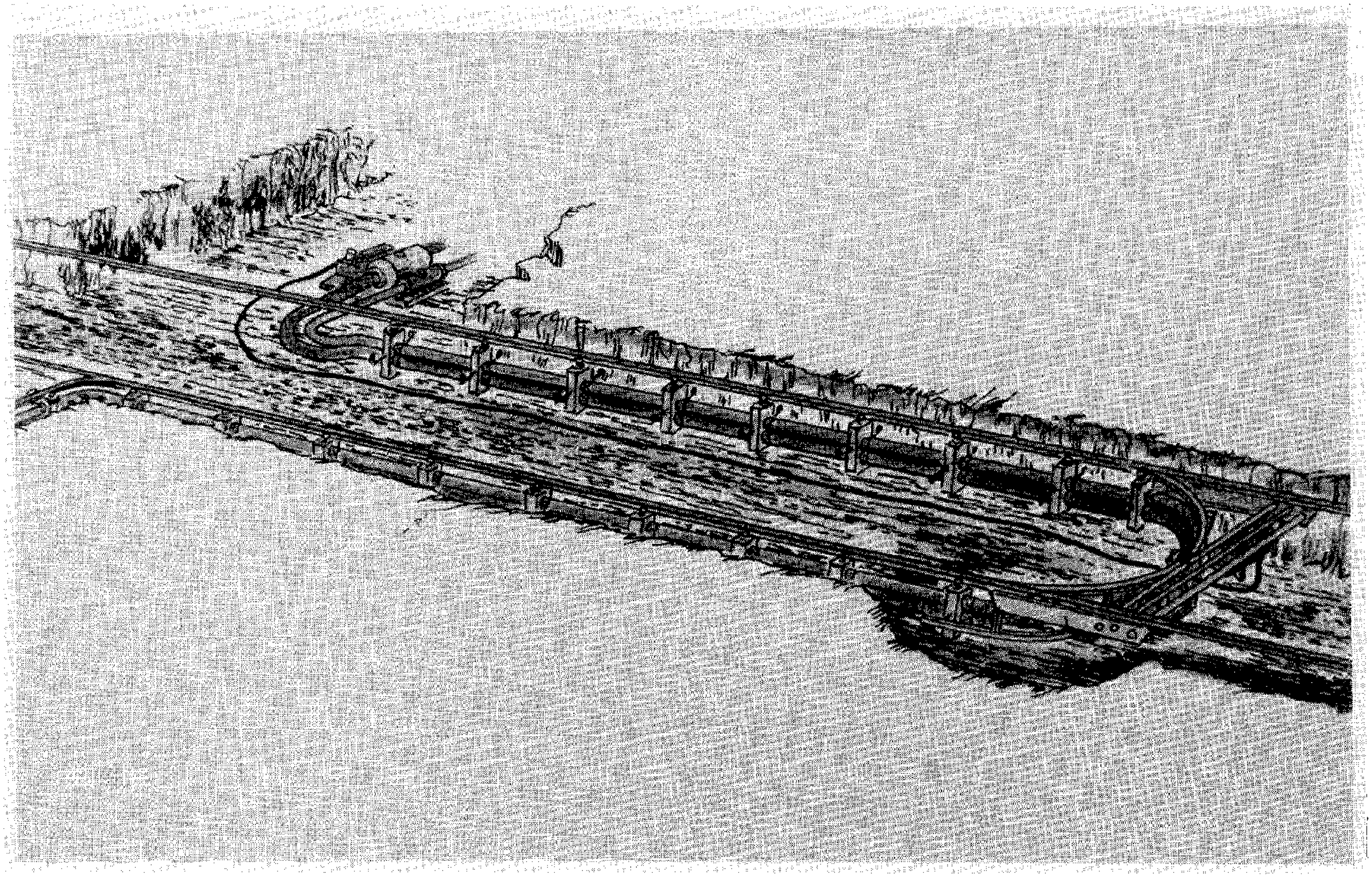


Figure EX-2: Artist's Rendition of the Downstream Flexible Hose System and Booster Pump

It was therefore the purpose of this concentrator research project to extend those present limitations and make pipeline transport more efficient and thereby more economical by enabling a greater throughput. Since test data showed that a pump system operated up to 60% concentration without mention of clogging, perhaps current limitations lie with other factors such as poor size control or unsteady flow rates.

1.4 Phase III - Mobile Jet Injector Development

In this phase, the engineering design of a full-scale, mobile jet injector vehicle was completed. The resulting vehicle is depicted in an artist's perspective rendition in Figure EX-3. Plan and side elevation views of the vehicle, showing the run-of-mine (ROM) coal being processed by the various, on-board subsystems and finally entering the slurry hydraulic haulage line, are depicted in Figure EX-4.

This tracked vehicle concept is capable of receiving ROM coal, sizing this coal to the desired upper limit (through breakers), injecting the coal into a pressurized slurry haulage line (by means of a hydraulic jet pump) and removing excess water from the injected slurry (in a centrifugal concentrator) to achieve the desired slurry concentration in the haulage line. The breakers produce 3" top sized coal for ultimate injection into an 8" slurry line.

Originally, a complete full-scale mobile jet injector vehicle was to have been fabricated and test evaluated underground. Program scope was modified, however, limiting development testing to two of the vehicle's critical subassemblies - namely, the jet pump injector itself and the two-stage coal breaker.

Design of the full-scale jet pump injector was based on analytical modelling and subscale (.55 TPM) testing performed in the earlier phases of the program.

The full-scale jet pump injector subsystem demonstrated a capability for injecting 6.4 TPM (384 TPH) of sized coal at concentration levels up to 43% (by weight). The injector was able to maintain substantially constant slurry velocity when solids input was varied between 0 and 350 TPH in a period of 5 seconds.

The feeder-breaker subsystem, also tested at full-scale, was based on double-roll breaker technology developed by the S&S Corporation, a subsidiary of Ingersoll-Rand Company. This work demonstrated that a small, compact, sizing-breaker subsystem, meeting assigned vehicle space limitations, could be designed to have a throughput of at least 6.4 TPM.

Drawings of the jet pump injector are on file at the Bureau of Mines' Pittsburg Research Center.

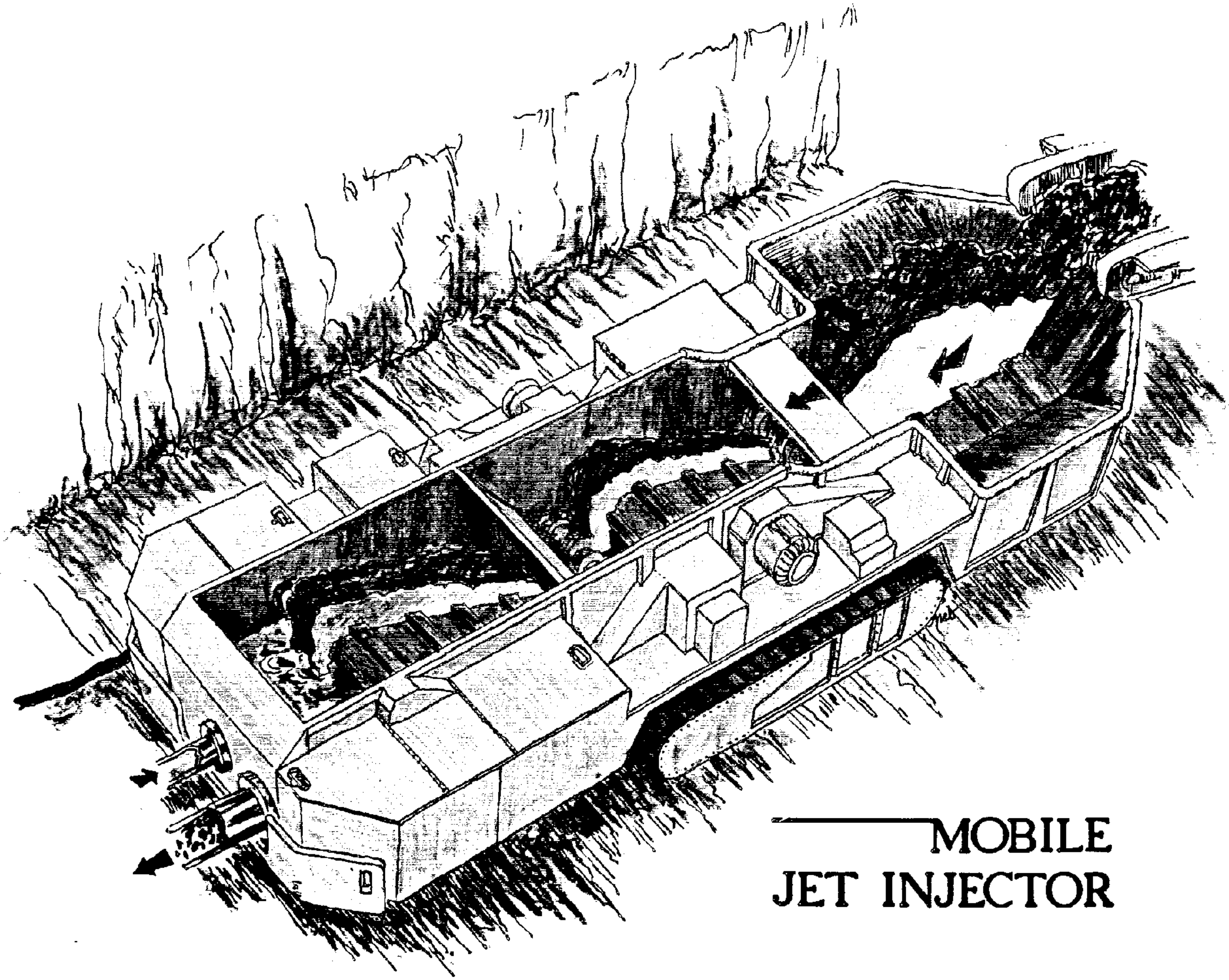


Figure EX-3: Perspective View of Mobile Jet Injector Vehicle

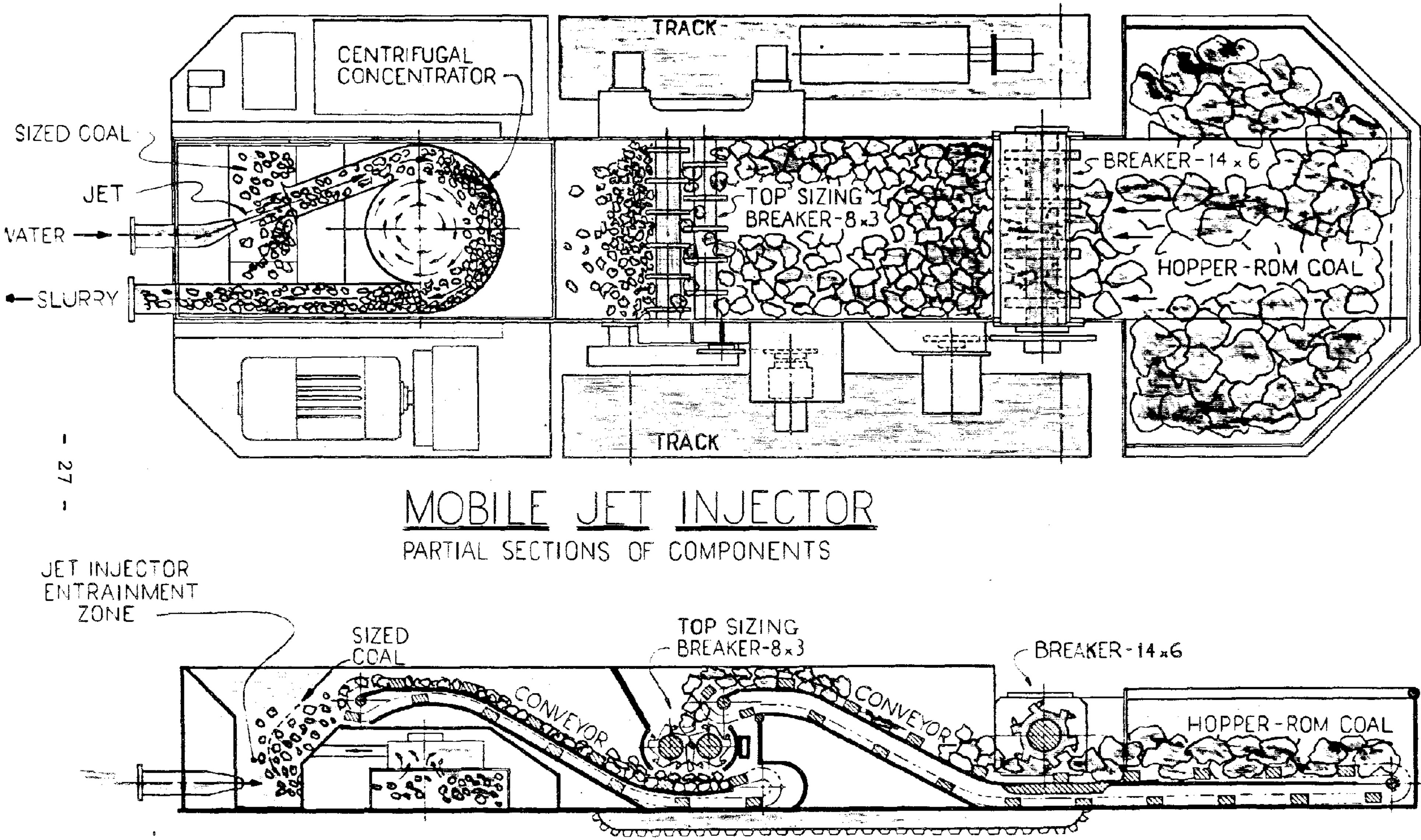


Figure EX-4: Plan and Elevation Views of the Mobile Jet Injector Vehicle

2.0 OBJECTIVE AND SCOPE OF WORK

2.1 Objective

Conceive, design, fabricate and test a device that will provide for controlled injection of run-of-mine (ROM) coal into an operating, hydraulic pipeline in an underground coal mine. The device shall inject the coal without degradation of the coal, without water leakage from the pipeline and without plugging of the pipeline. Controlled injection means that the feed rate can be regulated, ranging up to the output capability of continuous mining machinery, for the purpose of controlling slurry concentration. The device shall be adaptable to vertical, sloped, or horizontal pipelines.

2.2 Scope of Work: Phase I - Conceptual Studies

Conduct a survey of patents, literature and appropriate manufacturers to determine (1) the state-of-the-art with respect to solids pumping devices and (2) the applicability of any such device(s) to the requirements of this program. Preferred schemes to be documented in a Phase-ending report including diagrams, sketches or photographs.

Concurrently, develop original concepts for feed devices suitable for fulfilling the requirement of this program.

Make an objective comparison of all configurations by narrative discussion and by a comparative point rating system evaluation. All candidate concepts will be evaluated such as to show the potential of each device for meeting the following design criteria:

-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.

-Overall Dimensions:

The solids injector shall not exceed the following maximum dimensions: Length, 27 feet; width, 9 feet; and a height suitable for operation in a 4-foot coal seam. Smaller dimensions are highly desirable.

-Mobility:

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

2. The solids injector shall be sufficiently mobile to follow all operating maneuvers of the primary machinery of continuous, conventional, and longwall mining systems. Mobility is here defined as tramming speed (if powered), turning radius, and steering response speed.

3. 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.

-Capacities:

1. 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 per minute during time periods of up to 3 minutes in a 4-foot seam for continuous miners. (This task has since been made easier by later modifications which required only 6.4 TPM operating rate.)

2. 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 pipeline diameter, therefor oversize must be controlled by crushing or breaking.

3. 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.

4. 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.

5. The injector shall be designed for 6- to 18-inch 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-inch pipe as a small-scale prototype and 10- or 12-inch pipe for a full-scale prototype.

6. 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.

7. 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 of 250 psig, 500 psig, and 1000 psig shall be provided along with the standard design. If more than one model is necessary, the conclusions must be justified.

-Controls

1. 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.

2. 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.

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

-<u>Service and Economy Requirements</u>

1. 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.

2. Delicate or precision-adjustment mechanisms shall not be specified.

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

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

5. 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.

6. 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.

7. Lowest possible capital and operating costs shall be 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.

-<u>Safety</u>

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

2. The injector shall fail in a safe manner. This 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.

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

2.3 Scope of Work: Phase II - Design and Engineering

A jet pump injector, capable of receiving and processing dry coal, was recommended as the preferred concept on the basis of the results of the Phase I study. This recommendation was made on the basis of size, weight and adaptability, ruggedness, ease of maintenance and adaptability to a multiplicity of system configurations. This phase of the program was directed towards confirming the feasiiblity of the jet pump injector for the intended service and establishing a workable mining plan using a mobile, jet pump injector vehicle concept.

In Phase IIA, the scope of work was divided into two parts. The jet pump injector was to be modeled and was to involve the following elements of work.

Conduct subscale model jet pump injector testing in a closed loop test facility capable of processing one twelfth (1/12) of the full scale vehicle flow rate. The injector and its test loop were to meet the following upper limit specifications:

Coal Size	:	3/4"
Coal Rate	:	1 Ton/Min.
Water Flow Rate	:	250 GPM
Supply Water Pressure	:	125 PSIG
Discharge Line Size	:	4 Inches
Discharge Pressure	:	25 PSIG

Additionally, develop an analytical model of the jet pump injector to describe the interaction of the water jet and the coal lumps which is significantly different from more conventional slurry jet pumps having a much lower feed concentration. The model was to be used to size the subscale jet pump, permit jet pump scaling and study the physics of the energy exchange.

In Phase IIB the scope of work involved the following two elements of work.

Conduct a study to establish a workable mining plan integrating the mobile jet injector vehicle.

In conjunction with the closed loop test facility and experimental, subscale jet pump, develop a slurry concentration device capable of increasing the coal concentration in the delivered slurry to 40% (wt) or greater.

2.4 Scope of Work: Phase III - Mobile Jet Injector Development

In Phase III the scope of work was comprised of three efforts as described below.

Design and fabricate a full-scale jet injector subassembly in preparation for testing in a surface test circuit. Perform tests on this unit to verify analytical work done to predict design parameters and to verify the capability of the jet injector to meet minimum performance standards. An existing surface test circuit shall be modified to incorporate all features needed to test the jet injector subassembly, including several control system alternatives for controlling slurry concentration, line velocity and hopper water level.

Design and fabricate critical portions of the feeder/breaker subassembly and test (separately from the jet injector subassembly). Test this subassembly for ability to (1) control top size, (2) feed at the proper rate and (3) meet all detailed performance requirements. Flow medium for these tests to be from a limestone quarry, a slate quarry or other source of material that will assure the ruggedness of the feeder/breaker subassembly.

Prepare engineering layout-type drawings of the complete mobile, jet injector vehicle. These drawings to include the most current program technology with respect to the (1) jet pump, hopper and diffuser, (2) coal concentrator, (3) ROM coal breakers and (4) cross-over valving and controls.

3.0 PROGRAM SUMMARY AND CONCLUSIONS

Phase I: Conceptual Studies

The five month Phase I effort began with a patent search in which over 2000 related patents were screened, 63 were selected for review, and 2 were chosen for concept evaluation. The results of the search were published as separately bound Volumes II and III of Appendix D of the Phase I report submitted to the Bureau on December 15, 1975. A concurrent literature search was conducted of published information relating to injectors and feeders. A total of 32 technical papers, reports, and magazine articles were surveyed and separately bound in Volumes IV and V of Appendix E of the Phase I report. Similarly, a survey of feeder and slurry pump manufacturers was conducted in which 32 manufacturers were contacted and pertinent catalog information was accumulated and presented as a separately bound Volume VI of Appendix F of the Phase I report. In addition, original concepts for feed devices were submitted as part of the Phase I effort.

All configurations resulting from the survey and concepting efforts underwent objective comparison and were evaluated against the design criteria defined in the Statement of Work. The evaluation process led to the selection of four configurations for further detail evaluation. A jet pump injector capable of receiving and delivering dry coal to a pressurized hydraulic pipeline was judged as the preferred injection concept.

To further assess the viability of the jet pump injector concept, typical hydraulic face haulage systems were analyzed in terms of comparative capital and operating costs. Within the contract restraint that prohibited consideration of modifications to the continuous miner, the system embodying a separate vehicle containing the jet pump injector and a feeder breaker, provided the least cost per ton of coal delivered to a slurry hydraulic face haulage pipeline. However, all of the hydraulic face haulage systems studied offered economic advantage over the conventional shuttle car system.

Phase II A: Jet Pump Model Studies

Details of the Phase IIA injector model studies, computer analysis, and haulage system analysis were presented in a phase ending report on June 22, 1977. The following is a summary and a list of the salient conclusions.

A subscale model jet pump injector closed loop test facility capable of processing 1/12 of the full scale vehicle flow rate was fabricated and used to evaluate concept feasibility and parametize performance. The test facility had the capacity to evaluate a coal size and flow rate of up to 3/4" and 1 ton per minute respectively, a nozzle pressure and flow rate of up to 140 psi and 500 gpm respectively, and a discharge line size and pressure of up to 4" and 25 psi, respectively.

As a result of systematically investigating a variety of injector configurations and component sizes, the following conclusions were made regarding performance parametization:

- A horizontal gravity fed injector is superior to both a vertically oriented injector and a horizontal screw fed injector.
- The introduction of a small amount of supplementary water in the feed hopper increases the maximum solids injection rate capability and eliminates air ingestion.
- A shock phenomenon exists immediately upstream of the injector throat which creates a backflow, fluidizing the hopper solids and contributing to total system pressure rise.
- Varying diffuser angles from 4 1/2 to 10 degrees has no measurable effect on discharge pressure.
- A bell-mouth throat inlet results in a nominal 10% improvement in discharge pressure when compared to an oblique conical inlet configuration.
- A 3" mixing chamber throat length yields a nominal 15% improvement in discharge pressure when compared to 1" and 12" throat lengths for a gravel slurry. Since little difference is seen with a coal slurry, a 3" length is considered optimum from the point of view of compactness rather than performance.
- A 4" diffuser discharge diameter is recommended over a 3" diameter due to the slightly higher discharge pressure produced.
- A solid jet is superior to a hollow core jet due to its relative strength and resistance to disintegration in the entrainment zone.
- It is recommended that the ratio of nozzle throat area to mixing region throat area be kept in the range of .152 to .293, and that the ratio of secondary mass flow to primary mass flow be kept in the region of .65 to .75. (Secondary mass flow is the sum of coal tonnage rate and hopper water rate. Primary mass flow is the nozzle water flow). Within

these operating ranges, discharge pressure is fairly insensitive to mixing area throat diameter, and will therefore tolerate throat water without significant effect on performance.

- Although a 4" entrainment length yields a higher discharge pressure than either a 3, 6, or 8 inch length, it requires substantial amounts of supplementary hopper water to maintain a submerged jet. Therefore a 6" entrainment length is considered optimum when considering total injector efficiency as indicated by the nondimensional injector coefficient C . If supplementary hopper water can be extracted from the injector discharge, then optimum entrainment length would revert back to the 3" value. Longer entrainment lengths impair performance due to energy losses from jet disintegration.

In addition, operational testing led to the following observations.

- Neither the feed hopper nor the injector throat experience any coal accumulation or blockage problems.

- A hardened mixing chamber throat experiences diametric wear of 0.15" per thousand hours of operation.

- The injector performs well on gravel, and ingests tramp metal and wood without damage.

- Backflooding of the feed hopper occurs only under abnormal operating conditions and at such a slow rate as to render it controllable.

- Coal degradation is measurable and therefore accountable in the analysis of results.

- The hopper water level control system is effective in maintaining a submerged jet and thereby eliminating air entrainment.

- Although the injector had no means for directly controlling slurry discharge concentration, it was found that optimum mass concentration of 34% was achieved when operating with a secondary to primary mass ratio in the range of .65 to .75. (The sum of coal tonnage rate plus hopper water flow rate, divided by nozzle water flow rate).

A computerized analytical model of the jet pump injector was developed to describe the momentum and energy transfer phenomenon between the water jet and the coal particles, which is significantly different from more conventional slurry pumps having a much lower feed concentration. The model accounts for the pressure shock phenomenon at any position within the injector, and uses a momentum defect factor of 0.9775 as the ratio of actual to ideal momentum transfer to account for additional losses in the entrainment region resulting from operating with the jet submerged. Friction losses in the mixing chamber are accounted for by application of a friction coefficient for slurry transport through a pipeline, and energy losses through the diffuser were obtained from test results in which diffuser efficiency is related to a diffuser friction coefficient.

Analysis lead to the following observations:

- Optimum shock location was found to be in the mixing chamber where minimum pressure losses occur through the shock.

- Mixing chamber length was found to have little influence on discharge pressure, since mixing is initiated in the entrainment region and accelerated through the shock.

Discharge pressure and pressure profiles predicted by the computer program were compared with actual test results for a given injector geometry and a variety of operating conditions, varying from complete shutoff to nearly 1 TPM flow rate, and a deviation of less than 6% was achieved. Since the model does not predict the amount of supplementary hopper water required for a given operating condition, this information was determined from test data. Therefore, based on a known injector geometry and nozzle pressure, the analytical model can accurately predict discharge pressure for a given coal flow rate and supplementary hopper water flow rate.

Based on the results of the computer analysis and the subscale model tests, several design and operating factors were assessed for a prototype jet pump injector face haulage system servicing a 48" seam mine. The slurry velocities, line sizes, and conveying distances were determined for injector rates of 3 to 10 TPM. The following conclusions were drawn from the analysis:

- A 6 ton surge capacity can be accommodated with an oversize hopper, and this will result in a 20% reduction in required injector mass flow rate with a subsequent 50% increase in potential conveying distance.

- Maintaining the ratio of nozzle to mixing throat area in a range of .152 to .219 results in an estimated throat life of 1 to 6 million tons for mass flow rates 3 to 10 TPM, respectively.

- o The recommended injector control concept allows the water flow rate to increase whenever coal flow rate decreases, thus maintaining slurry line velocity above the design value.

- o The preferred haulage line booster pump control method uses a constant speed pump with a pressure control valve supplying supplementary water to the booster pump inlet as required.

- o The low concentration variable flow shut-off-bypass valve eliminates hopper overflow and maintains the the jet submerged thereby avoiding ingestion of air.

Phase IIB: Hydraulic Face Haulage System and Concentrator Development

Details of the Phase IIB hydraulic face haulage system design and laboratory concentrator tests were presented in a phase ending report on June 30, 1978. The following is a summary, and list of the salient conclusions.

A study was conducted in which the mobile jet injector vehicle was integrated into a workable mining plan. The haulage system consisted of the breaker-injector vehicle, 92 ft of ground transition hose on wheel carriages spaced 4 ft apart, a monorail system with suspended tractors and corner guides, and a powered loop-forming carriage which plays out or retracts hose much like a moving pulley. A time motion study was conducted on a standard 5-entry mining plan using this haulage system with a continuous miner and 2 dual arm roof bolters. The continuous miner was found to be the bottleneck to productivity because of time lost in moving from one place cut to the next. The miner is a larger and more cumbersome vehicle than the injector breaker vehicle and its power cables must be moved by hand. The injector breaker vehicle is compact and light in comparison and all control and power lines are suspended with the slurry hoses by means of the monorail. There are no ground cables to hinder movement, and the monorail suspended hose and power train can tram at reliable high speed because of the tracks.

Three configurations of linear concentrators were designed, manufactured, and tested on the 1 TPM subscale jet injector test facility. A single slot concentrator with a deflector vane placed upstream at 45 degrees from the main flow experienced plugging of the slot with only a 1% increase in concentration. An 18 slot concentrator with a constant cross sectional area and a 60 slot concentrator with a variable cross sectional area each demonstrated coal mass slurry concentrations approaching 54% with stable, reliable, and repeatable performance. The following conclusions were drawn from this test program:

- Sufficient understanding was gained during the program to predict the performance of the 60 slot variable cross sectional area concentrator.

- Control of the concentrator does not require any moving or special controls. All that is required is a restrictive non-plugging device in the concentrator overflow such as a cyclone separator.

- Fines circulation in the circuit consisting of the concentrator and jet pump injector do not tend to increase with time.

- With a coal mass slurry concentration of up to 40%, all water removed from the concentrator can be fed back directly to the jet pump hopper so as to increase the efficiency of the system. At concentrations above 40%, a small onboard centrifugal, clear water pump can be used to boost the pressure of a portion of the concentrator overflow and feed it directly to the nozzle.

Phase III: Mobile Jet Injector Development

Details of the Phase III injector vehicle design, subscale centrifugal concentrator tests, and full scale injector and secondary breaker tests were never summarized in a phase ending report. Instead this project ending report serves that purpose. The following is a summary of the salient conclusions from the Phase III efforts which were concluded in 1982.

A mobile jet injector vehicle was designed based on the S&S Corporation's Spartan SE II feeder-breaker vehicle with the following onboard subsystems:

- Surge Bin - A 3.5 ton capacity receiving hopper for ROM coal.

- Primary Breaker - Production-type S&S Corporation 18" single roll pick-type breaker capable of downsizing 14" ROM to 7". Utilizes a flywheel energy-storage drive system to reduce power needs.

- Primary Flight Conveyor - Delivers coal from surge bin to primary breaker and then on to secondary breaker.

- Scalper - A combination of parallel trapezoidal bars with recessed round spacer bars to allow properly sized coal to drop directly onto the secondary flight conveyor and bypass the secondary breaker, thus minimizing overbreakage and reducing breaker throughput requirement; for a more compact design.

- Secondary Breaker - Two 11" diameter, counter-rotating rolls with 8 TPM throughput capacity at 400 RPM safely located in a covered mid-region of the vehicle. Down sizes the coal from 7" to a maximum of 3" particle size with minimum fines production. Utilizes a fly wheel energy storage system for reduced power consumption.

- Secondary Flight Conveyor - Delivers coal from the secondary breaker to the jet injector hopper.

- Jet Injector Hopper - A holding capacity of approximately 2.5 tons in which coal immersed in water is supplied to the jet injector entrainment zone for induction into the high velocity water jet. Water level is maintained by coupling Drexelbrook level sensors with a low concentration variable water flow cut-off-bypass valve and a nominal 200 gallon reserve tank.

- Jet Injector Nozzle - Supplies a high velocity water jet through the entrainment zone into the mixing tube.

- Mixing Tube and Concentrator - Increases slurry concentration by partial water removal. The overflow water from the centrifugal separator is recirculated to the injector hopper.

- Cross-Over Valving - To eliminate water hammer during start up and enable excessive water supply to bypass the limited capacity injector nozzle.

A 1-to-6.4 scale model of a domed-disk cyclonic concentrator was fabricated from transparent acrylic for flow visualization, and tested with 30% inlet slurry concentration at flow rates up to 100 gallons per minute. 1/8" diameter Delrin pellets with a specific gravity of 1.4 served as models for 3/4" full scale coal particles. Test results yielded output slurry concentrations as high as 45%, thus proving the viability of the cyclonic concentrator design.

A full scale 6.4 TPM jet pump injector subassembly was installed and tested at a slurry test facility constructed under another Bureau of Mines contract at Framingham, Mass. Two nozzle diameters and 9 entrainment lengths were tested at a variety of nozzle pressures and flow rates. For all test runs, the jet pump injector subsystem demonstrated the ability to provide 6.4 TPM coal flow rate and did so at a consistant slurry concentration of 42% regardless of the operating conditions. Test results led to the following conclusions:

- Within the accuracy prescribed by the data scatter, nozzle diameter appears to have no effect on the relationship between slurry mass concentration and injector efficiency.

- On average, a tripling of coal slurry concentration results in an increase in jet pump efficiency of as much as 40%. The efficiency increase reduces as entrainment length is increased.

- A given change in discharge pressure requires a 5 fold change in nozzle pressure using a 56 mm diameter nozzle.

- A given change in discharge pressure requires a 10 fold change in nozzle pressure using a 63 mm diameter nozzle.

- For each nozzle, the effect of entrainment length is minimal, namely a 60 to 80% increase in entrainment length results in only a 10% increase in the effect of nozzle pressure on discharge pressure.

A full scale secondary breaker subassembly was tested at S&S Corporation using a corporate standard shale stock to simulate coal. A double roll breaker design was modified to enable a comparative evaluation of drag and attack picks. Throughputs as high as 8 TPM were achieved with downsizing to 3" by 0". Although differences in angle of attack and spacing between the attack and drag picks resulted in differences in bite size and bit stalling, each bit type achieved its sizing and throughput goal. The tests therefore showed that a compact design utilizing a 36" double roll configuration requiring approximately 50 HP, could in fact achieve a 6.4 TPM throughput of 3" by 0" coal.

Phases III, IV, and V: Fabrication, Demonstration, and Delivery of Jet Pump Injector Vehicle

The original program plan of manufacturing a complete jet injector vehicle and conducting an underground haulage evaluation was changed because of lack of funding. Instead it was replaced with a plan to evaluate the performance of the jet pump injector on single pass ROM coal at a coal preparation plant. Discussions with the Pittston Coal Company for the installation of test facilities at their Moss-III coal preparation plant led to the defining of a test program. However, the imposition of budget reductions on the part of the Pittston Company led to the curtailment of these plans also. Numerous other preparation plants were surveyed without finding an interested party willing to participate in a cooperative test program.

4.0 DETAILED DISCUSSION OF TECHNICAL PROGRESS

4.1 - Introduction

In 1975 Ingersoll-Rand Research, Inc., (IRRI) responded to Bureau of Mines RFP H0155097 for a study involving the design and development of a mobile, coal injector vehicle for use in a coarse slurry transport system. IRRI was subsequently awarded a five-phase program which, according to plan, was to lead from concepting through underground demonstration testing and delivery of a vehicle. The specific phases of the program were identified as follows:

Phase I -	Concept Development
Phase II -	Design and Engineering
Phase III -	Fabrication of a Prototype
Phase IV -	Prototype Demonstration
Phase V -	Final Report and Delivery of Injector

In Phase I, a jet pump capable of receiving dry, coarse coal was identified as the most promising device for injecting the coal into an underground, hydraulic, face haulage system. In Phase IIA a subscale jet pump was developed that successfully demonstrated concept feasibility as well as the desired performance characteristics. In parallel, an analytical model of the jet pump was prepared based on theoretical considerations. Correlation of analytical predictions with test model results was good, providing confidence for subsequent scale-up of the jet pump to a full-scale injector. Phase IIB explored various system configuration options for the integration of the jet pump into a fully operational, underground, coarse coal hydraulic face haulage system.

Finally, in Phase III a full-scale, mobile vehicle embodying a jet pump, coal sizing equipment and all other supporting systems and subsystems was engineered. Engineering drawings of the complete mobile vehicle were prepared to demonstrate space management to contract specifications. Fully operational, full-scale jet pump and feeder/breaker subassemblies were designed, fabricated and tested successfully.

Program realignment prevented the accomplishment of the underground Prototype Demonstration (Phase IV) and ultimate delivery of a mobile injector vehicle (Phase V).

This final report has been prepared in partial fulfillment of the requirements of Phase V. This section of the report documents the technical progress accomplished under Phases I, II, and III of the program.

4.2 Phase I - Conceptual Studies

4.2.1 Introduction and Summary

The hydraulic transport of coal was studied as a potential means for safer haulage and higher productivity. One important development required to assure the system's usefulness in underground coal mining was the availability of a coal feeder (an injection mechanism) that could inject run-of-mine coal directly into an operating hydraulic pipeline. To be truly effective and to provide a necessary breakthrough, the feeder had to be low in profile, minimize breakage during feeding and accept coal feed directly from the continuous miner. Ingersoll-Rand Research, Inc. (IRRI) received a contract from the Bureau of Mines to conceive and develop such an injection device. This section of the report describes the work performed under Phase I - Conceptual Studies to select and define a suitable concept.

A survey was conducted that was comprised of a patent search, a literature review and a screening of selected manufacturers to establish the state-of-the-art with respect to solid material feeding devices. The complete results of this activity were reported in the Phase-Ending Report for Phase I.

The concepts arising from this survey, together with concepts conceived by IRRI, were screened resulting in the selection of four injector concepts for detailed evaluation. Only the four selected concepts are described in detail in the present report. These concepts were objectively compared with the design criteria specified by contract. The concepts were evaluated by Group Delphi and Point Scoring Method.

In addition, face haulage systems incorporating the various injector concepts were defined and analyzed. The most promising system was identified and recommended for future development work. This system was considered to offer significant improvements in productivity and safety.

4.2.2 Background Surveys

This activity included a survey of patents, general literature and selected manufacturers to determine whether any publically identified, existing device could be applied to meet the ultimate design objectives. An original file of published documents related to the subject was provided by the Bureau of Mines. This information, along with new information obtained under this phase, was submitted as part of the Phase-ending Report for Phase I. Several interesting concepts and devices were obtained from this survey and examined during the initial screening process. None, however, were selected for final evaluation for reasons indicated later.

Patents

A patent search was carried out at the Washington, D.C. Patent Office. The search was concentrated in such classes as Conveyors: Fluid Current, Conveyor: Power Drive, Mining of In Situ Disintegration of hard material, Material or Article handling, Metallurgical: Apparatus and Distillation Apparatus. Approximately 2,000 patents were screened and 63 were selected as related to the subject. The pertinent patents from this survey were included in the Phase-ending report for Phase I.

The patents were surveyed in some detail to establish whether or not the concept had merit for this application. Two patents (#3,294,454 "Reciprocating Vane Type Rotary Pump" and #2,796,028 "Centrifugal Pump") were selected for the initial concept screening process.

Literature

Although a considerable body of literature is available on slurry transport, only limited information was found on the subject of injectors and feeders. The literature survey embraced published information on the subject from Russia, the United Kingdom, Germany, Poland, France, South Africa, Japan and the United States. In particular, selected translations from two Russian books on hydraulic mining and USBM #H0133037 report entitled "Feasibility of Hydraulic Transportation in Underground Coal Mines" were studied. A total of 32 technical papers, reports and magazine articles were surveyed and separately bound and formed a part of the Phase-ending Report for Phase I.

A number of directly applicable devices have been investigated in Russia and the United Kingdom. All of them, however, have a very low coal feed rate (up to 1 Ton/Min.) and require a large amount of space. The degree of success with all these units was not always apparent. However, several lockhopper concepts have been used successfully for hoisting in several countries. These concepts are less attractive for face haulage because of their large space requirements.

Manufacturers

Related equipment can be supplied by drum feeder, screw feeder and slurry pump manufacturers. Thirty-two (32) manufacturers were contacted by phone or in person. The project requirements and how they relate to their devices were discussed. Wherever possible and appropriate, catalog information on the product(s) of interest was obtained and included in the Phase-ending Report for Phase I. Of special interest were the drum feeders of the ESCO, Kamyr, Radmark and Bauer manufacturers and the low profile slurry pump of Moyno Pump Division of the Robbins & Myers Company. All of these units were included for the initial concept screening process but eventually rejected for reasons indicated later.

During the survey two interesting developments were noted. First, Combustion Power Inc., Menlo Park, California, under an EPA contract was investigating an ESCO feeder for feeding coal (-1/4") into a 60 psig pneumatic line. Problems were encountered with sealing and wear of the feeder. Second, Island Creek Coal Co., Lexington, Kentucky was investigating the hydraulic transportation of coarse coal (- 1 1/2") slurry from the washing plant to the railroad terminal. They were using a Kamyr Co. drum feeder in their experimental set-up to inject coal into a steel pipeline. A request to visit Island Creek Coal Co. was refused and we were unable to obtain further information from the Kamyr Co. on this application.

4.2.3 Collection of Concepts and Screening

Three principles can be used to inject coal into a high pressure line. They are: positive displacement, dynamic forces and shear drive. For each, the feed can be continuous or intermittent. The various concepts were classified in each category and are shown on the parametric chart of Figure 4-1.

All of the concepts were screened with the following methods:

- Group Delphi
- Point Scoring Method

Group Delphi Method: The groups were made up of project personnel, IRRI Underground Mining personnel, Lee-Norse Company personnel and several consultants. The operating principle, the approximate required dimensions, the advantages and disadvantages were studied for all the concepts. An opinion from each group was obtained.

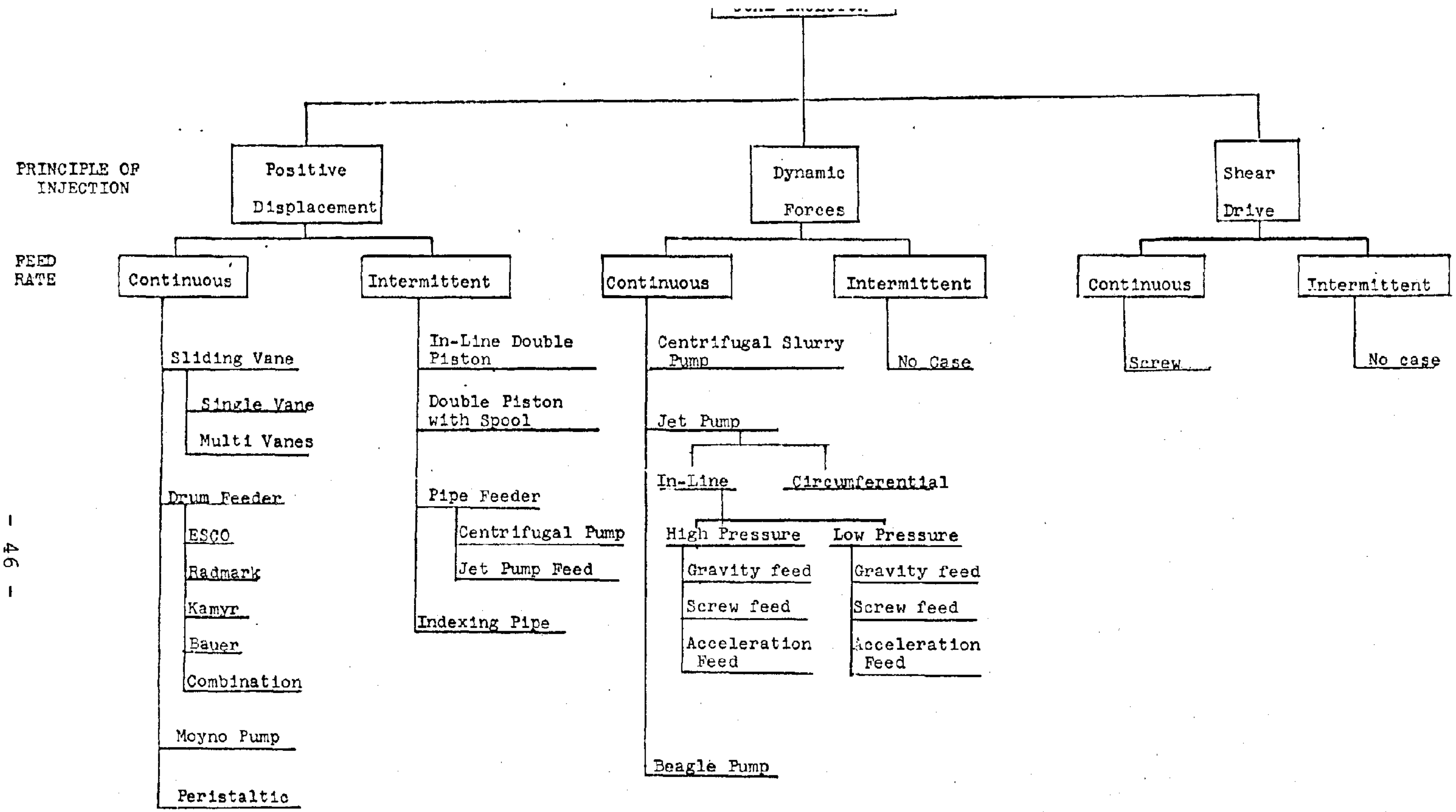


Figure 4-1: Matrix for Coal Injector Parametric Analysis

Point Scoring Method: The key evaluation factors were established and listed. Each factor was given a relative weight of relative merit which determined the maximum number of points a concept could receive for that factor. Each concept was then evaluated with respect to each factor and a score assigned to each combination. A total score for each concept was then obtained by summing each of its factor scores.

The results of both methods were integrated and the four top-rated concepts were selected for detailed evaluation as follows:

- -Single Sliding Vane
- -In-Line Double Pistons
- -In-Line Jet Pump (Low Pressure)
- -Circumferential Jet Pump

These concepts are described in detail in the next section of this report.

4.2.4 Discussion of Concepts

4.2.4.1 General

The four selected concepts were analyzed in depth. The analysis included the dimensioning of components in such detail as necessary to permit the determination of speeds, cycle times, flows, pressures and other performance-related information.

The contract specifications required that the injector should handle 12 tons of coal per minute for up to 3 minutes. Two design approaches are possible. In the first approach, the injector would be designed for the full 12 tons per minute. In the second approach the injector design feed rate would be reduced but adequate surge capacity would be provided. Practical considerations limit the maximum surge capacity to about 9 tons. With this surge capacity the injector coal feed rate could be reduced to approximately 9 tons per minute. Continuous miners are capable of cutting coal at the rate of approximately 12 tons per minute under ideal conditions. The average productivity, however, is much lower because of delays associated with roof bolting and haulage. There have been continuing efforts by the USBM and industry to improve productivity and much of the efforts are directed at these areas. Hydraulic mining and haulage are concepts for coal production in the late 1980's. It is expected that by that time the injector will be required to handle 12 tons per minute for perhaps longer than 3 minutes and a surge capacity of 9 tons will not help. Furthermore, it can be shown that for any concept a twelve ton per minute system is smaller than the combination of a 9 ton per minute system plus a 9 ton surge capacity. It is apparent, therefore, that the injector must be designed for the highest expected feed rate that can be sustained for several minutes. The injector coal design feed rate of 12 tons per minute was selected. However, systems with less feed rate and surge capacity are discussed in a subsequent section entitled, "System Considerations".

A pressure level of 125 psig was specified for the study and design changes for higher pressure are discussed.

A coal size of 4" (maximum) was established as a design goal. Ten percent of a continuous miner output will be of a +4" size. Investigations have indicated that it is possible to limit the output of a continuous miner this size with certain modifications to the cutting drum design or by the incorporation of a coal crusher. However, the contract requirement did not allow any modification to the existing miner in order not to jeopardize its acceptability for underground mining. On the basis of this restriction it was necessary to design and incorporate a feeder breaker on all injector vehicle configurations.

Coal from the continuous miner conveyor is dumped into the injector vehicle hopper. The injector vehicle can receive coal from the miner conveyor in any position. A variable speed hydraulic chain conveyor drive moves the coal from the hopper at a controlled rate into the injector vehicle's rotary pick crusher. The crusher drum was designed specifically for this application since none of the existing underground feeder breaker configurations can either handle the required high coal feed rate or control the size satisfactorily. The possibility exists that a future, novel design approach, offering a lighter and more compact crusher, will emerge which will reduce the overall dimensions and weight of the injector vehicle. It is estimated that a maximum of 75 horsepower will be required for crushing. The crusher drum is driven by an electric motor through a speed reducer at constant speed. Water is sprayed through nozzles in the crushing area to control the dust. The chain conveyor delivers the sized coal to the injector inlet at a controlled rate. A magnet is installed at this point to remove tramp metal from the coal.

However, tramp metal can be removed more effectively if a magnet is installed on the miner discharge conveyor.

All of the preliminary design sketches presented in the following sections were generated utilizing equipment of such dimensions as to permit operation in 4 foot seams with sufficient space from roof and floor remaining for maneuverability. Due to this height requirement it was necessary to design these vehicles with water cooled electric motors and hydraulic pumps and motors. The hydraulic equipment also provides speed control and reversible operation.

The injector vehicle must be sufficiently mobile to follow all maneuvers of the continuous miner. Mobility and maneuverability of the injector are extremely important and for this reason all designs are for self-powered vehicles.

The slurry concentration is controlled to 50% by volume as closely as possible and the method of control is described in the concepts description section.

The preliminary design sketches that follow show all major components (such as hydraulic motors and pumps, electric motors, crawlers, controls, belt and screw conveyors) in their relative sizes and positions. Standard available components were utilized wherever possible.

4.2.4.2 Single Sliding Vane Concept

This concept was generated during the course of the contract work and it consists of a casing, a rotor, a sliding vane, coal intake and outlet openings. As shown in Figure 4-2 the coal and water mixture fills the cavity between the casing and the rotor. The rotor rotates counterclockwise and displaces the mixture towards the outlet port. As it rotates, the mixture mixes with the supply water and pushes the coal and water into the slurry pipe. Sealing is provided by the sliding vane which maintains the desired clearances.

This concept as used for the coal injector vehicle is shown in Figure 4-3. The coal from the continuous miner is discharged into the hopper and fed through the crusher at a controlled rate. The crusher reduces the coal size to -4" and delivers the coal into the hopper where it mixes with enough water to fill the voids. The hopper is equipped with two

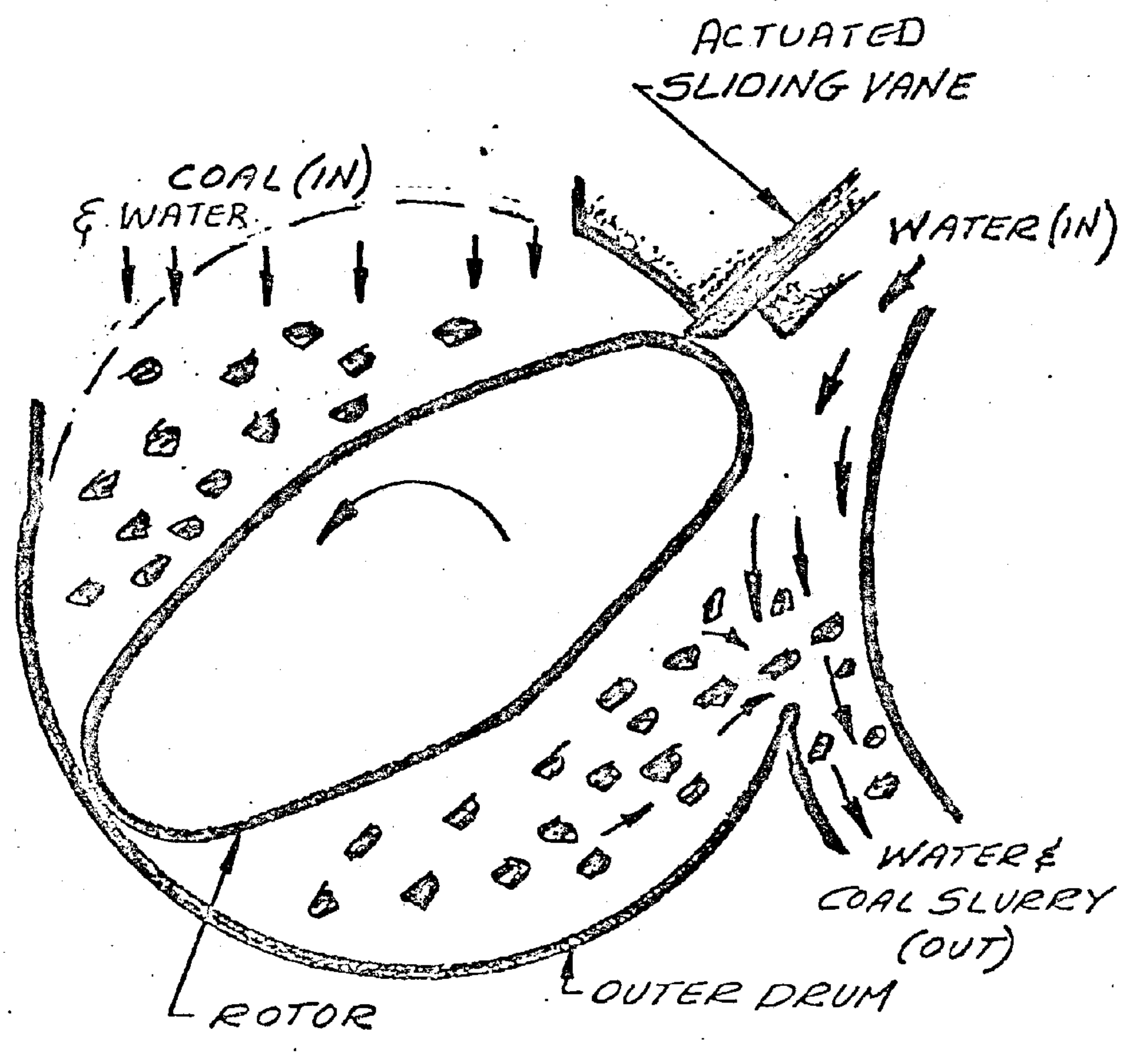


FIGURE 4-2 - Single Sliding Vane Concept variable pitch screws to insure force feeding the coal and water to the cavity formed by the drum and rotor. A 150 horsepower electric motor powers the crusher drum and the hydraulic pump powering the motors for the chain conveyor, the tram drive and the feed screw drive. The advantage of this system is that it can handle transient peak torques at any of the drives. The hydraulic motors give an infinite speed control.

The coal and water are displaced and pushed into the slurry line during rotor rotation as described before. One critical area in this concept is the sliding vane which requires an independent positioning control. Vane positioning is controlled with hydraulic actuation in such a manner that a positive clearance is maintained between the vane and rotor. The side pressure loading is taken by roller bearings and the vane thickness is designed to avoid excessive vane deflections. In addition, the complete vane package can be designed for quick removal and replacement. The sealing between vane and housing is achieved with packing and one side of the vane is flushed by the incoming clean water.

The rotor shape is so generated as to minimize the vane problems. The rotor is divided into two vertical sections separated by a disc. The shape of the rotor in both sections is the same but they are 90 deg. out of phase. This arrangement equalizes the torque, minimizes the bearing loading and delivers more uniform coal into the slurry line.

The rotors' major and minor diameters are 6' and 3' and the active height of the full rotor is 1 1/2'. The rotor speed is controlled from 0-22 RPM. The rotor drive consists of a bull gear and pinions, two hydraulic motors, two pumps and two 175 horsepower electric motors. A total of 500 horsepower is required for injecting 12 tons per minute. The slurry concentration is controlled by the water flow rate to the injector. A hydraulically actuated 3-way valve is used to bypass the water and isolate the system whenever needed.

The overall dimensions of the injector vehicle are 25 1/2'L x 9'W x 3 1/2'H. The estimated weight is approximately 25 tons.

The primary advantage of this concept, in comparison with other positive displacement devices, is that it has only a few moving parts, is simple in construction, does not displace water and can be designed to be extremely rugged in construction. In addition, it has the capability of handling

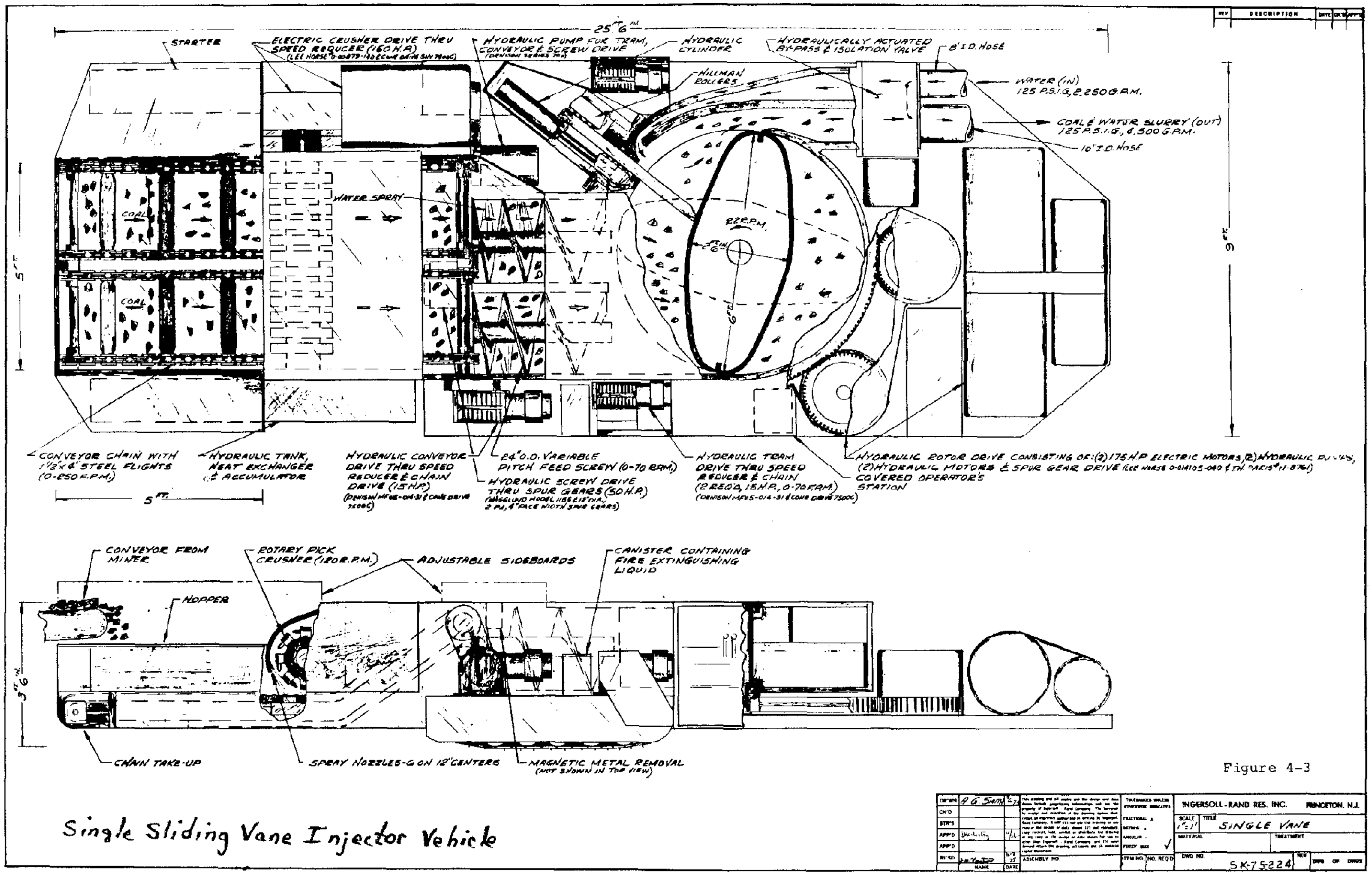


Figure 4-3

Single Sliding Vane Injector Vehicle

larger than 4" coal.

This device exhibits the several disadvantages that are enumerated below. It requires 500 horsepower at the face. The vane design is extremely critical. In addition, it requires sealing between (1) vane and rotor, (2) rotor and drum inside diameter, (3) rotor and vane ends and drum end covers and, (4) vane and drum. Considerable water leakage can be tolerated. However, it is anticipated that once in operation, wear and sealing problems are likely to develop. Coal will settle at the bottom end and cause uneven wear.

4.2.4.3 In-Line Double Piston Concept

This concept was originally conceived at IRRI for coal gasification systems to feed coal into a pressurized, coal conversion gasifier. It was modified to suit the requirements of this application. The concept is based on two pistons which can be actuated independently and which ride in a common, cylindrical housing. The housing contains an inlet opening for coal in one location and clear water inlet and slurry discharge openings in another location. Figure 4-4 shows a typical cycle of operations. Figure 4-4(a) shows the loading mode. The cavity between pistons #1 and #2 is filled by gravity in the manner shown. Note that piston #1 seals the water ports. Figure 4-4(b) shows the position of piston #2 after filling is completed. Piston #2 now has sealed the inlet opening. Both pistons move to the right, displacing coal into the flushing zone as shown in Figure 4-4(c). Once the coal is flushed out, the cavity remains filled with water. Piston #1 now moves to the left to displace this water. The new position of the pistons is shown in Figure 4(d). Finally, piston #2 moves to the left for the loading mode and the next cycle begins.

The coal injector vehicle, based on this concept, shown in Figure 4-5, requires five cylinders. The conveyor from the continuous miner delivers the coal to the injector hopper. The coal is reduced to -4" in a crusher and delivered to the piston housing opening. The chain conveyor must make two bends to achieve this delivery. The crusher drum is driven through a speed reducer with a 75 horsepower electric motor.

A piston diameter of 18" was selected. Five units, operating at 34 cycles per minute are required to achieve the maximum feed rate. The area around the inlet opening is designed to handle and uniformly deliver coal to all the units. Gravity feed is shown. However, a vibrator may be

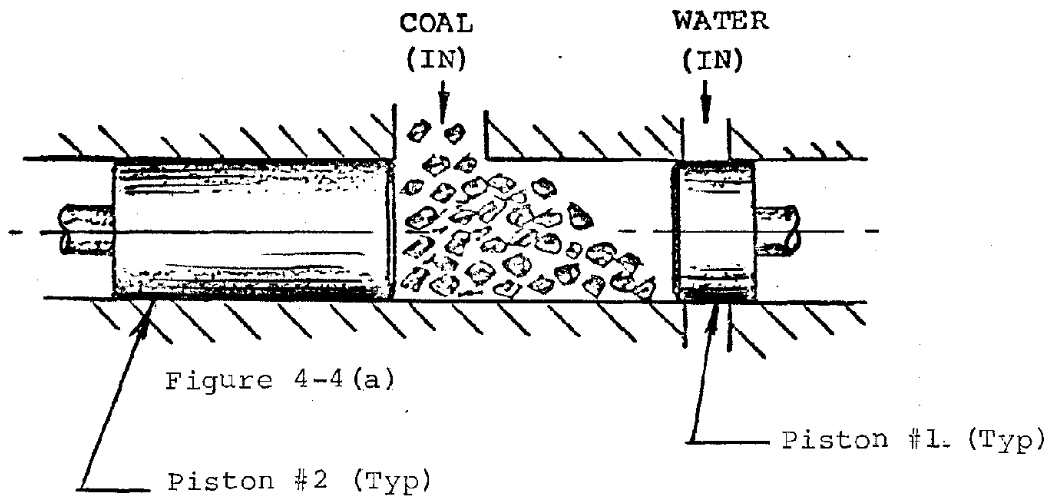


Figure 4-4(a)

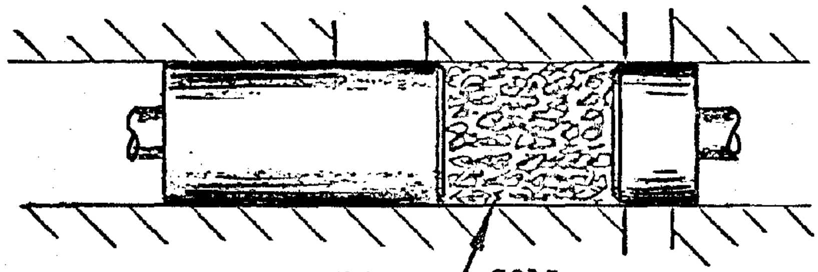


Figure 4-4 (b)

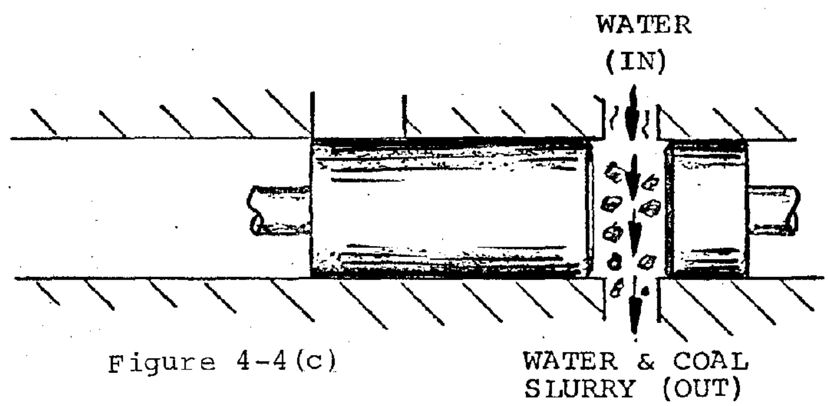


Figure 4-4(c)

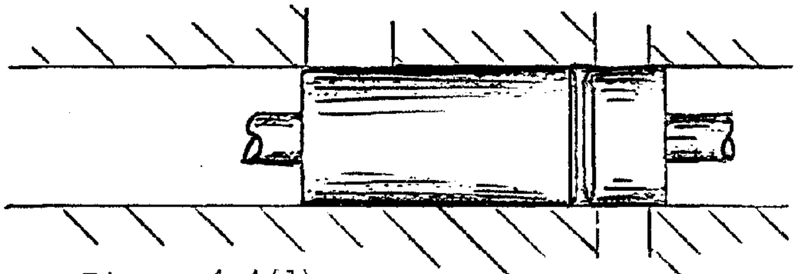


Figure 4-4(d)

Figure 4-4 - Typical Cycle of Operations for In-line Double Piston used to assist the gravity feed. The pistons are driven with hydraulic cylinders and can be triggered with limit switches or other appropriate devices. The two pistons in each unit are timed with each other and each unit is timed with respect to the other units so as to achieve uniform slurry concentration and nearly constant slurry flowrate. The water and slurry from each unit are manifolded. The coal feed rate is controlled by a variable speed chain conveyor and the piston speed. The slurry concentration is controlled by the water flow rate.

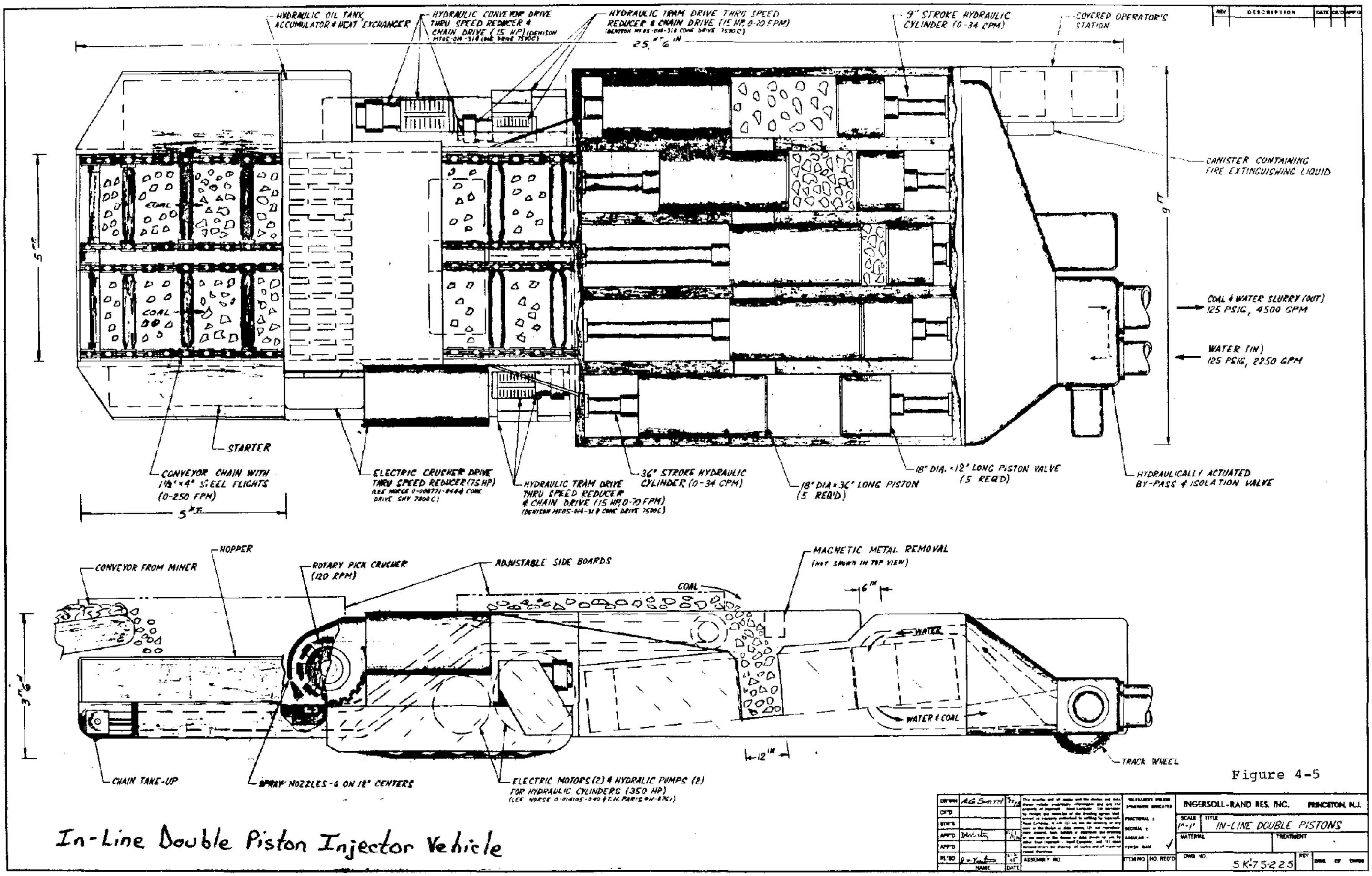
Two 175 horsepower motors and two hydraulic pumps deliver power to the hydraulic cylinders and hydraulic motors for the chain conveyors and tram drive. The total of 425 horsepower is required at the maximum feed rate. The overall dimensions of the injector vehicle are 24 1/2'L x 9'W x 3 1/2' H and the estimated weight is 28 tons.

The sealing problem in this concept is somewhat easier due to the cylindrical shapes of the pistons and casing. However, the pistons are heavy and must operate at high rubbing velocity (34 cycles/minute corresponding to the linear velocity of 5-6 ft/sec.). This will cause considerable wear and once in operation it will be difficult to maintain the clearance between pistons and cylinders. In addition, filling may be a problem. Repair and component replacement represent additional problems in operation.

4.2.4.4 In-Line Jet Pump (Low Pressure) Concept

Liquid jet pumps have been widely used in a variety of applications because of their inherent advantages. These low-initial-cost pumps have no moving parts, nothing to break, no troublesome packing or glands and require no lubrication. They can be noise insulated, are small in size and provide maintenance and trouble-free operation. Considerable work has been carried out recently in pumping slurry with jet pumps.

The principle of operation of a liquid jet pump is the transfer of energy and momentum from a primary to a secondary fluid through a process of turbulent mixing. Figure 4-6 depicts the operating concept of a jet pump. A schematic representation of a jet pump is shown in Figure 4-6(a). The primary fluid (water in this case) is introduced from an independent source and accelerated to high velocity in the nozzle. The secondary fluid (solids in this case) is entrained by and mixed with the primary fluid in the constant diameter throat section. The mixture then passes through a



In-Line Double Piston Injector Vehicle

Figure 4-5

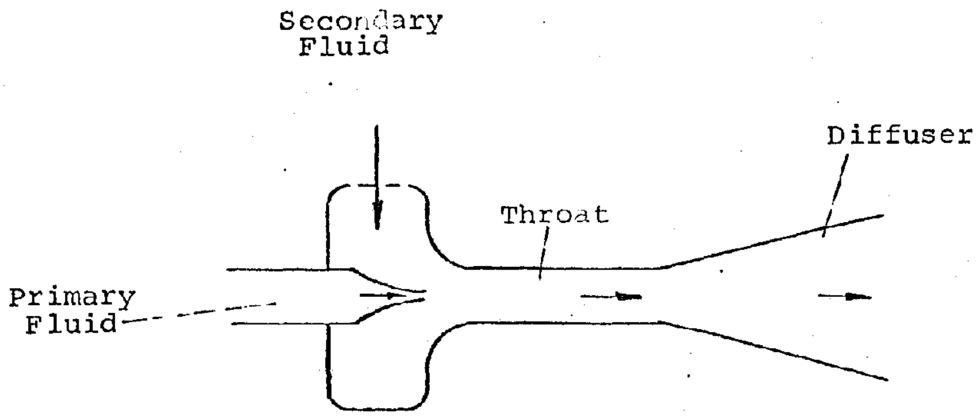


Figure 4-6(a) - Schematic Representation of a Jet Pump

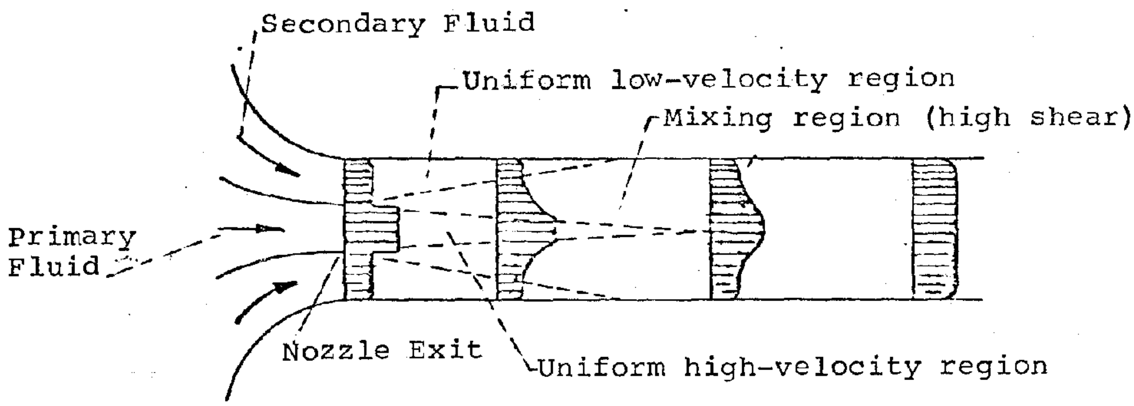


Figure 4-6(b) - Schematic Representation of Velocity Profile in The Throat of Conventional Jet Pump

Figure 4-6: Jet Pump Operating Concept

diffuser in which a portion of the velocity head is further converted to static pressure.

The fundamental mechanism of jet pump operation is the turbulent mixing process. A schematic representation of the mixing velocity profile development in the throat is shown in Figure 4-6(b). At the nozzle exit, the primary stream is essentially a core of constant high velocity water which is separated from the secondary stream by a region of high shear. Initially, the high shear region is made up of a thin sheet of vortices or eddies which give rise to mixing on the periphery of the high velocity core.

The principle used for the coal injector device differs from the conventional jet pump described above in that the secondary "fluid" consists virtually of solid coal particles that are entrained by a primary fluid, i.e., water.

The injector vehicle embodying the jet pump concept is shown in Figure 4-7. The conveyor from the miner delivers the coal into the injector vehicle hopper in the same manner as for the other configuration. A special designed screw feeds coal to the jet pump. The screw is a variable pitch design for complete filling. The screw is completely enclosed for one pitch length to permit circumferentially uniform coal delivery to the jet pump. The screw rotates around a hollow stationary shaft that is supported in a cantilever manner and serving as a conduit to the primary jet pump nozzle. In addition, four nozzles are installed outside the screw casing wall to facilitate the movement of coal into the mixing section. Preliminary design analysis indicated that nozzle and throat diameters of 2 1/2" and 7" respectively were required. The mixing section length would have to be designed for best performance and a diffuser section would be required to recover a portion of the remaining kinetic energy. The jet pump can be isolated with the hydraulically actuated bypass and isolation valve.

A gravity feed system should be considered as an alternative to the screw feed system. It would consist of multi-jet primary nozzles utilized to accelerate the falling coal sufficiently to move it into the mixing section and prevent blockage. This approach would reduce the overall injector vehicle length. The analysis and other details for this approach are discussed in the Phase-ending Report for Phase I.

Slurry concentration will be controlled by regulating

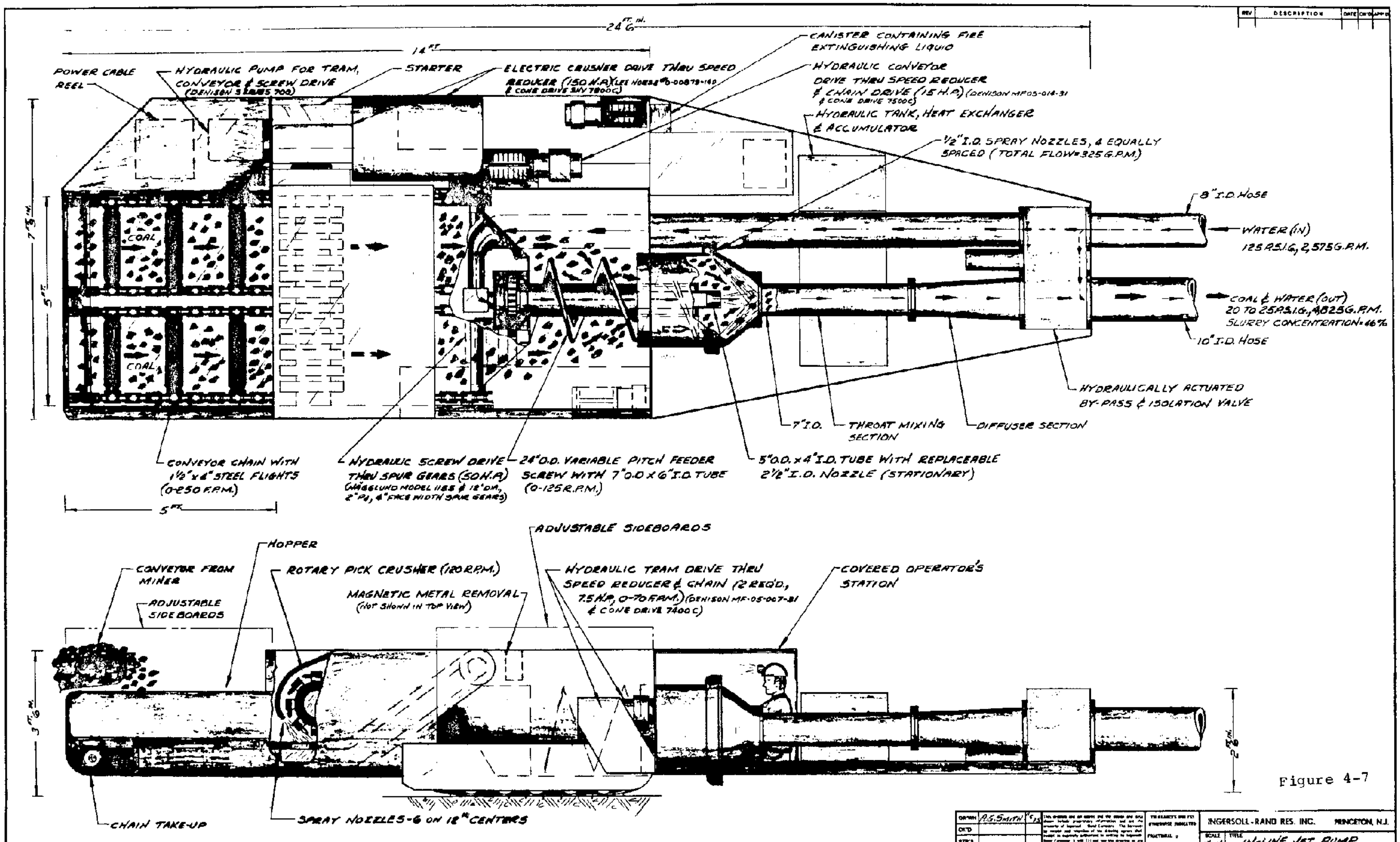


Figure 4-7

In-Line Jet Pump (Low Pressure) Injector Vehicle

INGERSOLL-RAND RES. INC. PRINCETON, N.J.
IN-LINE JET PUMP
SK-75-222

nozzle pressure and flow. The coal feed to the jet pump is controlled by means of a variable speed chain conveyor and screw.

The injector unit requires a 150 horsepower electric motor. The electric motor drives the crusher through a gear reducer and hydraulic pump as depicted in Figure 4-7. A single hydraulic pump powers the individual hydraulic motors for the chain conveyor, tram drive and screw. This single-pump, multi-motor arrangement can handle transient peak torque more efficiently.

The injector delivers the coal slurry pressurized to 20 to 25 psig into the discharge hose. This pressure will convey the slurry approximately 500' (Reference USBM Contract #H0133037, page A-16) from the face where it is delivered to the intake of a commercial centrifugal slurry pump. This slurry pump will then boost the pressure to approximately 100 psig which is comparable to the pressure provided by positive displacement injector devices at the same system location. The corresponding jet pump efficiency is about 28%. The pump is a stationary unit, driven by a 500 horsepower (approximately) electric motor and flow can be controlled with a discharge valve. The booster pump is semi-permanent set up to be moved as the face advances and as the flexible hoses are replaced with fixed pipes. The frequency of relocating the pump is estimated to be once a week in a procedure analogous to the current method of extending the belt on a conveyor system.

The overall dimensions of the injector vehicle are 24 1/2'L x 7 3/4'W x 3 1/2'H. Due to the vehicle configuration and size associated with the jet pump concept it has a smaller turning radius when compared to the units described previously. The estimated weight of the vehicle is 17 tons. With the gravity feed approach the vehicle dimensions and weight would be less. The length would be reduced by 4 feet.

The jet pump vehicle has a number of advantages over the positive displacement pump vehicles. Besides not requiring as heavy an electric cable, the vehicle is substantially simpler, smaller, lighter and easier to maneuver. Further, this concept has few sealing problems. Finally, the initial cost of the device is lower, and, because of its simplicity, it is readily and inexpensively repaired or serviced.

The main disadvantage of the jet pump is its lower efficiency and the fact that a booster station is required.

4.2.4.5 Circumferential Jet Pumps Concept

This concept was generated in the course of pre-contract work. Subsequently some analytical and experimental work was performed and the merits of the concept became more apparent.

It consists of a specially designed volute casing with several nozzles pointing circumferentially. High pressure water (125 psig) is accelerated to high velocity in the nozzles, creating a spinning vortex (or whirlpool) in the casing. The spinning fluid is capable of maintaining a pressure gradient between its center and its outer periphery and this pressure gradient would be used to pump the coal and water mixture into the downstream pipeline. The casing is shaped to convert a portion of the velocity head into static pressure.

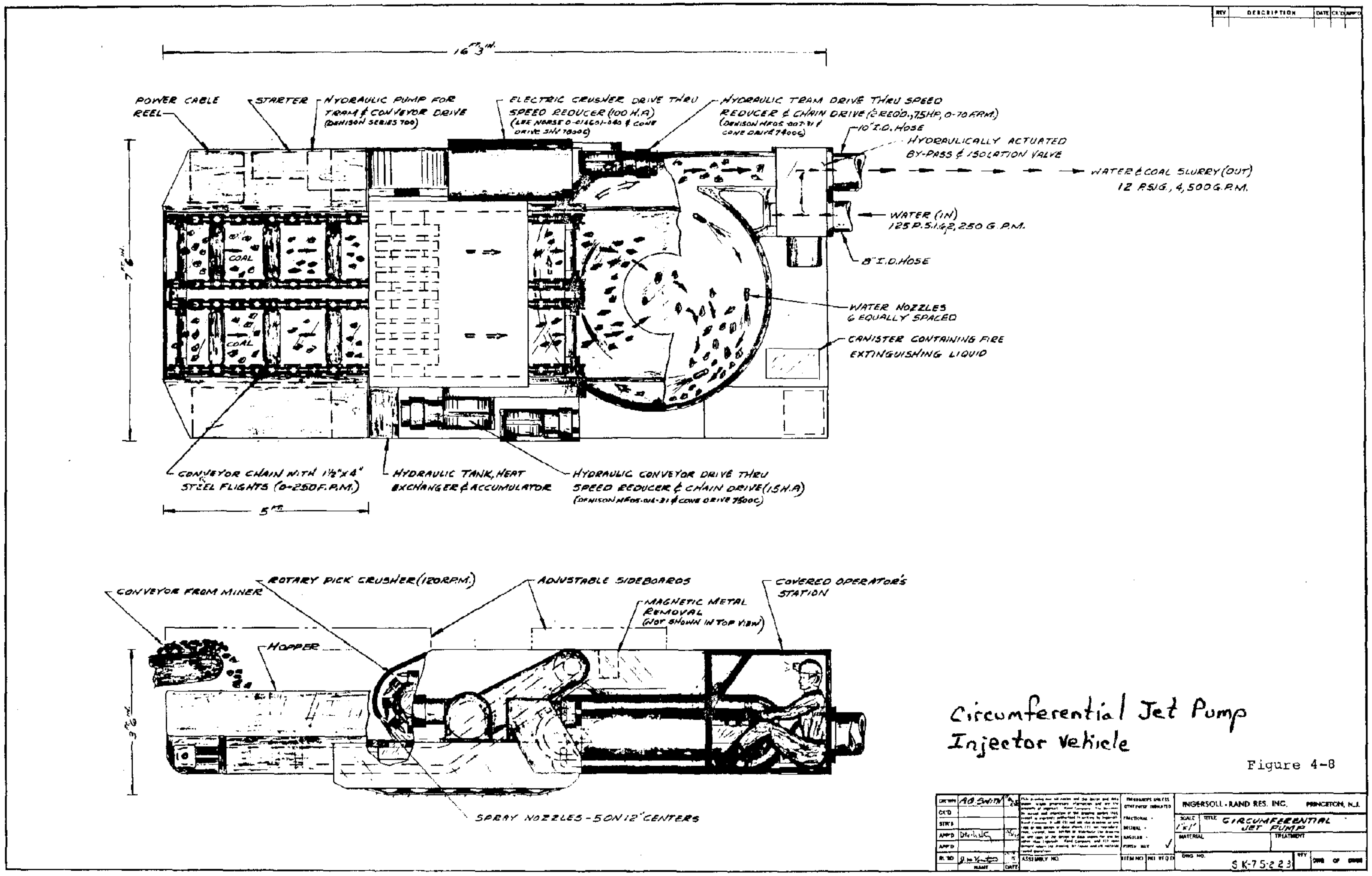
The overall injector vehicle is shown in Figure 4-8. The conveyor from the miner dumps the coal into the injector vehicle hopper. The chain conveyor pushes the coal through the crusher and conveys it into the casing hopper. The device can handle +4" lump coal and hence the crusher requirements are less critical.

A 100 horsepower electric motor operates the crusher (through a speed reducer) and a hydraulic pump to power the chain conveyor and the tram drive hydraulic motors. The electric power hook-up is similar to that for the in-line jet pump concept. The coal falls into the casing center by gravity, mixes with the water jets and the slurry is then conveyed into the pipeline. Analytical and experimental studies were made of the expected performance.

In effect, three separate efforts were undertaken to establish the suitability of the concept under no load and full load conditions, and the performance level that may be attained when pumping coal up to the specified rate. Details of the analysis and experiments are included in the Phase-ending Report for Phase I.

-Vortex Analysis

It is possible to describe analytically the behavior of driving the vortex under various conditions of loads. This analysis requires the satisfaction of the angular momentum laws in the mixture. It requires representation of the wall shear forces and the simulation of various rates



Circumferential Jet Pump Injector Vehicle

Figure 4-8

and distributions of water and coal as they penetrate the central area of the vortex.

This analysis, which is based entirely on theoretical grounds, shows that a pressure rise of 15 pounds per square inch would appear realizable in the full size, full flow design. It also shows that the vortex will operate such that the water inner surface will move inward, as the coal quantity is increased. If the coal quantity is decreased, the hose pressure drop also becomes less and it becomes possible to decrease the driving jet flow rate by throttling so as to maintain essentially constant discharge concentrations.

-Experimental Study

In order to verify the practical behavior of this design and verify the level of the loss coefficients, an experimental, circumferential ejector was built using a pump volute as a casing. Eight internal water jets were positioned on radial arms from the center.

The following observations were made:

- At no load, and with open discharge a powerful air water vortex was found to exist. The generated pressure rise was low, because the inside vortex surface was pushed outward and because the air water mixture exhibits a low mean density.

- As the back pressure was raised, the vortex became better behaved. It ingested less air, if any. The casing filled up with water and the discharge pressure rose. With the laboratory set-up and with a water pressure at the nozzle of 39 psig and flow rate of 83 gpm, a maximum pressure rise of 2.8 psi was obtained without any secondary flow.

- The vortex behaved well when the load was increased by pouring water, sand or coal into the inlet. In fact, it was found that increasing stability was reached at flow rate ratios (secondary to primary) up to approximately .5. Feed limitations external to the test set-up did not permit increasing this ratio to 1.0.

o Data Correlation

On the basis of these data it was possible to verify the analytical modeling to insure that it predicted properly the pressure rise from the test configuration.

The model discussed in the Phase-ending Report for Phase I involved a linear velocity vs. radius curve from which everything else could be calculated. The test data provided the means for obtaining a measure of the losses and the nature of the velocity distribution.

For the test apparatus, the model which provided satisfactory correlation was based on standard drag estimates for the nozzle arms and casing sidewalls. The method correctly predicted the effect of changing flow rates and the addition of the secondary fluid.

One may extrapolate this information to the configuration of a clean design capable of handling the full flow of the application under consideration. Under these conditions the analysis projects a pressure rise of 12 psi under fully loaded conditions (12 tons/min. of coal) with the slurry concentration of 50% by volume. This pressure would convey the slurry approximately 250' from the face where it would be delivered to the intake of a commercial centrifugal slurry pump in a manner similar to that described for the in-line jet pump subsection.

The overall dimensions for this injector vehicle concept are 16 1/4'L x 7 1/2'W x 3 1/2'H and the weight is approximately 13 tons.

Except for its low efficiency, this configuration was the most attractive of all the devices considered. It is simple, small, light and easy to maneuver. It has no moving parts, no heavy electric cables and no sealing problems. It can take +4" lump coal. The initial cost of the device is low and it is readily and inexpensively repaired and serviced.

The projected performance (pressure rise) is less than that theoretically attainable in an In-line jet pump. However, it is possible that increases in performance could be reached by improved radial location of the primary jet nozzles and by optimum sizing of the casing diameter and height. The additional analytical and experimental work required to optimize the indicated parameters was beyond the scope of work for Phase I and was not undertaken.

4.2.5 Objective Comparison of Concepts Against Contract Design Criteria

The four injector vehicle concepts described were compared with the design criteria listed under Section 1.4.0 of the contract which are presented in Section 3.0 of this report. It should be recognized that some of these are of such a nature that they can be evaluated completely and realistically only after detailed design has taken place which was beyond the scope of this program. However, for the remaining criteria it is a matter of designing the unit to meet the criteria. Nevertheless, the comparisons were made and the results are presented in Figure 4-9. The design criteria are listed with brief, evaluative descriptions given as a function of the pertinent section number of the contract. Wherever possible the concepts were rated poor, fair, good or excellent. Appropriate remarks are also included.

4.2.6 System Considerations

The most desirable system was considered to be one which could move the coal from the face at the same rate as it is produced by the continuous miner. If this could be accomplished the overall mine productivity would be maximized. Further, that system which required the least number of components and the lightest weight equipment at the face would increase the mobility and maneuverability of the system and hence its productivity. A mine face area is generally congested and the cut coal should be moved expeditiously from the area to the next transfer point some distance away. The shuttle car system which is the predominant method for moving coal from the face at this time is limited to about 500'. A common basis was used in comparing the various categories of hydraulic face haulage systems. They were narrowed down to five basic systems, capable of representing any injector configuration. All systems include provisions for handling the hose. However, a system which has a compact injector device can incorporate a hose handling device on the same vehicle. Each vehicle requires its own operator. The systems can further be categorized into one step or a split system depending whether the final pressure is attained in the injector or whether a booster pump is required.

The one-step system is possible for all positive displacement injector devices. The water is introduced at 125 psig and the slurry is delivered at the same pressure. But the injector device is heavy and requires considerable

DESIGN CRITERIA	SECTION NUMBER	DESCRIPTION	SINGLE SLIDING VANE	DOUBLE PISTON	IN-LINE JET PUMP	CIRCUMFERENTIAL JET PUMP
Application	1.4.1.1	Adaptability to other mining systems	fair	fair	good	good
Overall Dimensions	1.4.2.1	Overall dimension (length x width x height)	25½' x 9' x 3½'	25½' x 9' x 3½'	24½' x 7 3/4' x 3½' (with tail)	16½' x 7½' x 3½'
Mobility	1.4.3.1	Self-powered or towable	Self-powered	Self-powered	Self-powered	Self-powered
	1.4.3.2	Ability to follow continuous miner	fair	fair	good	excellent
	1.4.3.3.	Ability to travel on difficult terrain	good	good	good	good
Capacities	1.4.4.1	Coal feed rate (Tons/min.)	12	12	12	12
	1.4.4.2	Maximum coal size	4", possibility of handling-6"	-4"	-4"	4", possibility of handling -6"
	1.4.4.3	Ability to handle refuse	fair	poor	good	good
	1.4.4.4	Protection from metal scrap ingestion	good	good	good	good
	1.4.4.5	Adaptability to different size pipes	good	good	good	good
	1.4.4.6	Ease of clearing jams	fair	fair	good	excellent
	1.4.4.7	Higher pressure capability	good	good	fair	poor
Controls	1.4.5.1	Simplicity in control	fair	poor	good	good
	1.4.5.2	Concentration control	good	good	fair	fair
	1.4.5.3	Coordinating multiple injections	good	good	good	good
Service & Economy	1.4.6.1	Service	poor	poor	excellent	excellent
	1.4.6.2	Need for fine adjustment	fair	fair	good	good
	1.4.6.3	Reliability	fair	fair	excellent	excellent
	1.4.6.4	Wear & abrasion resistance	poor	poor	good	good
	1.4.6.5	Preventive maintenance	fair	fair	good	good
	1.4.6.6	Use of standard parts	good	good	good	good
	1.4.6.7	Capital & Operating Cost* ($/Ton)	0.38	0.38	0.23	0.23
Safety	1.4.7.1	Permissibility Requirements	good	good	good	good
	1.4.7.2	Ability to fail in safe manner	fair	fair	good	good
	1.4.7.3	New hazard	Electric power cable	Electric power cable & noise	Slurry back flow if gravity feed	Slurry back flow

Figure 4-9: Objective Comparison of Concepts, Against Contract Design Criteria

electric power at the face. There will be a need for a minimum of two operators, one for the injector and one for the hose handling device. The mobility and maneuverability of this system is poor.

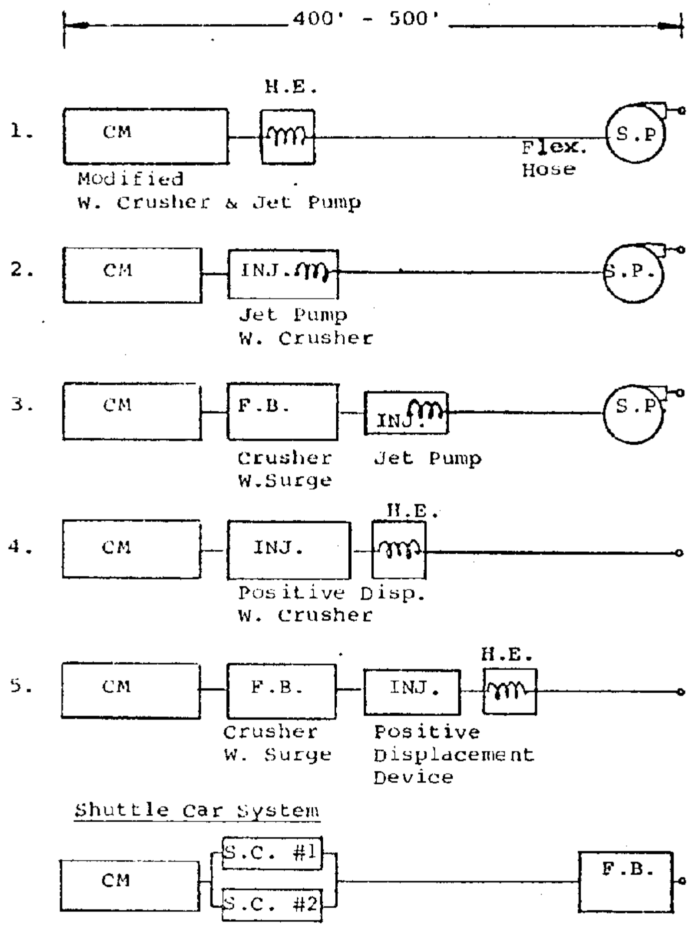
The split system consists of a jet pump injector which takes 125 psig water, mixes the coal and delivers the slurry at low pressure, but still enough to convey the slurry approximately 500 ft. The slurry is introduced into a skid mounted commercial centrifugal slurry pump which boosts the pressure to the final value. The booster pump is movable and is relocated periodically as the face advances. This system can be light and compact. It can alternately combine a hose handling device with the injector on the same vehicle. This will reduce the labor requirement and achieve superior mobility and maneuverability. It requires, however, more total electric power than the one step system.

Five typical hydraulic face haulage systems were analyzed in terms of comparative initial and operating costs and the results are shown in Figure 4-10. In addition, the shuttle car system is included for comparison. The cost shown includes only those costs which are associated with haulage.

The information in the Bureau of Mines 1975 Information Circular #8682 entitled "Basic Estimated Capital Investment and Operating Costs for Underground Bituminous Coal Mines" was used as the cost data source. The following assumptions were made for purposes of system analysis:

- The continuous miner sections produce 344 tons of coal per unit per shift with the shuttle car system. Thus, an output of 227,040 tons per year per section is achieved. The output for hydraulic haulage systems is assumed to be 500 tons per shift and 330,000 tons per year per section. A 50% increase over the shuttle car system is felt reasonable. This figure is consistent with the maximum tonnage capabilities in view of the fact that this injector is adaptable to other forms of mining as well.

- The operator wage is $55 per day. In addition, it costs $45 per day (40% of payroll) for overhead and union welfare benefits for each man. The mine operates 3 shifts per day, 5 days per week, and 220 days per year.



CM-Continuous Miner
Inj-Coal Injector
S.P.-Slurry Pump
F.B.-Feeder Breaker
S.C.-Shuttle Car
H.E.-Flexible Hose Handling Device

No. of Man Required	Labor Cost K	Added CM Cost-K	Injector K	F.B. & Surge K	Slurry Pump K	Added Op. Cost-K	Shuttle Car System Equip Cost-K	Total Cost K/Year	$/Ton
1	66	25	-	-	6	5	-	102	.20
1	66	-	40	-	6	5	-	117	.23
2	132	-	30	20	6	5	-	193	.38
2	132	-	60	-	-	-	-	192	.38
3	198	-	50	20	-	-	-	268	.54
2	132	-	-	-	-	-	40	172	.57

Output(Hydraulic System)-500 tons/shift
Output(Shuttle Car System) - 344 tons/shift
Figure represents only haulage cost.

Figure 4-10: System Analysis Summary

-The equipment cost reflects the amortization over four years, spare parts and repairs.

-The electric power cost is 2 cents per kilowatt hour.

Each of the five hydraulic face haulage systems that were analyzed is briefly described below to assist in the understanding of their mode of operation.

System No. 1

This concept is a split system requiring some modification of the continuous miner. The cutting drum is designed to cut the coal to a size that reduces the crusher load. The crusher and the injector form an integral part of the miner. Also, the selected injector device must be of light and compact design, such as the jet pump. The system is most attractive since it requires only one operator for the haulage function. However, contract requirements did not allow the consideration of modifications to the existing miner. In any case the system is included for future consideration should that retraint be removed at a later date.

System No. 2

This concept is also a split system, consisting of a compact injector device (such as the jet pump) matched with a crusher jointly mounted on a single and separate vehicle. As before, the slurry is raised to a pressure of 100 psig with a booster pump. Since the injector unit has the possibility of incorporating a hose handling device it is assumed that it will require only one operator.

System No. 3

This system is the same as System No. 2 except that it includes a feeder breaker unit having a 9-ton surge capacity between the continuous miner and the injector. The injector vehicle would be designed without a crusher and for a lower capacity of approximately 9 tons per minute. Thus, the injector cost will be less than that for System No. 2. However, it requires an additional operator.

System No. 4

This one-step system consists of a positive displacement injector device designed with a crusher. The injector device in this case is fairly complex and expensive compared to the jet pump devices. This system will require two operators.

System No. 5

This system is the same as System No. 4 except that a special design feeder-breaker unit having 9-ton surge capacity has been placed between the continuous miner and the injector. The injector vehicle would be designed without a crusher and for a low capacity of approximately 9 tons per minute. Thus, the injector cost will be less than that for System No. 4. However, this system requires an additional operator.

Shuttle Car System

For the purpose of comparison a system involving shuttle cars was also considered. This system requires two shuttle cars and two operators for a production rate of 344 tons per shift. Typically, the shuttle car requires more repair and maintenance than other underground equipment. The amortization, spare parts and repairs for two shuttle cars is estimated to be $60,000 per year. On the other hand, the absence of the hose results in a $20,000 per year saving.

In addition, the feeder-breaker of the shuttle car system is traded off against the flexible hose handling device, since the costs of these two pieces of equipment are approximately the same. Putting these figures together the shuttle car system equipment cost per year is shown to be $40,000.

The electric power cost was assumed to be the same as for System No. 4.

Study of Figure 4-10 indicates that System No. 1, utilizing a modified continuous miner, is the least expensive to operate on a dollars-per-ton basis. Since the contract did not permit the consideration of systems requiring miner modification, System No. 2 was the next choice. Further, from Figure 4-10 it can be seen that all of the hydraulic face haulage systems are less expensive to operate than the conventional shuttle car system on a dollars-per-ton basis.

4.2.7 Injector Concepts Evaluation

The four selected coal injector concepts discussed earlier were evaluated in detail, using the same two methods employed during the screening process. The evaluation criteria and their relative weights for the point scoring method were established and are shown in the first two columns of Figure 4-11. The remaining columns show the points scored by each of the four concepts. The results of both methods compared very well. Some general comments, related to each of the evaluation criteria noted on the figure, follow below.

First, mobility and maneuverability at the face are two of the most important aspects of any underground mining operation. In this application the factors that effect the mobility and maneuverability are: 1) number of vehicles, 2) the size of the electric cable, 3) handling of hoses, 4) vehicle, and, 5) dimensions and weight. It is essential that the number of vehicles behind the continuous miner be kept to a minimum. It should be stressed that all of the injector concepts under considerations can also be used as independent units without support vehicles. The positive displacement concepts will require 125 psig hoses for both water and slurry. The water hose is smaller in diameter but the slurry hose is large. However, the jet pump concepts will require a 125 psig hose only for the water, the slurry hose being low pressure. The lighter and smaller the injector vehicle the easier it is to move and maneuver. Both the jet pump concepts are smaller and approximately 40 to 50% lighter than the positive displacement concepts. All these factors directly related to productivity.

The availability of the injector vehicle is the second most important factor. Some of the important factors that affect availability are the time required for servicing, (as influenced by the size and weight of component parts), the overall design complexity of the vehicle, the susceptibility of critical parts to abrasion, seal and bearing failures, and, finally, problems associated with controls, electric and instrumentation. Both of the positive displacement concepts, in general, will have more component parts and are inferior in all respects. The jet pump concepts, on the other hand, have few moving parts subject to wear and access to all components for service is easier.

The economic ratings are derived directly from the study previously summarized in Figure 4-10. Each selected concept may be associated with any particular system of Figure 4-10. On this basis, the ratings of Figure 4-11 can be expressed as shown.

In recent years safety and health in the underground mining environment have been receiving wide attention in the coal mining industry. For example, the hazards associated

CRITERIA	ASSIGNED WEIGHT (POINTS)	SINGLE SLIDING VANE	IN-LINE DOUBLE PISTONS	IN-LINE JET PUMP	CIRCUMFERENTIAL JET PUMP
1. Mobility & maneuverability	60	20	20	40	44
a) Number of vehicles at face	30	10	10	20	20
b) Overall dimensions & weight	12	4	4	8	12
c) Ease of handling hoses	12	6	6	8	8
d) Handling of electric cable	6	0	0	4	4
2. Availability	50	15	10	33	38
a) Ease of clearing jams	15	4	4	7	10
b) Overall complexity	10	3	0	7	8
c) Abrasion & wear	10	4	2	8	8
d) Sealing and bearing	10	3	3	8	9
e) Electrical equipment & instruments	5	1	1	3	3
3. Economics*	35	17	16	26	26
a) Initial investment	10	6	6	8	8
b) Labor for operation and control	10	4	4	8	8
c) Labor for servicing, maintenance & repair	8	3	2	6	7
d) Spare parts	4	1	1	3	3
e) Electric power	3	3	3	1	1
4. Safety & Health	25	14	13	19	17
a) Hazard due to electric cables & motors	7	2	2	5	5
b) Hazard of pressure hose failure	7	3	3	5	5
c) Water back up & flying lumps of coal	5	5	5	3	1
d) Hazard of control system failure	2	1	1	2	2
e) Dust generation	2	1	1	2	2
f) Noise level	2	2	1	2	2
5. Ability to Generate Pressure in various Systems	15	15	15	10	5
6. Cost and Time for Successful Development	10	5	5	5	5
7. Adaptability to Other Mining Methods	5	4	4	5	5
TOTAL POINTS	200	90	83	138	140
Percentage (%) Points	100	45.0	41.5	69.0	70

Figure 4-11: Concepts Evaluation with Point Scoring Method

with the use of electric cabling represent one of the most frequent causes of underground accidents. Concepts such as jet pumps are most attractive because they have a more favorable cable arrangement. Another problem is pressure hose failure that can cause injuries and mine flooding. The higher the pressure level the more serious the problem may be.

Next, ability to generate pressure is a very important evaluation criteria for the practical system. The practical system in this sense is defined as one that has an inlet water pressure of 125 psig and yields a discharge slurry pressure high enough to transport the slurry away from the face a distance of at least 500 feet. The circumferential jet pump was considered to be sufficiently well understood with regard to its capability to whether it generates sufficient pressure to transport the slurry for 500 feet. Accordingly, it was ranked as poor with respect to this evaluation criteria.

Finally, the cost and time for successful development and adaptability of the injector to continuous, conventional and longwall mining methods were also utilized as evaluation criteria.

4.3 Phase II - Design And Engineering

4.3.1 Introduction and Summary

In 1976 Ingersoll-Rand Research, Inc. completed the Phase I state-of-the-art survey and design study on hydraulic face haulage injector concepts suitable for use in 4 ft. seam height, delivering a 12 TPM injection rate with coal having a maximum lump size of 4 in. A jet pump injector, capable of receiving and processing dry coal, was recommended as the preferred concept on the basis of the results of this study. This recommendation was made on the basis of size, weight and adaptability, ruggedness, ease of maintenance and adaptability to a multiplicity of system configurations.

Further, it was also recommended in Phase I that, prior to performing a full size injector design, a subscale model study should be made to include both operational model testing and analytical modeling to describe the physics occurring in the injector. Only limited information was available from the literature to facilitate the design of a jet pump handling the specified coarse dry solids feed and the unprecedented high concentration. Vital information regarding performance, control and wear was unavailable.

The objective of the complete model study was to (1) determine the essential design and performance parameters and (2) demonstrate the injector's feasibility including the evaluation of all potential operating problems that could be addressed in the test loop environment.

The operational model testing was to be conducted in a closed loop test facility capable of processing one twelfth (1/12) of the full scale vehicle flow rate. Accordingly, the injector and its test loop were designed to the following upper limit specifications:

Coal Size	:	3/4"
Coal Rate	:	1 Ton/Min.
Water Flow Rate	:	250 GPM
Supply Water Pressure	:	125 PSIG
Discharge Line Size	:	4 Inches
Discharge Pressure	:	25 PSIG

A test loop was designed and constructed so as to maintain the coal size and control the fines that naturally build up in a closed loop slurry system. These fines were removed by using a vibrating dewatering screen and a cyclone. A narrow size range of coarse coal was continuously added to replace the fines removed from the closed loop.

An analytical model was developed to describe the interaction of the water jet and the coal lumps which is significantly different from more conventional slurry jet pumps having a much lower feed concentration. The model was used to size the subscale jet pump, permit larger size scaling, and study the physics of the energy exchange. As subscale test data became available, they were used in turn to refine the basic model.

The test models included three different injector concepts which were designed with the assistance of the analytical model. The first was a vertically oriented injector which was tested to provide a basic understanding of the jet pump injector concept without the complications of a feeding mechanism. The second concept used a feed screw to feed the coal uniformly to the region upstream of the injector throat. It was horizontally oriented, as was the third concept, which was originally conceived to be gravity fed using multiple jets. However, early in the design of this concept it was recognized that multiple jet designs have reduced efficiencies as compared with single jet designs and thus, the concept was modified to use only one jet. All the jet pump components (throats, nozzles, diffusers) were designed to be interchangeable with all three concepts.

In summary, the following sections of the present report describe the experimental and analytical work performed at the subscale level. The feasibility of the jet pump injector for the intended service was demonstrated. Further, an analytical model describing the operation of the jet pump from the high pressure water nozzle through the slurry diffuser was developed. The latter model was adjusted in accordance with experimental results, leading to a very useful tool for subsequent scale-up of the jet pump injector to full size in Phase III.

4.3.2 Phase IIA - Jet Pump Model Study

4.3.2.1 Development of Model Jet Pump Injector

This section of the report describes the activities associated with the subscale model jet pump injector work. The test system, test measurements and injector designs are described. Next, the test sequences utilized to optimize performance and demonstrate life potential are discussed.

Finally, the test results achieved are reviewed.

4.3.2.1.1 Test System

A closed-loop test facility was designed and fabricated to meet the following program objectives for the subscale jet injector:

- Feed coal or refuse (stone or gravel) to the injector at uniform but controlled rates up to 1 TPM.
- Supply water to the injector model in excess of 250 GPM at 125 PSIG.
- Control the injector discharge pressure.
- Dewater the slurry discharge to achieve a maximum 10% surface moisture on the recycled feed entering the injector.
- Control the size of the solids entering the injector.
- Control the build-up of fines in the closed water loop.

The test facility, followed by a legend describing the various elements of the system, is depicted schematically in Figure 4-12. This facility met all of the program objectives enumerated above.

This system included seven major subassemblies having the following functions:

- <u>Vibrating Dewatering Screen</u> (S1): Removed fines (<10 mesh) and water from the recycled solids feed. Removed fines coarser than 40 mesh from the water. Fines smaller than 40 mesh passed through the dewatering screen with the effluent.
- <u>Water Tank</u> (reservoir T1): Received the water and minus 40 mesh fines as they left the dewatering screen.
- <u>Centrifugal Pump</u> (P1): Drew suction water from the tank and supplied high pressure

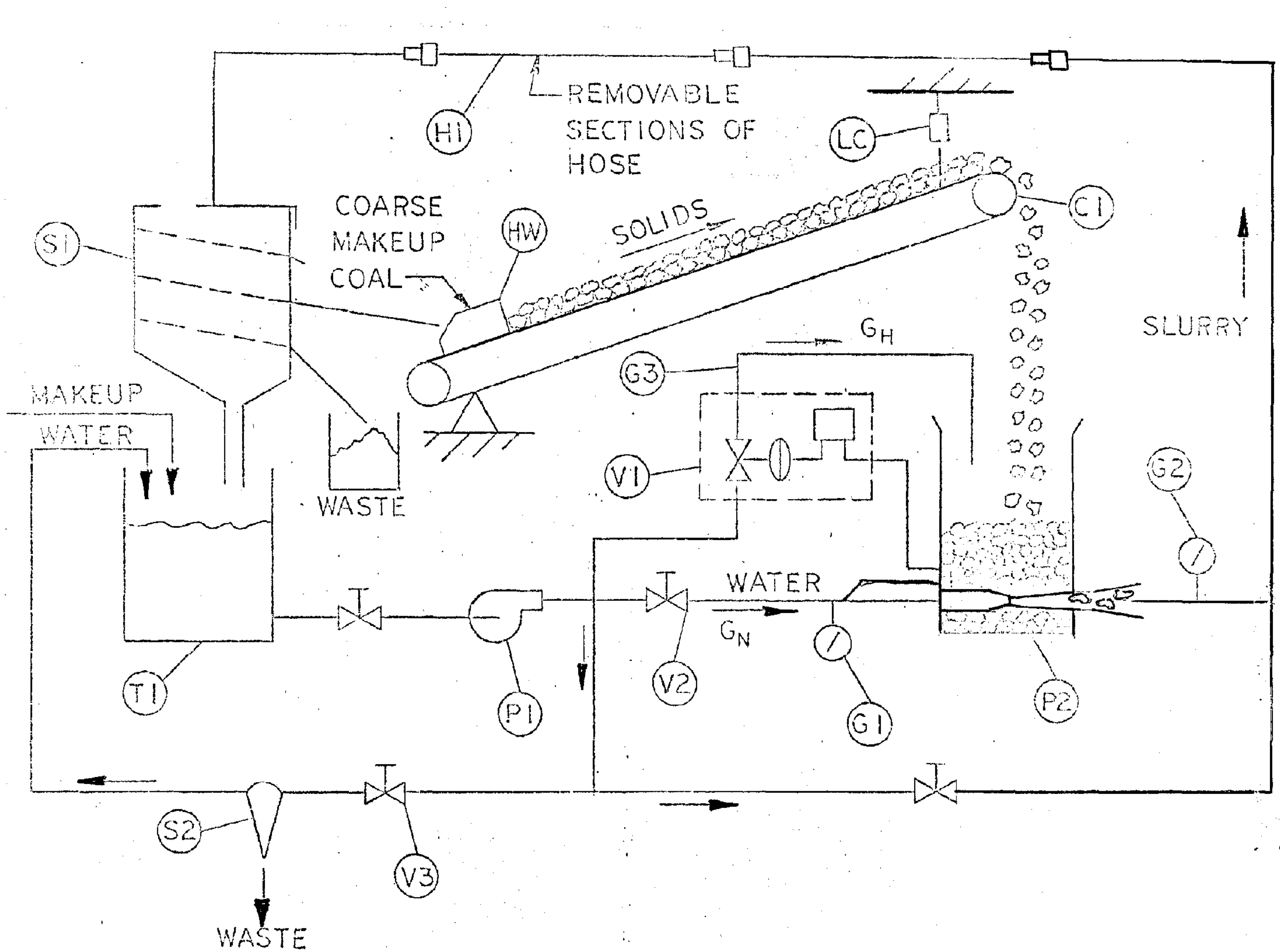


Figure 4-12: Schematic of the Sub-scale Jet Injector Closed Loop Test Facility for Coarse Coal Slurry

Legend for Test Loop Equipment (Ref. Fig. 4-12)

C1 - 24" wide reversible conveyor
26' long, 60 TPH
Manufacturer unknown

P1 - Centrifugal pump, 50 Hp
3 x 2 x 9 HC, 3550 RPM
Ingersoll-Rand, Inc.

P2 - Fabricated jet pump

S1 - Vibrating dewatering screen
Size 3' x 6' triple deck
Prater, S/N 1088

S2 - Cyclone separator, 125 GPM, 75 PSIG
Model No. D6B-12-878
Krebs

T1 - Water storage tank and support for Prater separator
Capacity of 750 Gals.

H1 - 3" Goodyear concrete hose

HW - Fabricated solids holdup weir

V1 - Automatic valve and level controller
1½" 1051-V100-4162
4162R-216 Controller with valve positioner
Fisher Controls

G1 - Pressure gage, 6" face, 0.5% error
0-160 PSIG, Spec. 1129
Span

G2 - Pressure gage, 6" face, 0.5% error
0-30 PSIG, Spec. 1126
Span

LC - Load Cell
SJ-F 3/8-16 NC-4"
350 OHM
Measurement, Inc.

water to the injector nozzle, cyclone separator, injector discharge line and hopper water level controller.

- Cyclone (S2): Removed fines from a portion of the total pump discharge.

- Surge Volume (HW): Mounted on the belt conveyor, this volume accommodated changes in solids hold-up in the injector feed hopper slurry line and dewatering screen. This device provided an important secondary function of producing a uniform cross section of solids (loading) on the conveyor belt.

- Belt Conveyor (C1): With a variable speed drive this subassembly transferred the solids from the dewatering screen to the injector feed hopper.

- Hopper Water Level Controller (V1): Sealed the injector throat and prevented air from being ingested.

During operation the discharge pressure of the injector could be adjusted in the following ways:

- By pinching the injector discharge hose with a clamp which restricted the flow.

- By introducing extra water into the injector discharge line, thus increasing the flow and pressure drop.

- By changing the length of the injector discharge line.

The first method was only used for tests with gravity where the maximum particle size was considerably smaller than with coal. The use of the second method was found impractical because of the effects of feed back on system pressures when a large amount of water was drawn from the centrifugal pump to change injector back pressure. The third method was most commonly used with coal because it was difficult to maintain steady pressures with the other methods.

4.3.2.1.2 Test Measurements

The following measurements were taken to determine the performance of the injector:

- Nozzle pressure was measured using a calibrated bourdon tube pressure gage.
- Nozzle flowrate was determined by nozzle supply pressure since the nozzle flow rate was calibrated against nozzle pressure.
- Injector discharge pressure was measured using a calibrated bourdon tube pressure gage. This reading was checked with a calibrated pressure transducer.
- Solids feed rate was calculated from measured belt loadings (LB/FT) and clocked belt speeds. The calculated feed rates included the surface moisture on the coal.
- Surface moisture on the injector feed was determined by collecting a wet solids sample and comparing its weight before and after air drying.
- Supplementary hopper water flow was determined by calibrating the flow through a large restriction in the hopper water supply line. Water flowrate was correlated with a pressure reading upstream of the restriction.
- Particle size distribution of the feed was determined by collecting a sample of wet solids as they discharged from the conveyor belt, air drying them, and screening using standard Taylor screens.
- Pressure distribution along the length of the injector was measured using a flexible static pressure traversing probe and a pressure transducer.

4.3.2.1.3 Injector Configurations

Three basic injector concepts were designed and tested, based on the following orientations and feed methods:

- <u>Vertically oriented, gravity fed.</u>
- <u>Horizontally oriented, screw fed.</u>
- <u>Horizontally oriented, gravity fed.</u>

All the concepts were designed to be tested with a fixed nozzle diameter and changes in nozzle/throat (mixing chamber) area ratios were achieved by changing the diameter of the mixing tube. Interchangeability of the jet pump components (throats, diffusers and nozzles), as well as other supporting parts, minimized the number of manufactured parts required.

The jet pump components for the injector concepts featured the ranges for the pertinent geometric design variables shown below.

-Nozzle Diameter, ϕ_N

- 0.878" Solid Core
- 1.050" OD x 0.577" ID Hollow Core

-Diffuser Angle, α_D 4.50, 5.5 , 7.5 , 10

-Diffuser Outlet Diameter, ϕ_D 3", 4"

-Mixing Chamber (Throat) Diameter, ϕ_M 1.625", 1.750", 1.875", 2.000", 2.250"

-Mixing Chamber (Throat) Length, L_M 1.00", 1.75", 3.00", 6.00", 12.00"

-Jet Entrainment Length, L_E Infinitely variable from 0 to 12"

-Mixing Section Inlet Configurations:

- Oblique Cone (Long and Short)
- Low Angle Cone
- Bell-mouth

Any combination of throat diameter and diffuser angle could be assembled using a diffuser transition piece between the outlet of the throat and the inlet of the diffuser.

The nozzle could be positioned anywhere from 0 to 12" from the throat. Specially machined alignment mandrels were used to insure good nozzle throat alignment. One end of the mandrel was positioned in the outlet of the nozzle and had approximately 0.0005" clearance. The other end projected through the socket in which the (mixing chambers) throats were centered and had about 1/8" clearance. The nozzle was positioned until the clearance was uniform between the mandrel and the socket.

Some of the specific design features of the three injector concepts are discussed below:

Vertical Oriented, Gravity Fed Jet Pump Injector

This injector was the simplest of the three concepts. The solids were actually funneled into the throat by gravity as shown in Figure 4-13. The conical feed hopper was fabricated from AISI Type 304 stainless steel to reduce wall friction on the solids, particularly the sticky wet coal.

Horizontally Oriented, Screw Fed Jet Pump Injector

This injector design, as shown in Figure 4-14, is the most complex of the three concepts. Referring to the figure, solids drop into the injector feed hopper where they are conveyed by the screw into the entrainment region. A concentric cone provided a transition zone between the screw outlet and the injector throat (mixing chamber). This cone forced the solids to move radially into the jet.

Two different cross-sections were considered for this transition piece. The simplest, which was tested, was a straight conical shape as shown in the figure. The second shape was nearly parabolic in cross-section and theoretically the solids would have a more uniform radial velocity component as they entered the jet.

Horizontally Oriented, Gravity Fed Jet Pump Injector

This injector configuration is illustrated in Figure 4-15, shows several features which were later found to be unnecessary. The original concept had the solids fall directly into the hopper and slide in a layer down the sloped hopper side walls to the hopper jet. A jet shield was included to protect the jet from direct impingement by the falling coal lumps. The water boxes on the sides were intended to flow a very small quantity of water across the entire bottom surfaces

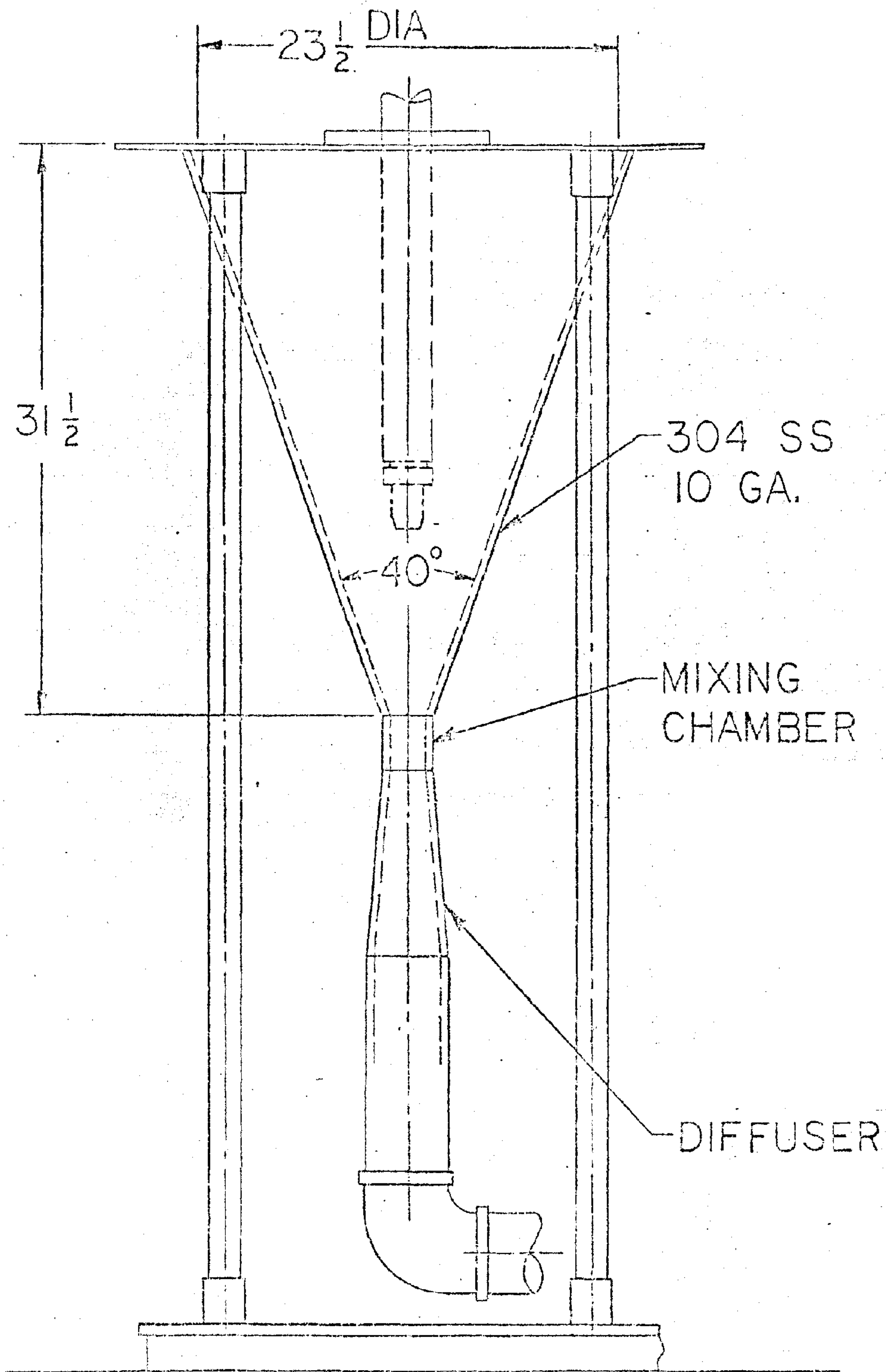


Figure 4-13: Vertically Oriented, Gravity Fed Jet Pump Injector

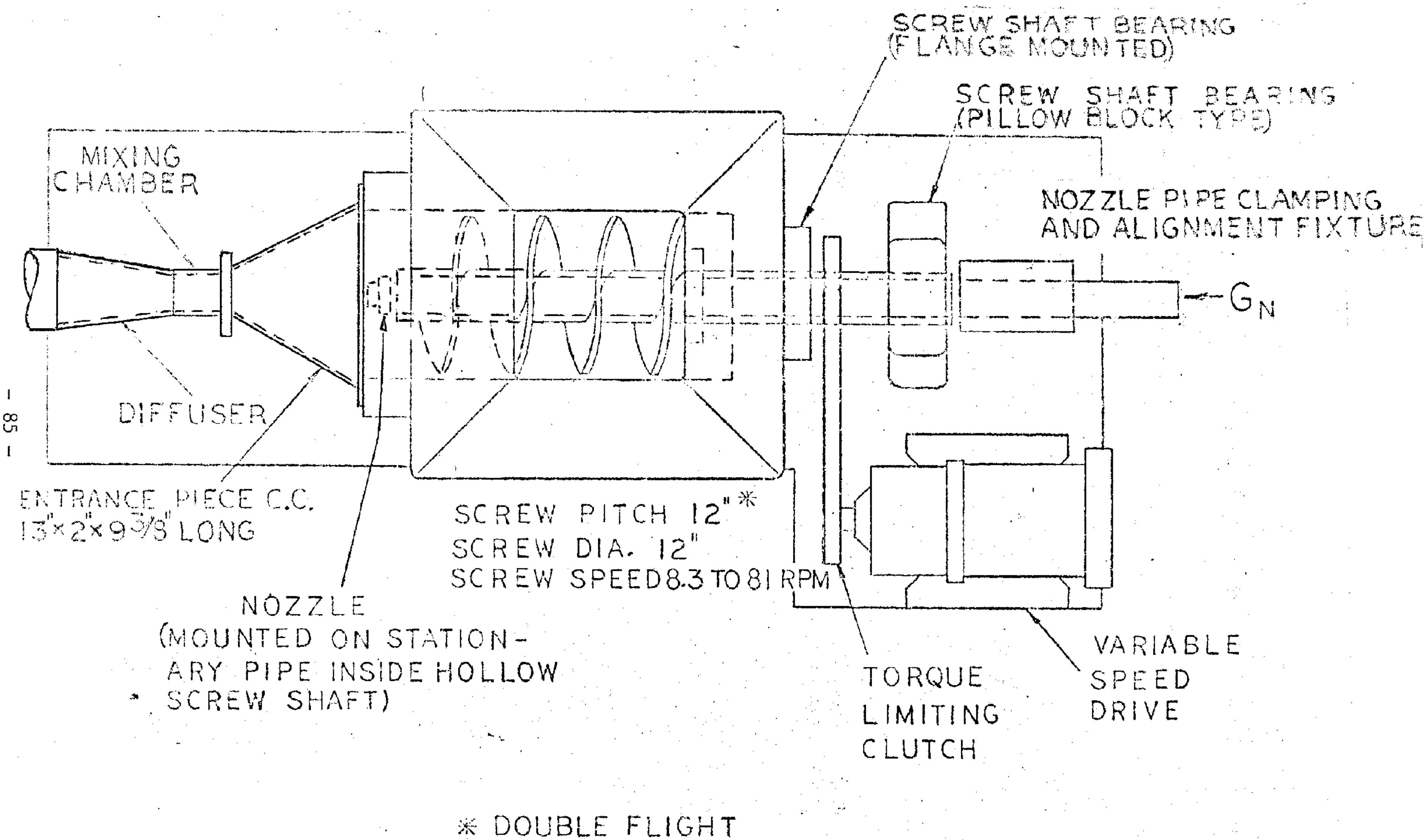


Figure 4-14: Horizontally Oriented, Screw Fed Jet Pump Injector

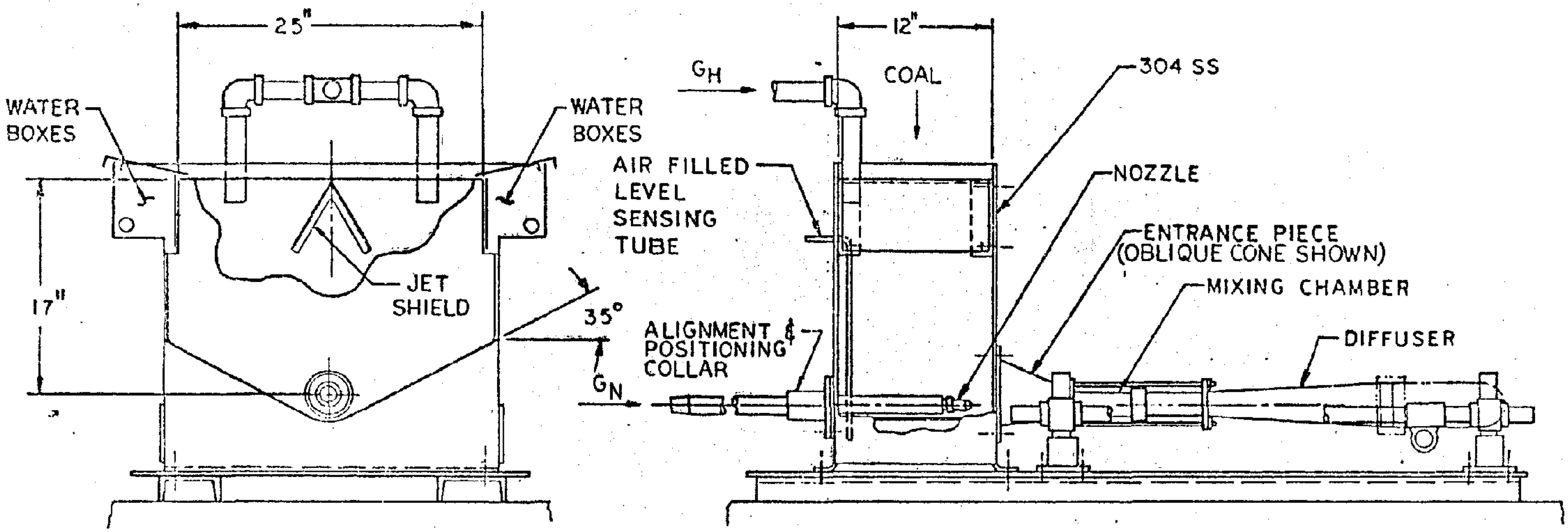


Figure 4-15: Horizontally Oriented, Gravity Fed Jet Pump Injector

of the hopper to assist in flushing the solids into the jet. During the test program it was found that neither of these two features was necessary. This design also used AISI Type 304 stainless steel to minimize the surface friction on the wet coal.

4.3.2.1.4 Sequence of Testing

Preliminary Testing

The objectives of the preliminary testing activity were to arrive at a basic understanding of the injector performance parameters, debug the test loop, and determine which of the jet pump configurations was the most promising. Most of this testing was done with gravel (rather than coal) which was more readily dewatered and less expensive. The gravel approach provided basic data for refuse injection and afforded a cleaner feed material with which to work.

The first tests were done on the vertical injector which provided a basic understanding of the performance parameters for this injector concept. The horizontal gravity fed injector was tested next, followed by the horizontal screw-fed concept.

The injectors were tested under a variety of operating conditions. Various geometrical configurations were also tested and, to a limited extent, some optimization of jet pump geometry was completed during these tests.

Only the significant results of these tests, as well as the others performed in this phase of program activity, are discussed below.

Optimization Testing

The objective of the optimization testing activity was to determine the best geometrical configuration compatible with the specified operating conditions, for the most promising injector concept.

A wide range of injector geometrical configurations and operating conditions were studied. The geometrical parameters that were studied included throat inlets, throat diameters, throat lengths, diffuser angles, diffuser outlet diameters, jet configurations and entrainment lengths. Operating conditions that were varied included nozzle pressure (60 to 140 psig) and capacities from zero (shut-off; i.e. no secondary feed) to the maximum solids feed rate which in some cases was limited by cavitation. Geometry, nozzle pressure, and solids feed rate (discharge pressure) were all varied systematically to obtain complete performance maps. This testing was done on coal feed having a size range of 3/4" x 10 mesh and a specific gravity of 1.58. Supplemental water was added to the feed to maintain a sealed throat condition.

Endurance and Operational Testing

Many potential operating problems were evaluated during this study including those defined in the Work Statement of Phase II and others identified during the model testing. The problems are listed below:

- Wear of critical components.
- Possibility of feed hopper back-flood while receiving dry feed.
- Capability of concept for handling coal and/or refuse.
- Capability of concept for handling continuous or intermittent feed rates.
- Injector behavior under rapid load changes.
- Susceptibility of concept to blockage.
- Control of coal concentration.
- Coal degradation rate.
- Air ingestion.

4.3.2.1.5 Test Results

Performance Testing Results

The preliminary testing work yielded several significant conclusions in addition to establishing the most promising injector concept. These conclusions were:

- The introduction of a small amount of supplementary water in the feed hopper increased the maximum solids injection rates and eliminated air ingestion. This phenomenon was found to apply to all injector configurations.

- A shock phenomenon exists immediately upstream of the injector throat which was found to create a back flow. This back flow migrated out of the throat entrance and into the feed hopper tending to fluidize the incoming "dry" solids. This shock also contributed to a significant portion of the total pressure developed in the injector.

The preliminary testing results indicated that the horizontal, gravity-fed injector concept was superior to the screw-fed concept. The screw-fed concept was abandoned because of 1) operational problems associated with fines breakage and screw jamming 2) inferior pressure-flow performance characteristics and 3) less desirable design and functional features.

As previously stated the final optimization of geometry was conducted on the horizontal, gravity fed injector.

The performance was found to be least sensitive to the following parameters:

- Diffuser angles ranging from 4 1/2 deg. to 10 deg.

- Throat inlet configurations ranging from an oblique conical inlet to a bell-mouth inlet.

- Throat lengths ranging from 1" to 12".

The performance was found to be moderately sensitive to the diffuser discharge diameter which included 3" and 4". The larger outlet produces slightly higher discharge pressures.

The performance was found to be most sensitive to the following parameters:

- Jet configurations which included both solid and hollow shapes.

- Throat diameters which ranged from 1 5/8" to 2 1/4", and nozzle/throat area ratios (R) which varied from 0.293 to 0.152 .

- Entrainment lengths (L_E), which were varied from 3" to 8", and L_E/ϕ_N ratios which varied from 3.42 to 9.11.

The solid jet outperformed the hollow core jet configuration. The hollow jet appeared to be considerably weaker and more susceptible to disintegration in the entrainment zone.

Changes in throat diameter (with a fixed nozzle diameter) produce the characteristic changes seen in typical jet pump (head-flow) family curves for various values of R. The curve for low area ratios produce the characteristic flat performance curves while higher area ratios have steep performance curves. The best range of R appears to be from 0.152 to 0.219.

Changes in jet entrainment length result in a complex interrelationship between 1) the maximum amount of solids that could be delivered, 2) the amount of supplementary hopper water that was required to maintain a sealed jet, and 3) the head-flow relationship, N-M curve. Short entrainment lengths like 3" (L_E/ϕ_N = 3.42) required substantial amounts of supplementary hopper water and the overall N-M performance curves resemble the results obtained for slurry handling jet pumps, like those of References 1, 2, and 3. However, the discharge slurry concentration suffered from the additional hopper water requirement which otherwise was necessary to achieve the highest pumping rates. With long entrainment lengths like 8" (L_E/ϕ_N = 9.11) the supplementary hopper water requirements were reduced substantially due to the disintegration (shedding) of the jet as it flows through the entrainment zone. In some cases the supplementary water requirement was reduced to zero; however, this long entrainment results in an overall performance which is seriously impaired by these energy losses. An intermediate entrainment length of 6" (L_E/ϕ_N = 6.83) was considered the optimum value. It provided a good balance between the solids delivery rate, supplementary water requirements, and the head-flow relationship.

However, if supplementary water can be extracted from the injector discharge then the optimum entrainment length would be reduced since the source of the supplementary water no longer would be the high pressure nozzle supply. This is true since the maximum solids injection rate increased slightly as the entrainment length was reduced but the supplementary water requirements increased substantially. This phenomenon is illustrated in Figure 4-16. If these substantial quantities of water can be extracted from the injector discharge, then

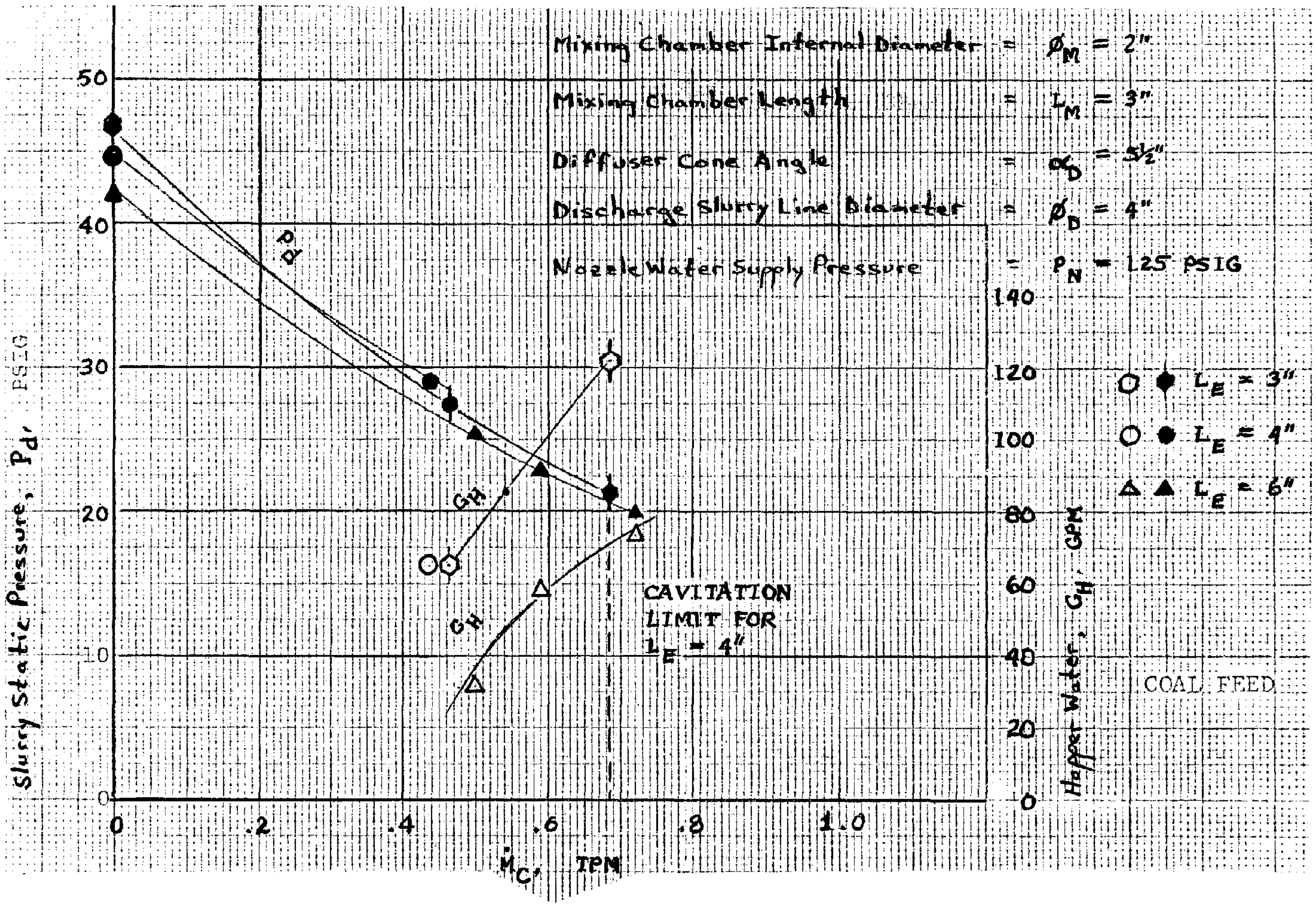


Figure 4-16: Effect of Entrainment Length on Discharge Pressure, Injection rates and Hopper Water Requirements.

higher injection rates can be achieved without the penalty of lower discharge concentrations.

The maximum continuous coal feed rate for the test model at 25 psig discharge pressure and 125 psig nozzle pressure was 0.54 TPM which required an additional 39 GPM of supplemental hopper water. For purposes of this report, the discharge pressure is expressed as a static pressure. The total pressure is approximately 0.7 psi higher. These conditions were obtained with 2" diameter by 3" long injector throat and a 10 deg. x 4" diameter diffuser with a nozzle entrainment length of 6".

Operational Testing Results

Coal Accumulation and Blockage

The accumulation of solids in the feed hopper did not cause the injector throat to block. Most of the tests were conducted with a substantial level of solids over the jet and nozzle without ever plugging.

The injector throat is not prone to blockage by coal lumps having a length to throat diameter ratio in excess of 1.0. Also, other non-coal solids passed repeatedly through the 2.0" diameter throat, including bolts, hose clamp, rocks and a "C" clamp, causing throat blockage. The "C" clamp could pass through the throat only when oriented in one way and then with about .020" clearance, yet it was cycled repeatedly through the loop.

Component Wear

The critical wear component of the jet pump injector was found to be the injector throat. Wear tests with Anthracite coal were conducted on the injector with a throat made of AISI Type 4130 steel hardened to Rc 63-66. The diametral wear rate was found to be 0.15" per 1000 hours. Nozzle and diffuser wear was insignificant.

Adaptability to Refuse Injection

All the preliminary testing was conducted on gravel (refuse). The injector performed better on gravel than on coal because of the higher specific gravity. It can also handle foreign materials such as tramp metal and wood without noticeable damage to the injector.

Back Flooding

Back flooding in the injector is defined as an overflowing feed hopper. This condition was occasionally observed in the test model. However, it was only brought about by abnormal operating conditions imposed on the injector such as when the slurry line plugged or while taking data points at shutoff. Whenever it occurred, the rate of increase in hopper level was always slow enough to allow sufficient time to sense the impending condition and automatically react to it before overflowing occurred.

Coal Degradation (Attrition)

The closed loop test facility was not well suited for isolating the breakage that occurred in the injector itself. However, the total breakage in the test loop could be determined easily by simultaneously collecting and analyzing samples of feed to the injector and fines from the dewatering screen and by knowing the size distribution in the make-up coal. A representative sample, reflecting the total breakage in the system, could be determined by blending these three known quantities mathematically and comparing the resulting particle size distribution with that going to the injector. This shift in particle size distribution is illustrated in Figure 4-17.

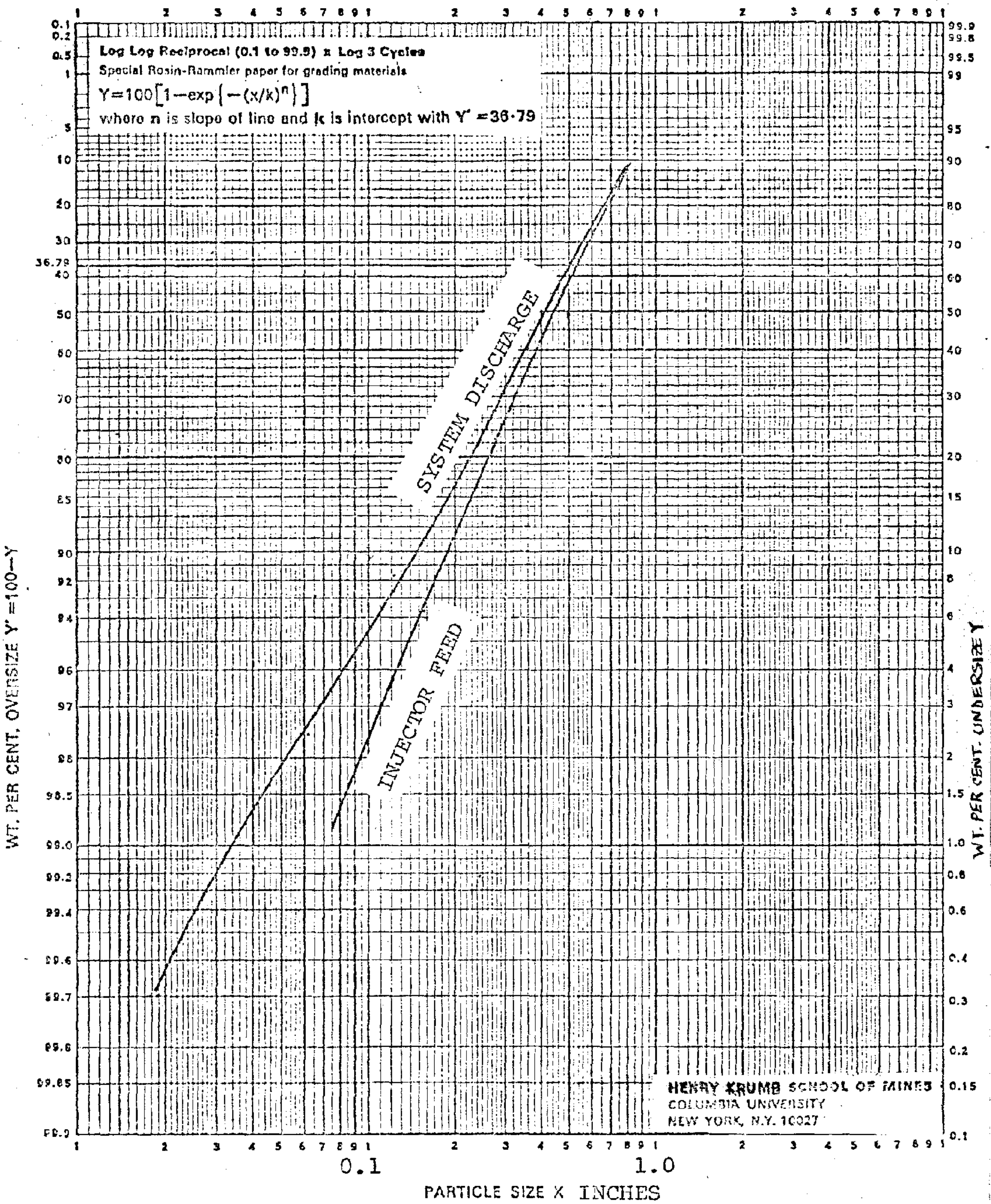
Slurry Concentration Control

This injector has no method for directly controlling the slurry discharge concentration. Generally, at maximum solids injection rates, the concentration increased as the discharge pressure decreased and the solids injection rate increased. The optimum configuration had a discharge concentration of 0.23 by volume when properly designed to operate between $M = 0.65$ and 0.75.

Air Ingestion

Tests indicated that a considerable quantity of air was being ingested into the injector while running unloaded and without the addition of supplementary hopper water to seal the throat. The results achieved are shown in Figure 4-18.

However, air entrainment was eliminated by maintaining a water level in the feed hopper that is above the jet and the throat. During operation this seal water is continuously



Log Log Reciprocal (0.1 to 99.9) x Log 3 Cycles
Special Rosin-Rammler paper for grading materials
$Y=100\left[1-\exp\left\{-(x/k)^n\right\}\right]$
where n is slope of line and k is intercept with Y' = 36·79
SYSTEM DISCHARGE
INJECTOR FEED
WT. PER CENT, OVERSIZE Y' = 100—Y
WT. PER CENT. UNDERSIZE Y
0.1
1.0
PARTICLE SIZE X INCHES
HENRY KRUMB SCHOOL OF MINES
COLUMBIA UNIVERSITY
NEW YORK, N.Y. 10027

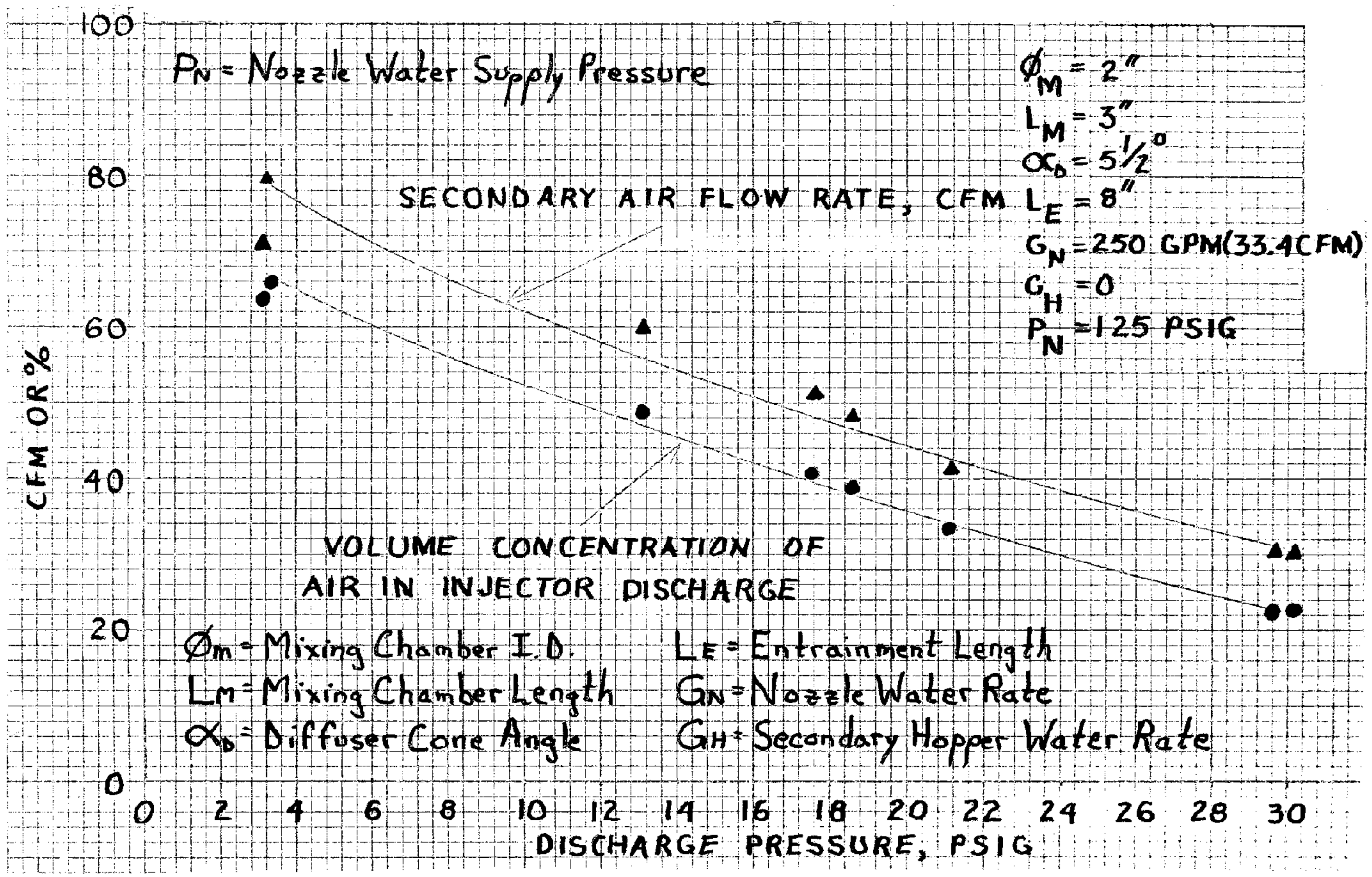


Figure 4-18: Air Entrainment Versus Discharge Pressure with a Solids Feed Rate of Zero

drawn out of the feed hopper by the jet and thus must be continuously replenished by a hopper water level control system. Required supplementary hopper water flow rates are a minimum when the injector is running at full load solids delivery rate and a maximum when the solids injection rate goes to zero.

A hopper water level control system was successfully demonstrated during this study which readily accommodated extreme changes in feed rate to the injector, including a step wise change in the feed rate from 0% to 100% and back to 0% again. This control system, shown in Figure 4-19, senses the water level in the feed hopper and automatically adjusts the hopper water flow rate as required by the injector to maintain a submerged jet and throat condition. The water level is sensed pneumatically by measuring the pressure in an open ended tube which bubbles air into the hopper water. As the water level rises (drops) the air pressure in the bubbler line increases (decreases) and the controller causes the pneumatically operated control valve to reduce (increase) the hopper supplementary water flow rate. This method of water level control proved to be trouble free, non-clogging and very effective.

4.3.2.2 Analytical Modelling

Details of the model and the computer program described below are on file at the Bureau of Mines' Pittsburgh Research Center.

4.3.2.2.1 Description of Model

The purpose of the computerized, analytical model was to define all the important performance parameters and develop a method for predicting injector performance as a function of these parameters. For a given feed rate, nozzle pressure, and injector geometry this model predicts the water and solids velocity history in the injector, the pressure profile, the discharge pressure and efficiency.

A schematic representation of the coal jet pump injector that was modelled is shown in Figure 4-20. The primary fluid is pressurized by a centrifugal pump and leaves the nozzle as a core of high velocity fluid. Due to the high turbulent shear region between the primary jet and the secondary stream of solids and hopper water, incomplete mixing takes place between the two streams in the entrainment region. By momentum exchange and continuity, a continuous expansion of the jet occurs as it flows from the nozzle.

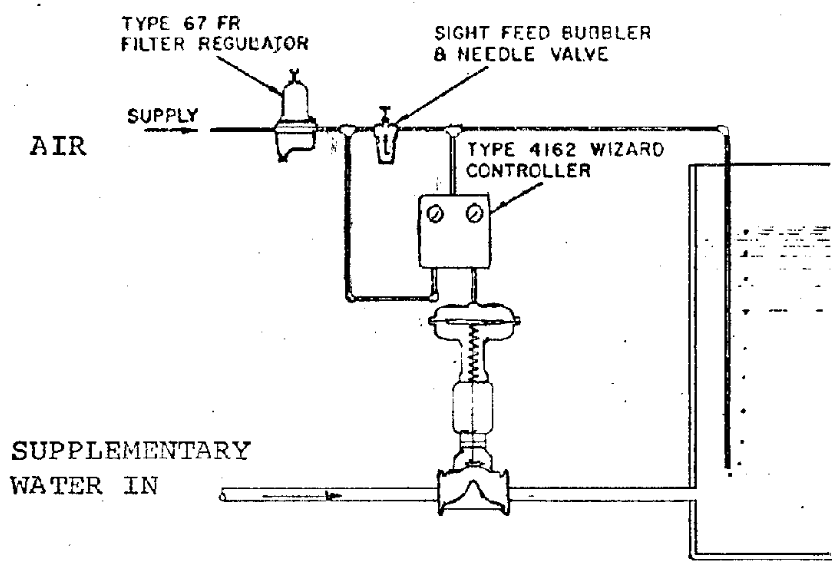


Figure 4-19: Feed Hopper Water Level Control System

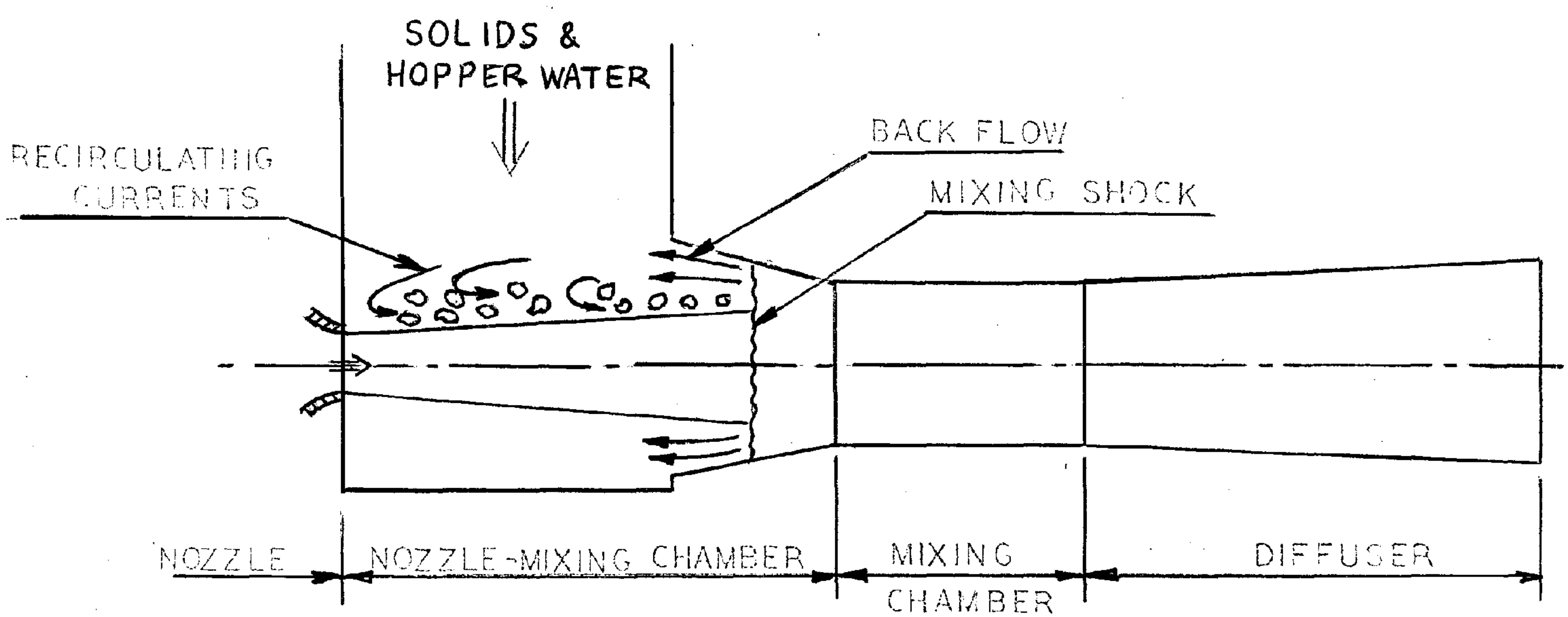


Figure 4-20: Schematic Representation of Coal Jet Pump Injector

The model describes the interaction of the solids and the high velocity jet in the entrainment region. It describes the history of the solids as they pass through the injector and determines the relative velocities between the liquid and solids at each axial position. It takes into account the size, density, and the drag coefficient of the solids and can account for an initial size spectrum and any jet pump geometry. The interchange of momentum between the liquid and solids accounts for the acceleration of the solids and deceleration of the water in the interacting regions.

The model includes a momentum defect factor which is applied to the entrainment region. This factor accounts for additional losses which result from the fact that the jet is operating in a submerged condition. This defect factor contributes to an additional spreading of the jet which is reported in other works such as in Reference 5.

This model includes a pressure shock phenomenon which describes a rapid build up of pressure observed in test data. The model can analyze the performance of the injector with the shock positioned anywhere within the injector, but the position of the shock is established by the jet pump back pressure level. It has the capability of calculating the localized pressure throughout the length of the injector.

4.3.2.2.2 Correlation With Test Data

The analytical model was correlated with test data in the following three ways.

First, the momentum defect factor was correlated under shutoff conditions with entrainment lengths ranging from 4" to 8" and with nozzle pressures which were varied from 60 to 140 psig. In this case the defect factor was adjusted until test data and analytically calculated results were in close agreement.

Secondly, the model was correlated with discharge pressure data for various nozzle pressures, coal flow rates, and hopper water flow rates.

Finally, the model was correlated against test data which determined the pressure profile along the length of the injector. In this manner the model was checked to determine whether or not the shock phenomena model was accurate.

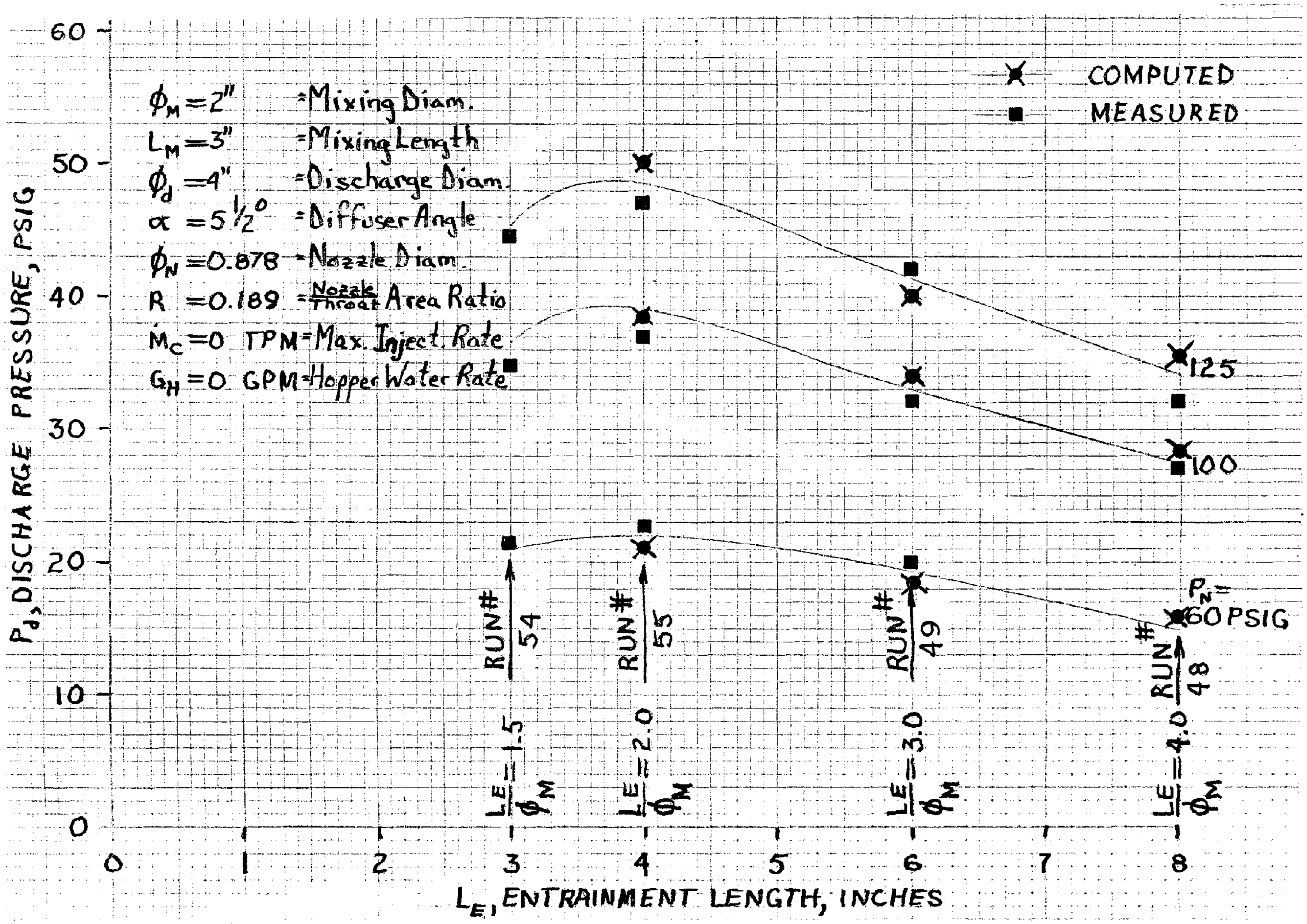


Figure 4-21: Correlation Results at Shut-Off

The results of the correlation at shut-off are shown in Figure 4-21 which indicates that a momentum defect factor of 0.9775 results in excellent correlation over a wide range of entrainment lengths and nozzle pressures.

Table 4-1 illustrates the correlation of injector discharge pressures for a wide range of solids flow and nozzle pressure conditions. The data shown indicates an average deviation of less than 6% between test and calculated results.

Figure 4-22 and 4-23 show pressure and velocity profiles with a shock positioned in the injector throat. Figure 4-22 shows how closely the analytical model is able to predict the discharge pressure.

Since good correlation with test data was achieved over a broad range of test conditions, it was concluded that the model was capable of very accurate predictions of injector performance. With a known injector geometry and nozzle pressure the model is able to predict accurately the discharge pressure for a given coal and supplementary hopper water flow rate. However, it was beyond the scope of the program to analytically predict the amount of supplementary hopper water required for a given operating condition. This information was determined from the physical model data. Except for predicting the supplementary water requirement, the model can be used for scaling the jet pump design or for anticipating the effects of changes in jet pump geometry or operating conditions, such as changes in feed material or supply pressure.

4.3.2.3 Prototype Injector Design and Operating Considerations

The purpose of this section is to present some of the more significant design and operating factors relating to the ultimate use of the jet pump injector in a face haulage system. The factors that are discussed are enumerated below.

<u>Design Factors:</u>

- <u>Design rate</u> of the haulage system for a 48" seam.
- <u>Significance of surge capacity</u> between the miner and the injector.
- <u>Feed hopper design</u> considerations.

TEST RUN NO.	NOZZLE PRESSURE (PSIG)	COAL RATE TON/MIN	HOPPER WATER (GPM)	DISCHARGE PRESSURE (PSIG)	
				TEST	ANALYTICAL MODEL
49g	60	0.39	22	12.5	12.52
49h	80	0.43	30	15.8	16.33
49i	99	0.51	41	18.75	19.39
49j	126	0.59	58	22.9	23.42
49k	140	0.62	61	24.8	25.8
491	148	0.65	63	26.1	27.26
50a	142	0.76	90	20.4	22.08
50b	125	0.72	79	18.4	18.66
50c	100	0.64	69	15.5	16.16
50d	81	0.57	36	13.3	15.43
50e	60	0.44	36	10.9	11.6
51a	140	0.534	41.5	27.7	27.0
51b	125	0.498	31	25.1	24.79
51c	99	0.425	20	19.9	21.07
51d	80	0.344	15	17.2	17.37
51e	60	0.260	2.5	13.9	13.56

Table 4-1: Analytical Model Predictions Versus Test Results

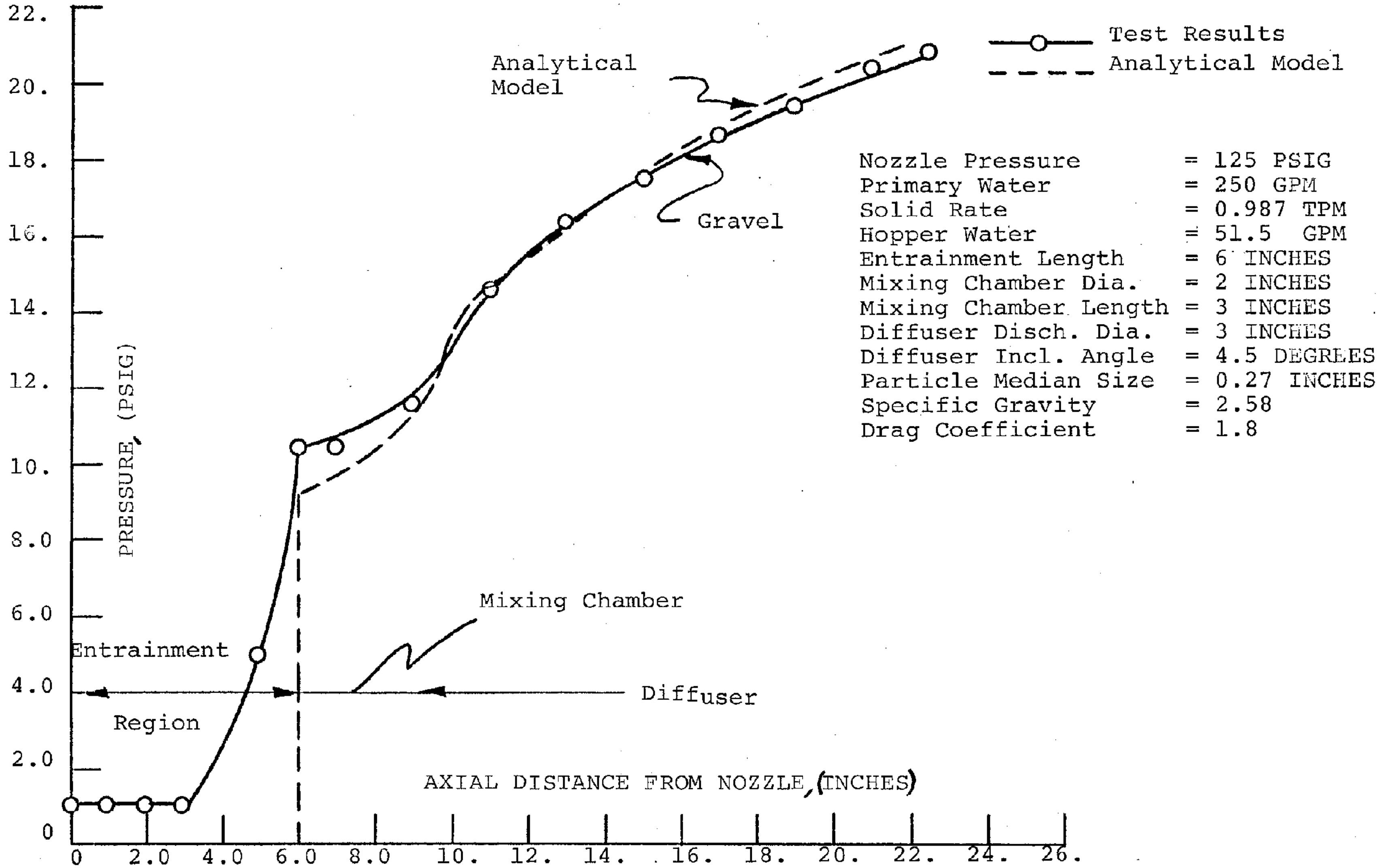


Figure 4-22: Pressure as a Function of the Axial Distance from Nozzle for the Analytical Model Versus Actual Test Results on Gravel

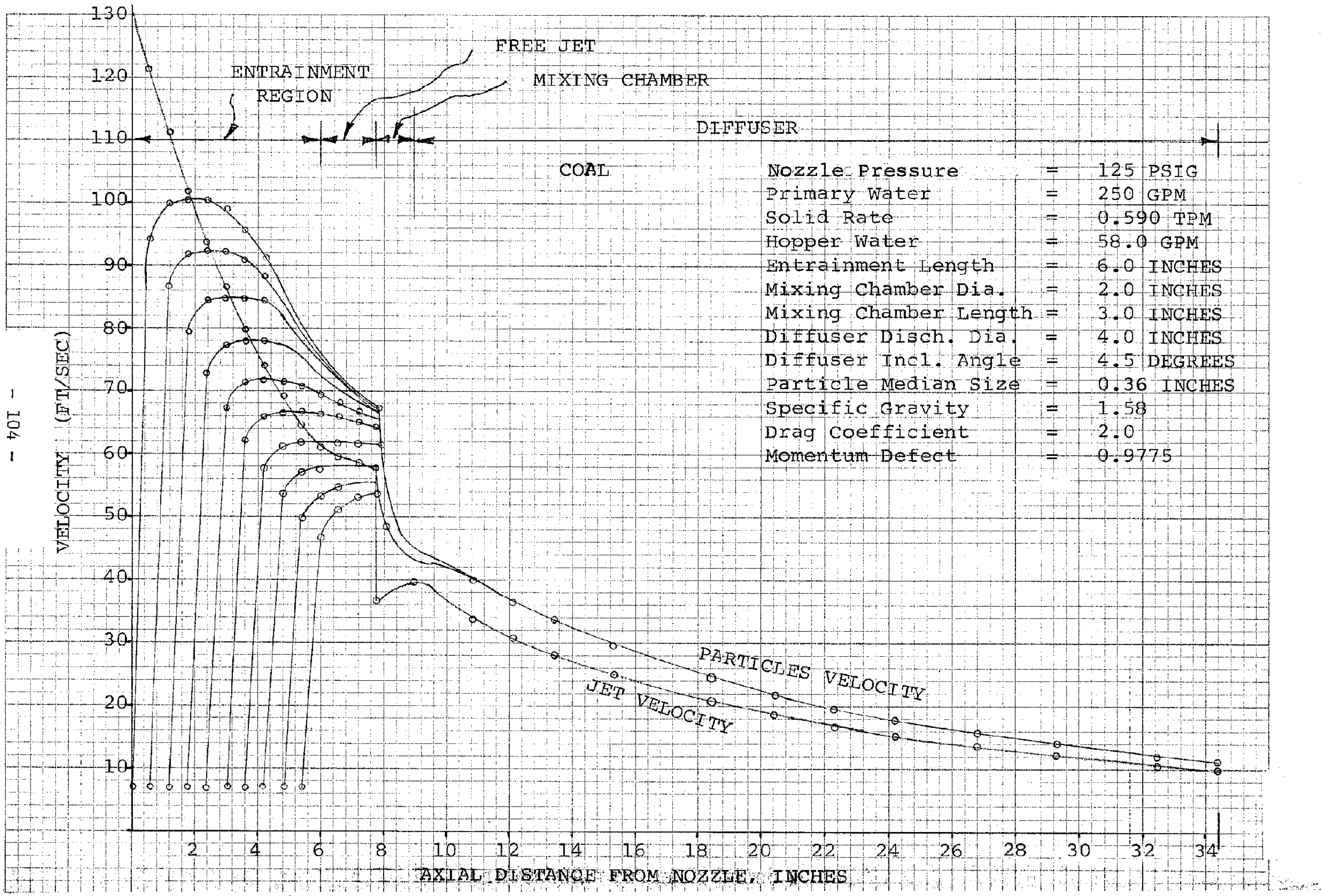


Figure 4-23: Particle Velocity and Jet Velocity as Functions of the Axial Distance from Nozzle, Based on Analytical Model, for Coal

- o *Service life* of the injector.
- o *Throat blockage.*

Operating factors:

- o *Injector Control Concepts.*
- o *Booster Pump Control.*
- o *Injector Safety Considerations.*
- o *Ingestion of air.*

4.3.2.3.1 Injector Design Rate

The design rate of a complete hydraulic haulage system is the most significant factor influencing its capital and operating costs. This parameter influences (1) water capacity requirements, (2) slurry line size, flexibility and mobility (3) maximum conveying distance and (4) size and horsepower of all booster pumps. All of these factors are reflected in the overall cost per ton of coal conveyed even though the jet pump proper is only moderately affected. There are inherent advantages towards minimizing the design injection rate and maximizing the full load operating time of the haulage system.

Table 4-2 illustrates how the design rate influences the line size, conveying distance and estimated relative hose life. In this table the Relative Hose Life Factor (RLF) is based upon a cubic function of the mean slurry velocity with all the hoses having the same rubber liner thickness. RLF = 1.0 for the lowest velocity case and estimated longest life. All cases were based upon an injector nozzle supply pressure P , of 125 psig, a flow ratio $M = 0.71$, a solids concentration in the secondary feed $C_{ws} = 0.78$ and a nozzle to throat area ratio $R = 0.193$. These parameters were selected as a conclusion to the discussion given in the subsequent section entitled *Sizing Injector Throat.* The discharge pressure is calculated on the basis of head ratio $(P_d/P_n - P_d) = 0.237$ *(N)*, but is adjusted for diffuser area ratios other than 4.0. Line length is calculated using a friction head loss factor for hose from Reference 7 and the slurry head loss equation from Reference 6. The minimum bend diameter is based upon 10 times the hose inside diameter.

The maximum continuous cutting rate of the miner is the

Design Rate TPM	Line Size IN	Minimum Hose Bend DIA-IN	Mean Slurry Line Velocity FPS	Maximum Conveying Distance FT	Relative Hose Life Factor
3	6	60	22.3	151	0.10
3	8	80	12.6	555	0.58
3.2	6	60	23.8	133	0.09
3.2	8	80	13.4	514	0.48
4.0	8	80	16.8	349	0.24
4.0	10	100	10.7	800	0.94
6.0	8	80	25.1	159	0.07
6.0	10	100	16.1	471	0.28
6.0	12	120	11.2	881	0.82
8.0	10	100	21.4	284	0.12
8.0	12	120	14.9	632	0.35
8.0	14	140	10.9	964	0.89
10.0	12	120	18.6	450	0.18
10.0	14	140	13.7	788	0.45
10.0	16	160	10.5	1054	1.00

Table 4-2: Influence of Design Haulage Rate on Line Size, Conveying Distance and Hose Life

key factor in establishing injector feed rate. This contract focused on a 12 TPM design which coincides with typical continuous miner conveyor capacities but is far above the cutting capabilities of present day miners. However, allowing for some advancement in the state-of-the-art of cutting, 4.0 TPM are also possible with surge capacity between the injector and the miner.

4.3.2.3.2 Significance of Surge Capacity

The amount of surge capacity between the miner and injector can have a significant influence on reducing the injector design rate; providing, it integrates well with the miner movement and the flexible hose handling system. Consideration must be given to the following points and questions:

- Size of the surge capacity relative to the injector design rate.
- Mobility of the slurry hoses.
- Can coal be injected while the miner is backing and getting into position to start the second cut?
- Is the surge vehicle mobile enough to keep from interfering with the mobility of the miner?
- Can coal be injected while the surge vehicle is changing place or in motion.

This surge capacity could take the form of a mobile-feeder breaker vehicle or a bolter/conveyor car which has the injector mounted on it and follows the continuous miner.

The following relationships establish how the surge capacity influences the required continuous injection rate. It assumes that the injector can operate continuously after the miner starts cutting and for a certain period of time after the completion of the second pass. Six tons of surge capacity is considered the maximum for a mobile vehicle operating in a 48" seam and is used in the following example:

$$\dot{M}_c = \frac{\Delta T_c \times \dot{M}_{ac} - M_s}{\Delta T_c + \Delta T_m} \quad ; \quad \Delta T_c = \text{Actual time spent per place cutting at the}$$

rate of $\dot{M}_{ac}$, MIN.

where:

T_m = Total elapsed time per place while not cutting but still injecting coal MIN. Such as when the miner repositions to make the second cut.

$$\Delta T_m < \frac{M_s}{\dot{M}_c}$$

if: $$\Delta T_m > \frac{M_s}{\dot{M}_c}$$

then: $$\dot{M}_c = \frac{\Delta T_c * \dot{M}_{ac} - 2M_s}{\Delta T_c}$$;

$\dot{M}_{ac}$ = Average cutting rate of miner, TPM. Assumed to be same for first and second passes.

M_s = Surge capacity, TON.

$\dot{M}_c$ = Maximum required, continuous injection rate, TPM.

If $\dot{M}_{ac}$ = 4.0 TPM, ΔT_c = 12.8 MIN, and ΔT_m = 1.2 MIN and M_s = 6 TON, then $\dot{M}_c$ = 3.2 TPM. This represents a 20% reduction in design rate. Referring to Table 4-2 and comparing the haulage distance for 3.2 TPM and 4.0 TPM designs using 8" hose, it can be seen that the 3.2 TPM case has the potential of conveying nearly 50% farther than the 4.0 TPM case. Lower line velocities at 3.2 TPM give potential hose life advantages too.

4.3.2.3.3 Injector Feed Hopper Design

The three main considerations important in the design of the feed hopper are discussed below.

- The volume of the feed hopper can serve as surge capacity since the injector can operate satisfactorily with solids piled over the jet and throat. Figure 4-24 illustrates a feed hopper sized for a 4.0 TPM <u>average</u> injection rate. This design assumes that no buffering of the continuous miner's cutting rate exists between its cutting head and the injector feed hopper.

- The volume of the feed hopper influences the hopper water level <u>control response time</u>

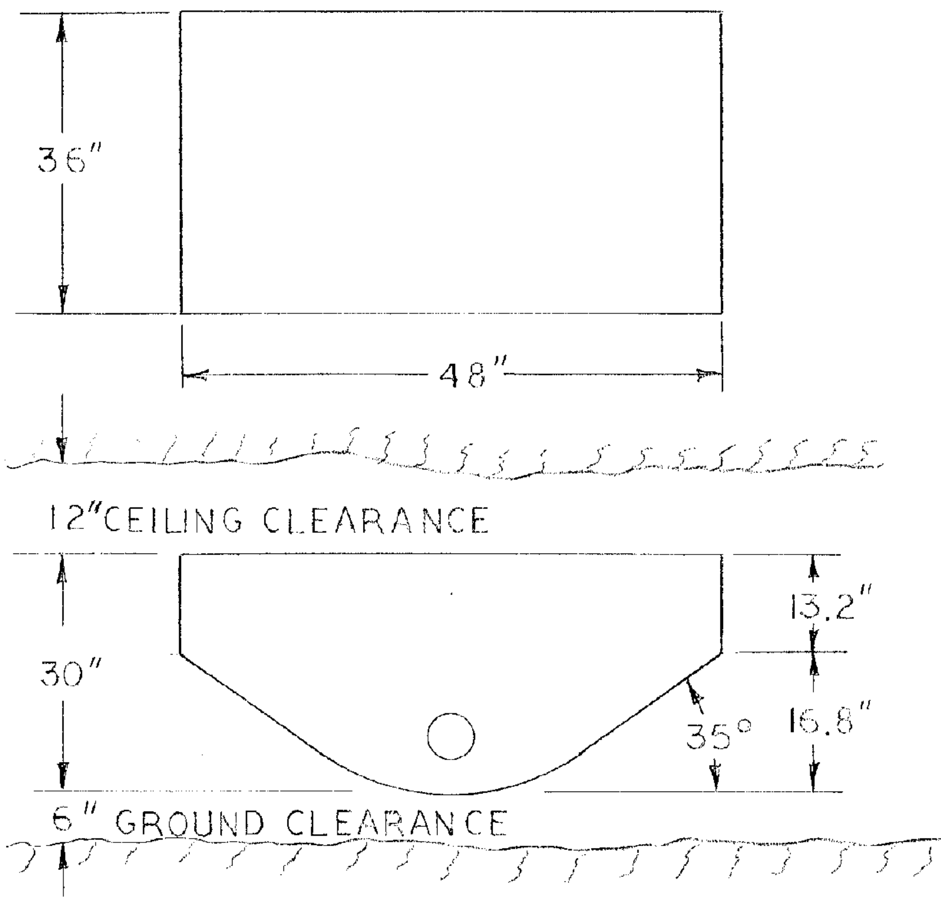


Figure 4-24: Injector Feed Hopper

requirements. The hopper illustrated in Figure 4-24 should be easier to control than the test model which was readily controlled.

- For a short period of time, while starting up a long slurry line, not all the nozzle water flow passes out the injector discharge. The hopper volume should be capable of accepting this extra water without overflowing. However, if the combination shut-off-bypass valve described below is incorporated in the injector design, then this consideration is eliminated.

4.3.2.3.4 Injector Service Life

The two factors which control the injector throat life are its wear rate and the chosen injector geometry/performance conditions. Of course, the material selection is critical for low wear rates but equally important is the sizing of the throat so that a substantial amount of material can be worn away without significantly reducing performance. These factors are briefly described below.

Injector Wear Rate

The wear that was observed in the jet pump injector test model occurred primarily in the throat piece. The test throat was manufactured from AISI Type 4130 steel tubing heat treated to Rc 63-66. While operating on Anthracite coal and 125 psig nozzle pressure the diametral wear rate was 0.15"/1000 hr. This wear rate can be expected on larger prototype injectors since the throat velocities remain the same. Excellent injector service life is anticipated at these rates.

In practice, it is recommended that the injector throat be designed to include the bell-mouth inlet and a small part of the diffuser inlet. In this way, the mating of new and worn parts is nearly eliminated at both ends of the replacement throat.

This design concept is illustrated in Figure 4-25. Nozzle wear was not found to be significant in the test model. The AISI Type 316 stainless steel nozzle experienced only a very slight polishing from the high velocity water flow near its outlet. It appeared that abrasion on the outside of the

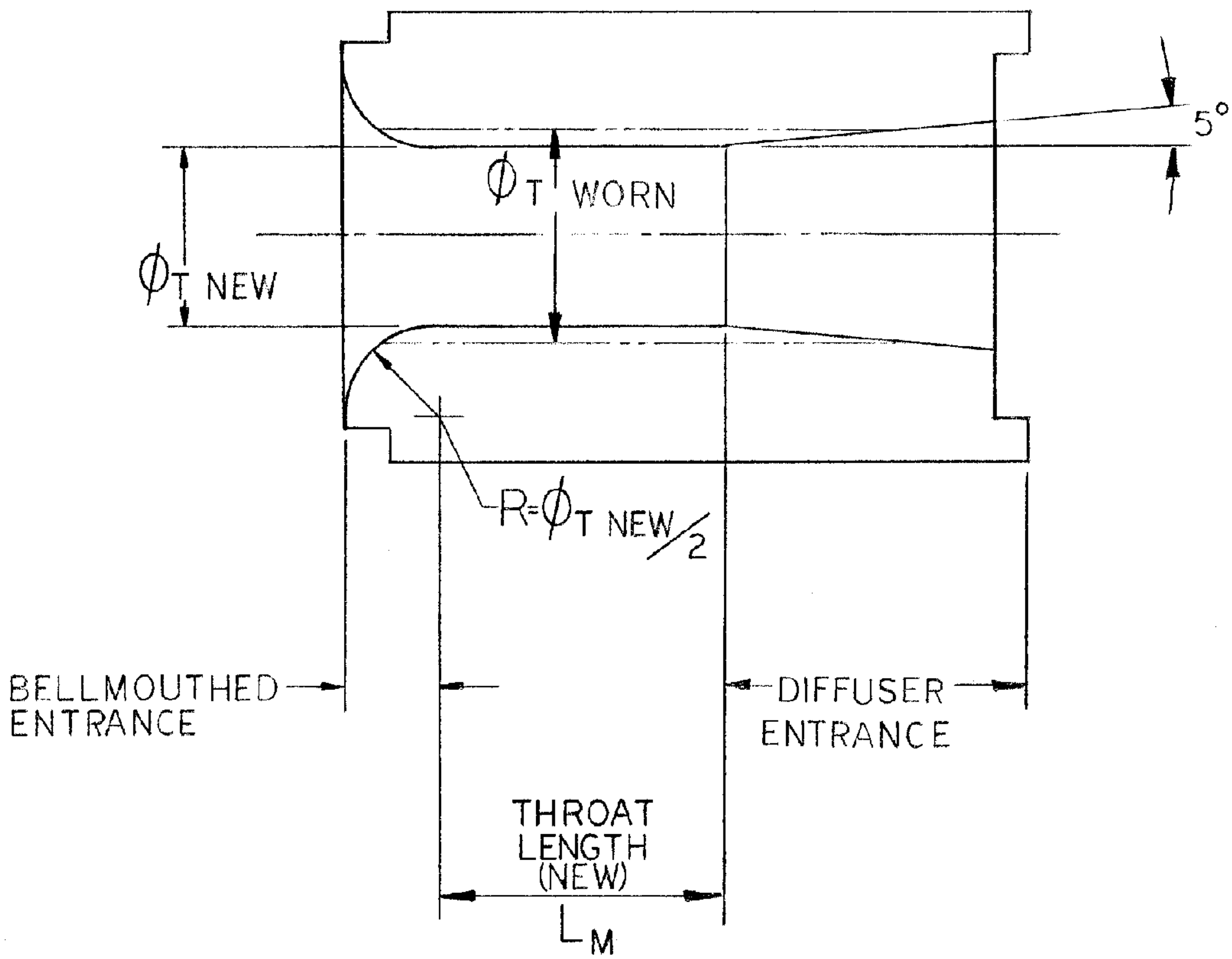


Figure 4-25: Recommended Injector Throat Configuration

nozzle could be more of a problem. Ultimately, a hardened design or a protective shield might be required. On dredging jet pumps, it has been a practice to incorporate an insert in the tip of the nozzle to reduce wear in this region. Diffuser wear normally is not a problem, even in dredging applications where cast iron gives acceptable life. These points are discussed in Reference 4. The test model diffusers were fabricated from AISI Type 8620 steel, case hardened to a depth of 0.060 inches. Wear was not a problem with these diffusers. However, it appeared that cast iron would be a better choice for the prototype.

Sizing Injector Throat, $\emptyset_N$

As the injector throat wears, the nozzle/throat area ratio, R, decreases thus changing the performance characteristics. It is essential to know how these characteristics change when designing the injector for a maximum service life since only a minimal change in performance is desirable.

Figure 4-26 shows the design curves of M-N, M-C_{WS} for injector configurations having a $L_E/\emptyset_N$ = 6.83 and $L_M/\emptyset_N$ = 3.42. This curve illustrates how the basic design parameters change with different area ratios. By inspection it is evident that at M = 0.71 the values of N are most uniform. However, the values of C_v are not as close together as they are at M = 0.55. Both C_v and N influence the shape of the injector performance curves. Both M and C_v control the shape of the system curve into which the injector discharges. Larger values of M and C_v are desirable to minimize the volumetric flow of slurry in the pipeline. Therefore, the whole system must be analyzed to determine how the actual maximum injection rates change with throat wear as R changes from 0.152 to 0.219.

The case described in Appendix E of the Phase IIA report can be used as a basis to illustrate how the steady state full load coal injection rate changes with throat wear. That case was calculated on the basis of M = 4 TPM, M = 0.71, R = 0.193, 0 = 10" which results in a G = 1737 GPM, G = 273 GPM, 0 = 5.268".

The corresponding throat diameter for R = 0.152 and R = 0.219 are 5.936" and 4.945" respectively. Determining the corresponding $\dot{M}_c$ values for these throat diameters we find that when $\emptyset_M$ = 4.945", $\dot{M}_c$ = 3.8 TPM (5% less) and in the very worn condition when $\emptyset_M$ = 5.936", $\dot{M}_c$ = 3.5 TPM (12 1/2% less).

In this case, the injector performance would start at $\dot{M}_c$

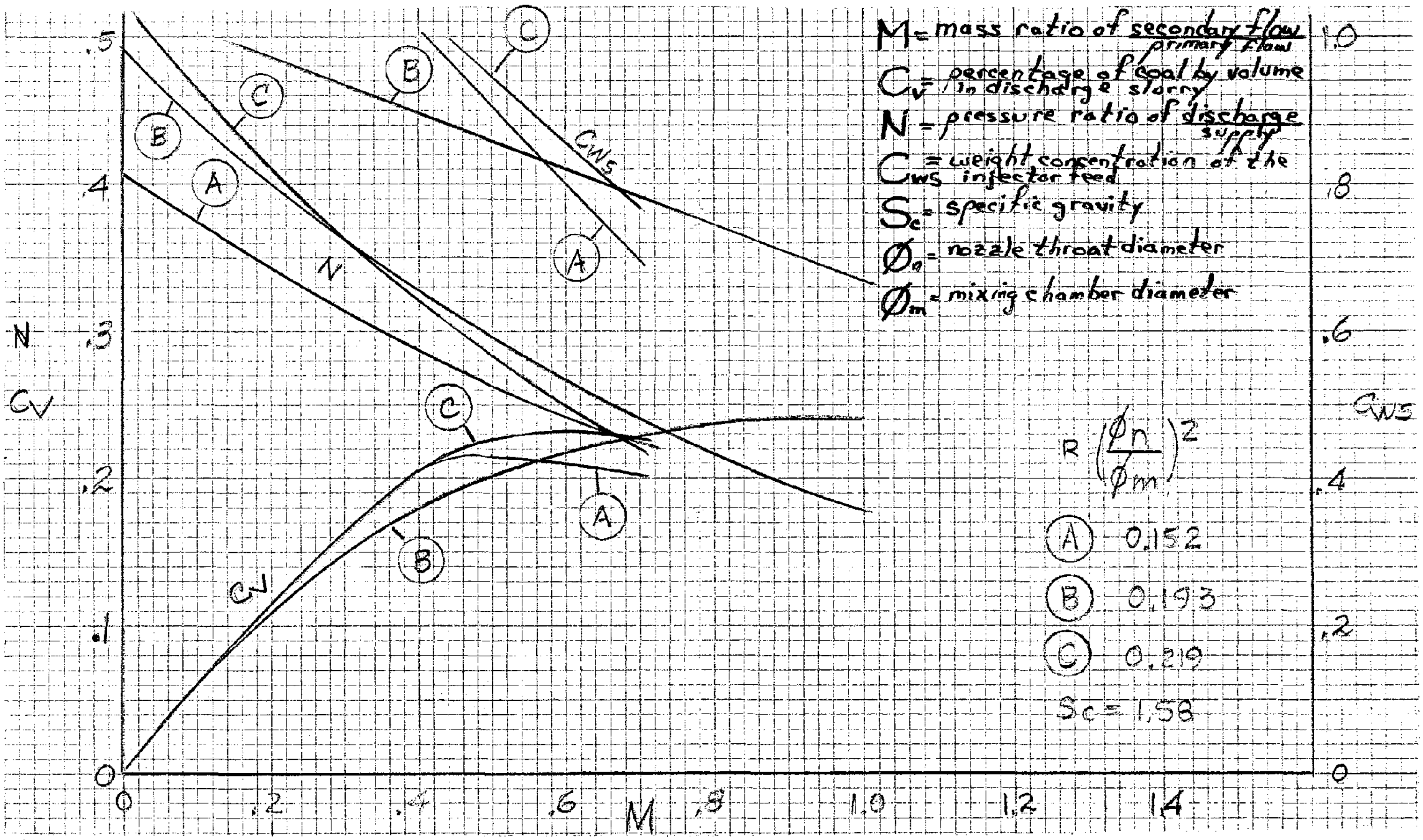


Figure 4-26: Combined Design Curves for $L_E/\emptyset_N = 6.83$ and $L_M/\emptyset_N = 3.42$

= 3.8 TPM and increase to 4.0 TPM and then drop again to 3.5 TPM when R = 0.152. If the application could not tolerate an injection rate less than 3.8 TPM then the throat would have to be changed when $\emptyset_M$ = 5.680" which would result in somewhat reduced life but would still be about 1.2 million tons of coal (or 5,000 hours of operation) when calculated on the same basis as that used in Table 4-3.

4.3.2.3.5 Injecting Into The Slurry Haulage System

This jet pump injector is particularly well suited to feed a slurry pipeline. The near right angle intersection of the injector performance curve and the system resistance curve enhances the stability of operation. As the system resistance increases the solids injection (discharge concentration) rate inherently drops until a stable condition is reached. When coal is no longer available to the injector it has the capability to increase the flow and pressure going into the pipeline. This feature increases the line velocity and aids in flushing out the haulage system before shut-down -- particularly important if more than one injector shares the same booster pump.

The injector can easily accommodate rapid changes in loading such as step-wise changes from unloaded to loaded and back to unloaded. This capability is particularly important since these conditions will be experienced in a hydraulic haulage system. During this rapid change in loading, the injector and the balance of the haulage system must be able to respond in a stable and reliable manner. Another desirable feature of this injector concept for this application is that the booster pump can be driven at a constant speed without concern that the conveying velocity will drop below the minimum safe value. Finally, the injector can be started up without a complex startup procedure. This feature is important if hose haulage system limitations or some other requirements dictate that more than one injector must be used in a section.

Throat Blockage

A rule of thumb used in a slurry pipelining design is that the maximum lump size should be no longer than 1/3 the inside diameter of the pipe. If this criteria is also adhered to in face haulage applications then it can be stated that the upper limit on lump size is controlled by plugging criteria in the pipeline and not the injector throat. A comparison of Tables 4-2 and 4-3 illustrates that practical injector designs

			ϕ_T Inches			Estimated Throat Life*
		R =	0.219	0.193	0.152	
$\dot{m}_c$ TPM	$\dot{G}_N$ GPM	$\dot{G}_H$ GPM	NEW	DESIGN	WORN	MILLION TONS
3.0	1303	205	4.283	4.563	5.141	1.0
3.2	1390	218	4.424	4.713	5.310	1.1
4.0	1737	273	4.945	5.268	5.936	1.5
6.0	2606	410	6.057	6.453	7.221	2.9
8.0	3474	546	6.994	7.450	8.395	4.5
10.0	4343	683	7.820	8.330	9.386	6.3

*Estimated Throat Life Based on a Wear Rate of 0.15"/1,000 Hours of Operation

Table 4-3: Injector Throat Sizes for Various Design Rates have a throat to line size ratio of about 1/2 or more.

Injector Control Concepts

There are two basic methods for controlling the injector. The first method involves using a constant water flow rate and injecting whatever quantity of coal that is available into that water flow. The second method allows the water flow rate to increase whenever the coal flow rate decreases thus maintaining the slurry line velocity above the design value. The second method is the preferred method since it always operates at lower line velocities at full load. These conditions are dictated by slurry line velocity requirements and not by the injector.

In any slurry haulage control system the minimum line velocity must be maintained at a safe margin above the critical line pumping velocity. For this example, a 15% margin of safety was used. In the case of run of the mine coal it is necessary to base the minimum operating line velocity on that required to convey the rock mixed with the coal due to its higher specific gravity and settling velocity.

Considering the alternating loaded/unloaded operation with a constant water flow system, the minimum conveying velocity is reached whenever the solids injection rate goes to zero. Yet certain sections of pipe will still have coal in them and a certain line velocity is required. The maximum line velocity coincides with the maximum solids injection rate. At this condition, the line velocity will greatly exceed the minimum required velocity. Wide changes in line velocity and pressure drop can occur with this system. The magnitude of this problem escalates rapidly as the design slurry concentration increases.

Table 4-4 illustrates the difference in the two control concepts. Three cases are displayed -- one with the variable water flow condition and two with the constant water flow concept. A 4 TPM coal injection rate is common to all three cases. Cases A-1 and R-1 were developed on the basis of 23.5% C_V slurry concentration. In the case of R-1 a non-standard line diameter was used to maintain consistency of design rates and velocity criteria. Note its higher pressure requirements to convey the same distance.

Concept R-2 illustrates the impact of high slurry con-

CONCEPTS

	A-1(6) Full Load	A-1(6) No Load	R-1(7) Full Load	R-1(7) No Load	R-2(7) Full Load	R-2(7) No Load
Hose Diameter, In.	10"		8.97" (5)		6"	
Velocity of Deposition, fPS (1), Vd	9.1		8.6		10.3	
Minimum Slurry Velocity, fPS (2)	10.5		9.9		11.9	
Solids Volume Concentration	0.238	0	0.238	0	0.365	0
Mean Slurry Velocity, fPS, V_m	10.7	13.6	13.0	9.9	19.0	11.9
Est'd 800 Ft Hose Pressure Drop, PSI (3)	24.0	21.8	32.9	13.4	89.1	30.5
Est'd Relative Hose Cost	1.00		0.84		0.49	
Est'd Relative Hose Life (4)	1.00		0.52		0.17	

1). Based upon rock being conveyed in the line at C_v = 0.238 for A-1 and R-1 and at 0.365 for R-2.

2). Allowing 15% margin of safety over deposition velocity.

3). Based upon 800 ft. of haulage.

4). Estimated on basis of hose wear rate being proportional to (mean velocity)^3. Some hose manufacturers literature suggest that it may be closer to (mean velocity)^4.

5). Hypothetical line diameter used to keep coal flow rate constant for all cases.

6). Concept based upon design discussed in Appendix E of Phase IIA Report

7). Concept based upon constant water flow rate during loaded and unloaded operation.

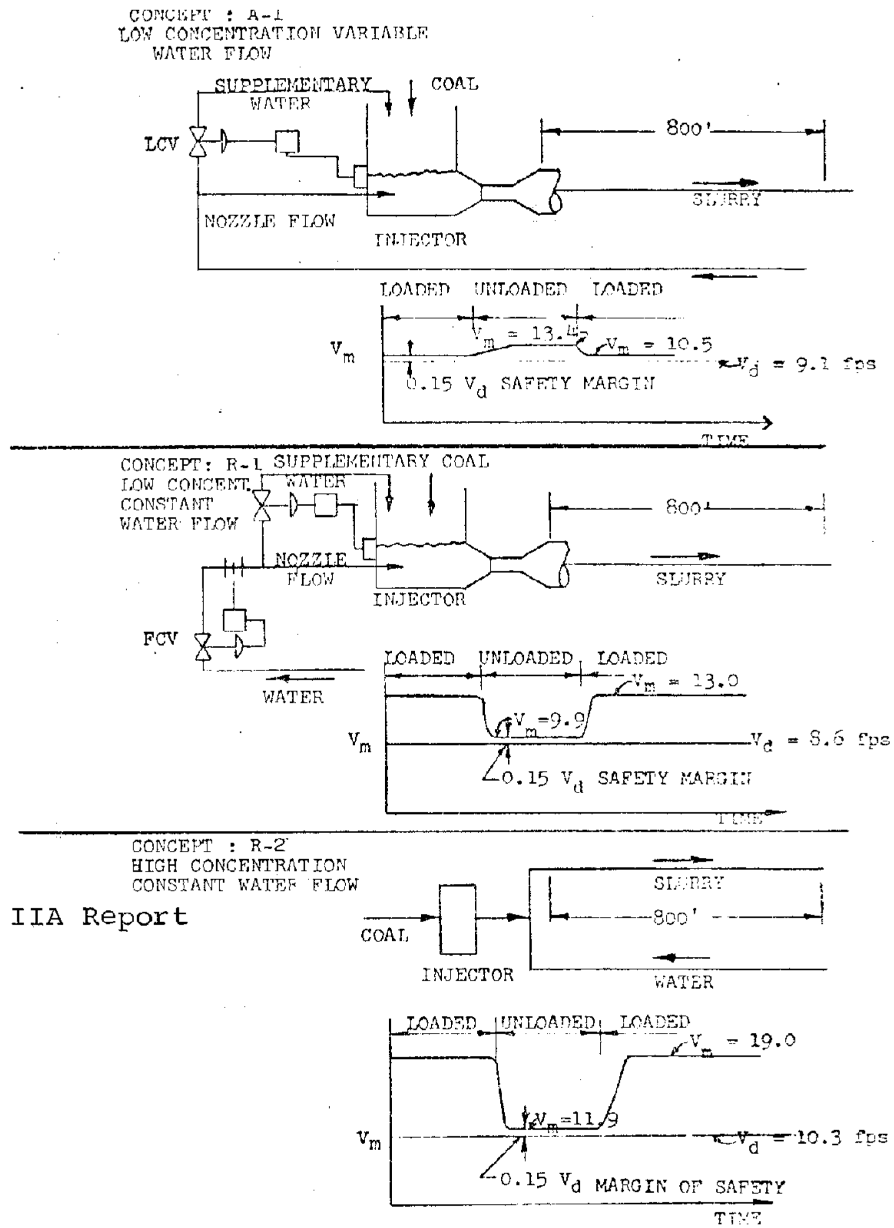


Table 4-4: Control Concepts for Interrupted Coal Injection at 4 TPM

centration on pressure drop when the solids are transported by a constant water flow. The line size is much smaller since less water is required to convey the same quantity of coal at high concentration; however, booster pump speed control would definitely be required to accommodate these severe conditions. Large traverses in line velocity and line resistances make the control of this system extremely difficult.

Booster Pump Control

Several options are available for the booster pump control. One method uses a constant speed pump with a pressure control valve supplying supplementary water to the booster pump inlet as required. A second method involves the use of a variable speed booster manually regulated to allow for gross changes in line resistances such as those resulting from adding or removing lengths of haulage line. This method also would utilize the pressure control valve at the pump inlet. A third method involves a variable speed drive controlled automatically from the booster pump suction pressure. As the pressure increased, the speed would also increase to keep pace with the injector feeding the booster.

A fourth method involves a variable speed drive controlled automatically from the flow sensed in the pipeline.

The above control methods have been listed in the order of increasing capital cost and complexity and in a descending order of reliability. The jet pump injector developed in this program interfaces excellently with the first method which is the preferred method.

Injector Safety Considerations

The absence of rotating shafts, electric motors and variable speed drive components greatly simplify the safety considerations for this injector. Overflowing of the feed hopper in the event of a plugged throat, haulage line, or other like form of malfunction, constitutes the only real safety consideration for this injector. This problem is easily solved.

The best solution to the hopper overflowing problem involves a system which senses an excessive hopper water level and automatically reacts when it is reached. The water flow to the injector nozzle (and supplementary hopper water valve) would be bypassed into the injector discharge line to keep the slurry flowing in the haulage lines. This entire operation could be accomplished with a three-way valve diverting the water through an orifice from the nozzle supply line to the primary haulage line. This valve could also be used in the normal shut-down and startup sequence.

Ingestion of Air

It was demonstrated during the model studies that substantial quantities of air can be ingested into the injector and slurry haulage line under certain unloaded conditions. It was also demonstrated that all the air could be excluded by sealing the injector throat and jet. However, it was never determined whether or not it is essential to completely exclude all air.

Some air introduced into the inlet of centrifugal pumps, such as the slurry line boosters, actually retards the damage which normally occurs when they operate in a cavitating condition. Consequently, there are some benefits to be derived from having small quantities of air in the slurry line. However, it is not known how severe the resulting changes in operating characteristics would be in the haulage line with the ingestion of substantial amounts of air. With the ingestion of substantial air volume, the slurry flow would have to be treated as a compressible fluid where line velocities would become a function of the local static pressure. Overall, a very complex three-phase flow pattern would emerge.

The water level control concept for completely excluding air is considered the preferred mode of operation for the jet pump injector.

4.3.3 Phase IIB - Hydraulic Face Haulage System And Concentrator Developments

4.3.3.1 Coarse Coal Hydraulic Face Haulage System Development

The IRRI hydraulic haulage system that is capable of receiving run-of-mine coal and transporting it away from the working face evolved from Phase I and Phase IIA work and consists of four major subassemblies, namely:

Feeder -

-A Breaker that sizes and meters run-of-mine coal to the injector.

-An Injector device that injects all coal into a pressurized slurry hose.

-A pair of Water and Slurry Hoses.

-A Hose Handling Means that supports and moves the hose pair to follow the Injector.

This section describes the complete Hydraulic Haulage System that was developed based on the use of the Jet Pump Injector vehicle combined with a Monorail Suspended Hose Handling System. This system is shown in plan view in Figure 4-27 and in a perspective rendition in Figure 4-28. The method by which this system was selected and the mine criteria used in the selection process are also presented.

The major factor influencing the design of the hydraulic face system is the method of handling the slurry hose. Hose handling is second in importance only to the injector device because the hose handling equipment determines the mobility, pressure loss, and extensibility of the slurry hoses. The rationale used in selecting the hose handling method will be discussed.

All methods of hose handling that were envisioned, that permitted the required expansion and contraction of the hose train, involved the forming of bends or loops in the hose train. Since there was no existing data related to head loss for coarse coal slurry through such loops and bends a full scale slurry test loop was constructed. The test set up

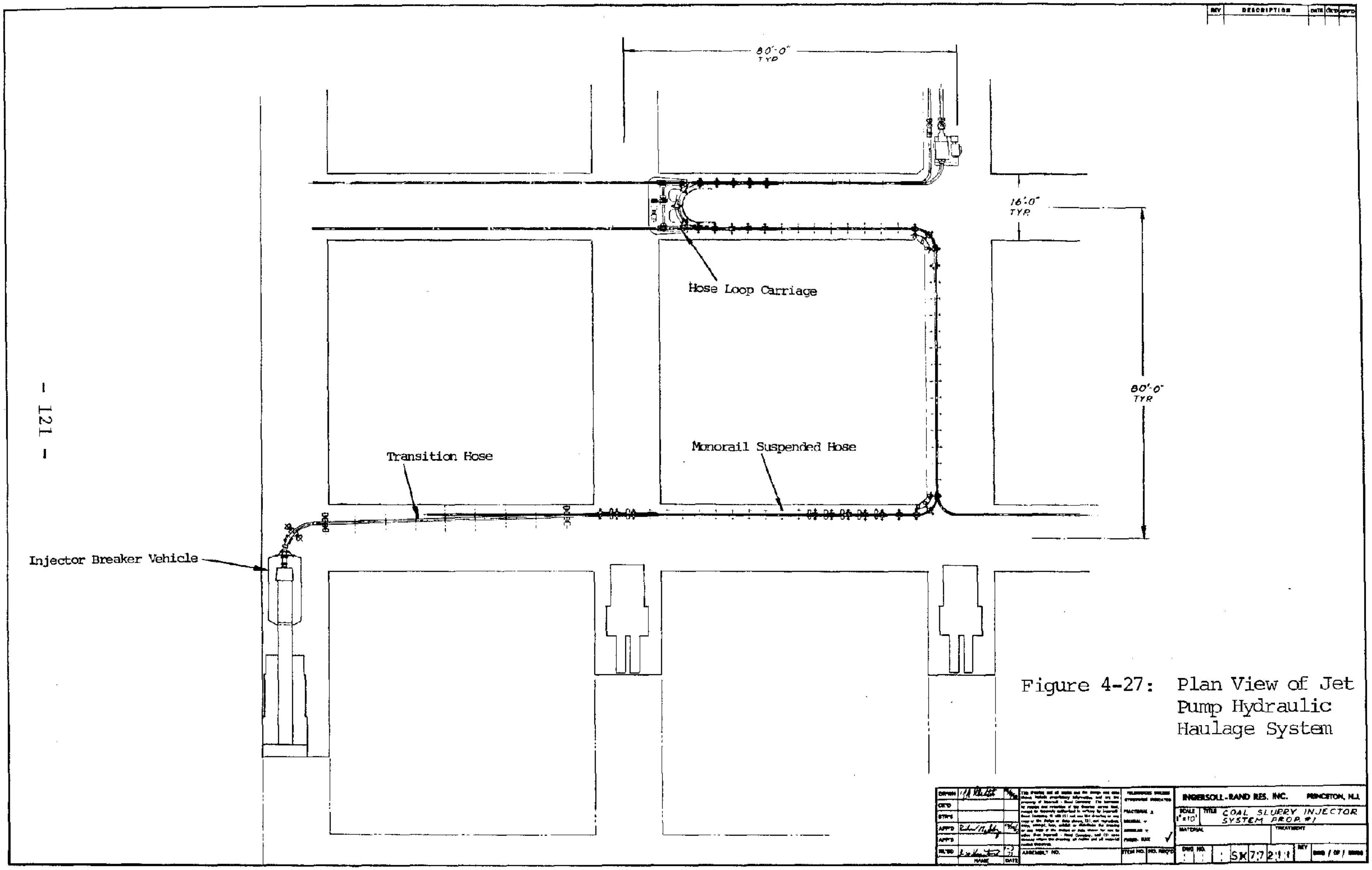


Figure 4-27: Plan View of Jet Pump Hydraulic Haulage System

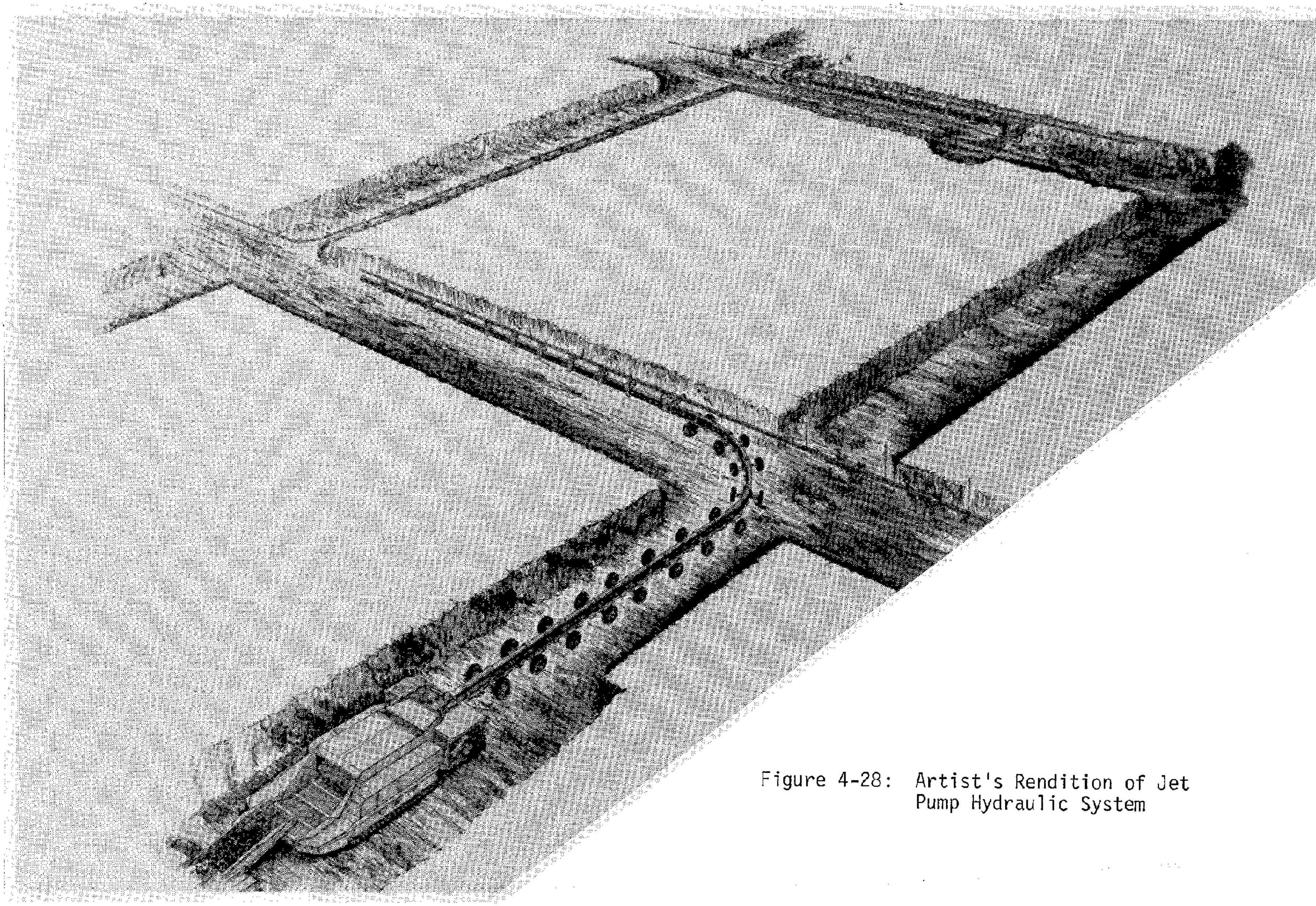


Figure 4-28: Artist's Rendition of Jet Pump Hydraulic System

permitted the evaluation of head loss in 90, 180, 360 and 720 degree hose bends.

4.3.3.1.1 System Design Criteria

A Coarse Coal Face Hydraulic Haulage System must satisfy the requirements of both (1) the mining practices utilizing continuous miners and (2) the physics of hydraulic haulage of coarse particles. The coarse coal haulage concept was originally planned to be a system that could be directly integrated into an existing mine plan with minimum restraints on existing operations. This philosophy was maintained when setting the following criteria that were met by the Jet Pump Injector Hydraulic Face Haulage System.

<u>Mine Operation</u>

The Hydraulic Haulage System should meet the following overall design and Operating specifications.

- 4-Foot seam capacity
- 5 Entry extensibility
- The ability to receive run-of-mine coal from a continuous miner in room and pillar mining.
- Maximum tramming speed of 50 feet per minute.
- Manueverability to track an unhindered continuous miner.
- The injector vehicle must be less than 27 feet long and 9 feet wide. These dimensions were established to permit maneuvering in an 18 ft. wide entry.
- There are to be only two vehicles at the face: a continuous miner and an Injector Breaker Vehicle.
- The Hydraulic Haulage System must be designed to fail in a safe manner.
- The haulage system must not increase the generation of dust.

- No interruptions of roof support or ventilation.

- System to comply with existing and proposed Federal and State health and safety standards.

Design Criteria - Hydraulic

The hydraulic haulage of solids makes the following two demands upon the system:

- Sufficient slurry velocity must be maintained to provide a homogenous slurry mixture to avoid solids settling out of suspension.

- The injector device must generate enough pressure to overcome the head loss through the slurry hose line.

Solid particles in a slurry are held in suspension by fluid turbulence associated with high Reynolds number flow. There are three regimes which are capable of transporting solids without plugging. They are:

1. "Moving bed" regime which occurs at lower velocities. The bulk of the solids lie in the bottom of the pipe as a thick slurry and move with the flow at a slower rate than the water. The characteristic axial profile of this regime has the appearance of rippling dunes with spacing of the dunes on the order of 1-2 pipe diameters.

2. Heterogeneous flow occurs at higher slurry velocities and is characterized by a segration of particles with uniform suspension of fines and mid-range particles, and rolling movement of large particles along the bottom of the pipe.

3. Homogenous flow is characterized by complete mixing of the particles throughout the pipe cross-section.

A slurry line which must contend with a mixture of coal and rock must operate at such a velocity as to maintain at least the moving bed regime of movement in order to avoid plugging. The minimum slurry velocity is called the saltation point and is a function of particle size, particle density, liquid density, and pipe diameter. Because the specific gravity of rock is approximately 2.6 as compared to approximately 1.5 for coal, rock particles will tend to settle out of suspension at lower velocities than coal. For this reason a high slurry velocity sufficient to transport rock must be maintained.

A maximum particle size distribution has been determined by experience which indicates that to avoid plugging no more than 10% of the solid particles in the line can be greater than 1/3 the inside diameter of the pipe. Given this data the minimum slurry velocity will be determined by particles of rock 4 inches in diameter with a specific diameter of 2.6. The saltation point for solids of this size is approximately 9 1/2 feet per second. For safety the minimum recommended mean slurry velocity will be on the order of 11 feet per second. This number, of course, greatly exceeds the 5.5 ft/sec minimum velocity required for a 100% coarse coal slurry.

The choice of a coal slurry velocity is limited by two important design parameters, namely, standard, available hose sizes and the coal tonnage rate of the system. The slurry velocity can be adjusted over a small range by varying the coal concentration, but this is limited to a range of approximately 25-35 percent coal by weight. Coal concentrations higher than this value have a greater propensity towards plugging especially at dips in the pipe. Coal concentrations lower than this are uneconomical in that they require larger hoses and pumps, and increased water for a given coal tonnage rate.

A third important consideration in choosing a coal slurry velocity is wear rate. The wear rate is a strong function of slurry velocity and varies directly with the third power of the velocity. Most wear appears along the slurry line bottom as there is a tendency at all concentration levels for larger solids to settle and move along the bottom. As a result, concentration has not been shown to be a strong factor on wear.

4.3.3.1.2 Proposed Concept - Monorail Suspended Hose System

The proposed hydraulic face haulage system consists of the following major assemblies, depicted in a plan view and an artist's rendition in Figures 4-27 and 4-28 respectively.

Beginning after the continuous miner, the haulage system consists of a:

1. Mobile Injector Vehicle

2. Short ground transition hose

3. Length of monorail suspended hose

4. 180 Degree loop carriage for hose advancing and retracting

5. Centrifugal booster pump

System Operation

The mobile Injector Breaker will operate by positioning its front surge hopper beneath the continuous miner discharge conveyor to receive the output of run-of-mine coal and refuse. The coal will be water sprayed continuously as it is fed from the surge hopper to a primary, single roll crusher. All coal, except for a small number of flat slabs "hugging" the conveyor, will be sized to minus four inches, and will fall through an open coarse grate when conveyed over the jet pump hopper. The inadvertent over-size slabs will be broken and fed to the hopper by a smaller comb-crusher combination located at the upper rear of the jet pump hopper.

The jet pump will then inject the sized coal into the flexible hose attached to the jet pump discharge where sufficient pressure is generated to convey the coal slurry through the low pressure flexible hose to the booster pump.

The hose train consists of 1) a ground transmission hose length, and 2) a monorail suspended hose length.

The transition length consists of paired 92 foot long hoses ground-supported by non-powered wheel carriages. A high pressure (125 psig) supply water hose supplies the Jet Pump nozzle, and a low pressure (25 psig) slurry hose receives the Injector output. The purpose of the transition train is to allow for a variation in tramming speeds between the Injector Breaker Vehicle and monorail suspended hose train.

The monorail suspended hose train is advanced with the Injector Breaker by means of one or more monorail suspended tractors which travel with the hose train.

The complete hose train is retracted by both monorail tractors and the powered loop carriage.

The loop carriage functions as a means for extending and retracting the hose train. Stationary hose from one side of a cross cut is fed through the 180 degree track of the loop carriage to the opposite side where the hose is advanced to follow the advance of the Injector Breaker Vehicle. By this means excess connected hose is stored in a cross cut away from the face.

In summary, the Injector Breaker coupled with a highly mobile monorail tracked hose will support the production of an unhindered continuous miner by accepting all run-of-mine coal at the maximum continuous cutting rate of a continuous miner.

Injector Breaker Vehicle

The Injector Breaker Chassis must fulfill two functions:

- Receive run-of-mine coal including refuse and break all particles to minus 4 inch.

- Inject the broken coal into a slurry hose line at sufficient pressure to convey the coal slurry through the flexible hose portion of the face haulage system.

Early in the program it was established that a continuous miner will pass large slabs of coal and slate regardless of pick pattern or carefully chosen cutting rates. Roof falls and coal break-out from the face will often create particles greater than 4 inches on at least one dimension. The breaker then must become an integral part of the haulage chain as both the surge capacity and particle sizer of the system.

The Injector Breaker Vehicle, which is comprised principally of a Breaker and a Jet Pump Injector, is shown in Figure 4-29 and in an artist's perspective rendition in Figure 4-30.

A conventional single roller breaker which is widely used to feed panel conveyor belts cannot guarantee 100% of minus 4 inch coal discharge. This is especially true when handling large quantities of rock. Although a low percentage of plus 4 inch coal would pass through a single roll breaker when the continuous miner is mining 100% coal, this is not a common situation.

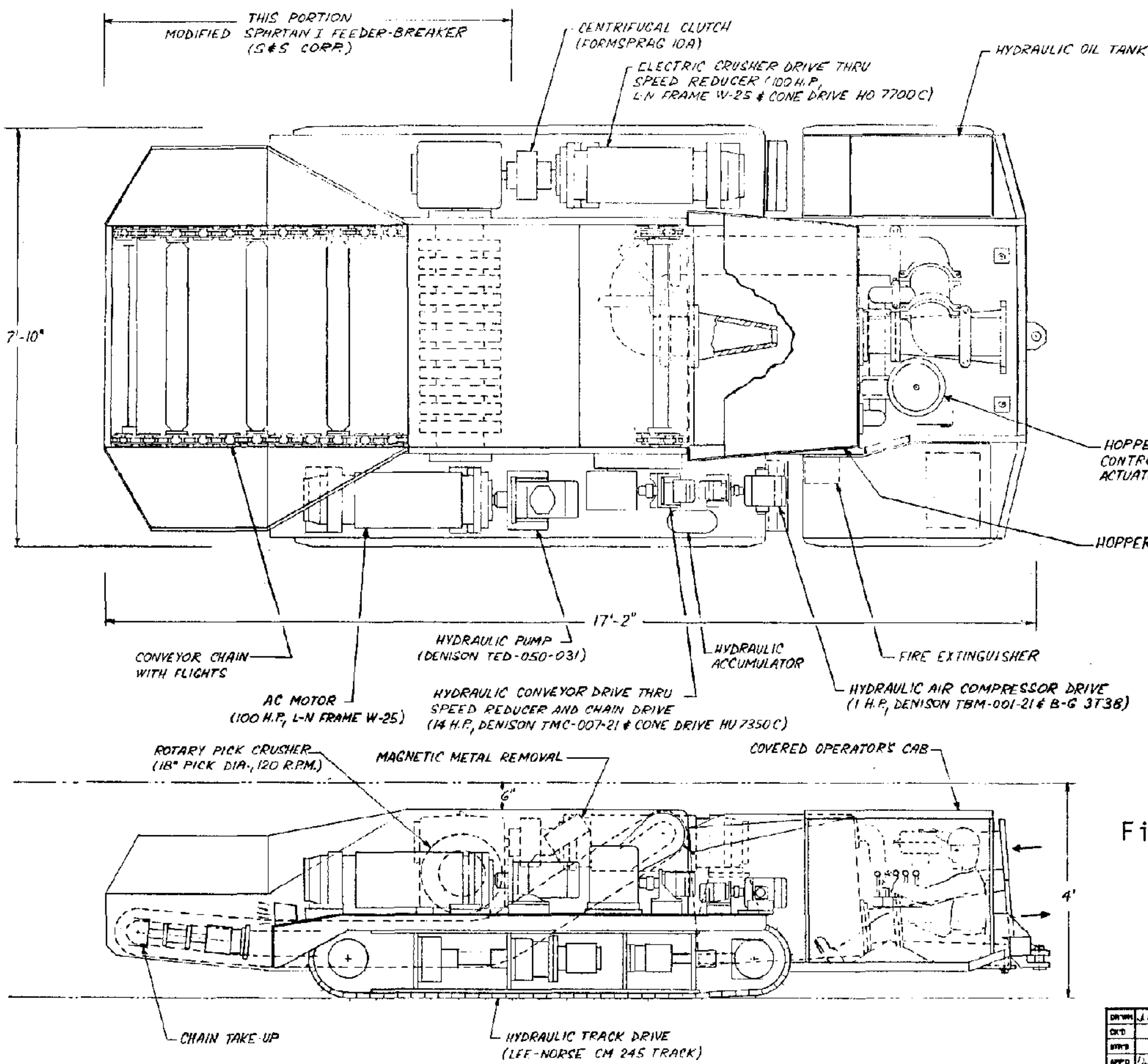


Figure 4-29: Plan and Elevation Views of Injector Breaker Vehicle

INGERSOLL-RAND RES. INC. PRINCETON, N.J.

SCALE 1"=1' TITLE COAL SLURRY INJECTOR-PROP #1

SK 77209

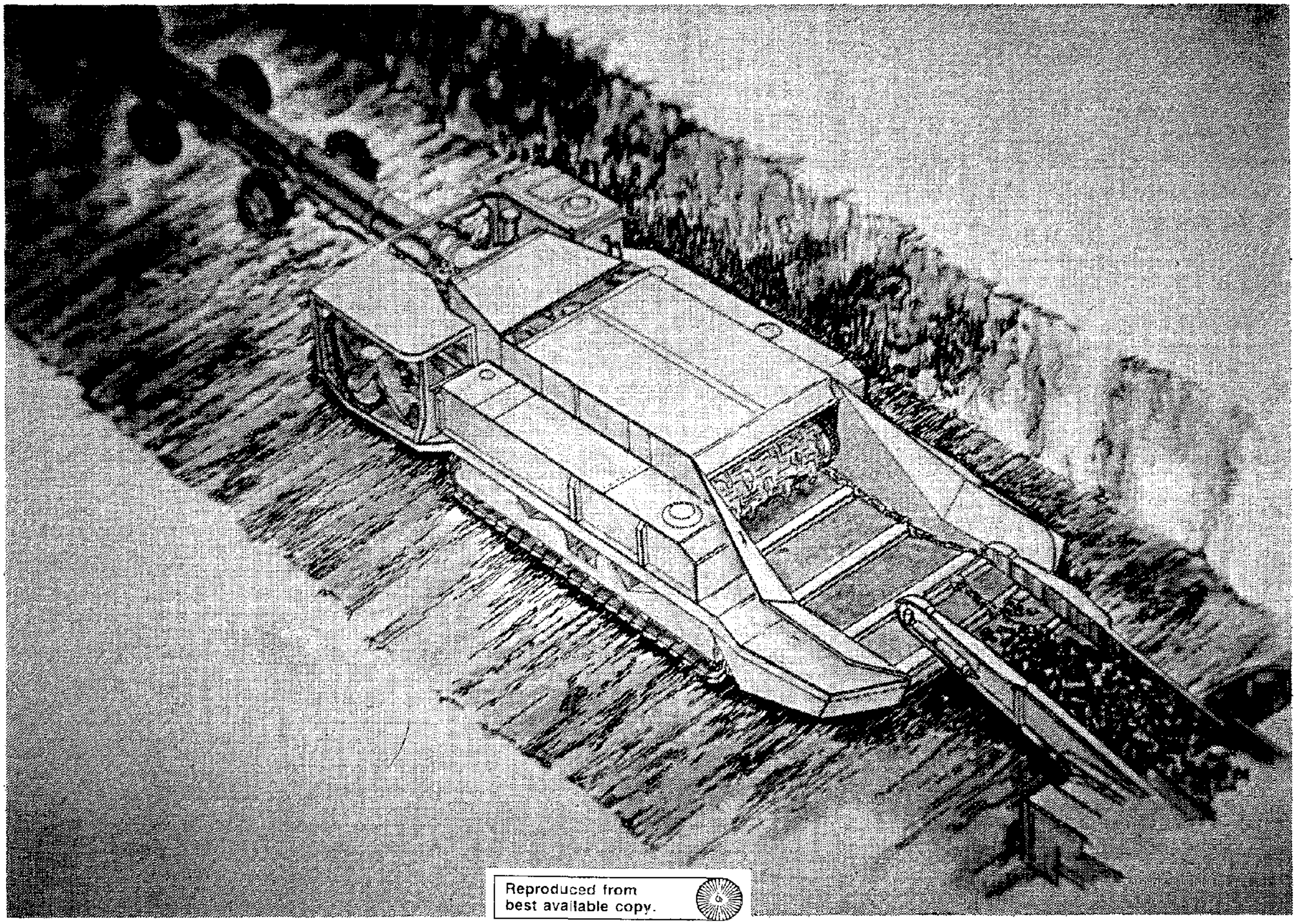


Figure 4-30: Artists Rendition of Injector Breaker Vehicle

Therefore, as shown in Figure 4-31, a small secondary breaker will be required for the small proportion of plus 4 inch stock which leaves the primary breaker. The minus four inch will fall through the openings of a coarse grate lying directly over the jet pump hopper. The secondary breaker has drum mounted breaker picks and a comb shaped spacer plate whose gaps are spaced such that only minus 4 inch pieces can fit within the area created by the teeth and spacer plates.

The full-scale jet pump injector is a scaled-up version of the 1 ton per minute sub-scale injector whose development was described in Section 4.3.2.1 of this report.

This Jet Pump Injector consists of:

1. A nozzle which receives high pressure water (125 psig) and projects it as a straight jet at approximately 136 feet per second.

2. A hopper which mounts coal around the jet.

3. A bell mouth mixing chamber which receives the mixed coal and jet water.

4. A diffuser section which receives coal slurry at approximately 50 feet per second and decelerates the coal slurry to approximately 1 feet per second thereby converting the dynamic head to a static head of approximately 25 psig.

The Jet Pump Injector is a fixed geometry device with no moving parts. The Jet Pump is equally capable of entraining water or coal or a mixture of the two. Most efficient operation occurs when injecting the maximum amount of coal which has all voids filled with water. This occurs when the Jet Pump Hopper is kept filled with coal and a water level is maintained several inches above the Jet Pump axis.

When less than 100% of the design coal rate is fed to the jet there is insufficient coal to maintain a level of either coal or water inside the hopper. Additional water will be required by the jet pump to maintain a fixed water level above the jet pump axis. There will not be a coal level at part load. The control of the secondary water which is added to the jet pump hopper in lieu of coal is the key to controlling the jet pump itself. This, of course, assumes that the nozzle supply water pressure is held reasonably constant.

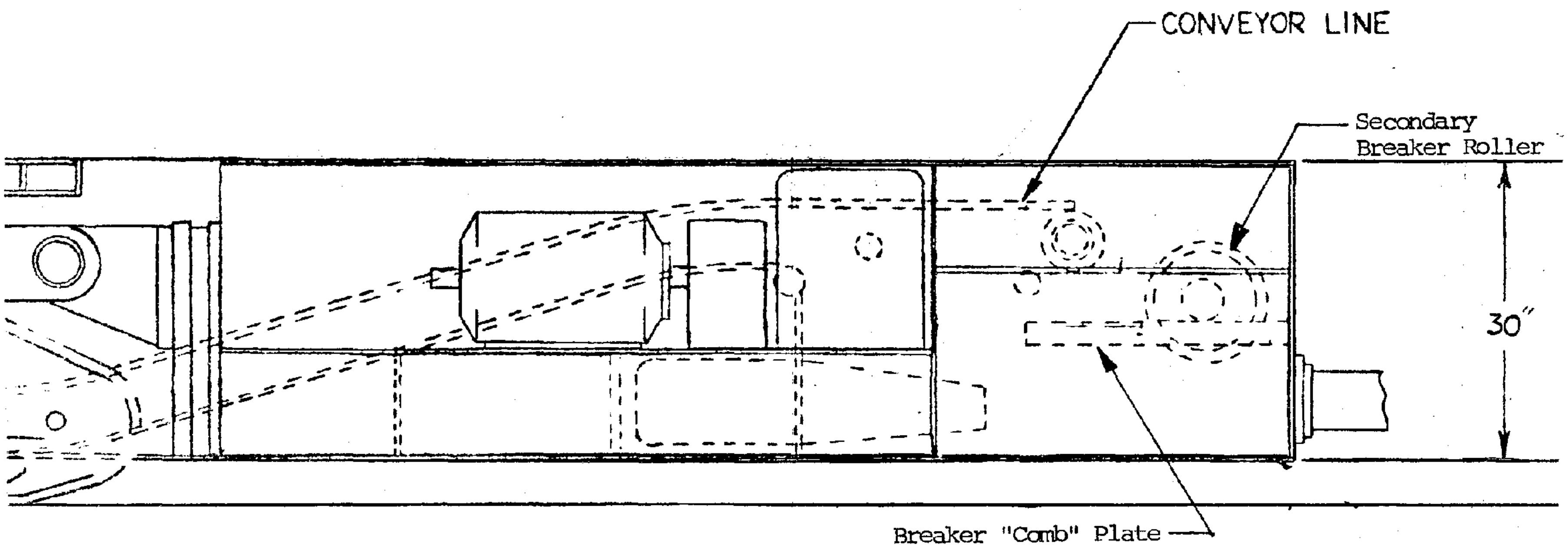


Figure 4-31: Secondary Breaker

Breakage of the jet pump is hard to envision, however it has been established empirically at IRRI and in the technical literature that jet pumps do wear at the entrance to the mixing chamber. This wear rate is not excessive nor is the replacement part expensive to purchase or install. Based on wear tests with 100% coal, the projected throat life in a 4 ton per minute injector would be on the order of 1 million tons of coal. This is a result of an inate feature of the jet pump that the efficiency of the jet pump is somewhat insensitive to mixing chamber diameter at maximum tonnage rates.

4.3.3.1.3 Injector Breaker Vehicle Specifications

The major advantages of the coarse coal Jet Pump Injector are simplicity, compact size, and inherent ruggedness. The design goal of the Injector Breaker Vehicle is to capitalize on these advantages to obtain a highly maneuverable, compact, and rugged vehicle.

The vehicle size was reduced in comparison to a standard Feeder-Breaker by:

- Reducing the surge hopper capacity, to match the surges of a single continuous miner.
- Eliminating the panel conveyor belt equipment.
- Reducing the coal throughput to that of a single continuous miner.

The reliability and resistance to typical mine-environment hazards of existing Breakers was to be maintained on the Injector Breaker Vehicle by utilizing the proven, low seam Breaker technology of the S&S Division of Ingersoll-Rand. Further, the Injector Breaker was to be designed and fabricated to the design and operating specifications given in Table 4-5.

4.3.3.1.4 Monorail Suspended Hose Train

<u>Hose Supports</u>

The major constraint to free movement of the jet pump

TABLE 4-5

INJECTOR BREAKER VEHICLE DESIGN AND OPERATING SPECIFICATIONS

Breaker Capacity	Coal	8 TPM
	50% Coal 50% Rock	5 TPM
Jet Pump Capacity	Coal or Rock in any Proportion	8 TPM
Tramming Speed (Maximum)		50 FPM
Tramming Grade Capability		20% Grade
Overall Vehicle Dimensions	Length	18 Ft.
	Width	8 Ft.
	Height	3.5 Ft.
Ground Clearance		6 Inches
Overhead Clearance		6 Inches
Number of Operators Required		1
Automatic Secondary Water Control for Injector Operation		

injector chassis would be the trailing hose train which connects the injector with the fixed pipe haulage line which passes out of the mine. A monorail network fixed to the ceiling of the central entry and all crosscuts would allow the hose train unrestricted movement in the entries between the Injector Breaker and the fixed pipe haulage line. As shown in Figure 4-32 the hose supports consist of standard monorail components which have proven commercially successful in such demanding environments as foundries.

The monorail train would consist of non-powered, free wheeling hose carrying frameworks spaced four feet apart throughout the hose length. The perimeter frame supporting the hose would consist of three parts: a main frame and two sub frames each of which holds only an upper or lower hose. This arrangement would allow easy removal of a short length of damaged hose without requiring disassembly of a greater portion of the train. Each perimeter frame cradles the hose through the bottom 180 degrees. This specification and the four foot spacing is per hose manufacturer's recommendations to obtain maximum hose life.

Two idler wheels are also mounted on each side of the perimeter frame to receive sideloads which may develop at either of the two corners of the central entry. This sideload will develop because the monorail train is not propelled uniformly along its length and lateral movement will be restrained by metal corner guides.

The monorail train is powered in the forward direction (towards the injector) by means of monorail suspended tractors, as shown in Figure 4-33, which are interspersed at suitable intervals along the monorail train. High traction is developed by the use of rubber-pinch rollers which bear against the bottom of the monorail track. This train is retracted by a combination of the above tractors and a loop-forming, take-up carriage which transmits its thrust to the hose through the side mounted idler rollers.

Tension is maintained throughout the hose train by the use of short lengths of parallel steel cable offset at equal distance from the hose train center line. Steel cable is used to withstand tension because slurry hoses are not sufficiently strong to withstand tension of the magnitude expected. Two offset parallel cables were used in place of one because of a problem which develops with single cables when going through turns.

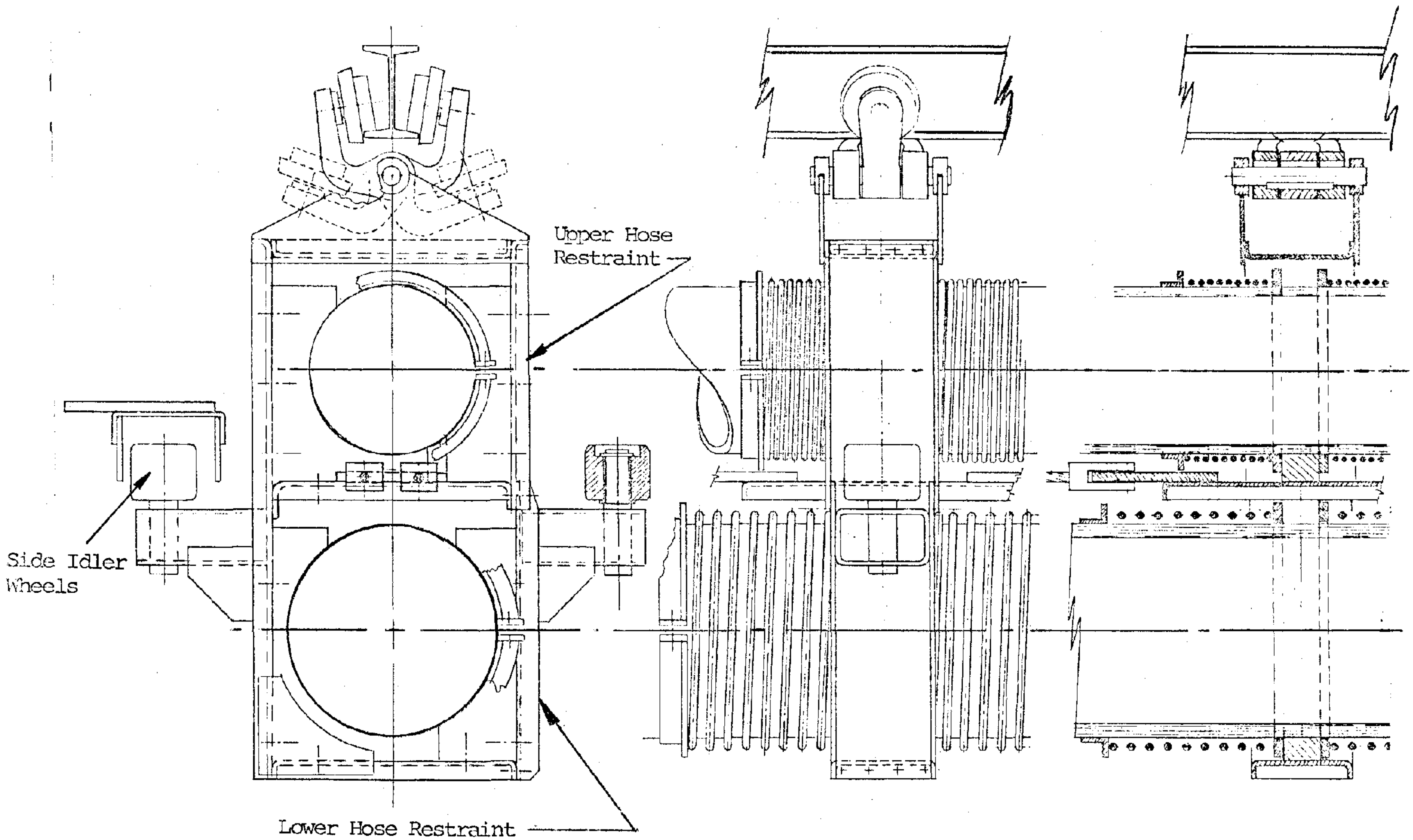


Figure 4-32: Hose Supports (Typical)

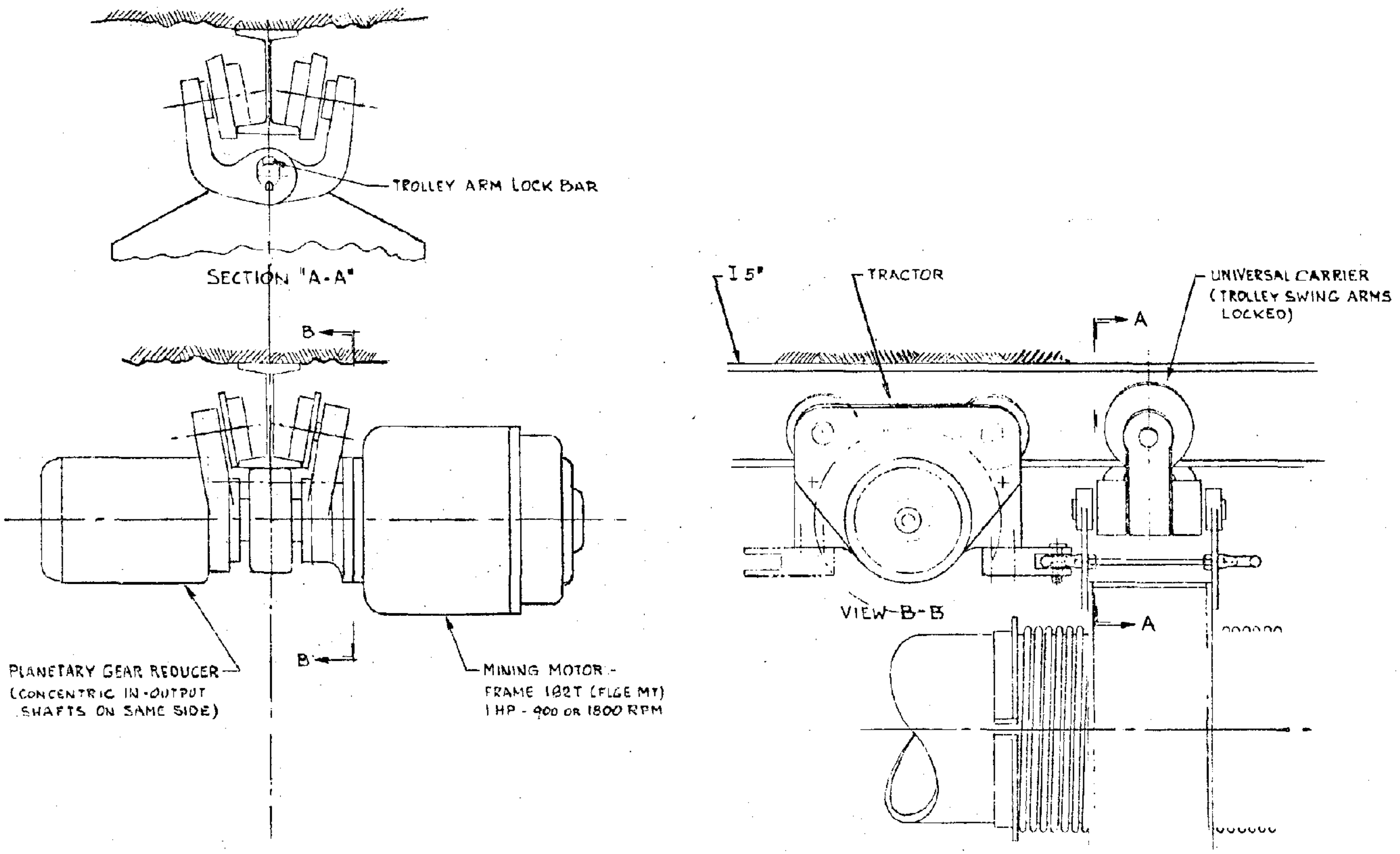


Figure 4-33: Monorail Tractor (Typical)

In a turn the steel cable would tend to straighten and form a cord of the turn arc. Because the hose and cable must be of the same length, buckling of the hose would occur. By utilizing two offset parallel steel cables this effect can be negated.

The last carriage at the injector end of the monorail-suspended portion of the hose train is unique in that it is a self powered tractor and has two suspended wheels which clear the mine roof by a few inches. The outrigger wheels are to limit side-sway caused by unusual side loadings at the monorail end such as side-pull by the ground transition hose length.

Corner Guides

Because of the nature of the mining plan used for multiple entries there are only three locations where the Monorail Suspended Hose must round a corner. They are located on both sides of the central entry, the corner connecting the central entry, and the crosscut containing the looped hose. The corner guides utilized are shown in Figure 4-34 and would consist of three modified Roof Support Jacks which would locate a curved piece of channel steel which matches the turn radius of the monorail mounted idler rollers. This configuration was chosen because it is light, easy to install, inexpensive and reliable. The three corner guides, once installed, would remain in place until the entire monorail system is advanced one or two crosscuts forward.

Extensibility - Loop Carriage

Extensibility of the monorail train is created by the loop-forming carriage which plays out or retracts hose much like a moving pulley. The two functions of the loop carriage are to support the hose train loop and provide tension to retract the monorail-suspended hose train. This element of the system is shown in place in Figure 4-27. Additional detail of the loop carriage is given in Figure 4-35.

The carriage is powered by a single electric motor with an extended shaft such that traction is supplied to both suspended monorails. Power is received by a coiled cable which is played out and retracted as the carriage traverses back and forth in the crosscut, thus avoiding dragging its power cable over rough ground. The monorail hose train measures 230 feet excluding the ground transition hose. The

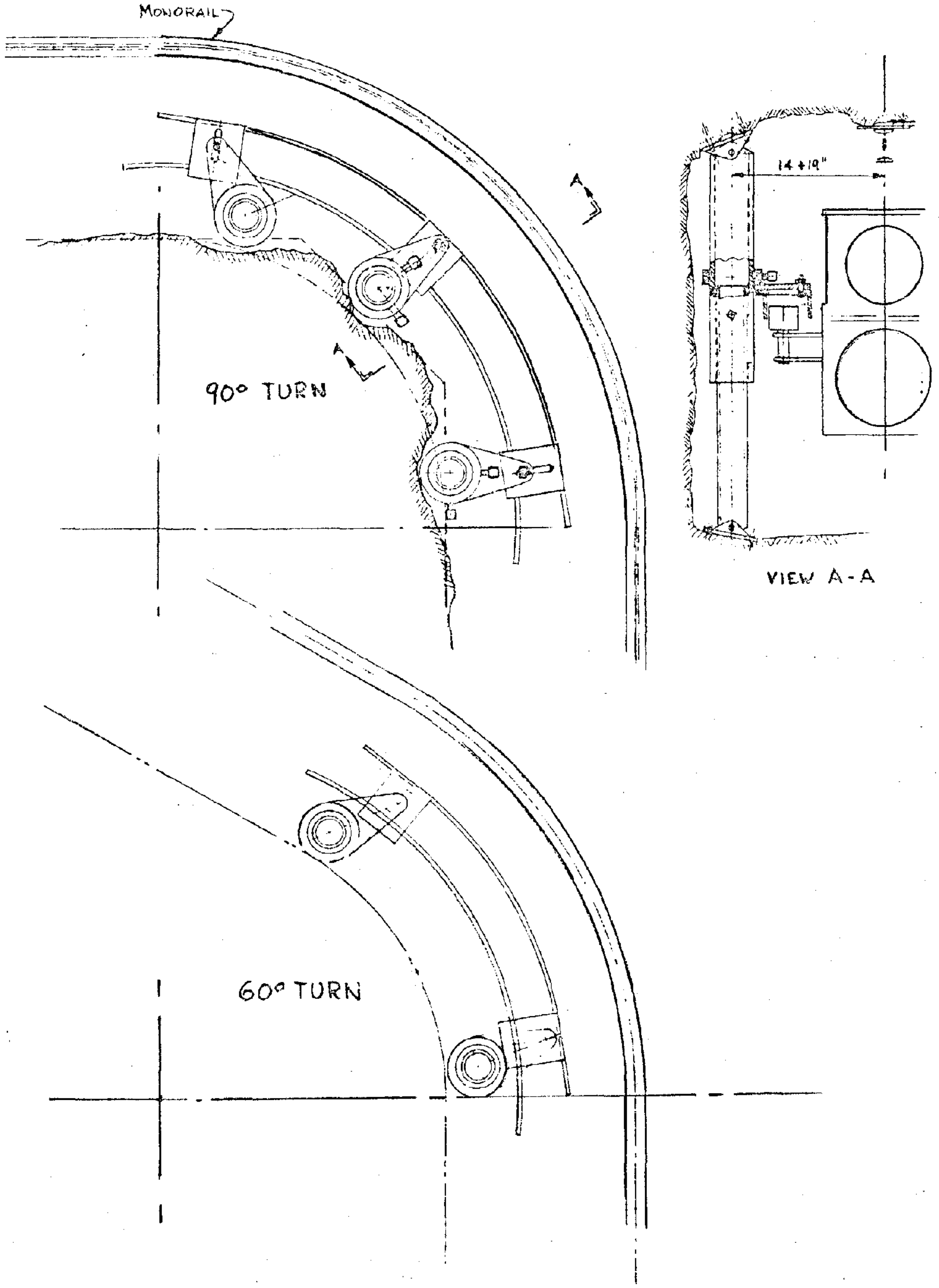


Figure 4-34: Corner Guide (Typical)

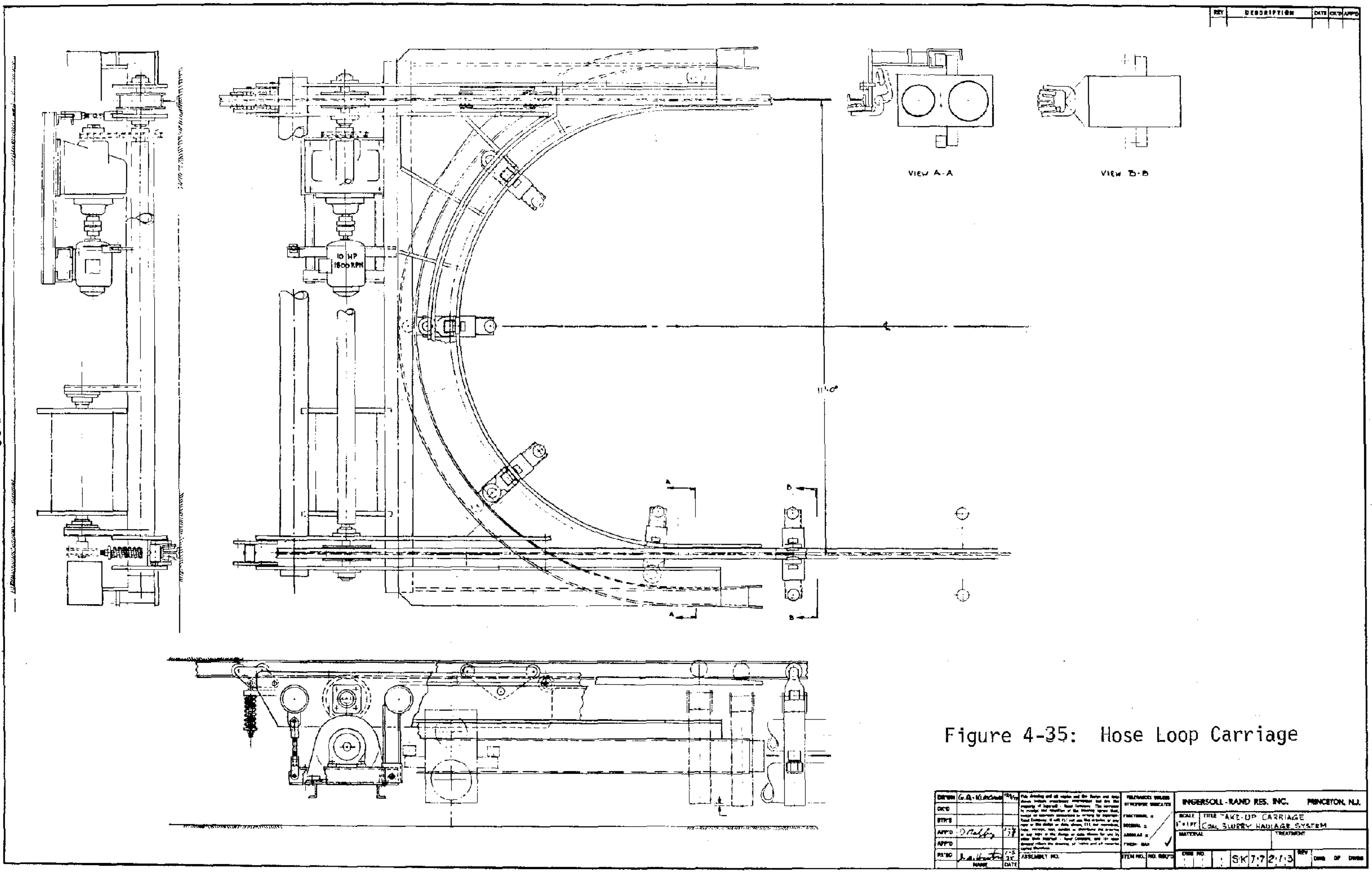


Figure 4-35: Hose Loop Carriage

160 feet of the monorail train closest to the booster pump must be capable of being decoupled from the monorail to pass around the loop carriage. This decoupling will take place automatically by camming wedges which unlock one side of the monorail wheels allowing them to sweep free. The hose supporting frameworks can then be lifted and removed from the moving side monorail to pass across the loop carriage and be remounted on the stationary side monorail where the moveable idler wheels are swung back and locked back into position. The front half of the monorail train does not require this feature as it does not pass across the loop forming carriage even when fully retracted.

Transition Hose

Several problems arise when attempting to connect the freely moving Injector Breaker Vehicle to the monorail suspended hose train. A 92 foot long length of ground mounted transition hose was devised to alleviate these problems. This element of the overall system is shown in place in Figure 4-27. Additional detail of the transition hose is shown in Figure 4-36.

The transition hose is supported by approximately 20 rubber tired carriages which are connected but free to pivot. Their minimum bend radius is limited by chains mounted a specific distance to each side of the centerline. Carriages are supported by free wheeling two foot diameter pneumatic wheels with 8 inches tread width. The wheels are oversized to allow their use on rough and wet bottom. The hose transition length must follow the squirm steering movement of the Injector Vehicle without receiving excessive side loads on the free wheeling ground wheels. This accommodation is attained by means of a pantograph support which extends from the rear of the injector chassis to the first ground supported carriage. The length of this cantilevered pantograph arrangement will be determined by experience although it is anticipated that a length of 8 feet would be sufficient. The Injector Breaker Vehicle can minimize this movement by centering itself in the crosscut, thus allowing the continuous miner to make both the first and second box cuts with the Injector Breaker having only to back up and advance in a straight line. The Injector Vehicle can still receive coal by orienting the "stinger" of the continuous miner such as to always feed the centrally located hopper on the injector.

4.3.3.1.5 Monorail Haulage System Advance

Monorail track would normally be installed continuously

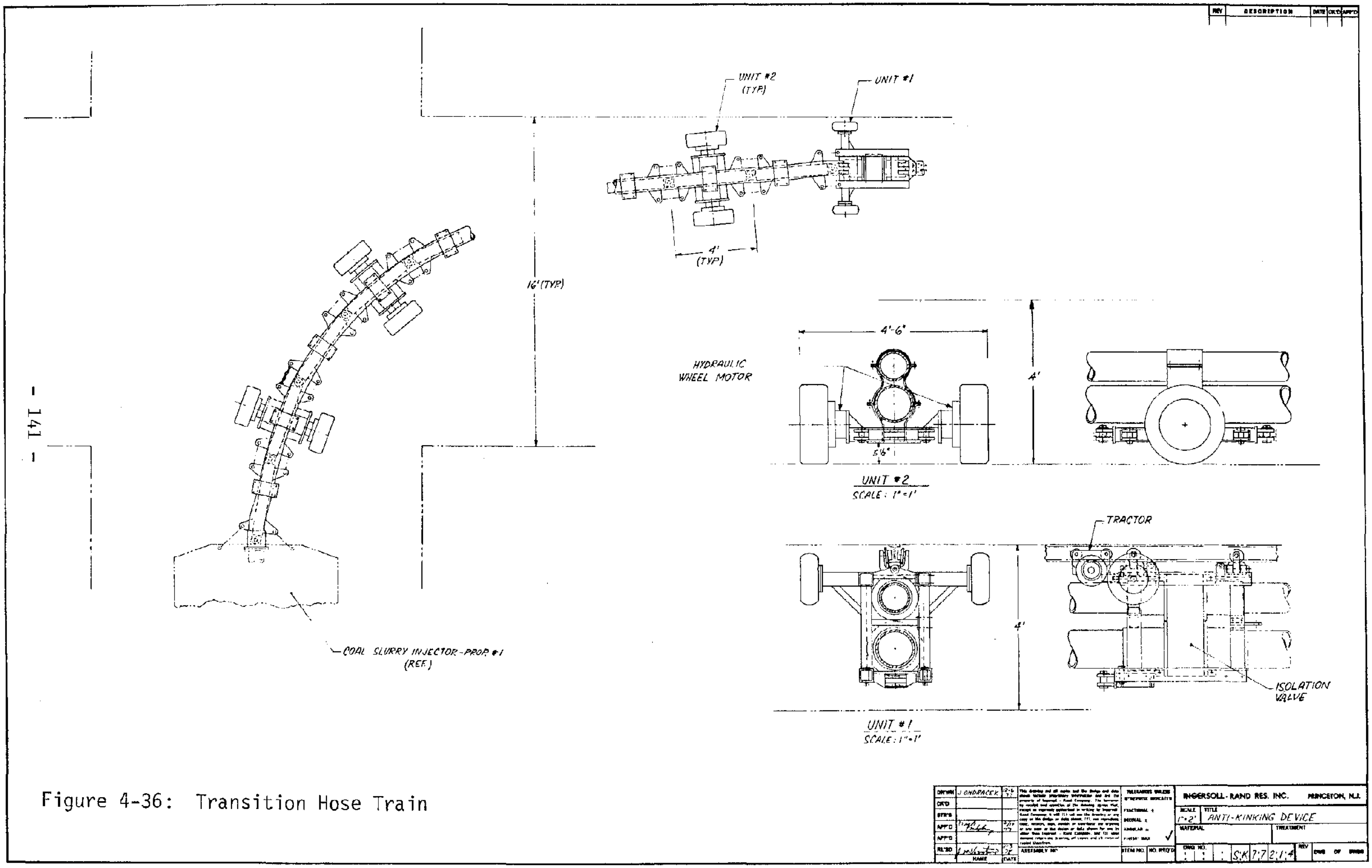


Figure 4-36: Transition Hose Train

at the working section by a single arm roof bolter. Advance of the monorail system, including the loop-forming carriage, would take place as a third shift maintenance operation. The advancing move would consist of four basic steps as follows:

-Disconnect the hose train at the booster pump, and at each end of the transition hose. Spillage would be avoided through the use of pinch valves.

-Advance the pump loop carriage to the next crosscut and advance the monorail hose train along the monorail track.

-Advance the rigid pipe line and connect the pump.

-Reconnect the hose train to the Injector and booster pump.

While the haulage system could be advanced by as few as four men, the use of six men would be preferred. The crew would include the continuous miner operator, mechanics for decoupling the quick disconnect hoses and support helpers. The actual sequence of events would proceed as follows:

1. Fully retract the complete haulage system. Close three pairs of air or hydraulically actuated valves (Figure 4-37).

2. Separate the hose pairs and pump at three locations (Figure 4-38).

3. Remove the hose from the loop carriage. This is done by advancing the hose 40 feet and driving the loop chassis 160 ft. off the end of the paired monorails (Figure 4-39).

4. Advance the pump.

5. Install the hard pipe.

6. Tram the continuous miner to the loop carriage and lower the loop carriage to the ground.

7. Connect and tow the loop carriage through one or two crosscuts and mount it on the

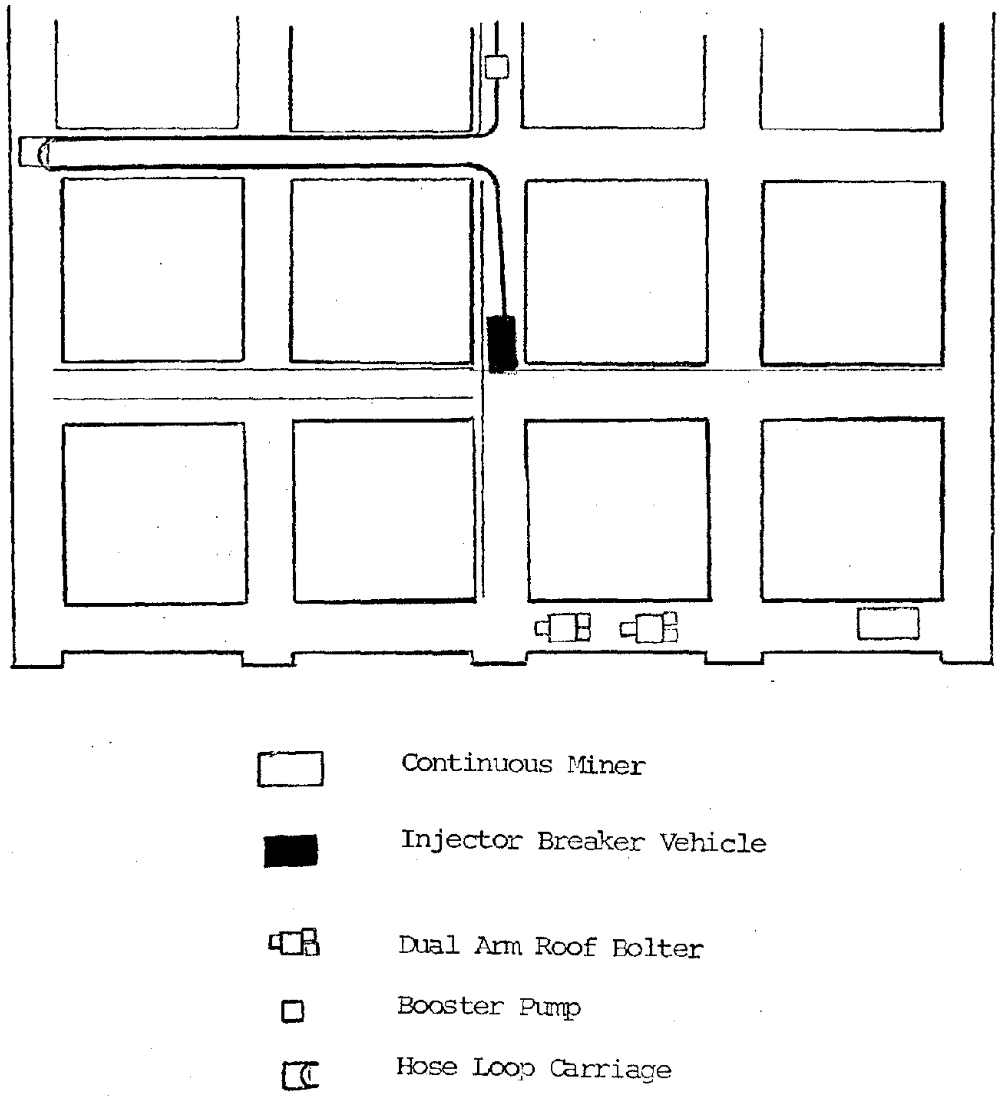


Figure 4-37: System Advance, Step No. 1

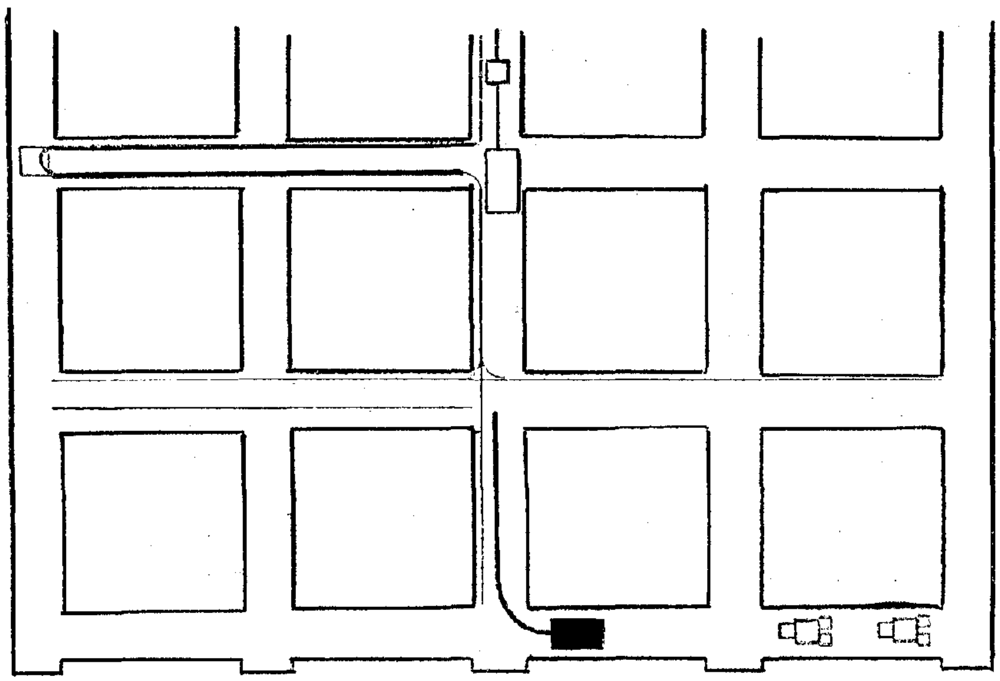


Figure 4-38: System Advance, Step No. 2

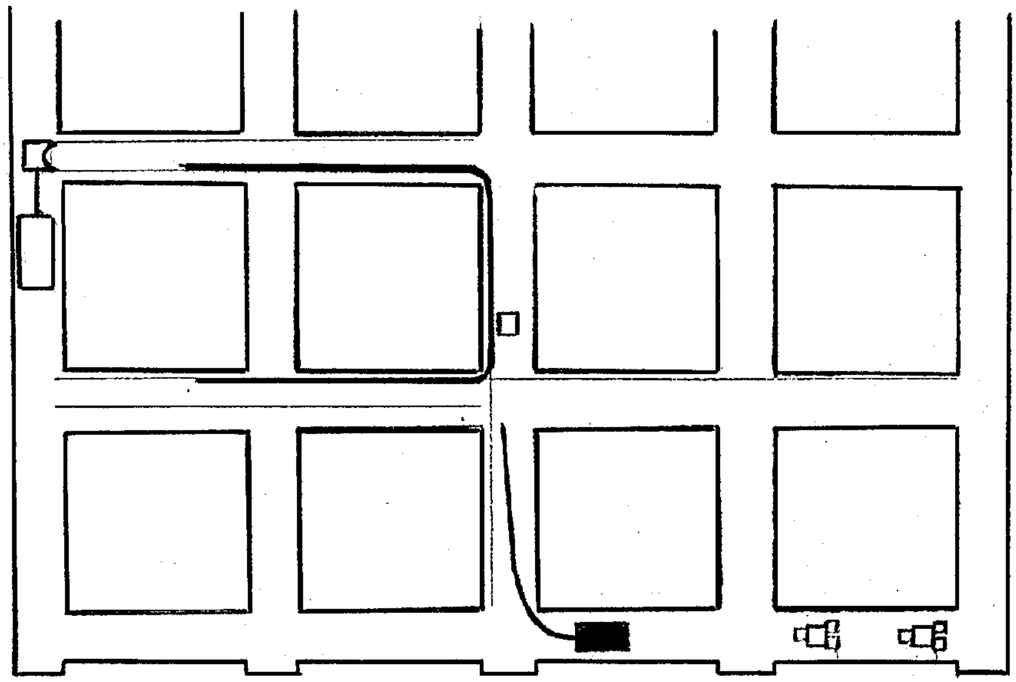


Figure 4-39: System Advance, Step No. 3

new monorail (Figure 4-40).

8. Tram the injector vehicle forward and also advance the hose train forward (Figure 4-41).

9. Connect hoses.

The loop carriage can be lowered or raised by 4 hydraulic jacks which are permanently mounted to the carriage. The centrifugal slurry booster pump is skid-mounted and is light enough to be towed by a roof bolter or continuous miner. Similarly, the loop carriage is fitted with two removal skids for dragging forward to the advancing crosscut.

4.3.3.1.6 Mining Plan

A multi-entry advance mine plan was used to:

1. determine the maximum potential productivity of a Hydraulic Haulage System, and

2. discover and correct equipment blockages, conflicting or unsafe procedures. A new mine plan was then developed for the Monorail concept.

A major factor in the acceptance of a Hydraulic Haulage System will be the ease in which the system can be integrated into a standard 5 entry mining plan. For this reason a study was undertaken to simulate the operation of the monorail suspended Hydraulic Haulage System in a conventional mine plan with 5 entries. The mine plan could be characterized as follows:

5 entries

80 ft. entry centers

18 ft. entry width

20 ft. box cuts

16.6 minutes per box cut for a continuous miner, including 3 minutes of maneuvering for the second cut

26 minutes per box cut for a dual arm

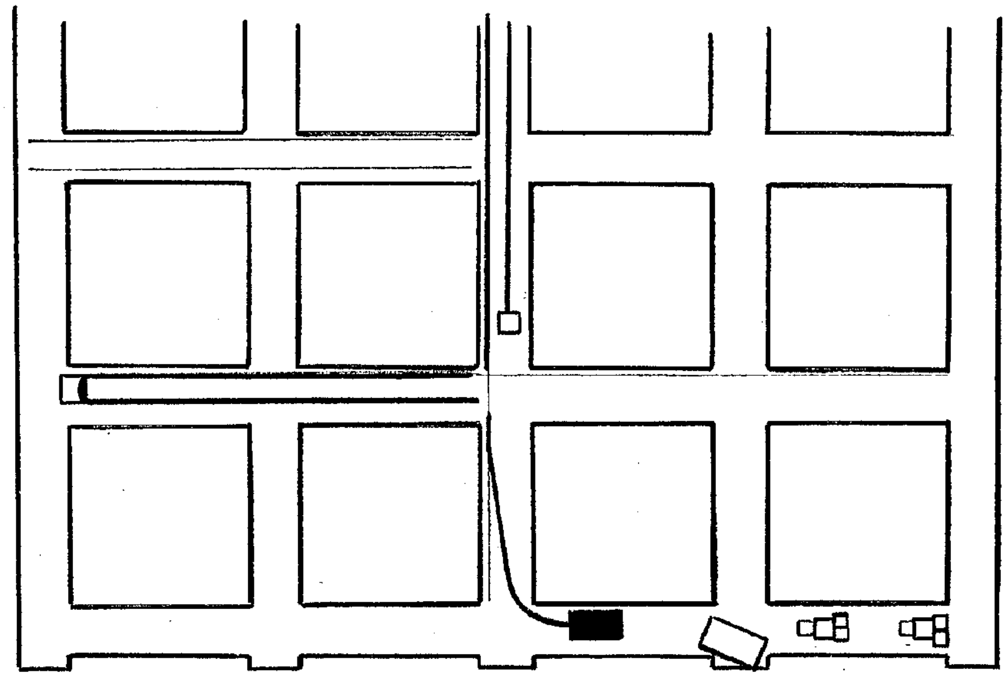


Figure 4-40: System Advance, Step No. 7

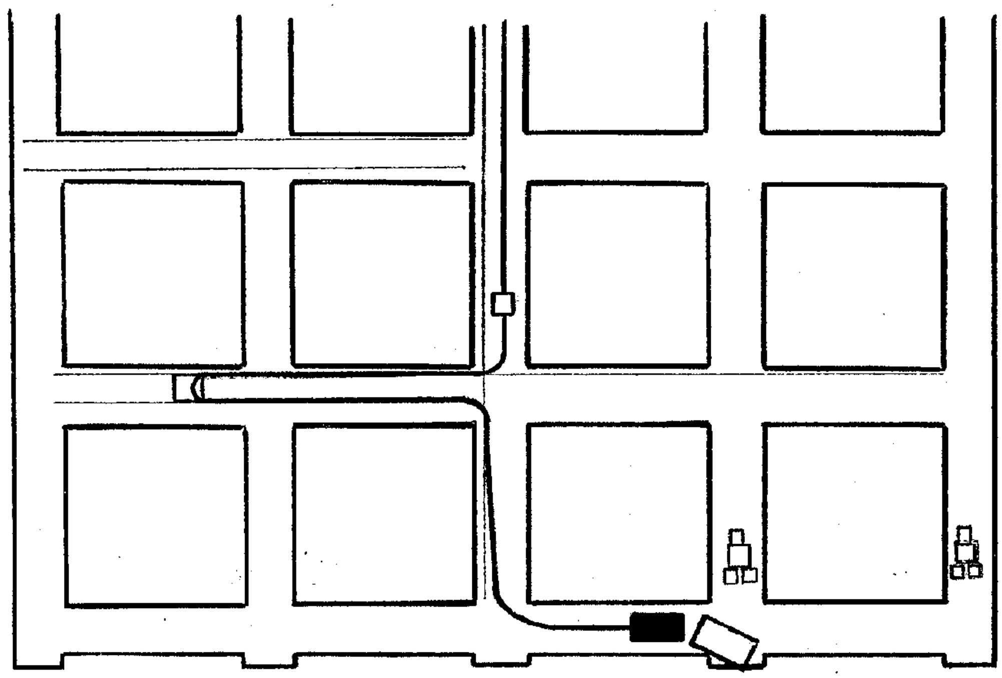


Figure 4-41: System Advance, Step No. 8

roof bolter

4 foot seam heights

Tramming speed of 40 feet per minute for a continuous miner

Two dual arm roof bolters

Figures 4-42 through 4-51 depict a continuous miner, a Hydraulic Haulage System and two dual arm roof bolters advancing a 5 entry system through the distance of one crosscut. In addition, one means of ventilating the mine plan is also shown. Not shown are rock dusting and other service functions.

As shown in Figure 4-43 the monorail hose train is retracted almost completely to allow the continuous miner to make the first box cut number 1. Two dual arm roof bolters are parked in entries 4 and 5 waiting on the continuous miner to bolt the first and second box cuts.

In Figure 4-45, the continuous miner is in the second entry beginning the third box cut while both dual arm roof bolters are in the first and second box cuts respectively. It should be noted that the injector chassis receives coal and injects it only when positioned behind a continuous miner. The injector can receive coal and advance to follow the miner as it cuts, but coal injection normally does not take place while tramming between box cuts.

Figure 4-47 illustrates the haulage system in its most fully retracted position. It can be seen that the loop forming carriage has passed fully into the number 5 entry.

Figure 4-48 shows the haulage system fully extended. The value of its length is shown here in that the transition hose allows the injector chassis to reach the continuous miner working in the advancing cross cut without the use of monorail in any but the center entry.

The remaining figure, Figures 4-49, show the repeated pattern used for completion of the 5 entry advanced mining plan.

Utilizing the mining plan illustrated in Figure 4-42, a time motion study was completed to evaluate the maximum potential productivity of the Hydraulic Haulage System. The

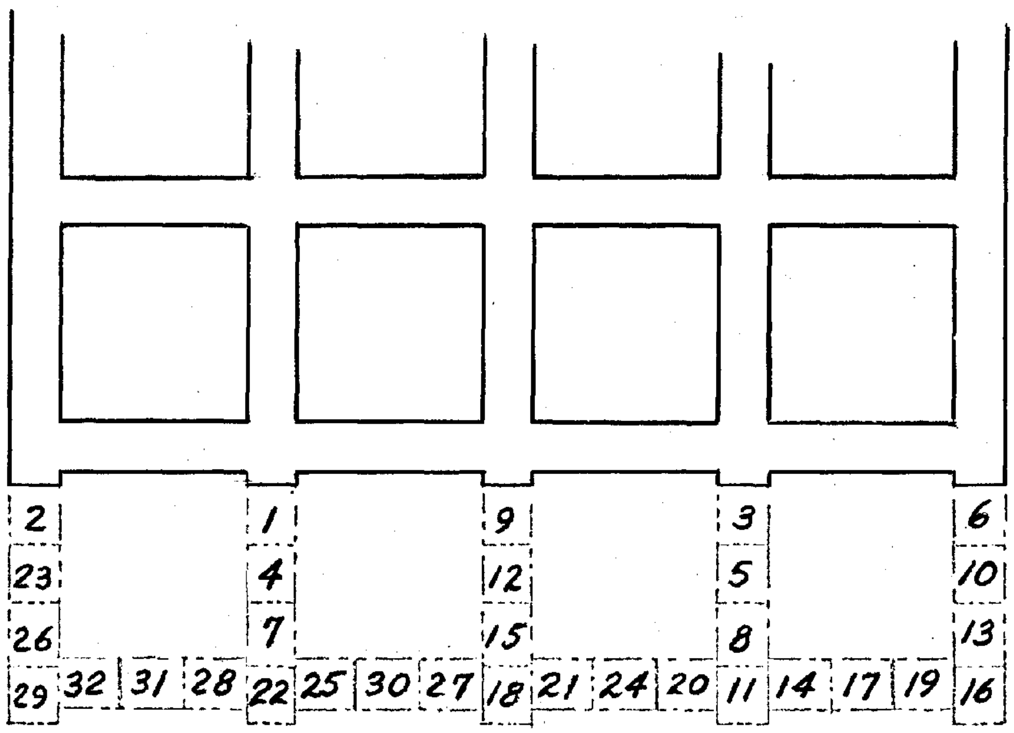


Figure 4-42: Mine Plan; Sequence of Cuts for Five Entry System

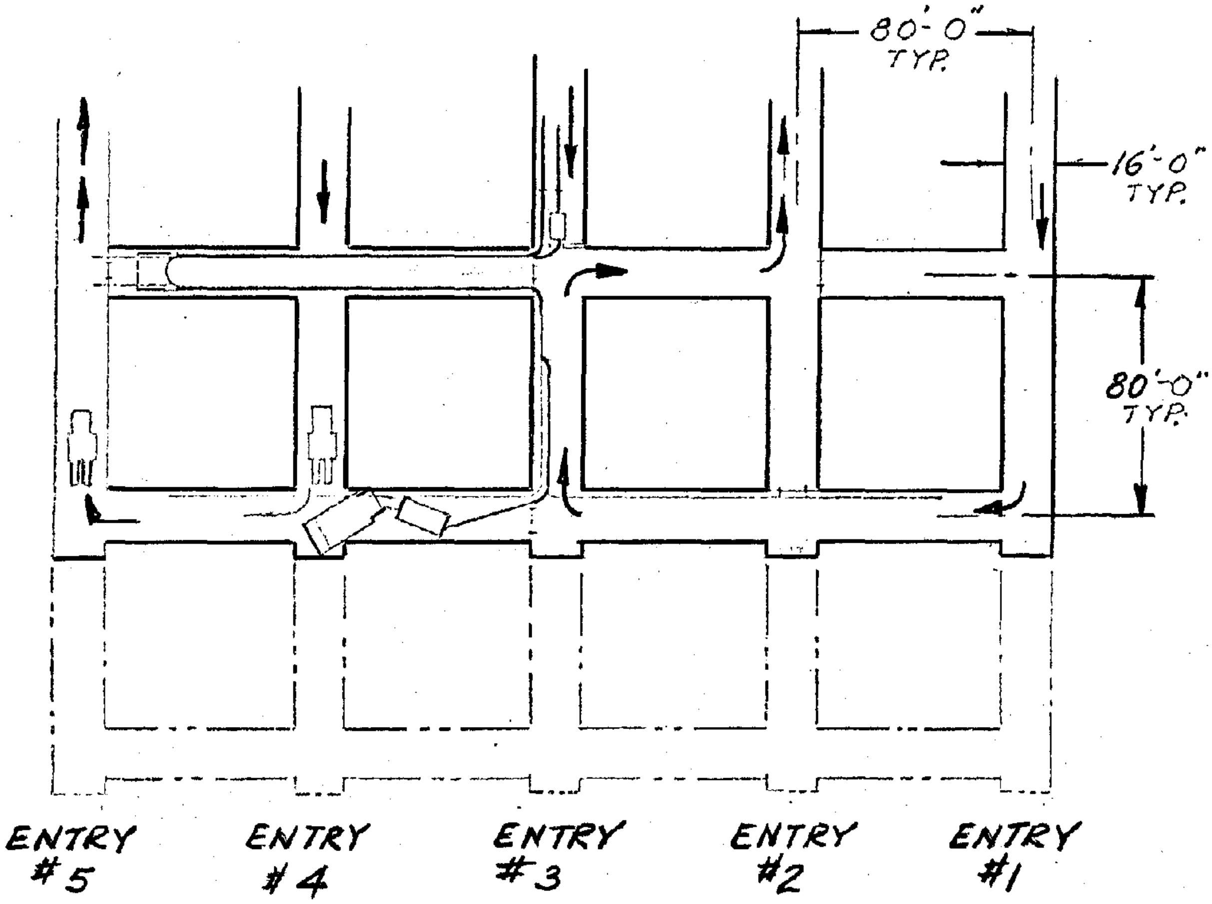


Figure 4-43: Mine Plan; First Cut in Entry No. 4

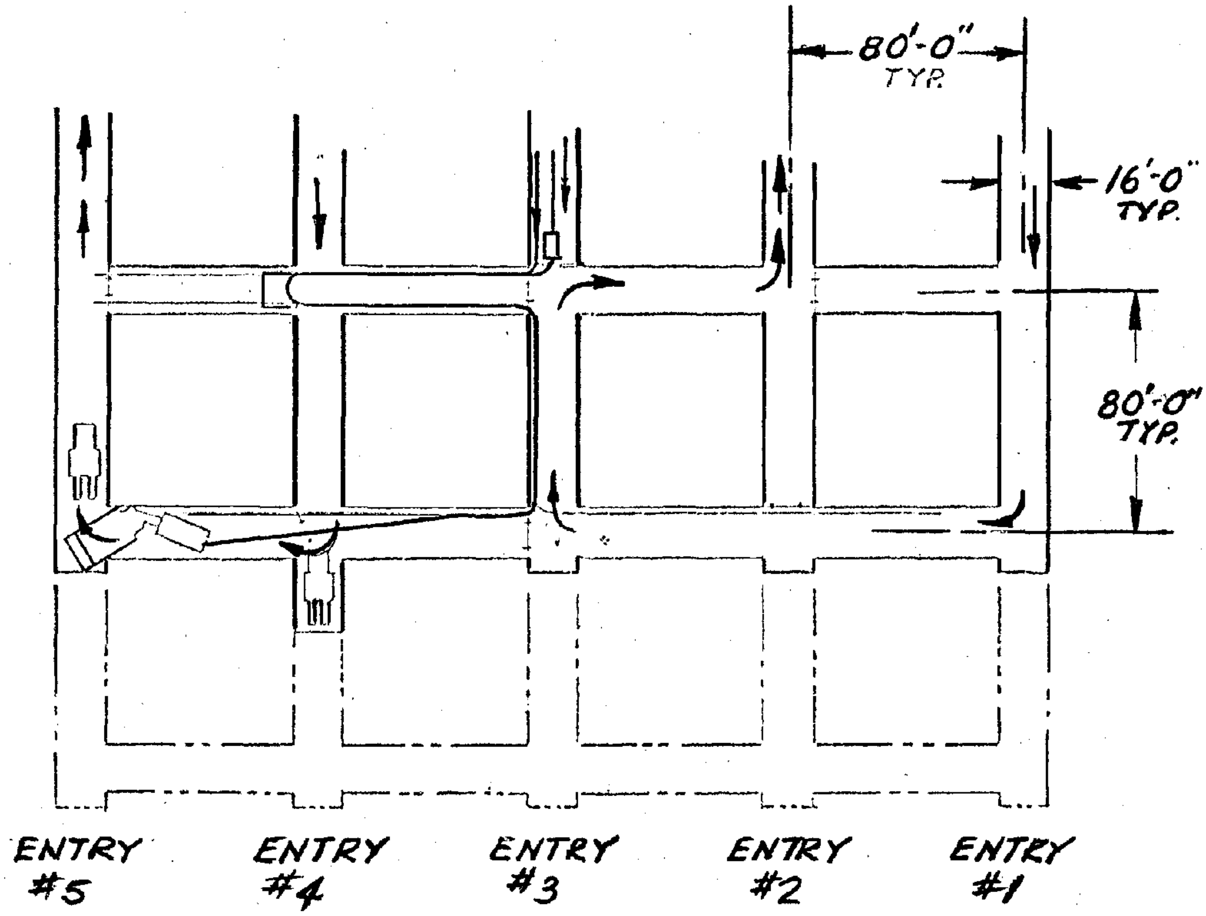


Figure 4-44: Mine Plan; First Cut in Entry No. 5

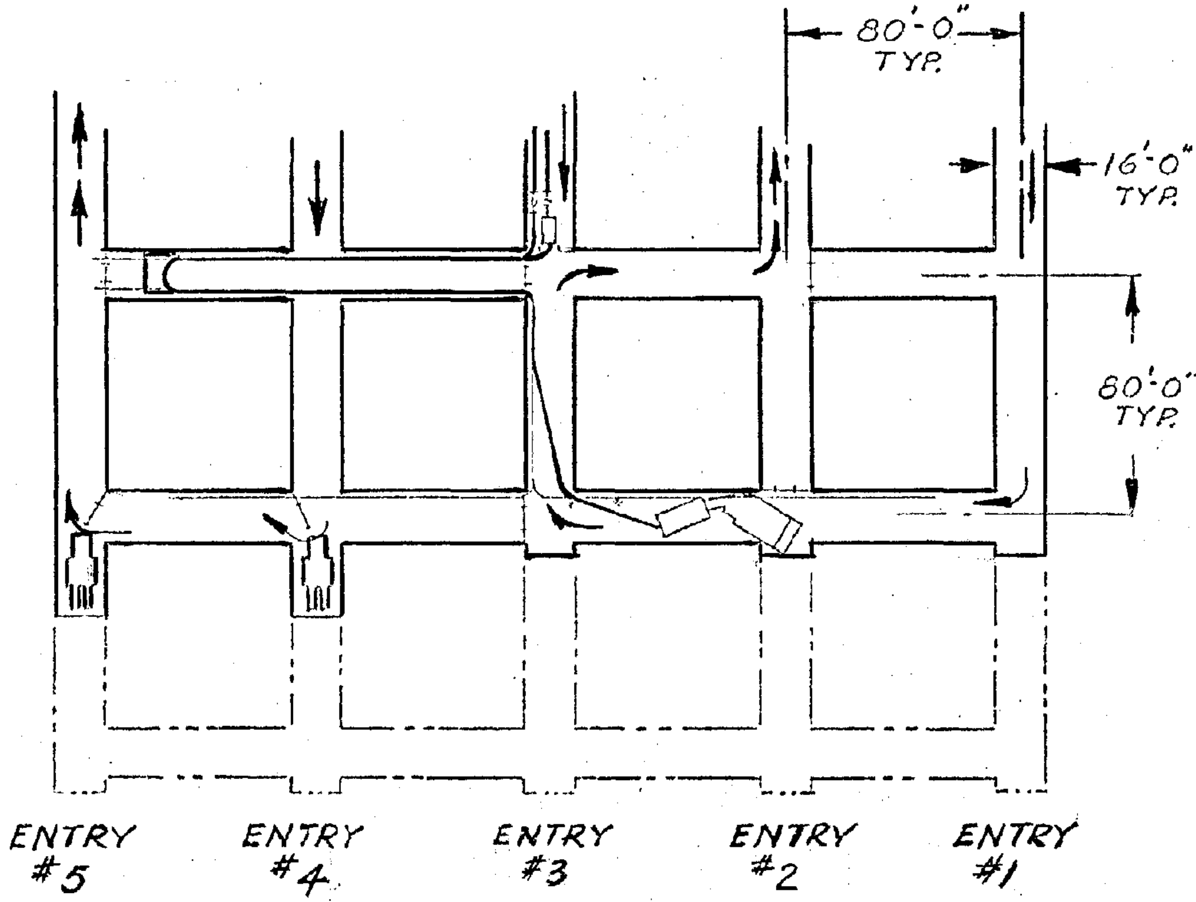


Figure 4-45: Mine Plan; First Cut in Entry No. 2

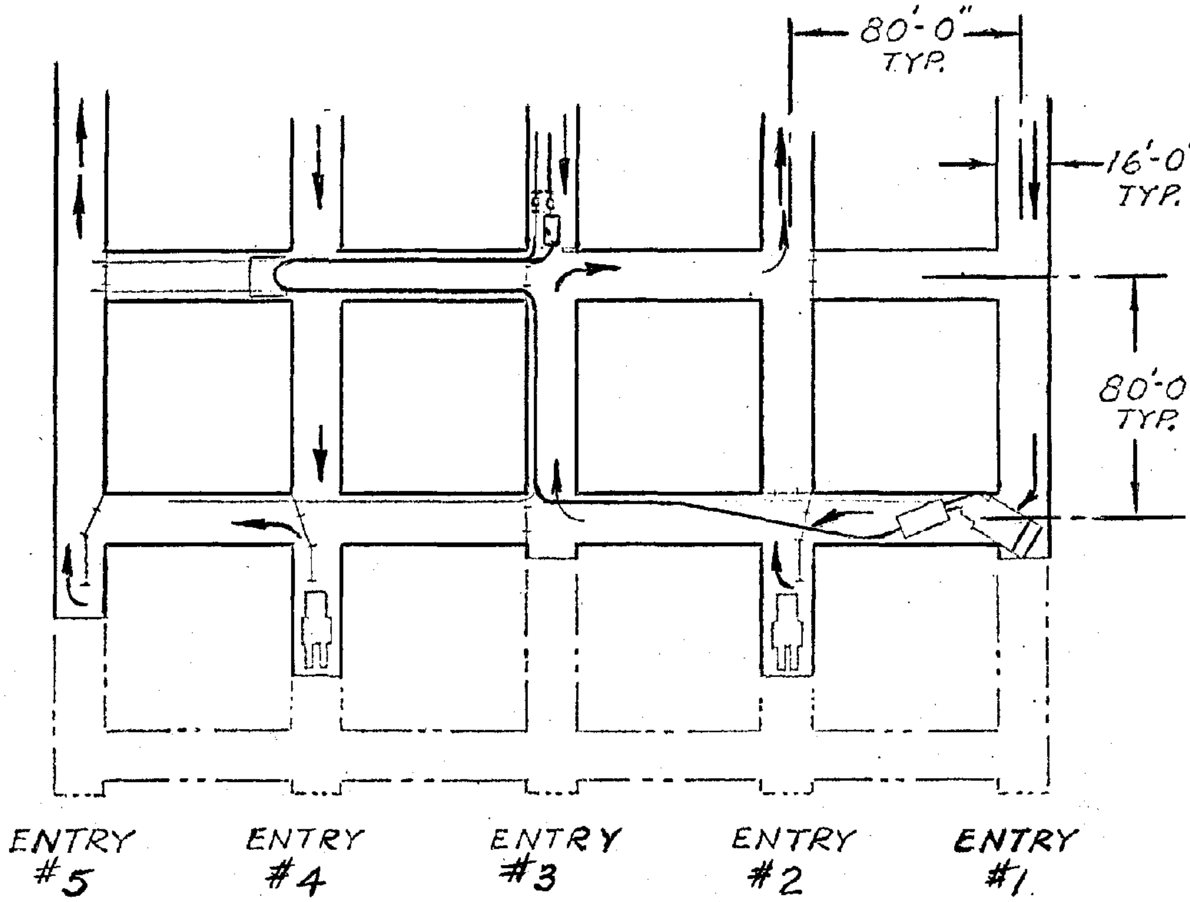


Figure 4-46: Mine Plan; First Cut in Entry No. 1

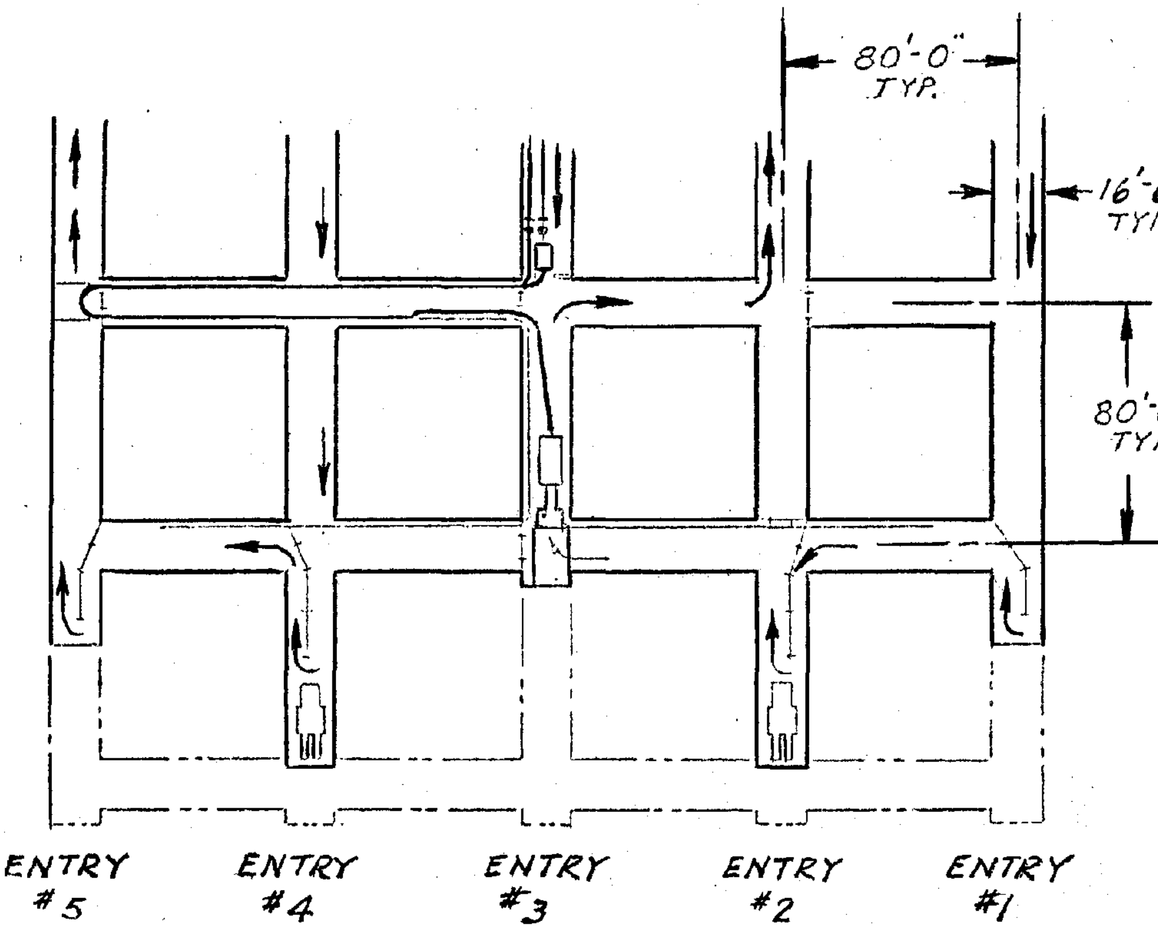


Figure 4-47: Mine Plan; First Cut in Entry No. 3

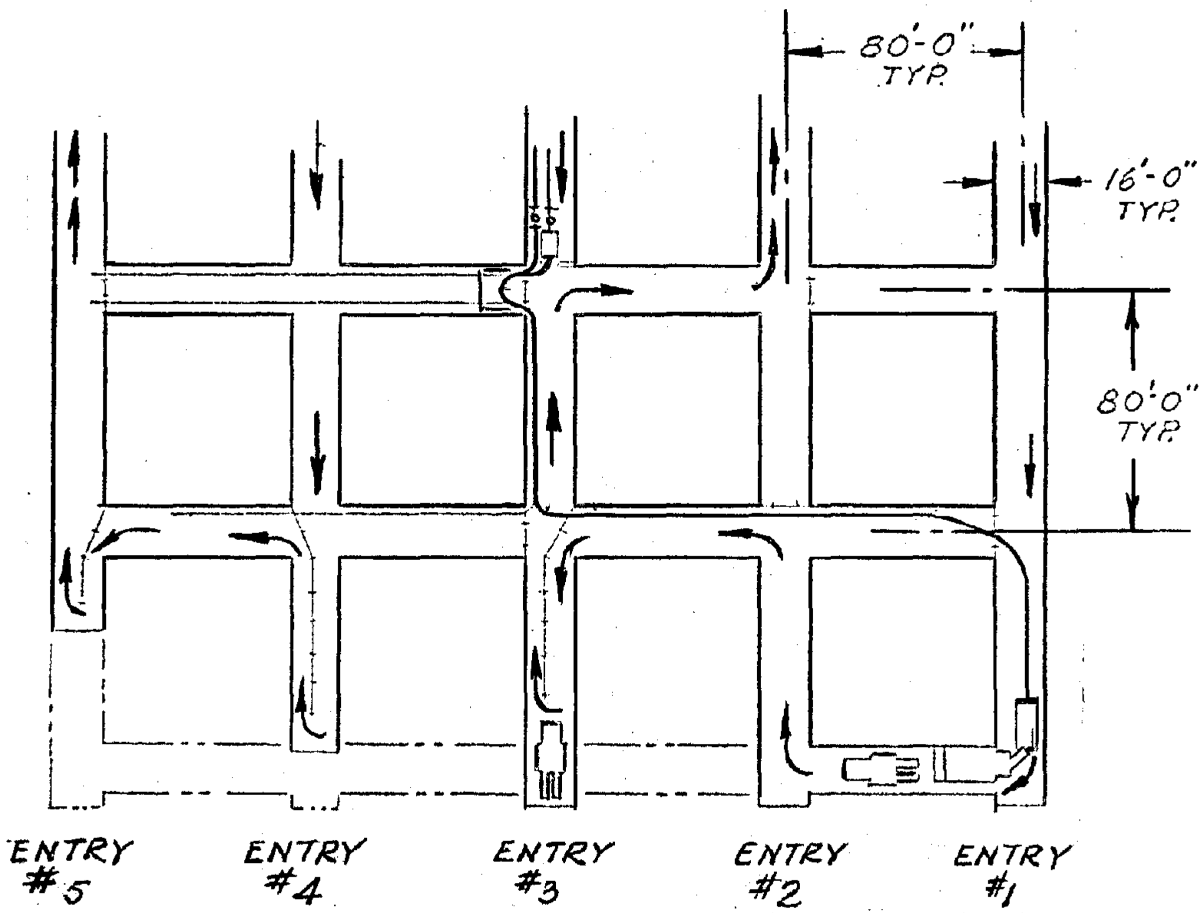


Figure 4-48: Mine Plan; Last Cut in Entry No. 1

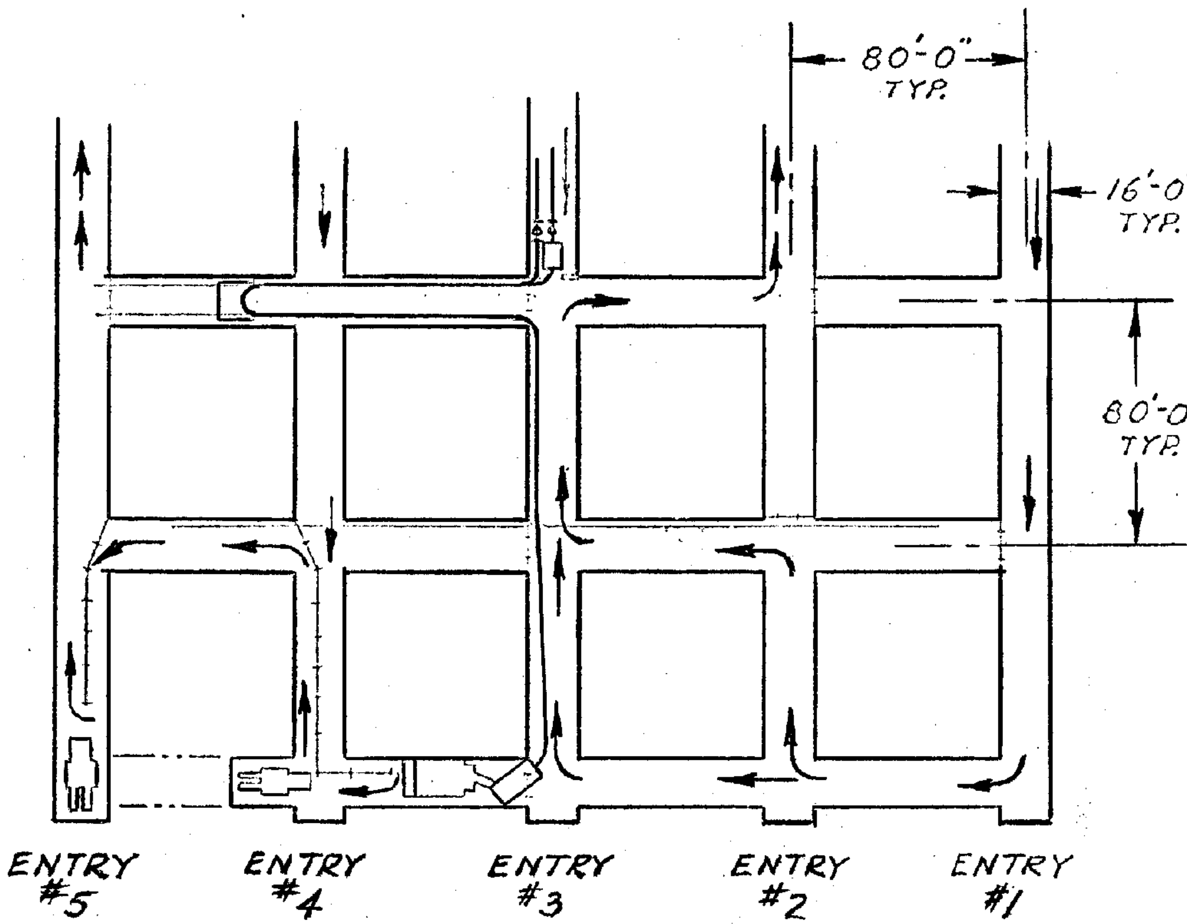


Figure 4-49: Mine Plan; Last Cut in Entry No. 3

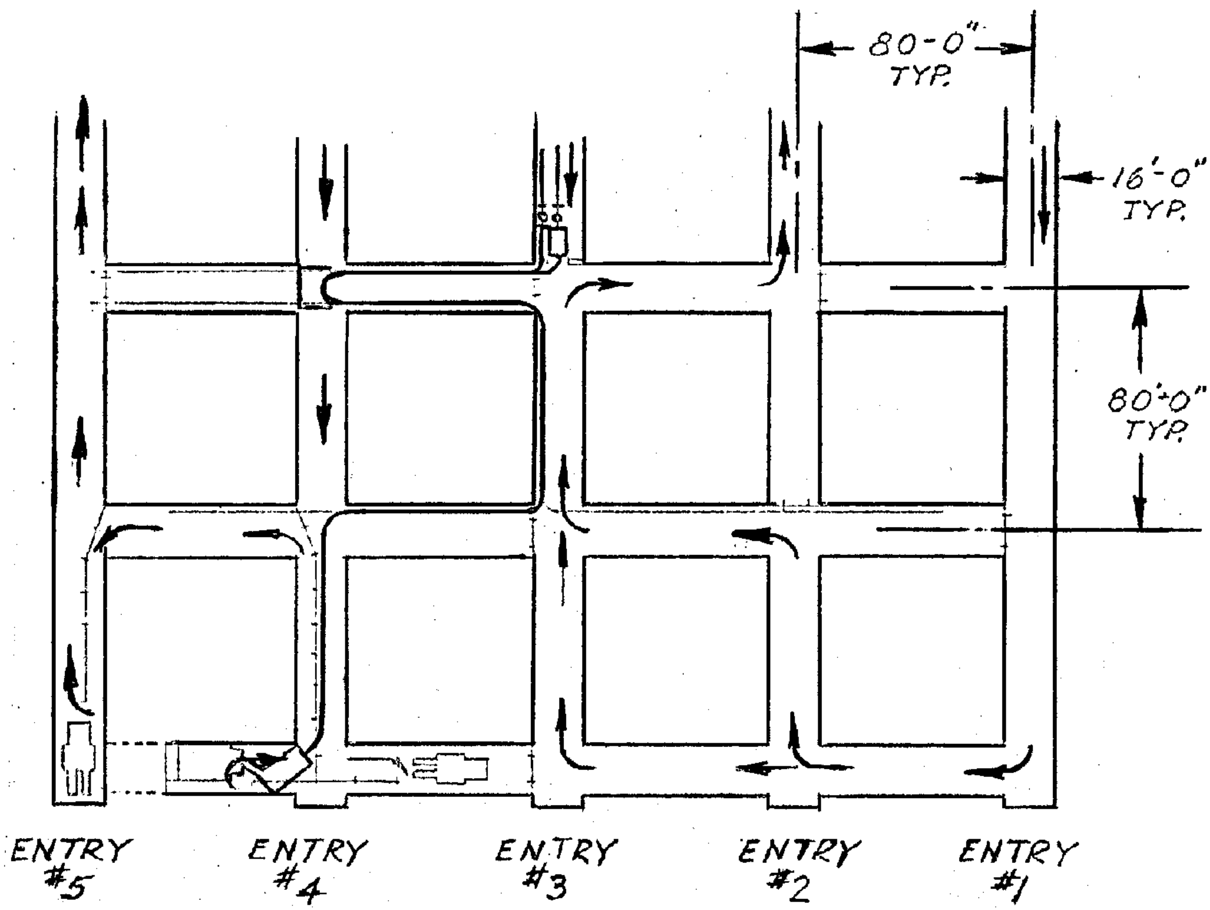


Figure 4-50: Mine Plan; Last Cut in Entry No. 4

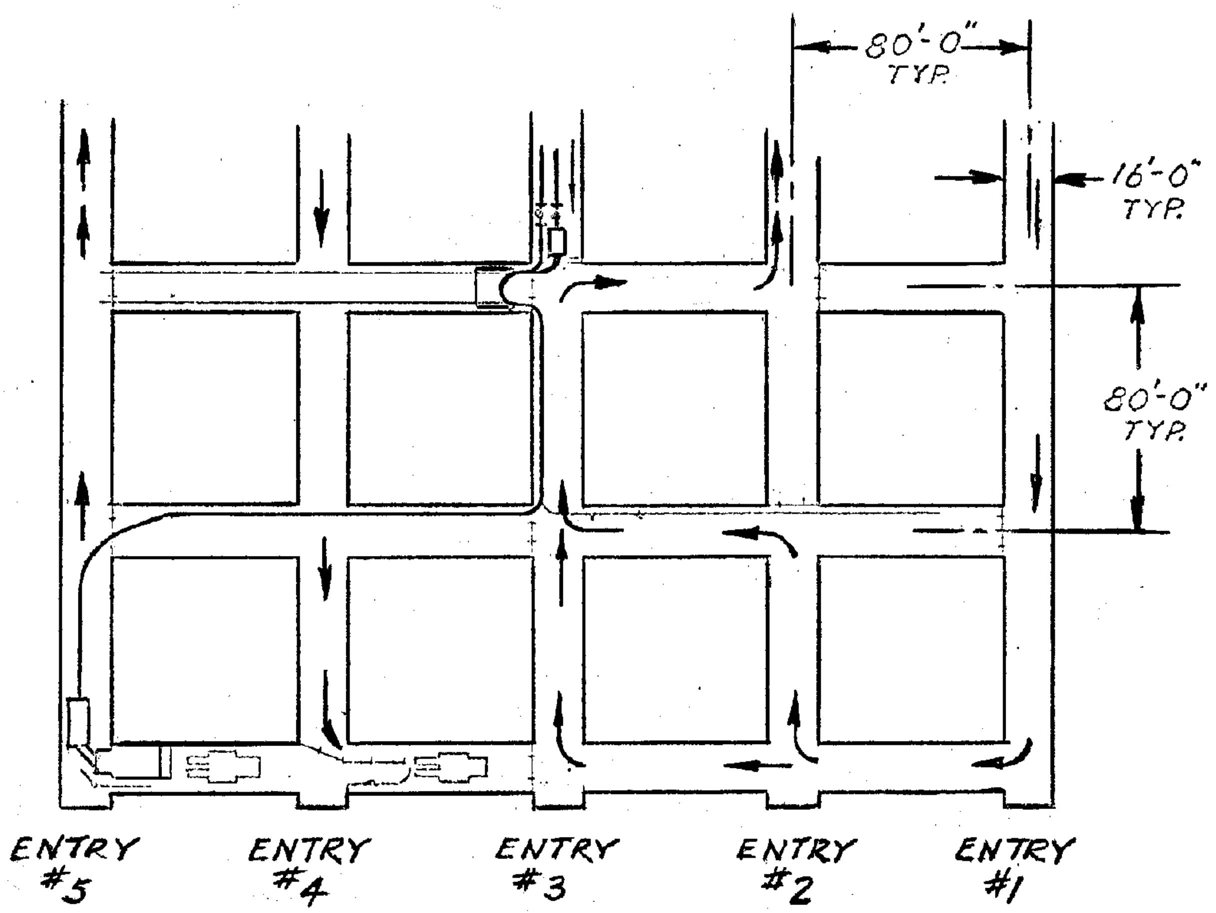


Figure 4-51: Mine Plan; Last Cut in
Entry No. 5

continuous miner was viewed as the bottleneck to productivity because of time lost in its moving from one place cut to the next. The continuous miner is a larger and more cumbersome vehicle than the Injector Breaker Vehicle and its power cables must be moved by hand. The Injector Breaker Vehicle is compact and light in comparison to the continuous miner. All control and power lines are suspended with the slurry hoses via the monorail, there are no ground cables to hinder movement, and the monorail suspended hose and power train can tram at reliable, high speed because of the tracks.

Two dual arm roof bolters are employed because a single bolter could not maintain pace with the continuous miner. By utilizing two roof bolters, each roof bolter will have to wait approximately 6-7 minutes for the continuous miner to complete a box cut.

The average tramming speed of a continuous miner is about 40 feet per minute including cable handling. Ninety degree turns require considerable manuevering by the continuous miner; and as a result tram at only 20 feet per minute through turns on essentially a 24 foot radius. Therefore, it takes approximately 1.9 minutes per 90 degree turn. In addition, the side cuts such as cuts 1, 2, 9, 3 and 6 on Figure 4-42 are penalized an additional 1.9 minutes for being equivalent to an additional turn.

As illustrated in Figure 4-42 typical place changing time includes the following:

1. Time to back cut from the previous place cut, number 5 in this example.

2. Time for first turn.

3. Time to go forward 62 feet.

4. Second turn.

5. Time to tram forward 62 feet.

6. Time for the third turn which is into the 6th box cut.

7. Time to cut the 6th box cut.

A similar analysis was done for cuts 1 through 32 and a running total of cumulative time was kept. This accumulated time was, in turn, broken down into time expended in tramming, cornering and cutting. Using the above criteria, the total cumulative working time to advance 5 entries was found to be 865 minutes. Of that time, 165 minutes was spent in negotiating corners, 168.5 minutes in performing straight tramming and 531 minutes in accomplishing cutting. Based on these results, a continuous miner has the potential to cut coal 61.5% of its time in a working section. This time estimate does not include effects of rest stops, breakdowns or any other interruptions. It is simply the ideal maximum productivity limit of the proposed mining equipment and mine plan.

The variance in ideal productivity in contrast to that normally attained does not occur while cutting at the working face itself. Most lost time occurs while tramming to the following box cut. This is due to problems associated with tramming; e.g., stoppage to check equipment, etc.

4.3.3.2 Concentrator Development

4.3.3.2.1 Introduction

This section of the report presents the analysis, design, construction, and testing of four versions of a coarse coal slurry concentrator, the concept of which was originated by Dr. D. T. Kao (Reference 11). The slurry concentrator is a device attached to the discharge of a Jet Pump Injector in the slurry line that removes part of the water from the coarse coal slurry thus creating a higher slurry concentration.

Figure 4-52 shows a schematic of the test facility in which the several candidate concentrators were evaluated.

There are no moving parts either in the Jet Pump or in the slurry concentrator. Each device functions solely because of the dynamic characteristics of the solid-liquid flow through the device. As shown in Figures 4-53 through 4-57 the slurry concentrators consisted of one or more slots cut transverse to the direction of flow and located within the top 90 degrees of the slurry pipe and, in one case, the slot was continuous around the periphery for 360 degrees. Removed water that passed through the slots was collected in a housing and returned to the water reservoir acting as the suction source for the high pressure pump of the slurry test loop.

The rationale for developing a slurry concentrator was to decrease the capital and operating cost of the haulage system. The objective of the test program described below was to demonstrate a concentrator with the ability to produce a

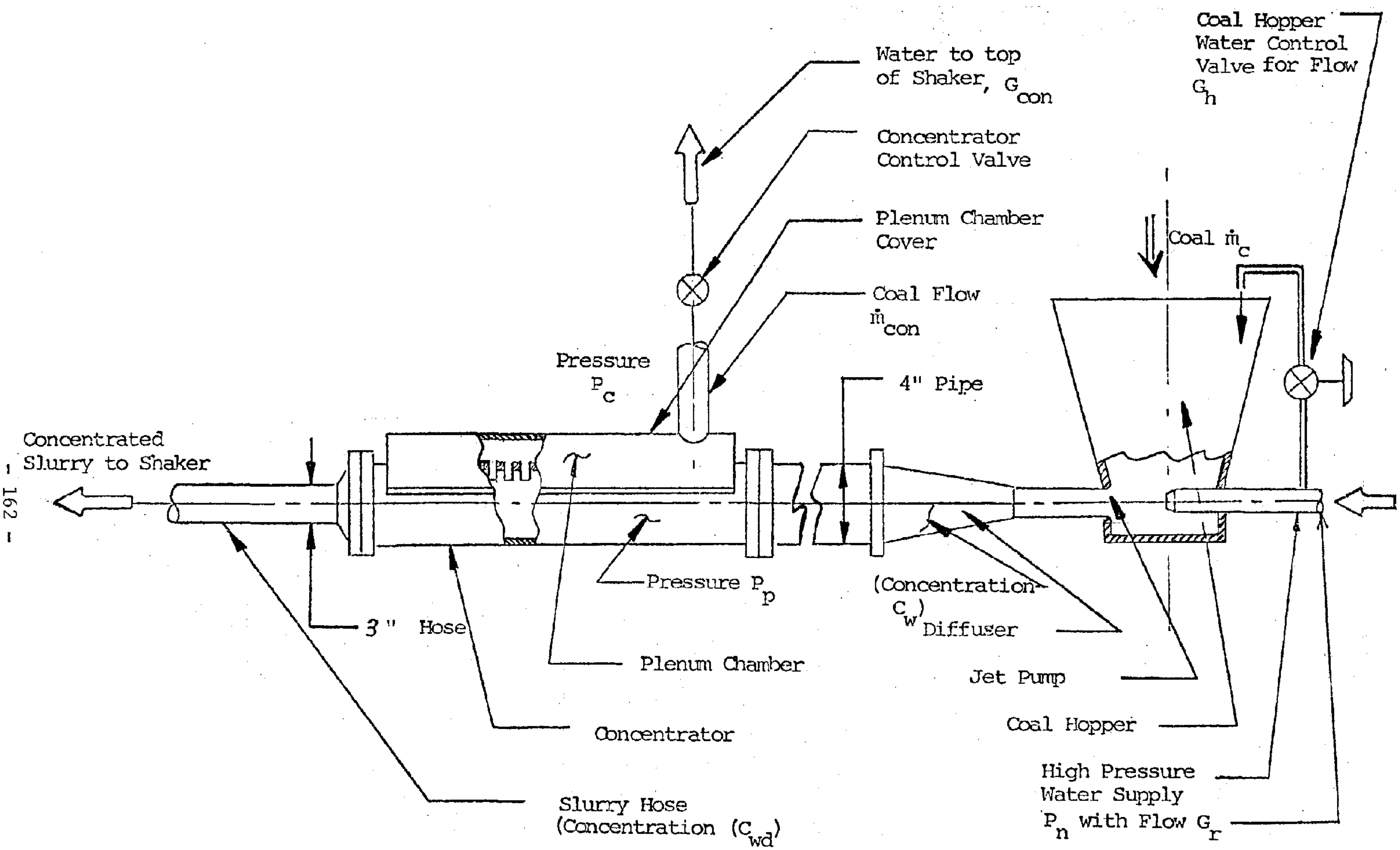


Figure 4-52: Schematic Diagram of Candidate Concentrator Installed in Test Loop

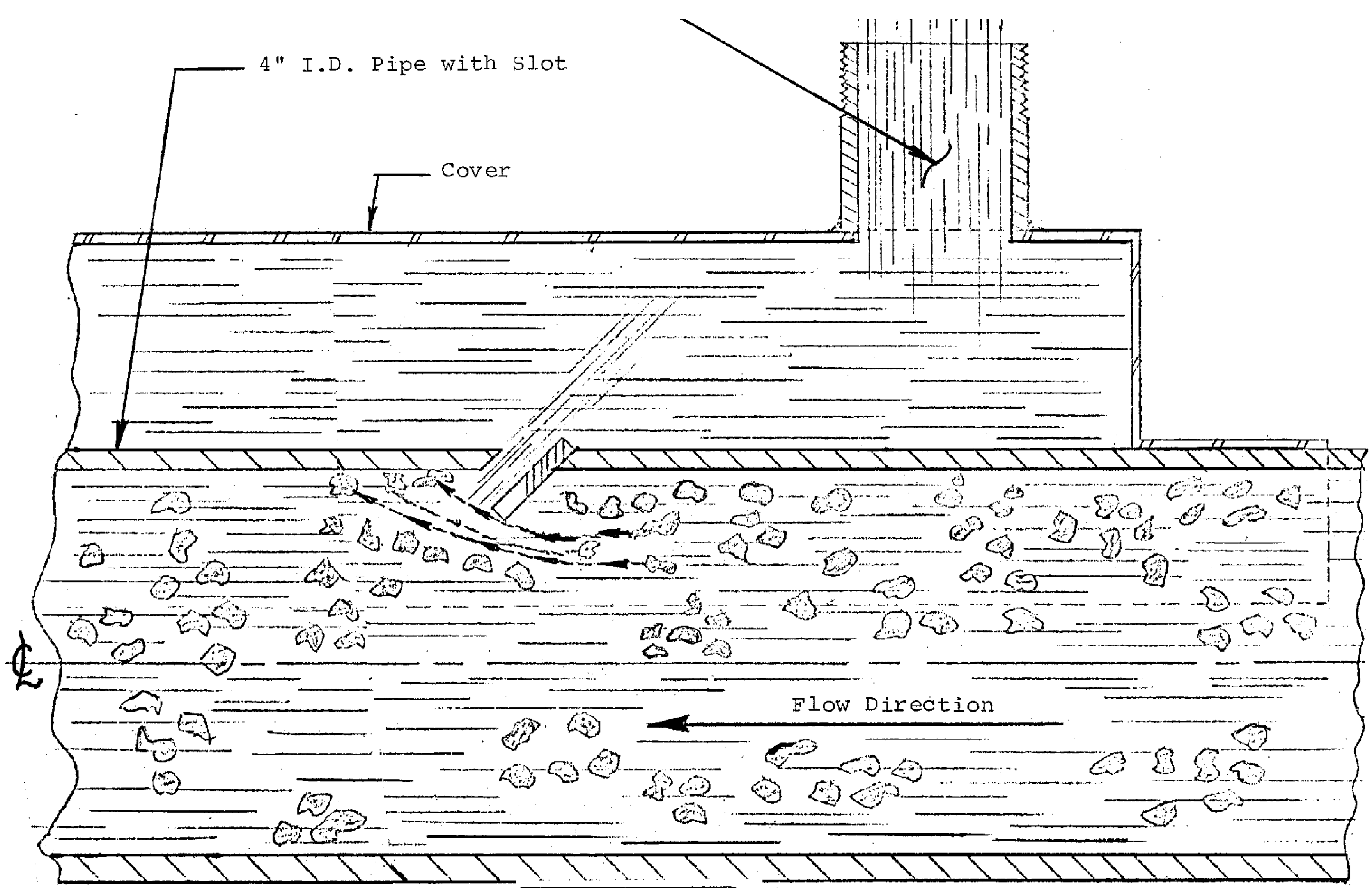


Figure 4-53: Functional Diagram of 18 Slot Concentrator (Reference 11)

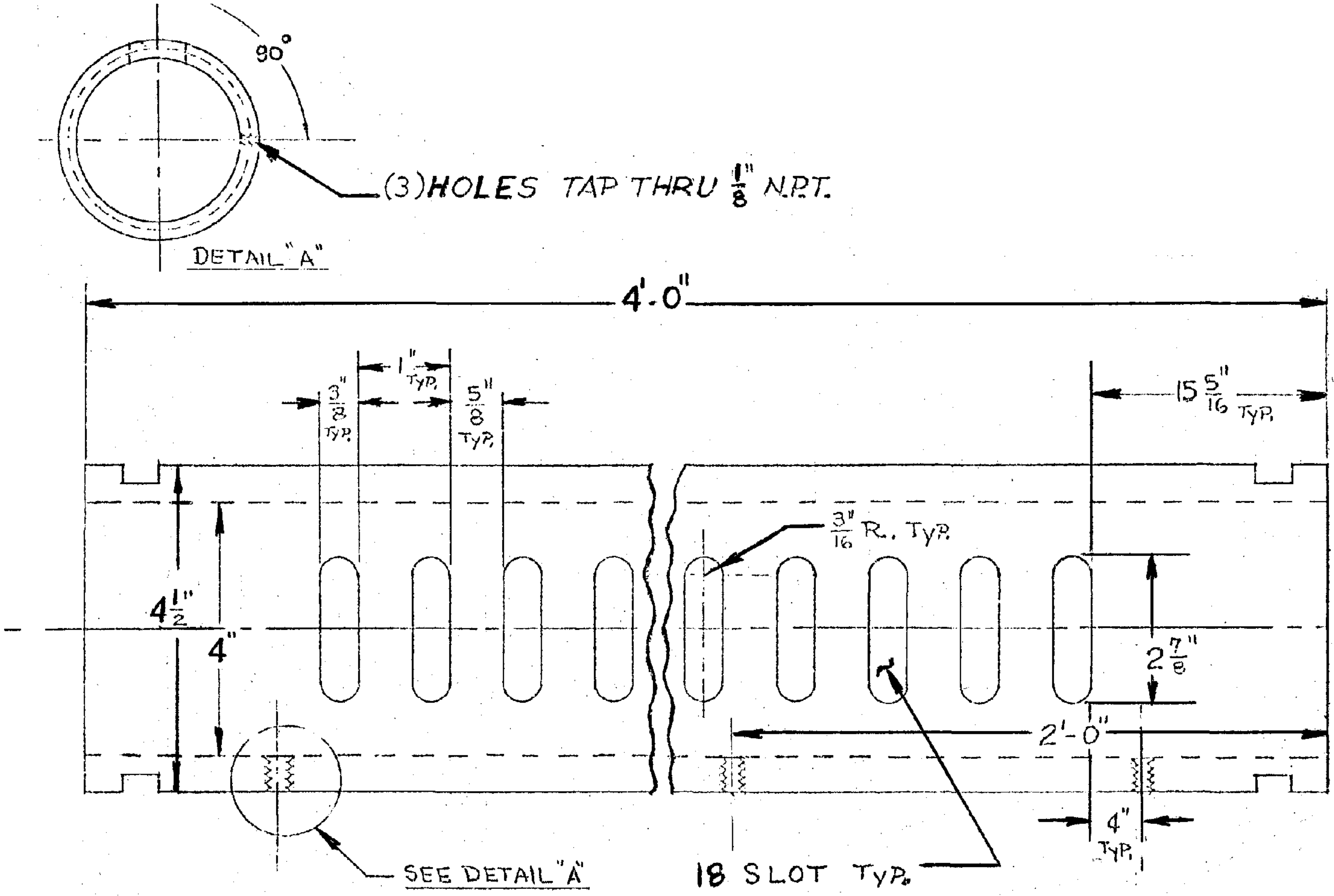


Figure 4-54: Dimensional Details of 18 Slot Concentrator

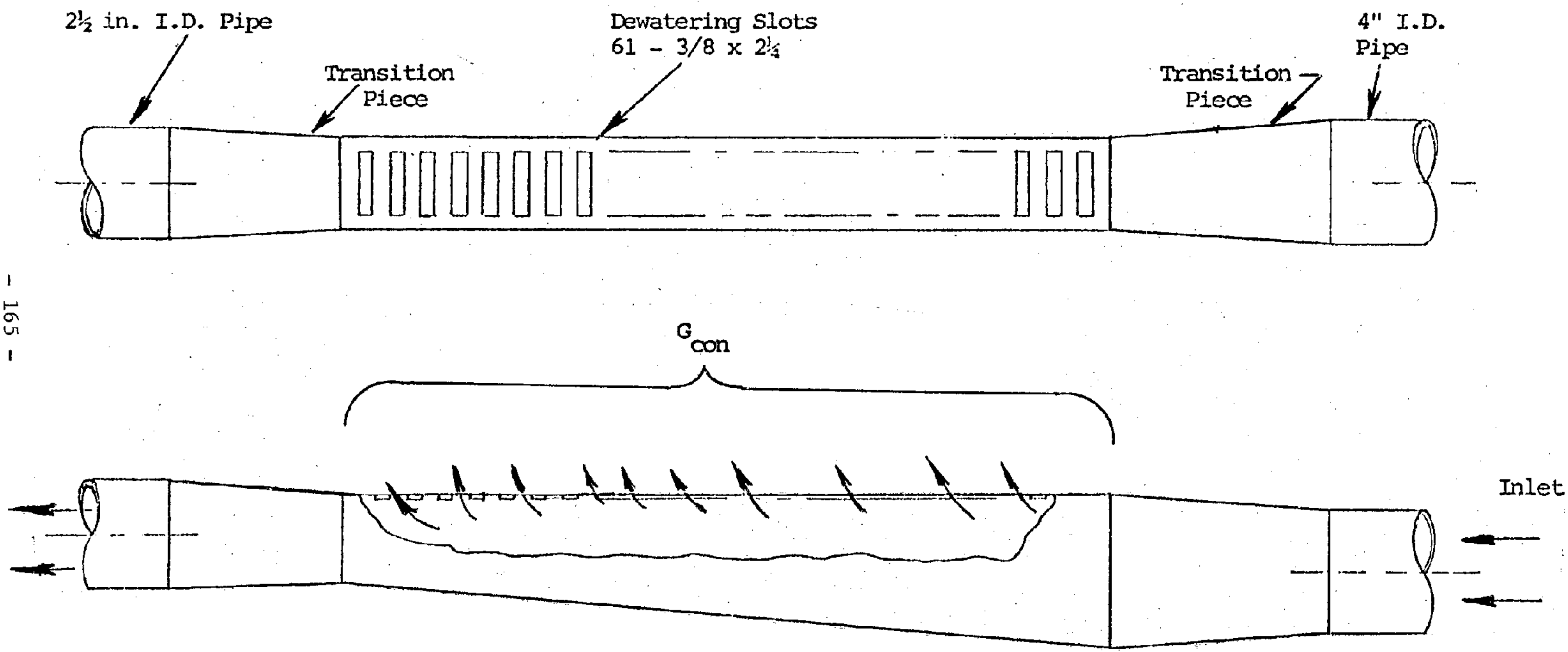


Figure 4-55: Reducing Area, Constant Velocity Concentrator

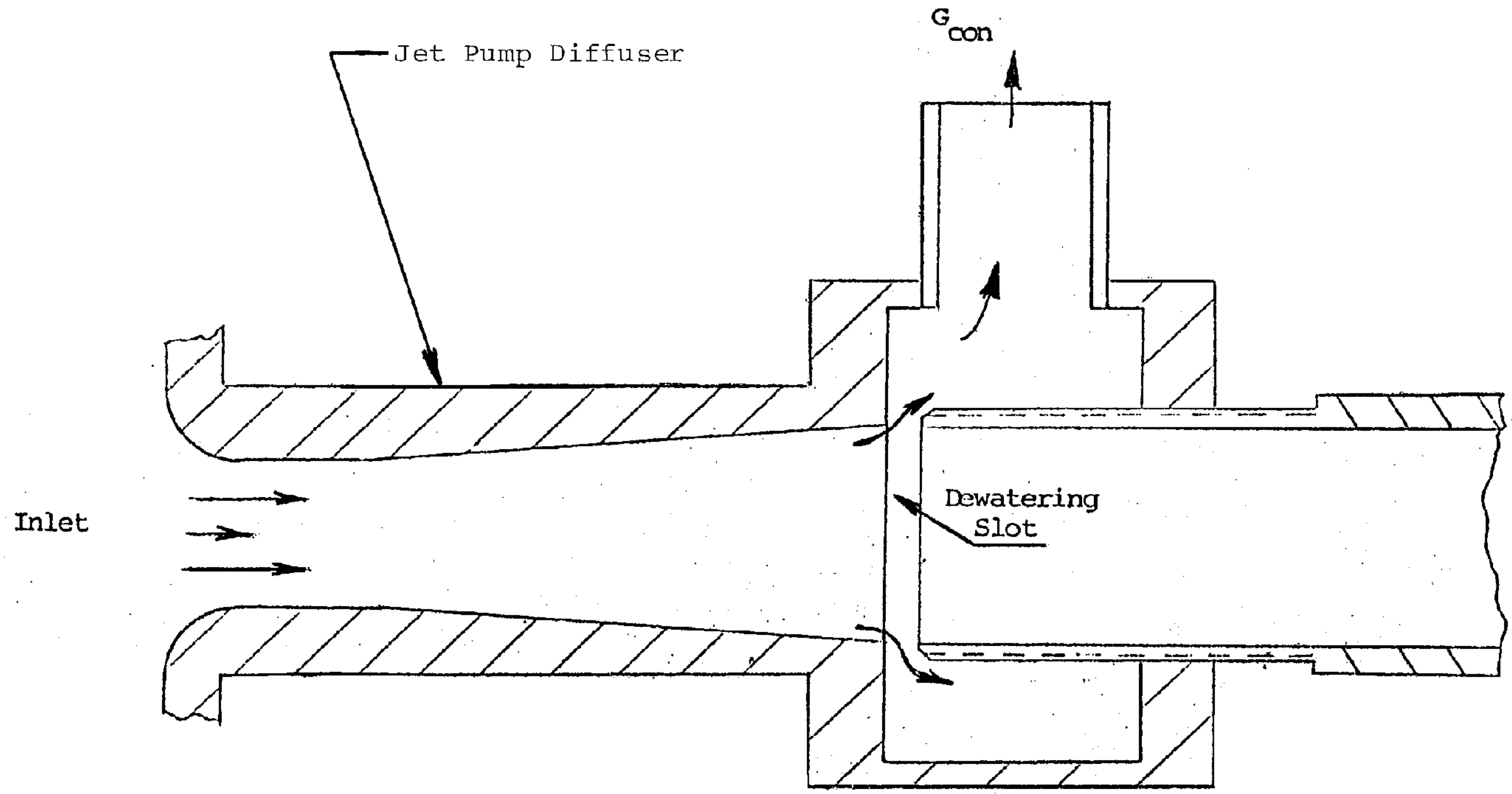


Figure 4-56: Diffuser Concentrator

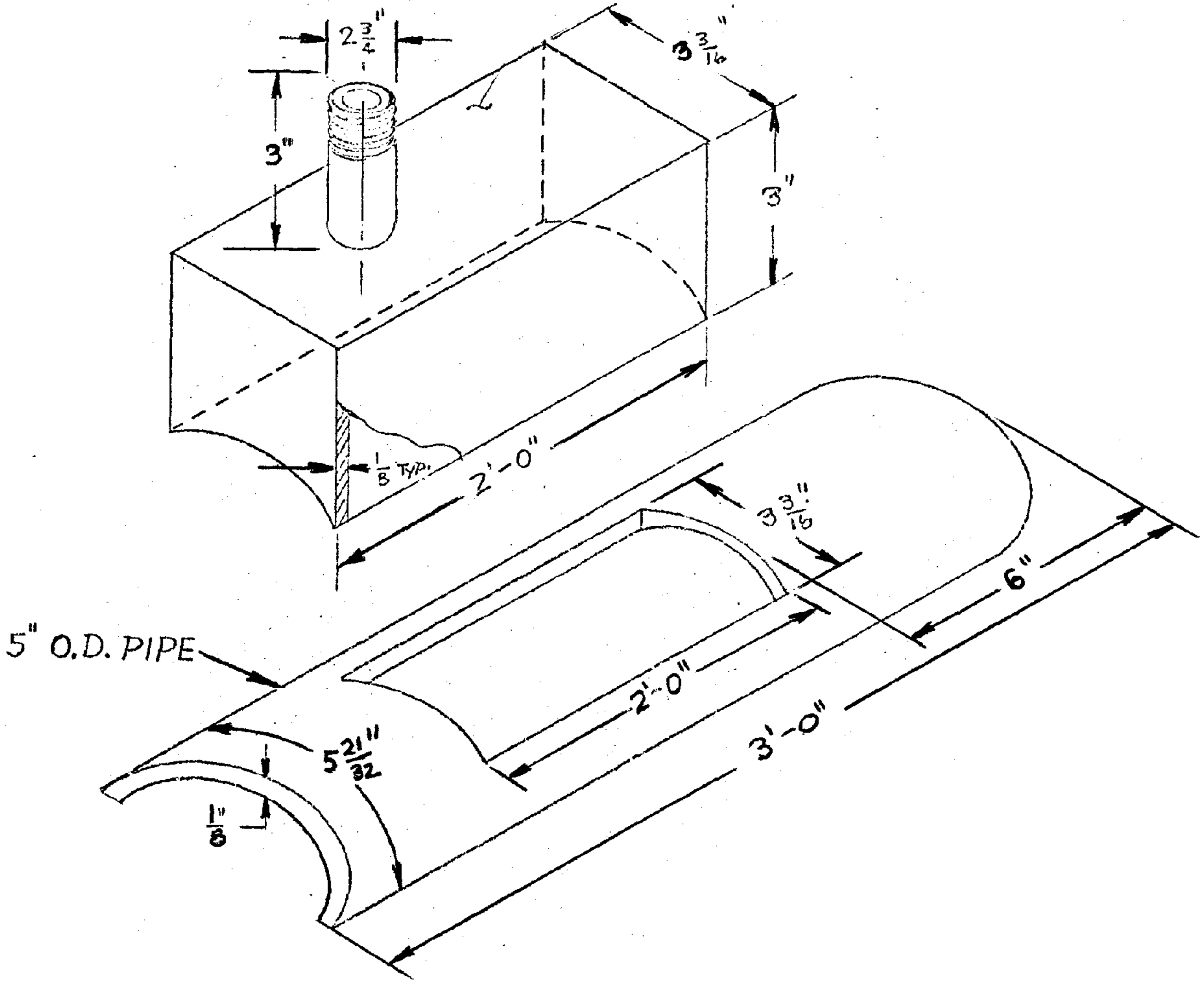


Figure 4-57: Concentrator Manifold Cover

slurry with a 50% mass concentration of coal given a 32% inlet concentration.

It would be desirable to operate with a high percentage of solids concentration, assuming plugging would not occur, because of reduced equipment cost. The cost reductions would result from reduced pump and slurry line sizes, lower wear rate and reduced pumping and water treatment requirements.

The jet pump injector is physically limited to a maximum delivered slurry weight concentration of about 32%. This condition occurs because a jet pump operates by surrounding a high velocity straight water jet with a mound of solid particles whose voids are water filled. This secondary slurry mixture in the hopper is entrained by the high velocity jet of water which further dilutes that mixture. As was determined during the Phase IIA experimental effort, a jet pump injector is capable of injecting a maximum of 32% of coal (by weight) into a slurry line pressurized to 25 psig. Such a slurry density is desirable for haulage pipe lines that must change grade, be of some length, or have any other propensity for plugging. Coal preparation plants traditionally operate their slurry lines at a concentration of approximately 30%.

The potential exists to utilize an in-line concentrator to adjust the slurry concentration, and thus the slurry velocity, to an optimum design operating point employing existing hose sizes. Therefore, an appropriate use of the concentrator device with a jet pump injector would be to use the concentrator as a slurry concentration adjuster keeping the concentration in the range of 30-40% of coal (by weight) regardless of hose size or tonnage rates.

One operational advantage of the concentrator would be to supply water which is added to the coal as it is fed to the jet pump hopper. This water was initially taken from the high pressure supply line, entrained with the coal by the water jet, and pumped out of the mine. This added secondary water is necessary for efficient operation of the jet pump, but it is wasteful to use water from the high pressure, 125 psi, water line.

The slurry concentrator acts as a recirculator for this secondary water by recycling water back to the jet pump hopper to fulfill the secondary water function as an air seal. By recirculating this water within the injector chassis, both the supply and slurry hoses may be reduced one hose size thus reducing bulk and hose cost. Water removed by the concentrator in excess of that needed for the secondary water function can be repressurized and used as a portion of the primary water to the nozzle.

4.3.3.2.2 Description of Test Apparatus

All testing of the slurry concentrator was conducted in the 1 ton per minute continuous slurry test loop, as described under Phase IIA activities in this report. To summarize the design of the test loop it consisted of:

- -A Prater vibrating screen separator.
- -A jet pump injector.
- -A 24 foot long, 18 inch wide feed conveyor and metering conveyor.
- -A 50 horsepower 450 gpm Ingersoll-Rand HC centrifugal pump.
- -A 200 gpm Krebs cyclone.
- -A large 600 gallon screen support and holding tank.

The test loop consisted of two continuous material paths which originated in the hopper of the feed conveyor that utilized a metering gate to lay down a uniform coal bed on the conveyor belt and a variable speed drive on the conveyor to set the delivery rate to the jet pump hopper. The coal was entrained in the jet pump and passed through a small diameter, high velocity slurry rubber hose to develop a high frictional head in a short distance. The slurry then passed through the top of the vibrating screen separator where water was removed and the coal shaken off the screens back to the hopper of the conveyor.

The water was fed from a large holding tank to the inlet of the 50 horsepower centrifugal pump. The flow was split such that part of the water flowed to the nozzle at 125 psig and the remainder of the water was used as secondary water flow to the jet pump as control water. This formed the slurry water with the coal. Coal and water were mixed inside the jet pump, and passed through the rubber slurry hose to the top of the vibrating screens to the holding tank beneath the separator screen, thus starting the cycle over again.

The concentrator test sections were constructed from 4 inch pipe sections, 4 feet long which were installed immediately downstream of the jet pump diffuser outlet.

The 4 inch pipe was used as a basis for all testing as this gave a slurry velocity of 11 feet per second which was identical to the desired design rate of the full-scale Hydraulic Haulage System.

For this entire test program, the jet pump configuration remained constant and had the following dimensional characteristics:

Nozzle Diameter	=	.978 Inches (Solid Core)
Diffuser Cone Angle	=	10 Degrees
Diffuser Outlet Diameter	=	4 Inches
Mixing Chamber Diameter	=	2 Inches
Mixing Chamber Length	=	3 Inches
Jet Entrainment Length	=	6 Inches

Bell Mouth Mixing Chamber Inlet Configuration

Four slurry concentrator configurations were designed and fabricated as tabulated below:

-A <u>single slot with a deflector vane</u> placed upstream at 45 degrees from the main flow as shown in Figure 4-53.

-An <u>eighteen slot concentrator</u> with <u>constant cross-sectional</u> area for the main flow as shown in Figure 4-54.

-A <u>sixty slot concentrator</u> with a <u>variable cross-sectional</u> area shown in Figure 4-55.

-A <u>combination diffuser-concentrator</u> with one slot for 360 degrees which becomes an integral part of the jet pump as shown in Figure 4-56.

The hollow housing shown in Figure 4-57 was placed over the slots to serve as a manifold to collect the water that passed through the slots. All water and entrained fines were passed through a Krebs Cyclone separator. The solids underflow from the separator was fed to the jet pump inlet to be reinjected. The water overflow from the separator was fed back to the main water tank of the test loop.

During later testing increased amounts of water were passed out the underflow of the separator to satisfy all secondary water requirements of the jet pump. This procedure reduced the amount of overflow water diverted back to the main tank. For final main line slurry concentrations of 40 percent or less, all water and fines removed by the concentrator were fed back to the jet pump. This operating sequence would be the identical procedure to be followed with a full scale underground system.

For main line slurry concentrations of 40-55 percent, more water is removed by the concentrator than is required for the jet pump secondary flow. This excess water would have to pass through a small onboard centrifugal pump and then to the jet pump nozzle in an underground system.

The single slot deflector vane version of the concentrator was first tested to gain an analytical understanding of the separation process and determine the required number of dewatering slots. The results of the single slot test were used to design the multiple, simple-slot configuration.

Provisions were made at several axial stations in both the slurry pipe and the concentrator housing to permit a determination of the pressure drop across the slots throughout the concentrator.

The measurements made and the instruments used were:

Hopper Flow: Barco Venturi (2 inch) model BR-12636-32-31 and a manometer utilizing mercury under water.

Nozzle Flow: Nozzle flow G, as a function of nozzle pressure was determined from the experimentally derived expression G = 22,36P (where G is in gallons/minute and nozzle pressure P is in psig).

Concentrator Flow: Barco Venturi (2 1/2 inch model BR-12750-40-31 and a manometer utilizing mercury under water.

Concentrator Pressure Drops: Dwyer Inclined Manometer Model 246 using mercury under water (with appropriate factors to account for inches of water manometer design).

Pressure, Upstream and Downstream of Slots: Standard pressure gages.

Coal Flow: Measurement of the conveyor belt loading to the injector hopper. The conveyor belt was calibrated by operating the belt and dumping the coal into a barrel for a known number of belt feet running at a known belt speed. The weight of the coal in the barrel provided the belt loading per foot. This information allowed the calculation of the coal delivery to the injector hopper.

Effluent Coal Content: Samples of concentrator effluent were weight-analyzed to determine coal content by weight. The samples were weighed, dried in an oven, reweighed, the coal removed, and reweighed. This allowed calculation of the mass fraction of coal.

After Run 122, the fines removal weight was determined by weighing a timed amount of underflow from the Krebs Cyclone separator and calculating the specific gravity of the effluent and thereby the fines removal rate.

4.3.3.2.3 Theory of Operation

There are two distinct physical effects which can be utilized in the operation of a coarse coal slurry concentrator. These effects are gravitational separation and inertial separation as discussed next.

Gravitational Separation:

The first effect, or hypothesis, is that the water velocity through the dewatering slots can be maintained less than the settling velocity of a chosen particle size thereby preventing the particle from being removed with concentrator water flow. Solid particles which happen to be located within a dewatering slot channel will settle out of the channel at a faster rate than the net water rate through the channel.

The pressure drop across the dewatering slots is limited by the maximum allowable water velocity through the slots. The experimental program has indicated that design of the concentrator should be such that the maximum velocity across the concentrator dewatering slot should not exceed 2 ft/sec. 1.0 foot per second is the settling velocity of a coal particle 3/16 inches in diameter. Therefore, it would be expected that a 3/16 particle of coal would have a 50% chance of either passing through the dewatering slots or not passing through the dewatering slots if it arrived at mid-channel. If the velocity exceeds the design value, particles large enough to partially block the slot can accumulate, thereby reducing the slot area. This situation increases the velocity further and leads to complete plugging.

In the simple slot design there is an additional mechanism for removing large coal particles which would tend to lie across the entrance of the dewatering slot. That mechanism is a net overturning force caused by the mainstream drag which is at a velocity head which is two to three times the drag due to the net flow through the dewatering slots. Thus, there would be a net force across the coal particle, due to flow through the slots, which would tend to hold the particle against the entrance to a dewatering slot. The maximum force would simply be the maximum pressure drop across the slot which is on the order of 0.3 psi. However, the mainstream velocity head is on the order of 0.8 psi, a factor of 2.67 greater. Assuming equal drag coefficients in both directions it can be seen that there is a component which would tend to overturn the particle thus clearing the slot.

The overturning moment also depends on the particle diameter. Smaller particles will rest deeper in the slot thereby reducing the overturning arm moment which is the distance from the particle's center of gravity to the edge of the slot. Of course, the nature of the concentrator is such that the settling velocity of the particle would aid in the removal of the particle from the slot.

Inertial Separation:

The second hypothesis is based on an inertial approach which states that a moving mixture of solids and liquids will tend to separate when the direction of flow is changed. The solid particles will tend to have different steamlines than the water. The objective of this approach to dewatering is to move the slurry at such a rate as to permit the water to make the turn and exit through the dewatering slots while the particulate continues along in the slurry matrix without passing through the dewatering slots. This approach was tried with a single slot shielded by a deflecting vane. In the concentrator concept based on inertial separation, it was envisioned that high velocity water movement would be maintained through a single slot promoting inertial separation of the solids from the liquids.

An apparent advantage of the single inertial slot was felt to be its compactness resulting from the high velocity through the slot. However, if two or more such slots are required it is felt that they will have to be spaced a considerable distance apart in order to allow some particle settling.

4.3.3.2.4 Test Procedure

Tests were conducted by starting up the test loop with the maximum coal rate fed to the jet pump thus developing a minimum slurry to the concentrator of 32 percent of coal by weight. Increasing amounts of water were removed through the dewatering slots until operational difficulties occurred such as plugging of the slots or slurry line. At high slurry concentrations (over 45 percent) it was necessary to switch from the 3 inch slurry hose to a 2 1/2 inch hose to avoid plugging.

Originally, the concentrator overflow was regulated by a gate valve which had to be nearly closed to develop the required 25 psig pressure drop. This practice was discontinued because of fines buildup in the small valve opening required with the single slot inertial separator.

The remaining three concentrator configurations were controlled by a large gate valve in series with the cyclone separator.

After allowing several minutes of operation, so as to reach system equilibrium, the data were recorded and samples were taken of the cyclone underflow. Representative samples were dried and screened to determine size distribution.

4.3.3.2.5 Results

The test data for the exploratory program to demonstrate the attainment of a 45%-50% coal mass concentration in a coal slurry is tabulated in Appendix B. Tests were performed on three of the four concentrator configurations noted earlier as follows:

-Single Slot (Figure 4-53): Test Runs 1-44

-18-Slot, Constant Cross-
Sectional Area (Figure 4-54): Test Runs 45-143

-60-Slot, Variable Cross-
Sectional Area (Figure 4-55): Test Runs 143-171

Because the target mass concentration level was conclusively demonstrated on the third configuration noted above, no testing of the diffuser concentrator (Figure 4-56) was necessary.

Referring to Appendix B, the data of Runs 1 through 44 represents the performance results of the inertial, single slot concentrator. The objective of this configuration was to allow a high head loss, high volume flow which would turn so abruptly as to preclude any large solid particles from also turning and passing through the slot. The turning vane was added to generate local high acceleration especially at the vane tip to increase the solid-liquid separating mechanism.

Testing of this single slot concentrator, however, showed plugging of the slot at any pressure drops greater than 0.5 psi. This situation corresponded to a flow through the slot of approximately 6.0 gpm and only about a 1 percent increase in concentration. The hypothesis advanced for the low-flow plugging was that flow separation occurred at the vane tip and the resulting streamlines became aligned with the vane tip, bypassed the slot and allowed no separation. The natural next step was to try a single slot concentrator without a turning vane. However, the 18 slot concentrator was tested next to gain insight into the mechanism of gravitational separation after operational problems with the inertial separator became apparent.

The data for Runs 45 -143 documents the performance results of the 18 slot, "short", constant cross-sectional area concentrator. Of these, Runs 45-74 are developmental runs to debug the test loop equipment. Runs 75-143 represent repeatable data typical of this concentrator. The results obtained from testing show maximum stable operation in the range of 46 to 48 percent weight concentration.

The 18 slot concentrator had a constant area cross-section which caused the main flow velocity to slow as water was removed through the slots. This decelerationof the main flow caused a pressure rise equal to the change in dynamic head. A pressure distribution was thereby maintained both in the main concentrator section, and in the water gathering manifold above the slots. Runs 109, 110, and 114 showed evidence of backflow through the manifold into the main slurry. This effect suggested that a much smaller number of identical slots would be just as effective as the 18 slots when operating at approximately 40 percent concentration. Of course, at higher flow rates through the slots and correspondingly higher concentrations there always would be a net positive flow from main stream through the slots. Even in the high flow cases, though, the velocity distribution along the slots would not be uniform.

The data for Runs 144-171 documents the performance results for the 60 slot, variable cross-sectional area concentrator.

Two separation mechanisms appeared to occur in the 18 and 60 slot concentrators. The basis of the design was gravitational separation and its affect was apparent in the concentrator except at the leading, or upstream, slots. As evidenced in the pressure drops across the leading slots the slot velocities are 2 to 3 times greater than the average slot velocity. A certain amount of separation due to the momentum of the solid particles occurred. Evidence of this was seen in the reduced fines carry over for velocities of the observed magnitude.

No experimental means were devised to determine fines carry over as a function of slot position, however, the amount of fines appeared to correlate with the average slot velocity. Exceptions to the above statement occurred when water was removed from the manifold at upstream positions. In most cases, the fines carry over correlated with the maximum local velocities.

The primary results of the concentrator study for the 18 slot and 60 slot concentrators are given in Figures 4-58 through 4-62.

Figure 4-58 depicts the performance of the 18 slot concentrator as measured by the mass concentration of coal in the slurry downstream of the concentrator as a function of concentrator water flow. Figure 4-59 shows the mass concentration of coal in the concentrator overflow as a function of concentrator water flow for the 18 slot concentrator. The final plot for the 18 slot concentrator, given in Figure 4-60, displays the mass coal concentration in the slurry downstream of the concentrator as a function of average slot velocity for

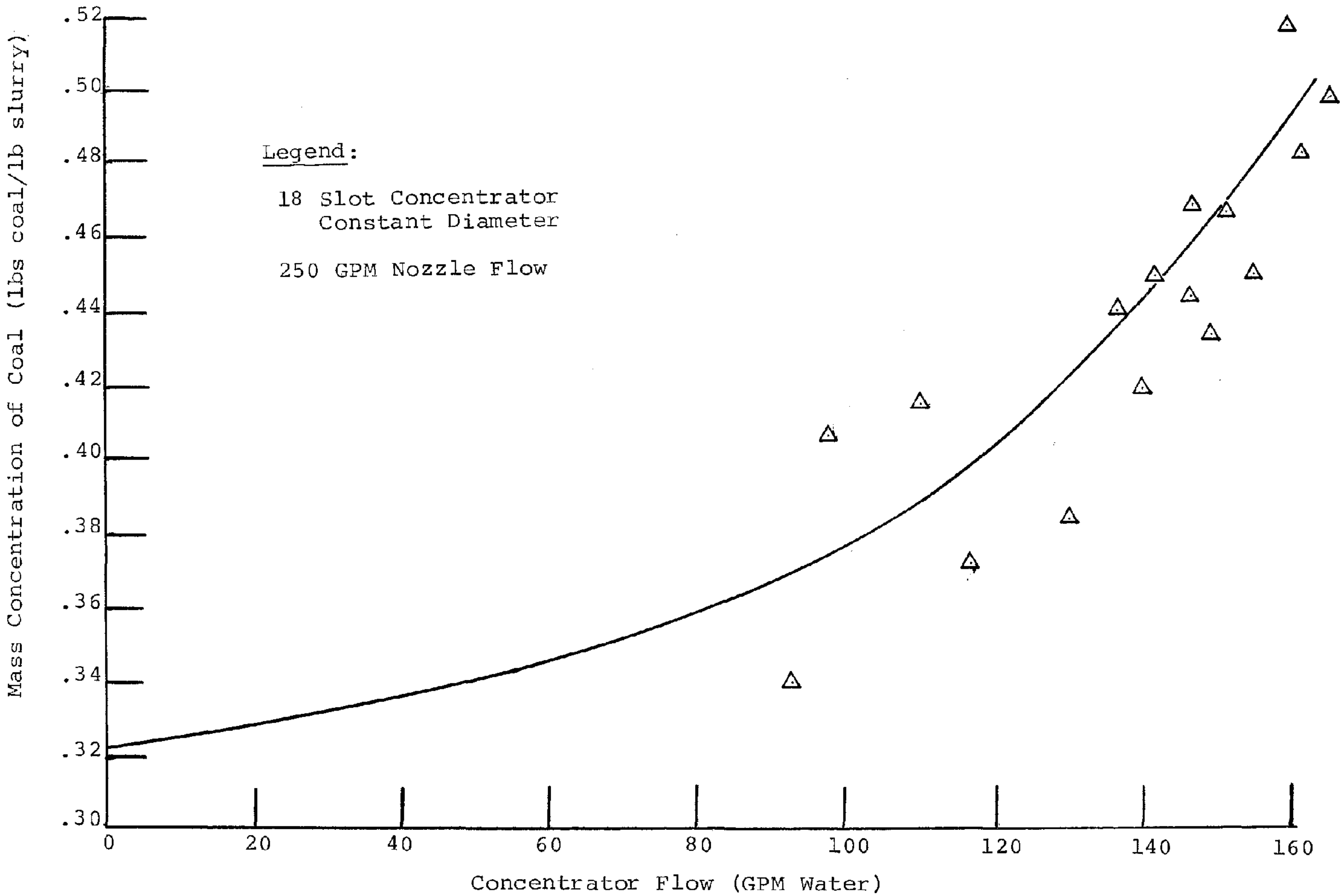


Figure 4-58: Mass Concentration of Coal Downstream of Concentrator vs. Concentrator Flow for the 18 Slot Concentrator

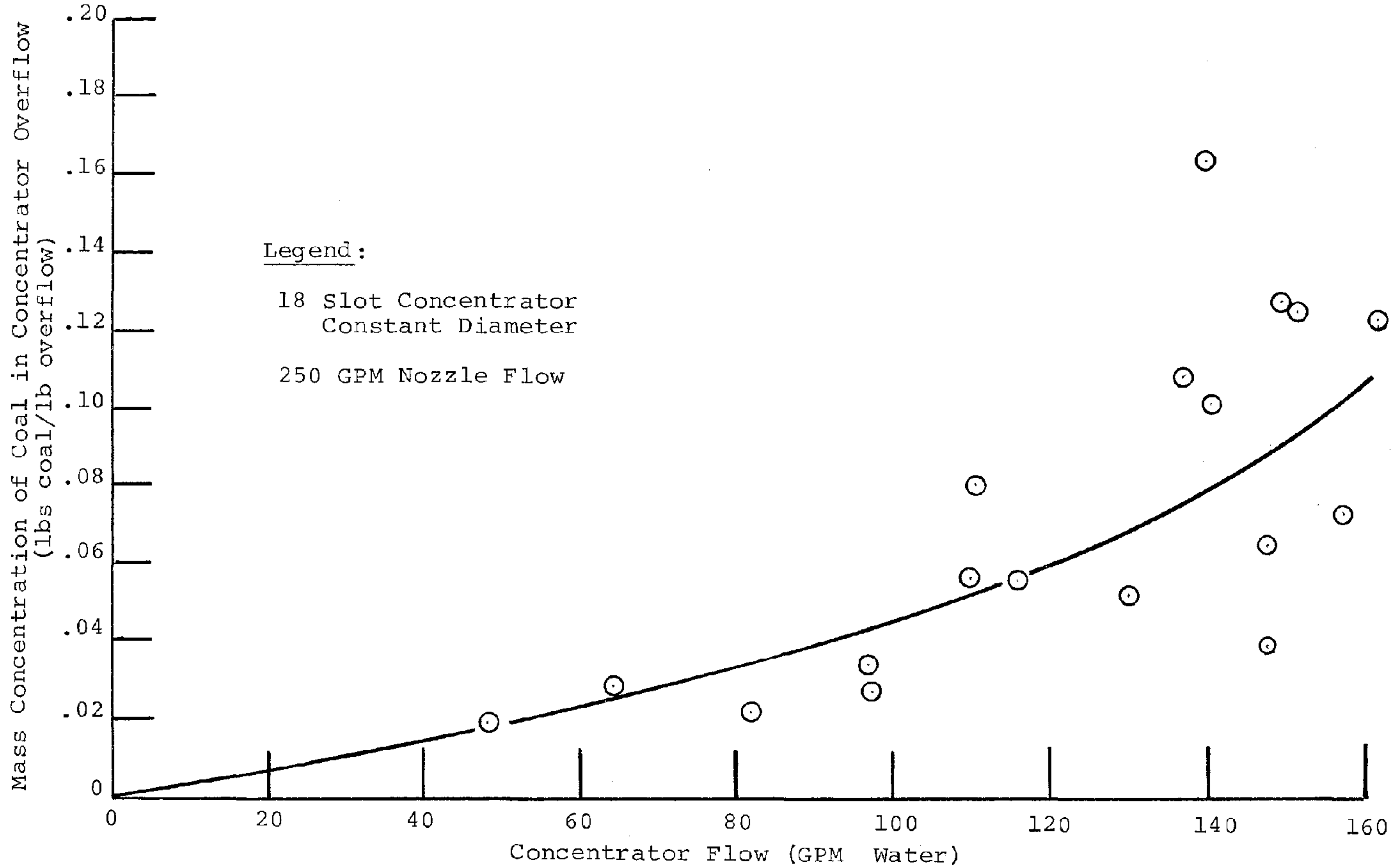


Figure 4-59: Mass Concentration of Coal in Concentrator Overflow vs. Concentrator Flow for the 18 Slot Concentrator

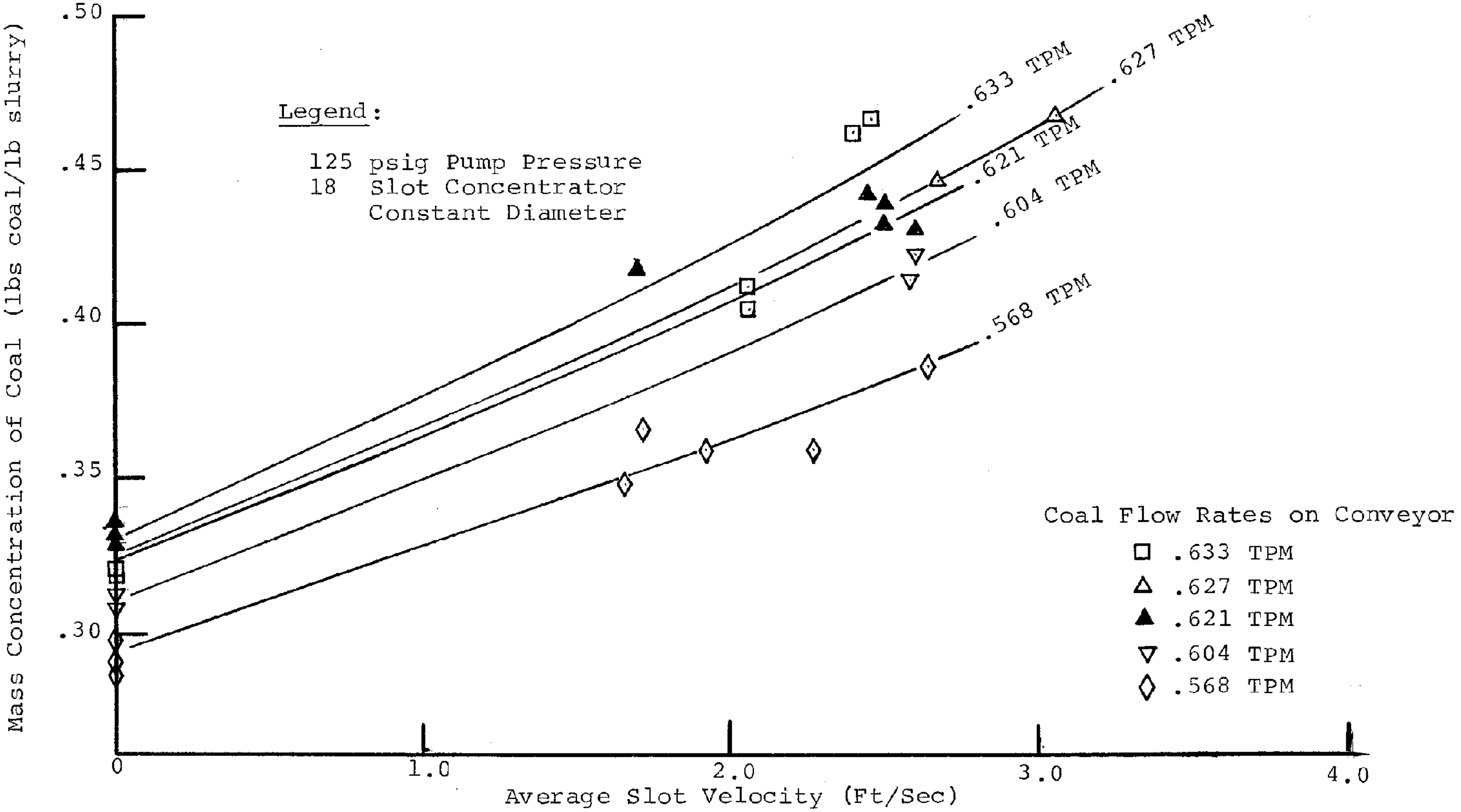


Figure 4-60: Mass Concentration of Coal Downstream of Concentrator vs. Average Slot Velocity for the 18 Slot Concentrator

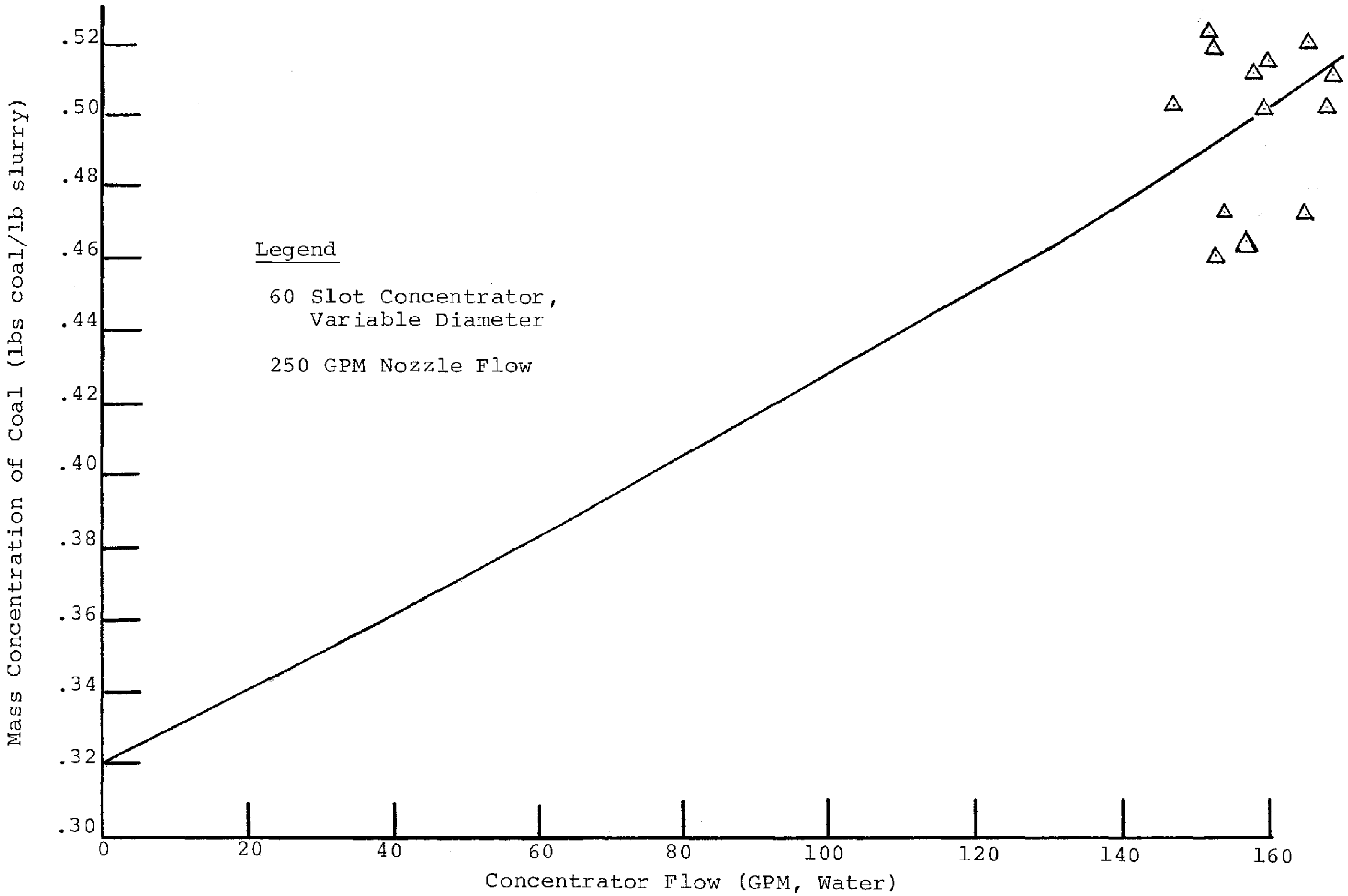


Figure 4-61: Mass Concentration of Coal Down Stream of Concentrator vs. Concentrator Flow for the 60 Slot Concentrator

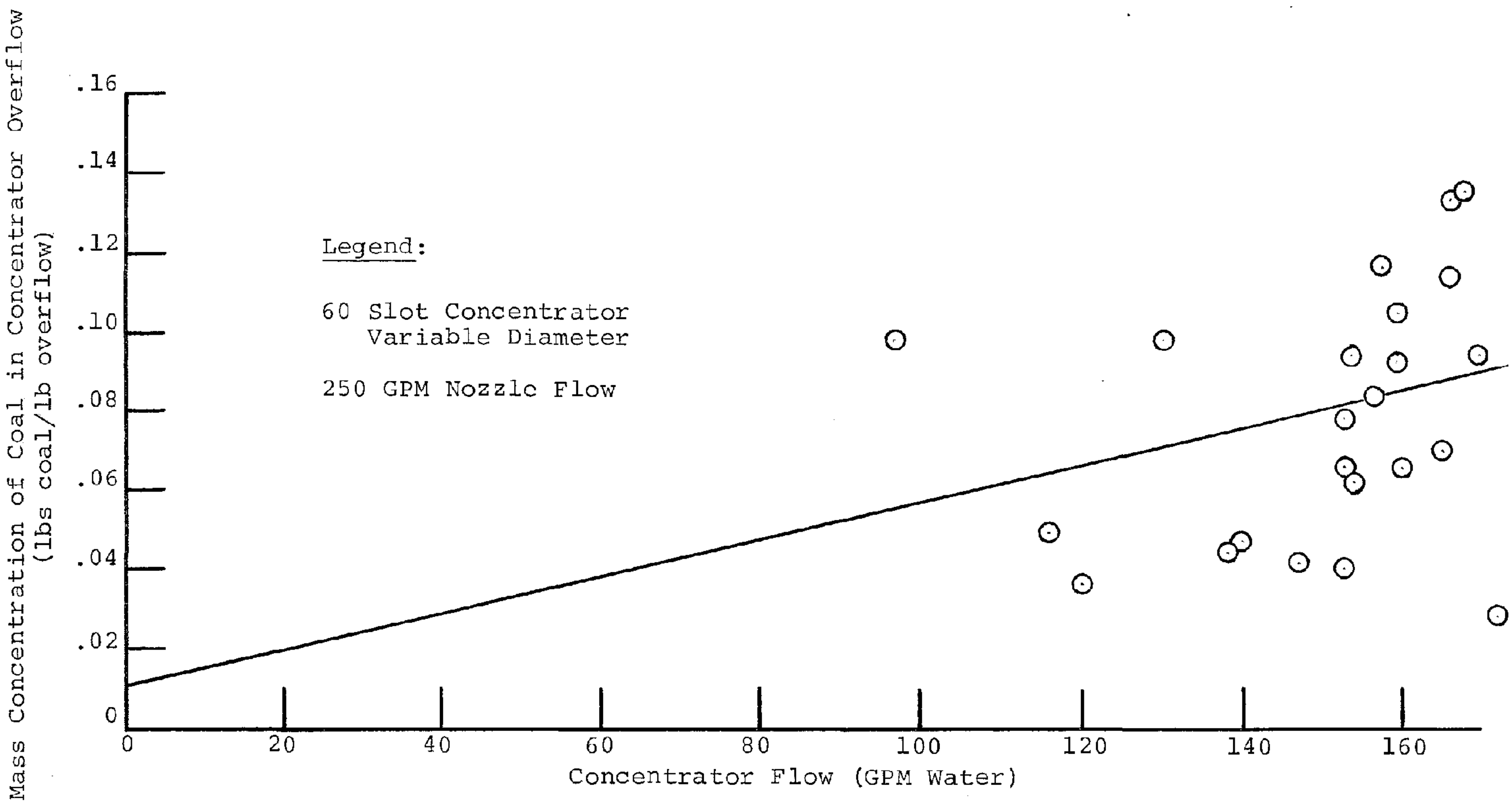


Figure 4-62: Mass Concentration of Coal in Concentrator Overflow vs. Concentrator Flow for the 60 Slot Concentrator

various coal flowrates. This particular plot is most useful from a design standpoint.

Similar performance plots are shown for the 60 slot concentrator in Figures 4-61 and 4-62.

These graphs show that 46 to 48% slurry mass concentration can be achieved by withstanding from 90 up to 175 gpm from the slurry which originally contained approximately 320 gpm of water. This represents a removal of 28% to 54% of the slurry water. The spread of the required water removal is due at least in part to the rate of coal injected into the original slurry.

4.3.3.2.6 Conclusions

On the basis of the subscale concentrator development work performed, it was concluded that a successful, full-scale concentrator, delivering up to a 50% mass coal concentration to the haulage pipeline, could be designed with confidence. The specific conclusions to be drawn from the subscale concentrator work that support this contention are:

- Coal mass concentrations in a coal-water slurry approaching 54% were demonstrated with stable, reliable and repeatable operation.

- Sufficient understanding was gained during the program to predict the performance of the 60 slot, variable area concentrator.

- Control of the concentrator does not require any moving or special controls. All that is required is a restrictive, nonplugging device in the concentrator overflow such as a cyclone device.

- Fines recirculation in the circuit consisting of the concentrator and the jet pump does not tend to increase with time.

- Up to approximately a 40% coal mass concentration in the slurry, all water removed from the concentrator can be fed back directly to the jet pump hopper so as to increase the efficiency of the haulage system. At concentrations above 40% a small onboard centrifugal clear water pump could be used to boost pressure of a portion of the concentrator overflow and feed it directly to the nozzle. This procedure would not change system efficiency. It merely would move a small portion of the pumping equipment to the face.

4.4 Phase III - Mobile Jet Injector Vehicle Development

4.4.1 Design of Mobile Jet Injector Vehicle

4.4.1.1 Overall Description of Vehicle

The final arrangement of the complete vehicle is presented schematically in Figure 4-63 (inside elevation) so as to acquaint the reader with the orientation of the various, major on-board components and subsystems. Referring to this figure, the coal is processed at the various locations in the following sequential steps:

Step 1: Surge Bin - Receives ROM coal from a continuous miner or other source.

Step 2: Primary Breaker - Reduces nominal 14" x 0" ROM coal to 8" x 0". This unit is an S&S Corporation Spartan SE unit that uses a flywheel energy storage system. The breaker itself is an 18" single-roll, pick-type breaker.

Step 3: Flight Conveyor - Delivers coal from the surge bin (1) to the primary breaker (2) and then to the secondary breaker (4).

Step 4: Secondary Breaker - A high-speed, double-roll breaker that uses the same basic size-limiting geometry that was tested successfully with respect to top-sizing and throughput.

Step 5: Secondary Flight Conveyor - Delivers coal from the secondary breaker to the jet pump hopper.

Step 6: Jet Pump Hopper - Contains coal immersed in water, to supply slurry to the "entrainment zone" in which the high-velocity jet entrains slurry carrying it into the mixing tube.

Step 7: Jet Pump Nozzle - Supplies a high-velocity water jet through the entrainment zone into the mixing tube.

Step 8: Mixing Tube and Concentrator - Adjusts slurry concentration by partial water

1 - Surge Bin
2 - Primary Breaker
3 - Flight Conveyor
4 - Secondary Breaker
5 - Secondary Flight Conveyor

6 - Jet Pump Hopper
7 - Jet Pump Nozzle
8 - Concentrator
9 - Slurry Outlet

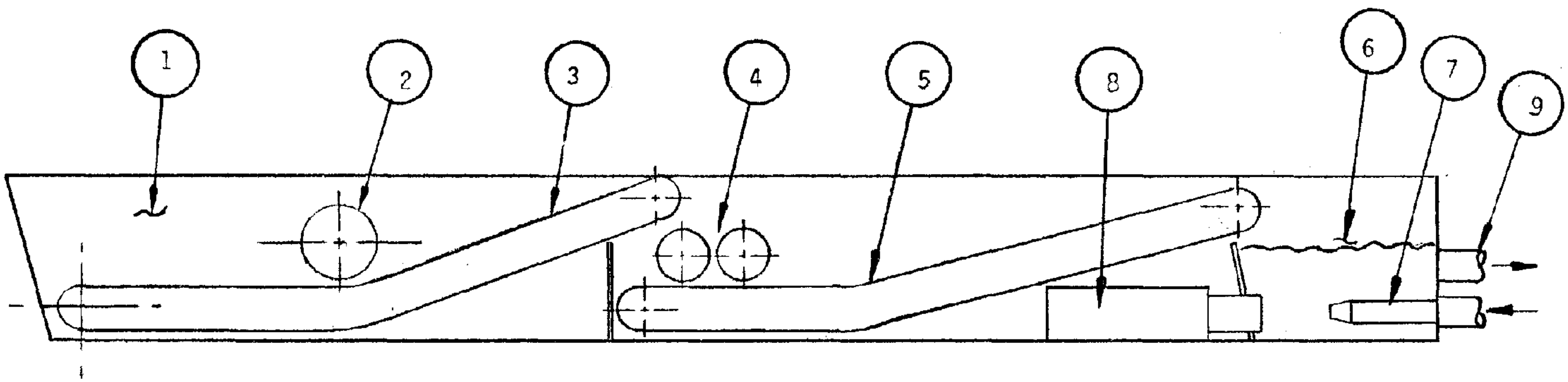


Figure 4-63: Side Elevation Schematic of Mobile Jet Injector Vehicle Showing Major Functional Components

removal.

Step 9: Slurry Outlet - Delivers concentrated slurry to the output slurry line.

Overall engineering layout views of the jet pump injector vehicle are presented in Figure 4-64. This vehicle is based on the S&S Corporation's Spartan SE II feeder-breaker vehicle. The front portion of the vehicle is shown in side elevation in Figure 4-65 (exterior view), Figure 4-66 (cross-sectional view) and Figure 4-67 (plan view).

Incoming coal produced by a continuous miner in front of the vehicle enters a receiving hopper, which has a capacity of 3.5 tons of coal. Chains spanning the hopper are installed as shown to level the coal bed as it passes through the hopper. A flight conveyor takes the coal into a primary breaker whose purpose is to break down 18" x 0" coal or rock to a 5" x 0" size. The primary breaker used here is the standard breaker from the Spartan SE II and is utilized intact. It is electrically driven through a right-angle gear box that also houses the S&S Corporation's energy storage system.

The aft portion of the vehicle (aft of the primary breaker) is illustrated inside elevation in Figure 4-68 (exterior view), Figure 4-69 (cross-sectional view) and Figure 4-70 (plan view).

After passing through the primary breaker, the coal is carried up a slope, over the hydraulic coal reservoir, and over a short scalper. This device is a slotted section of bedplate that allows fine coal to drop through. Scalped coal lands on a secondary converyor underneath.

Coal that does not fall through the scalper is dumped onto a double-roll, secondary breaker. The function of this breaker is to ensure that the injector system never encounters a piece of coal or refuse larger than 3 inches in any dimension. At the same time, by design it prevents further breakage of pieces that are already sufficiently small.

Following the topsizing operation performed by the secondary breaker, the coal is transported by the secondary conveyor to the jet pump injector hopper. Here the coal is slurried with water, directed by the hopper walls toward a high-velocity jet of water, and entrained by the jet.

The coarse coal slurry then enters a mixing tube followed by a centrifugal concentrator, developed by IRRI as

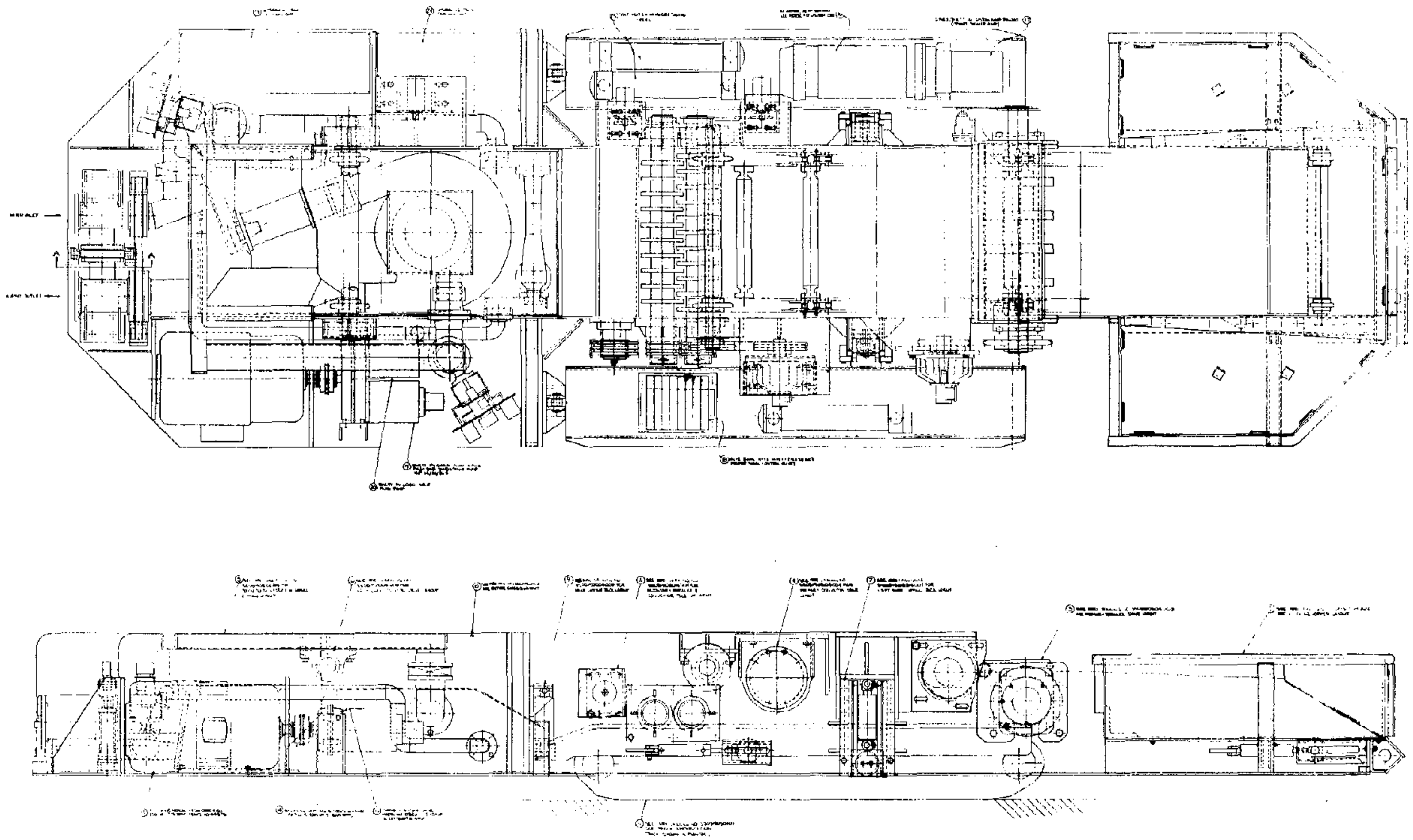


Figure 4-64: Ingersoll-Rand/S&S Jet Pump Injector Vehicle, Overall Views (Larger scale drawings are on file at the Bureau).

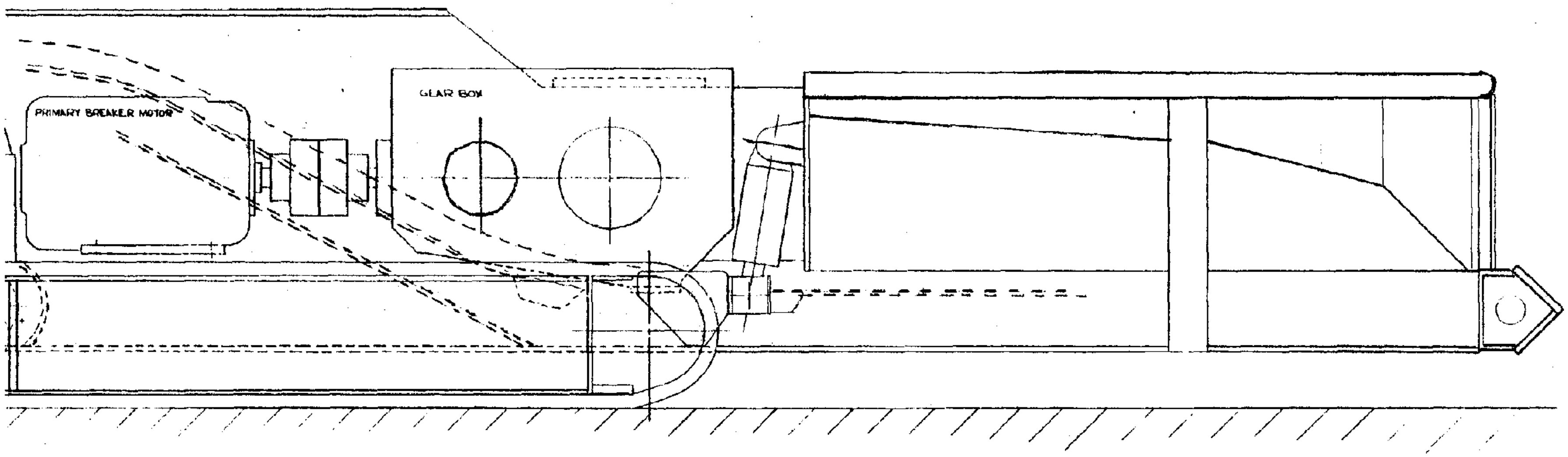


Figure 4-65: Ingersoll-Rand/S&S Jet Pump Injector Vehicle
Right Front Side View

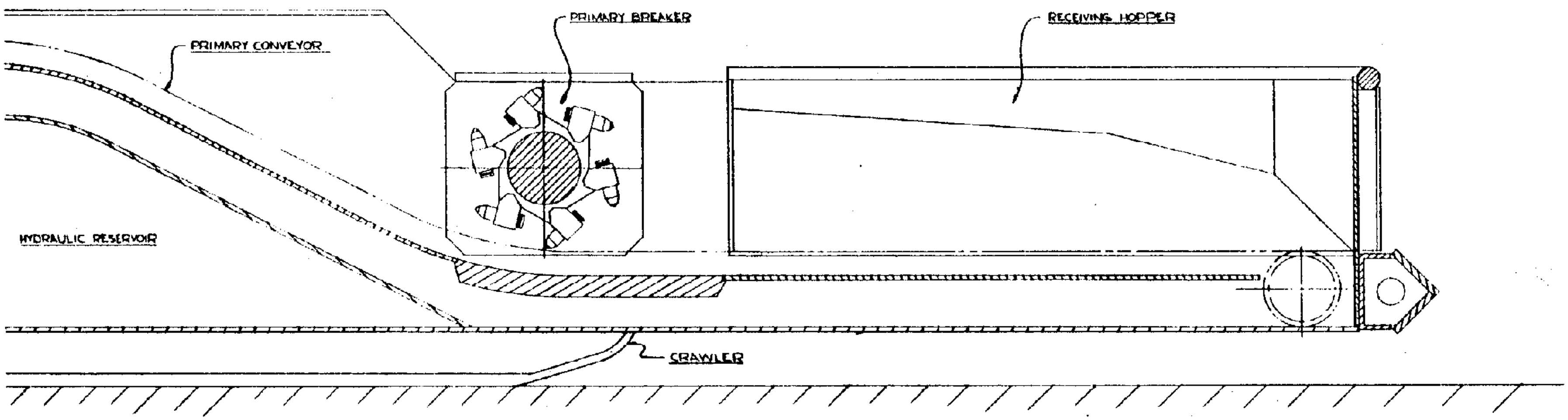


Figure 4-66: Ingersoll-Rand/S&S Jet Pump Injector Vehicle
Right Front Sectioned View

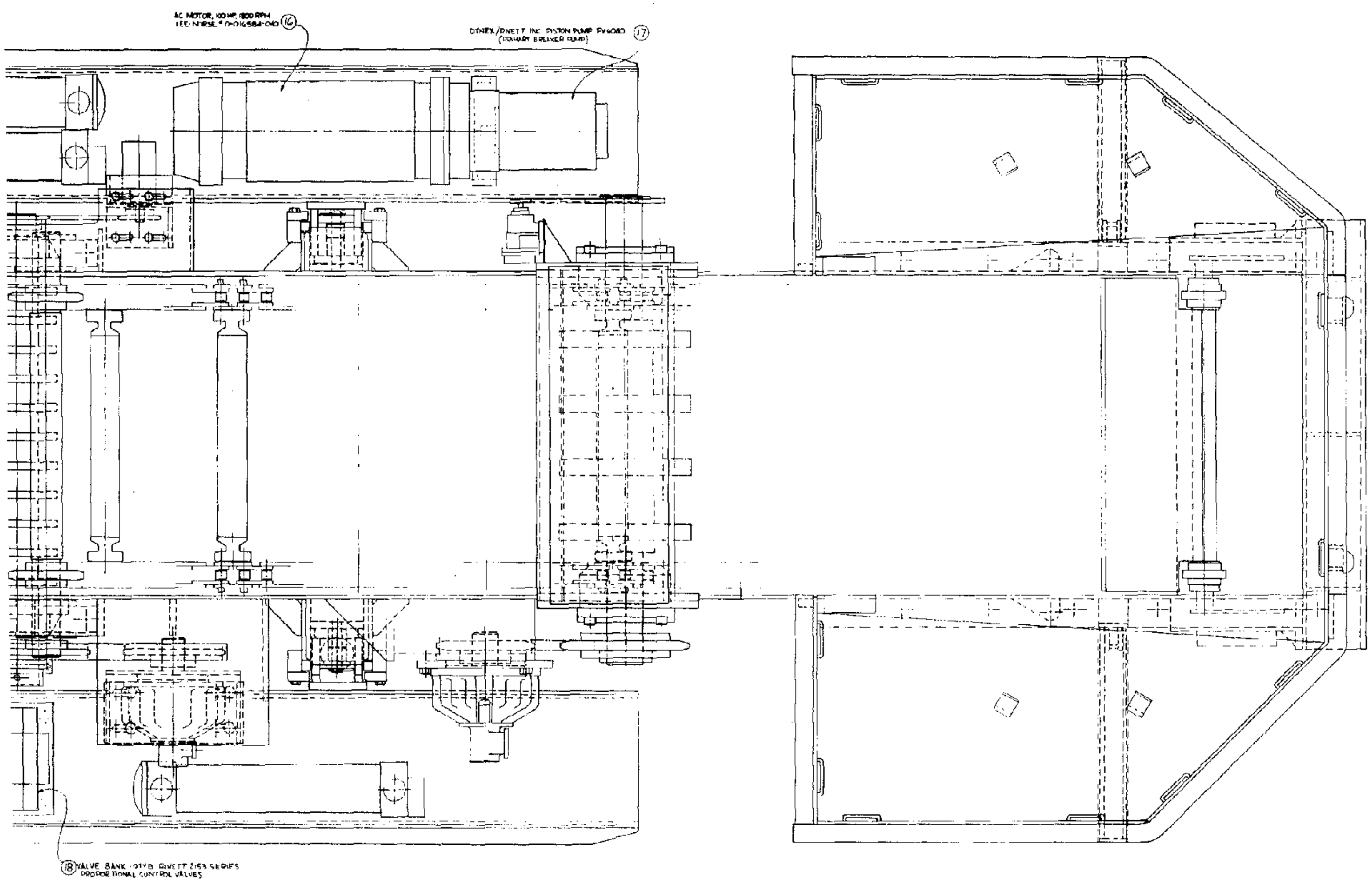


Figure 4-67: Ingersoll-Rand/S&S Jet Pump Injector Vehicle, Front Top View. A copy of the full scale design drawing is on file with the Bureau of Mines.

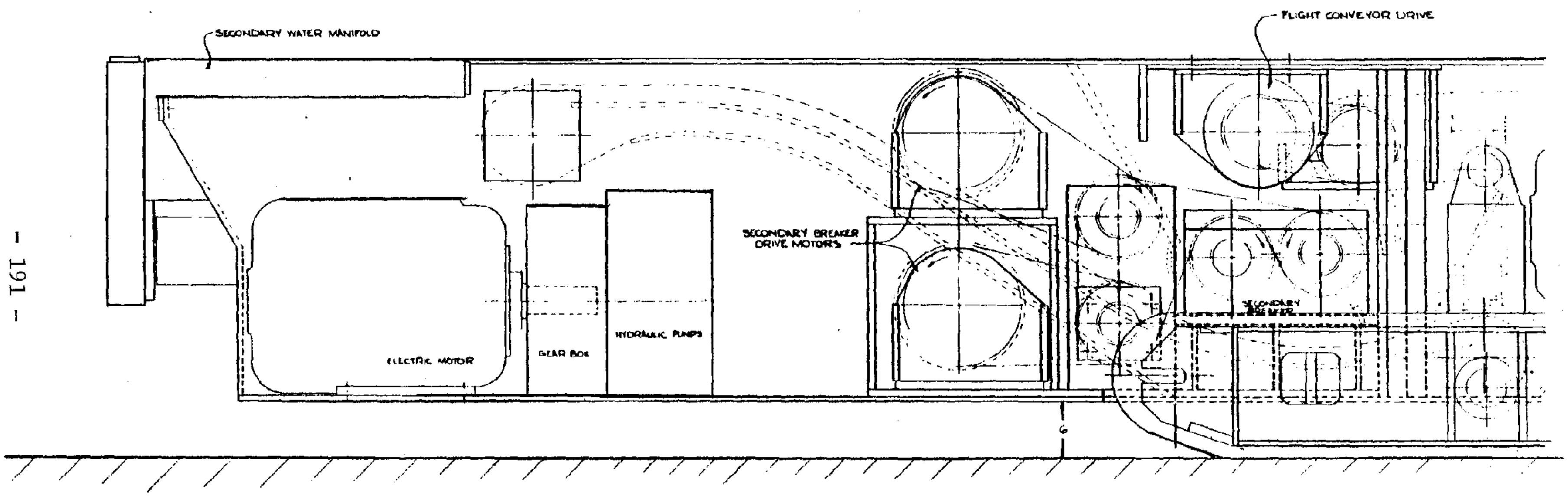


Figure 4-68: **Ingersoll-Rand/S&S Jet Pump Injector Vehicle Right Rear Side View.** A copy of the full scale design drawing is on file with the Bureau of Mines.

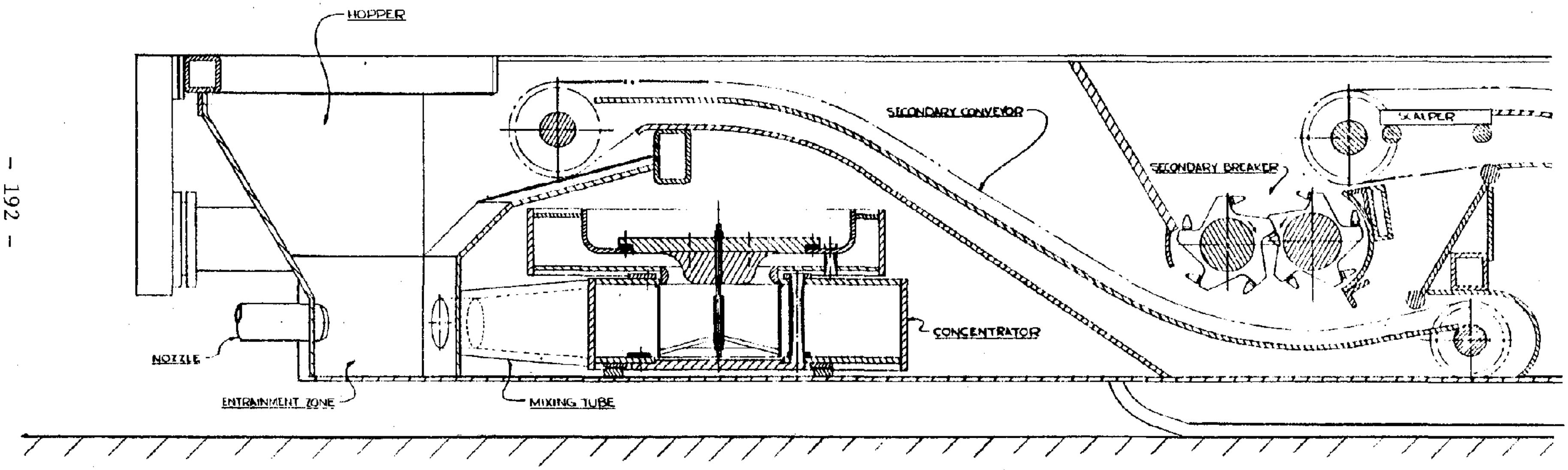


Figure 4-69: Ingersoll-Rand/S&S Jet Pump Injector Vehicle
Right Rear Sectioned View

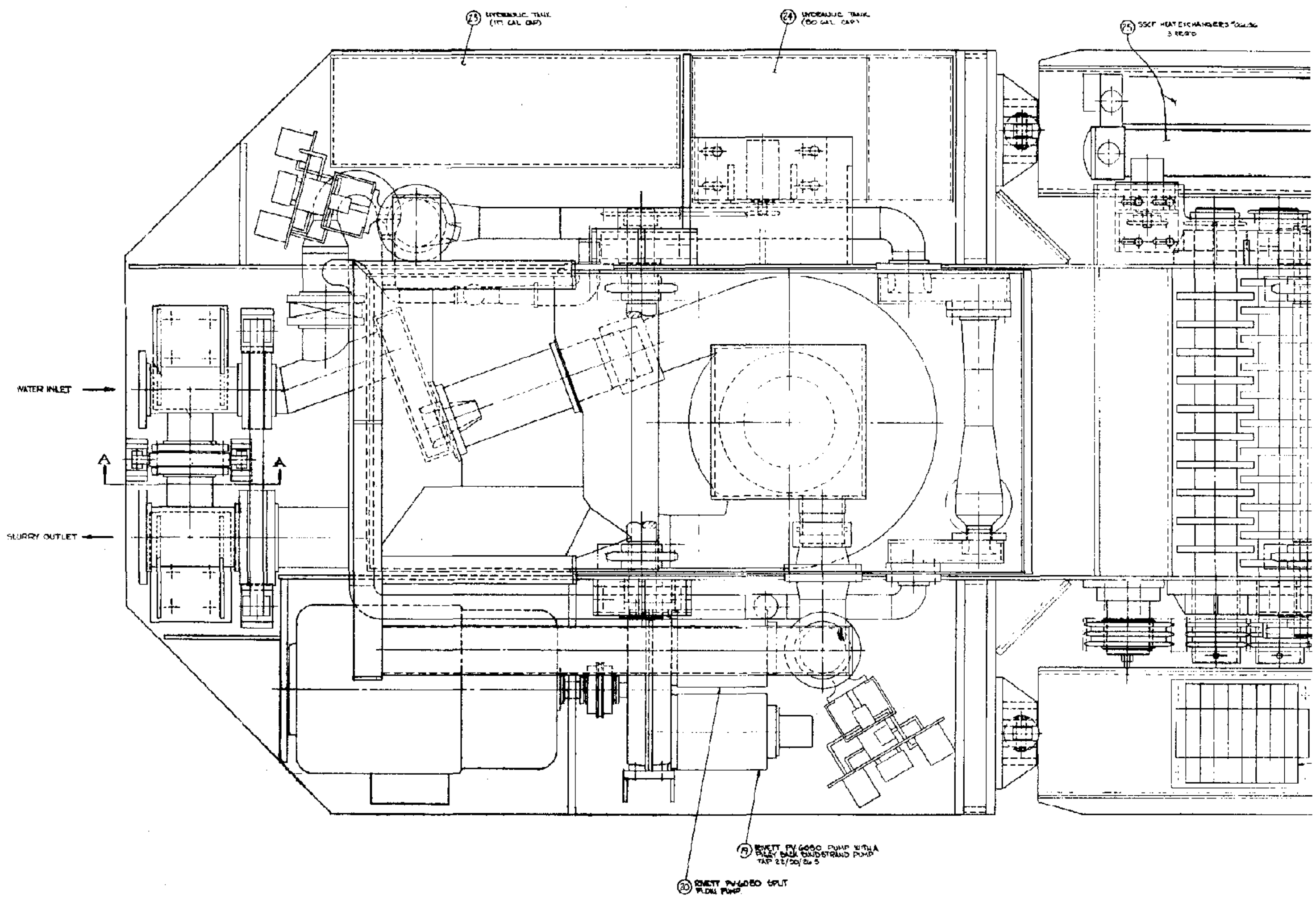


Figure 4-70: Ingersoll-Rand/S&S Jet Pump Injector Vehicle, Rear Top View

part of this project. The concentrator separates the incoming dilute slurry into concentrated slurry and water. The water is removed from the device through an overflow port located on the top center of the concentrator body, is diffused in a volute to raise its pressure, and is recirculated into the hopper through spray manifolds surrounding the top of the hopper. The ratio of the overflow water extracted controls hopper water level and concentration increase. Overflow is set by the position of a valve in the overflow duct. The concentrated slurry is discharged into the slurry transport line.

Since coal mass flow rate into the injector system is highly variable, a reserve water supply is provided in order to smooth the resulting flow rate and pressure transients in the line. This water is contained in the volume surrounding the concentrator and can serve as a heat-exchanging medium for the hydraulic oil.

The final design and operating specifications for the Mobile Jet Injector Vehicle, as designed and described above, are given in Table 4-6.

Additional information and detail on several of the major components of the system, providing the rationale for configuration and selection, is contained in the following sections. Alternative concepts are also discussed.

4.4.1.2 Slurry Concentrator

The slurry concentrator removes the excess water present in the diffuser section of the jet pump injector. This component is required to raise the mass fraction of coal in the delivered slurry to specification value which would not otherwise be possible.

The concentrator is of value if solids concentrations over about 30% are needed, particularly where high solids throughput is desired in small slurry lines.

The concentrator contributes to a higher overall efficiency with the jet pump by providing, as an essentially free by-product of the concentrator, the secondary water that is entrained by the jet, along with the solids. In the absence of a concentrator, the secondary water in all probability would have to be supplied from the same high-pressure line that supplies the jet nozzle which would reduce the jet pump overall efficiency.

The concentrator in this case is located on the same

Table 4-6: Final Design and Operating Specifications for the Mobile Jet Injector Vehicle

<u>Basic Feeder Breaker Vehicle:</u>	
Length	25' 4"
Width	9' 0"
Height	3' 6"
Weight (Empty)	38,000 Lbs.
Weight (Loaded)	48,000 Lbs.
Track Ground Pressure (Empty)	13 psi
Track Ground Pressure (Loaded)	17 psi
Torque (Each Side)	12,000 Ft-Lb
Draw Bar Pull (Max.)	25,000 Lbs
Top Speed (No Load)	80 Ft./Min
Top Speed (Max. Draw Bar Pull)	30 Ft./Min
Grade Climb (with a Rolling Resistance of 250 Lbs/Ton)	10%
<u>Jet Pump Injector Subassembly:</u>	
Coal Hopper Capacity	3.5 Tons
Jet Diameter	56mm
Coal Entrainment Length	230mm
Slurry Line Size	8"
Nozzle Pressure	200 psi
Coal Flow Rate	6.4 TPM
Coal Concentration in Slurry	43% Min. (wt)
Slurry Line Pressure	20 psi
<u>Breaker System:</u>	
Primary Breaker	From 14" ROM to 7"x0"
Secondary Breaker	From 8" to 3" x 0"
<u>Power Requirements</u>	
Primary Conveyor (at 56 RPM)	50 HP
Secondary Conveyor (at 110 RPM with an 18:1 Gear Reduction)	30 HP
Secondary Breaker (at 400 RPM)	50 HP
Main Water Pump	250 HP

vehicle as the jet pump. Alternatively, the concentrator could be located remotely, for example, at the first boost pump. In this case the secondary water would have to be fed back to the jet pump vehicle which would require a third line in addition to the lines for high pressure water and for slurry.

In some applications, concentrations of less than 30% may be acceptable and a concentrator would not be required. In such cases, the jet pump geometry would be chosen to minimize the use of high pressure secondary water.

For the present application a centrifugal slurry concentrator was selected for the full-scale vehicle design. Although work in an earlier phase of the program evaluated and demonstrated the feasibility of a linear concentrator configuration, the centrifugal configuration was considered more attractive on the basis of operating and packaging efficiencies.

To support this design selection, subscale performance tests were conducted on a transparent, centrifugal concentrator model. This design and test activity is described in Section 4.4.2 and fully documented in Appendix E. The results of this work confirmed that the centrifugal concentrator concept selection was correct for the application.

4.4.1.3 Hopper - Jet Pump Subassembly

The coal hopper and water powered jet pump together form an integrated subassembly that receives the on-board processed coal and provides the energy to inject the coal into the hydraulic slurry haulage pipeline.

Early, quarter-scale model testing of the hopper was used to develop a design configuration that would avoid major stagnation regions. The full-scale hopper, shown in Figure 4-69, is modelled after the sub-scale test unit and basically consists of a short, broad tank where all the coal and water to be delivered is brought together just prior to injection.

The hopper's length near its base has been confined to the length of the high velocity water jet from the nozzle discharge exit plane to the entrance plane of the mixing tube. The sides of the hopper are vertical except for sloping plane sections that converge on the coal entrainment zone. Care has been taken in the design of the hopper to assure, as far as possible, that all coal entering the hopper experiences a short, uniform residence time. Provisions are made to facilitate the movement of the coal through the hopper by sweeping the side walls with water. The holding capacity of the hopper is approximately 3.5 tons.

The hydraulic jet pump is positioned at the base of the hopper. It consists of a nozzle discharging a high velocity water jet, a coal entrainment zone (where incoming coal surrounds the water jet), a coaxial mixing tube (into which the coal-water slurry is driven) and a diffuser section (where the slurry is decelerated before entering the concentrator).

The physical dimensions used in the design of the full-scale jet pump (e.g. throat diameter, entrainment length) were based on the subscale model and analytical model activities of Phase IIA. Parametric studies of the pertinent design variables, as they affected performance, were carried out for optimization purposes. Adjustments in the analytical model were made to reflect the test results obtained. In this manner a design method was developed that permitted the confident design of the full-scale jet pump.

4.4.1.4 Reserve Tank

It is of paramount importance in the functioning of the jet pump injector that the mixing tube inlet remain submerged during operation. Thus, the water level in the hopper must be maintained within a specified range by a control system that is sensitive to water demand while avoiding hopper overflow. The control of the water level would be a comparatively easy matter if the hopper volume were large enough to allow for slow adjustments of input and output conditions. In the actual situation, however, water throughput rates must be made in a fraction of second. Since the luxury of a large hopper volume was not possible (due to vehicle size restraints) nor indeed desirable considering the required shape of the hopper in the coal entrainment zone, a separate water volume was considered appropriate.

Accordingly, a separate water storage volume was created by including what has been identified as a "reserve tank". This tank serves as water surge storage and is located adjacent to the jet pump injector. Its function is to either supply or receive water instantaneously with no time delay due to acceleration or deceleration of the water in a long line leading to the main source of water for the jet pump injector. It has been estimated that a time delay of about 5 seconds minimum would be required to double or halve the water flow from a remote source 1000 feet away from the Jet Injector. This is based on a round-trip pressure-transit time of about 1 second for 1000 feet. Five step-wise changes or a continuously-varying flow at the average rate of change would be satisfactory to avoid unwanted pressure spikes (water-hammer) due to rapid variations of flow.

For purposes of design, an engineering analysis of the required tank volume indicated that it should have a capacity at least equal to the volume that would flow in 5 seconds at maximum water flow rate. This design criteria met the space limitations and was shown to be adequate during the experimental use of the jet injector. During normal operation (but not at the maximum possible flow rates) water level could be controlled well within the chosen limits. Frequently, the variations were held to values in the range of plus or minus two inches during the stopping and starting of the flow of coal. This represents the largest normal transient that the equipment would experience.

4.4.1.5 Cross-Over Valving

The source of water needed to operate the jet injector will typically be remote from the jet injector itself. It may, for example, be a centrifugal pump located at the reservoir at which water used for the slurry transport system is stored. It has been assumed that the typical location may be 1,000 feet or more from the jet injector. In this case, start-up of the centrifugal pump would be at no-load and it is possible that the pump would generate a flow substantially higher than the jet pump nozzle could accept at the desired peak pressure.

To avoid the possibility of water hammer under these circumstances, four techniques were considered and are described below.

First, a restrictor, sized to allow a flow equal to or less than the desired nozzle flow would be placed in the line. After nozzle flow was established, the restrictor would be bypassed to avoid energy loss during running and provide for other flow demands.

A second approach would utilize a nozzle bypass system located at the jet injector site capable of accepting the full flow of the remote water pump without undue pressure rise when the flow arrives at the jet injector. The bypass device would be slowly closed as soon as full flow has been achieved.

In a third approach the bypass system noted above could be supplemented by a second bypass having an orifice flow control just equal to that provided by the nozzle itself. This arrangement could be used effectively during any period of time in which the bypass flow might be switched rapidly between the nozzle and the bypass orifice. The principal uses of this bypass would be preparation for switching established flow to the nozzle, for start-up of the jet injector, or, conversely, switching flow from the nozzle to the slurry line in the event that high-pressure water was needed to clear a clog (or an impending clog) shown by a rise in the water level in the hopper of the jet injector. This approach would be the fastest way to switch flow without causing pressure spikes.

The fourth and final method considered (but not implemented) involved the use of an axially-traversable nozzle to achieve the flow-switching just discussed. Normally, the jet injector nozzle is spaced from the inlet to the mixing tube by a fixed distance of several inches which creates the "entrainment" length of flow of the high-velocity nozzle output through the mixture of coal and water in the hopper. An arrangement could be made to move the nozzle forward to enter the mixing tube and to engage a shut-off seal located a few inches back from the nozzle opening. The effect of moving the nozzle forward would be to gradually increase the pressure in the exit line up to the full exhaust pressure of the nozzle, (as a limit) or to whatever lower value might be needed to clear a clog or clear coal from the slurry line.

It can be seen that ample means are available to make the jet injector a workable device in a real dynamic system. It should be noted that a number of the features discussed above represent the summation of thinking related to the control problem and were not in place during the experimental test work performed. In fact, during testing, substantial initial problems were caused by "water-hammer" that was not adequately controlled. The major problem was the failure to either limit flow at the pump or provide for accepting full flow at the jet injector without adequate means to bypass and then reduce the start-up flow. These problems were eventually handled in the field by throttling flow from the pump through a variable area valve, which therefore gives credence to the belief that any of the four techniques described above would be adequate in avoiding the water hammers.

The equipment actually used for controlling the "cross-over" function during jet pump injector testing utilized three valves. The first valve controlled the "cross-over" from the incoming water line to the outgoing slurry line. The second valve shut off water flow to the nozzle. The third valve shut off the slurry exit from the jet pump to the outgoing slurry line. This function was required to prevent backflow of high-pressure water into the jet injector. This valve in effect served as a slurry valve, since the line being closed could contain solid particles.

Because of the problems of matching valves and the synchronization of their opening and closing times, a decision was made to use a common valve actuator that would operate all valves during bypass. This arrangement gave positive assurance that the valving was synchronized, and was incorporated into the final vehicle design.

4.4.1.6 Coal Sizing and Conveying Equipment

A feeder-breaker system is required on board the mobile jet injector vehicle for the purpose of top-sizing and moving the coal or other product for final injection into the hydraulic slurry transport line. The upper limit on particle size suitable for injection is typically set at approximately one-third of the slurry line inside diameter. It is also true that no unnecessary breaking should occur since all fines produced during transport or crushing are very expensive to clean in prep plants and much can be lost. (Note: The cost to clean fines is about 4 times the cost to clean coarse coal.) Accordingly, design of the coal handling and sizing system for the vehicle was based on maximizing 3" product and transporting 3" x 0" product preparatory to injection into the 8" slurry line size.

The system utilized to accomplish the breaking and conveying tasks consists of the following elements which are described in the following paragraphs:

- Primary Breaker (Single-Roll)
- Secondary Breaker (Double-Roll)
- Scalper
- Flight Conveyors

Primary Breaker

A production-type breaker, designed by the S&S Corporation was used as the primary (or first stage) breaker. This conventional, single roll unit is capable of handling 14" ROM material and sizing it to 7" material. The basic breaker has been widely used on feeder-breakers manufactured by Ingersoll-Rand Company. A new energy-storage drive system greatly reduces the size and cost of the drive system, while improving the capability of the breaker itself.

Secondary Breaker

Design of the secondary breaker was substantially constrained by the space allocations on the mobile vehicle. However, it was found to be possible to achieve the top sizing of the coal from the 8" material (delivered from the primary breaker) to the maximum 3" particle size required for injection and transport utilizing a breaker design featuring two 11" diameter, counter-rotating rolls. This breaker design offers high throughput per unit of breaker volume and bit structures permitting material top sizing with minimum fines production. High throughput (up to 8 TPM) is achieved with a 400 RPM rotational speed. This high operating speed could be utilized safely because of the enclosed location available near the middle of the vehicle. During test, very little "kick-back" occurred, especially with the dragbit design.

The secondary breaker, positioned as it is at midships of the mobile vehicle, offers the following characteristics:

-Top-sizing geometry: Any coal or rock particle that cannot fit between breaker elements will be broken, usually in transverse rupture.

-Minimum crushing: Any coal or rock particle that fits between breaker elements will tend to pass through without crushing. Opposing flat areas are held to a minimum so that almost all of the undersize particles can pass through without being trapped and rebroken.

-Maximum feed-through: "Paddle-wheels" tend to feed undersize material through the bight without further breaking.

-Personnel safety: The covered, midship location of the breaker, with no direct path for kickback of solid particles provides for operator safety. Operation with cover removed must be interlocked to assure that the breaker cannot start while under inspection or service.

-Low power required: Since only about 10-15 per cent of the mass feed will require breaking and peak loads are met with stored energy, the breaker will consume very low power for a device of its throughput rating of 5.8 metric TPM.

Scalper

The scalper is utilized to assist in limiting fines generation by preventing overbreaking of the coal. Properly sized coal is separated and allowed to pass through the scalper and flow directly to the jet injector hopper, thus bypassing the secondary breaker. In addition to minimizing overbreakage, this device reduces the size (and throughput requirement) of the secondary breaker, permitting a more compact design.

The scalper that was developed features a combination of parallel, trapezoidal bars with recessed, round spacer bars. When used in conjunction with a flight conveyor the scalper is effective in preventing slabs from "diving" through. When a slab had a tendency to pass through, the next conveyor flight either rotated the slab out of the groove causing it to continue on into the secondary breaker or broke it down to suitable size.

The scalper concept, developed during the program in conjunction with the breaker testing activities, worked successfully. However, it probably would not be utilized in a final vehicle design because of space constraints on the vehicle and because it was found that the secondary breakers could pass fines freely.

Flight Conveyors

Conveying of the coal from station to station within the mobile vehicle is accomplished by flight conveyors propelled by electric motors. The units specified for use in this application are of the same, heavy duty configurations currently utilized by the S&S Corporation in many of their current underground mining product lines.

Two separate flight conveyors are required to service this vehicle - a primary conveyor and a secondary conveyor.

Coal from the continuous miner enters the surge bin. The sloped walls of the bin direct the ROM coal to the primary flight conveyor which moves the coal to and through the primary breaker. The rough-sized coal is then carried by the conveyor to vehicle midships where it is deposited between the rolls of the secondary breaker.

After passing through the secondary breaker, the final-sized, 3" x 0" crushed coal is deposited by gravity onto the secondary flight conveyor. This conveyor moves the coal from vehicle midships to the jet pump injector hopper in preparation for injection into the hydraulic slurry haulage line.

The secondary flight conveyor is of conventional design. It is noteworthy, however, that it can be operated quite differently from the primary flight conveyor. The primary flight conveyor must feed coal into the single roll, primary breaker and is, in a sense, part of the breaker. The speed of the primary conveyor must be held to 40 m/min, maximum. At this speed, the rated tonnage produces a fairly thick layer, about 240 mm thick.

In contrast to the primary flight conveyor, there is substantially less loading on the secondary flight conveyor because the upper limit on conveyor speed is much higher. In principle, a much thinner layer of material may be conveyed.

4.4.1.7 Control System

The purpose of the mobile vehicle, on-board control system is to regulate the input and output flows of water and solids as they progress through the various components of the vehicle. The control of these flows would not be difficult if large, buffer storage volumes could be provided on board the vehicle. For example, retention volumes for storage equivalent to approximately one minute of maximum flow would be considered reasonable. If such space was available, the required correction rates to material flows could be accommodated by a wide variety of existing equipment in the form of instruments and actuators.

In the actual hardware, however, flow rates are typically in the range of 2000 to 3000 gallons per minute, while the storage capacities available are about one-tenth these values. The small storage volumes are, of course, the result of the limited space remaining on the vehicle after other equipment needs have been met. The maximum vehicle height is 42 inches, the minimum ground clearance is about 6 inches, the available width is about 36 inches, and the available water hopper length is no more than 36 inches.

Considering the available space it was, therefore, necessary to control the flows into and out of the jet injector. Originally, it was intended to control flows directly, using flow measurements of high pressure water, overflow water, and slurry flow rate. Although considerable effort was devoted to achieving the desired flow measurements, this plan was finally abandoned in favor of a scheme to control the system based on fluid level measurements rather than flow measurements.

It should be noted that it was always possible to obtain stable, correct flow measurements on the incoming high-pressure water. This measurement alone, however, was insufficient to control the operation of the jet injector. It was, however, useful in the determination of total flow rates, in conjunction with the weight of the incoming solids as determined by the weigh-belt readings.

Some discussion of the experimental testing of the full-scale jet pump injector subsystem is warranted to explain, in as much detail as possible, how the jet pump injector flows would have to be controlled on a full-scale mobile vehicle of the future. The discussion that follows relates to water level measurements and hopper water level control within the test system itself.

Level Measurement

Water level measurements were made using Drexelbrook level sensors. This system was tolerant of the "dirty" water containing suspended solids that were mostly coal. However, it was found necessary to reduce water sloshing and extremes of aeration so that there was a tranquil water level that could be measured. This effort required substantial baffling of flows in the various water volumes concerned as described below.

First, the control scheme involving water level should be reviewed. Incoming water was derived from a single source - namely, the centrifugal pump located at the water source. In the system used, the water source was a few hundred feet from the jet pump injector subassembly. The controls provided were chosen to be adequate with the water source 1000 feet from the jet injector. As a base condition, it was assumed that flow changes could be made at the rate of five percent of the flow in a time period equal to 0.4 seconds which was approximately the time needed to travel to and from a water source 1000 feet away. In actual practice, it was not necessary to change at this rate in order to smoothly control the system.

It was possible to demand increases or decreases in flow without any variation of valves or controls at the source of the water. This capability was possible because of the operating characteristic of the centrifugal pump which had been chosen for the water supply. This pump, an Ingersoll-Rand Model 8X23SF is rated at 500 feet of head at 3000 gpm.

It was determined from scale up of the subscale model flow rates, that a nozzle flow of 2000 gpm would be adequate to achieve the solids flow rate of 6.4 TPM set forth in the specification. Sizing of the main nozzle provided for this rate of nozzle flow at 200 psi, or approximately 500 feet of head. The characteristic of the chosen pump is nearly flat in the region from 200 gpm to 3000 gpm. Accordingly, all necessary adjustments of flow could be accomplished at the jet injector, provided only that flow rate changes not be made more rapidly than about 200 gpm per second, to allow for transit time with distances up to 1000 feet to the high-pressure water pump.

Water-Level Control in the Hopper

The hopper is of the same type as that found to be optimum in the Phase II subscale work. The space limitations in the full-scale device, however, did not allow a direct geometrical scale-up of the configuration tested in Phase II. This situation was due, in large part, to the fact that high priority had to be given to the correct placement of the concentrator inlet and discharge plumbing. Consequently, the hopper was minimally modified to align its discharge chute with the concentrator inlet, as shown in Figure 4-70, in order to handle the following flows:

1. Coal, at a rate from 0 to 6.4 TPM (~ 1000 gpm).

2. Overflow water, at a rate from 0 to 600 gpm.

3. Auxiliary water, at a rate from 0 to 1000 gpm.

In summary, the hopper design underwent evolutionary configuration improvements, as the test activities progressed, that led to the definition of a completely operational, full-scale jet pump injector subassembly. This integrated sub-assembly included the hopper, the jet pump injector and the control system.

From this experimental work certain important design features emerged that should be recognized and embodied in any future, complete, full-scale mobile jet injector vehicle design. These features are noted and discussed below.

First, all incoming flows should be immersed, so that they do not tend to aerate the contents of the hopper. In practice, this involves distributed sheet flow of incoming liquids, except for the nozzle flow which must be a solid cylindrical flow on the axis of the mixing tube. In order to obtain the desired sheet flows, manifolds must be used to receive incoming flows from the regulating valves and their round exit pipes. Exit flow from the manifolds should be controlled by one or more slits that produce a distributed "sheet" flow tangential to the bottom of the hopper. This scheme proved to be effective even in the worst condition which occurs with water flow without coal in the hopper.

Next, the Drexelbrook probe that senses water level in the hopper should, as far as is possible, be exposed to the average water level in the hopper. Conditions to be avoided by design are water jets directly impinging on the probe itself or on adjacent hopper surfaces and any undue restriction of water flow to or away from the probe. Adherence to these criteria will prevent time delays from being created at the sensor with respect to the true rising or falling of the water level in the hopper. Consideration should be given to the use of two or more level-sensing probes positioned at different locations in the hopper so that their outputs could be averaged to achieve the best measure of water level.

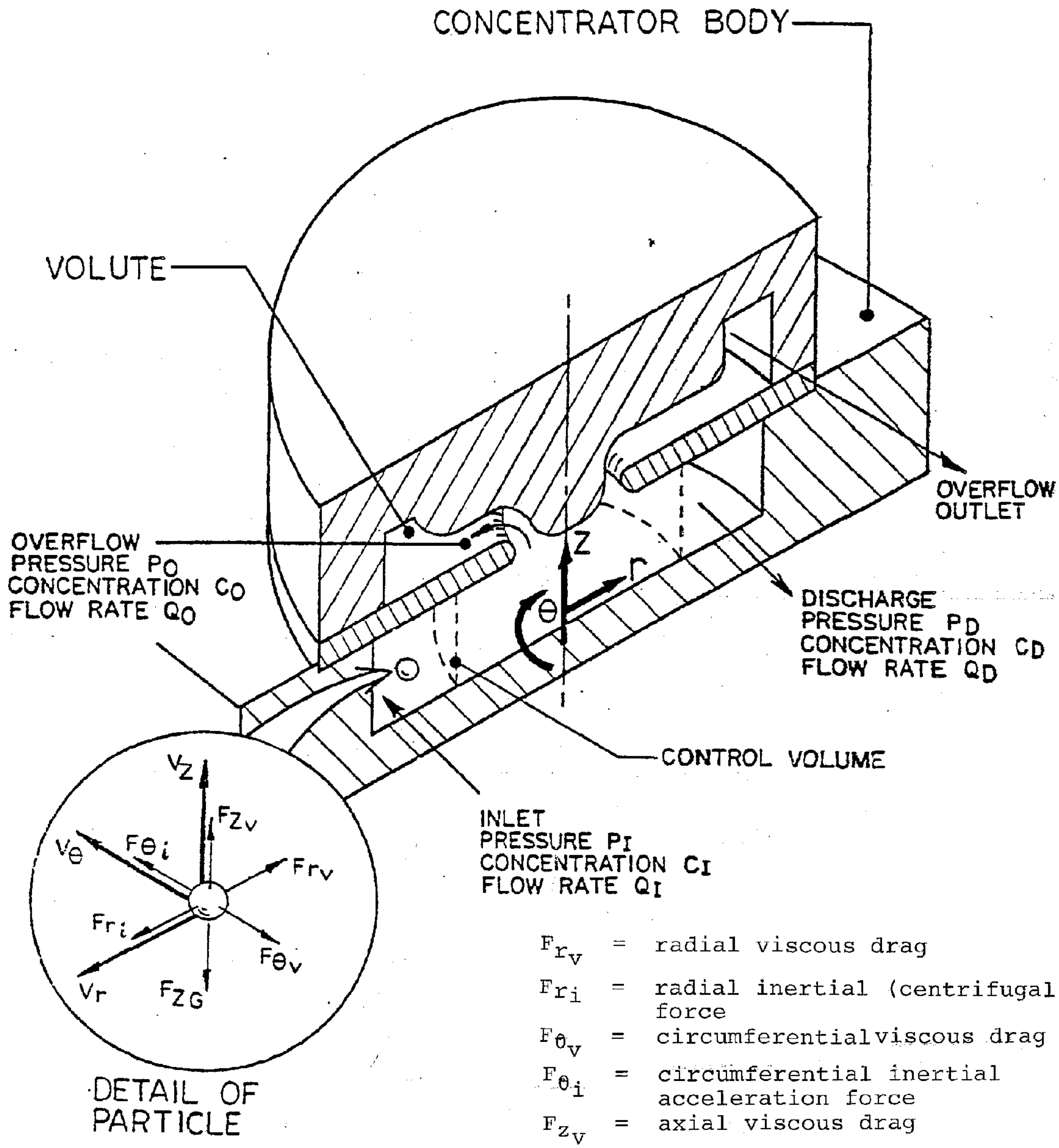
The level system described above worked well with aerated coal and could therefore be refined to be an accurate, trouble-free control element in any future, full-scale mobile jet injector vehicle design.

4.4.2 Design Fabrication and Test of Subscale Concentrator

During the full-scale mobile jet injector vehicle design effort of Phase III, it became apparent that on-board space constraints mitigated against the use of the linear concentrator concepts generated and tested in Phase IIB (see Section 4.3.3.2). The problem of accommodating the required axial distance within the vehicle was the determinant. Accordingly, a new approach had to be taken in order to conform with the space available for a concentrator on board the vehicle in the vicinity of the jet pump.

Consideration of a centrifugal (cyclonic) concentrator was pursued, since its natural shape would best fit the available volume on the vehicle. The selected concept is shown schematically in Figure 4-71.

Since information and design criteria, suitable for establishing a full-scale concentrator for integration on the vehicle was not available, a subscale injector development



F_{r_v} = radial viscous drag

F_{r_i} = radial inertial (centrifugal) force

F_{θ_v} = circumferential viscous drag

F_{θ_i} = circumferential inertial acceleration force

F_{z_v} = axial viscous drag

F_{z_g} = axial gravitational force

Figure 4-71: Schematic View of Centrifugal (Cyclonic) Slurry Concentrator activity was undertaken to develop the necessary data.

The operating conditions that the concentrator would have to meet in the jet pump injector system were:

A. Coal flow rate: 6.4 tons per minute. The coal would be run-of-mine Appalachian coal with a large fraction of included dense refuse. The materials' average specific gravity would be approximately 1.8.

B. Maximum solid particle size: Approximately 3 inches. Occasional, somewhat-larger particles may be encountered and should not present difficulties.

C. Hopper water level: Maintenance of a minimum water level in the hopper to ensure a completely submerged jet operating at optimum performance.

An accurate analytical description of the physical principles governing the operation of the centrifugal concentrator was not possible due to (1) the heavy turbulence of the flow pattern, (2) its nonhomogeneity, (3) the wide variation in size and shape of the solid particles, (4) variability of solids concentrations and (5) the interactions between solids. However, an approximate analytical description was attempted and found useful.

The criterion determining whether any given particle will migrate toward the periphery of the concentrator (separate), toward the center (entrain), or toward neither, staying at a constant radius, is the radial force balance on the particle depicted graphically in Figure 4-71. The two opposing forces acting radially are inertia and viscous drag. The tangential and vertical forces have no first-order effect on particle separation.

A detailed presentation of this analysis is given in Appendix E.

To evaluate the effectiveness of the centrifugal separator concept, so as to provide confidence that a full-scale unit would be operationally satisfactory for the mobile jet pump injector vehicle, a subscale concentrator model was designed, fabricated and tested.

A 1-to-6.4 scale model of the domed-disk cyclonic concentrator was fabricated and installed in the small-scale test loop illustrated schematically in Figure 4-72. The scale factor of 1 to 6.4 was selected to permit the use of a 1 1/4 inch inside diameter hose to represent the 8 inch ID inlet and discharge pipes of the prototype vehicle.

The model concentrator, made from transparent acrylic for flow visualization, was supplied with slurry from a 4 ft-diameter by 5-ft high sump tank which normally contained 100 to 200 gallons of liquid. A roughly homogeneous solids concentration was maintained in the tank by a propeller-type mixer. An electrically-powered centrifugal pump delivered slurry from the sump tank to the concentrator at flow rates up to 100 gallons per minute or at pressures up to 50 psi. A positive-displacement pump would have been preferred so as to allow flow rate and pressure to be set independently. However, a positive-displacement pump that was both small enough and able to handle coarse slurry was not available.

Inlet and discharge stream flowed through transparent, flexible polyvinyl chloride hose. The hose was squeezed with C-clamps to set back pressure which, like all pressures was measured with Bourdon-tube type pressure gauges. The overflow stream moved from the concentrator into a box fabricated from transparent acrylic sheet which in turn emptied into a weir channel for volumetric flow measurement. The weir was necessary because the overflow stream is by then open-channel (at atmospheric pressure) and conventional flow gauges were not usable. For a V-notch weir, such as was used here, flow rate is proportional to the 5/2 power of the height of the water level above the notch bottom. The weir channel used here had a capacity of 50 GPM. It emptied into a collection drum whose contents are returned to the sump tank by a recirculating pump.

Measurement of the flow rate of the discharge stream was not straightforward. Since the discharge contained a high concentration of coarse solids, the only way to measure volume flow accurately was with a collection vessel and stopwatch. A second drum was therefore included in the apparatus. This drum was calibrated on the inside with paint marks indicating every 5 gallons. A three-way valve at the end of the discharge hose allowed flow to be directed either immediately into the sump tank or into the collection drum. To measure discharge flow rates the flow was directed into the collection drum and the time to fill it to a preselected volume mark was measured by stopwatch. Simple arithmetic yielded flow rate. The same recirculating pump that emptied the overflow drum returned the collected slurry to the sump tank.

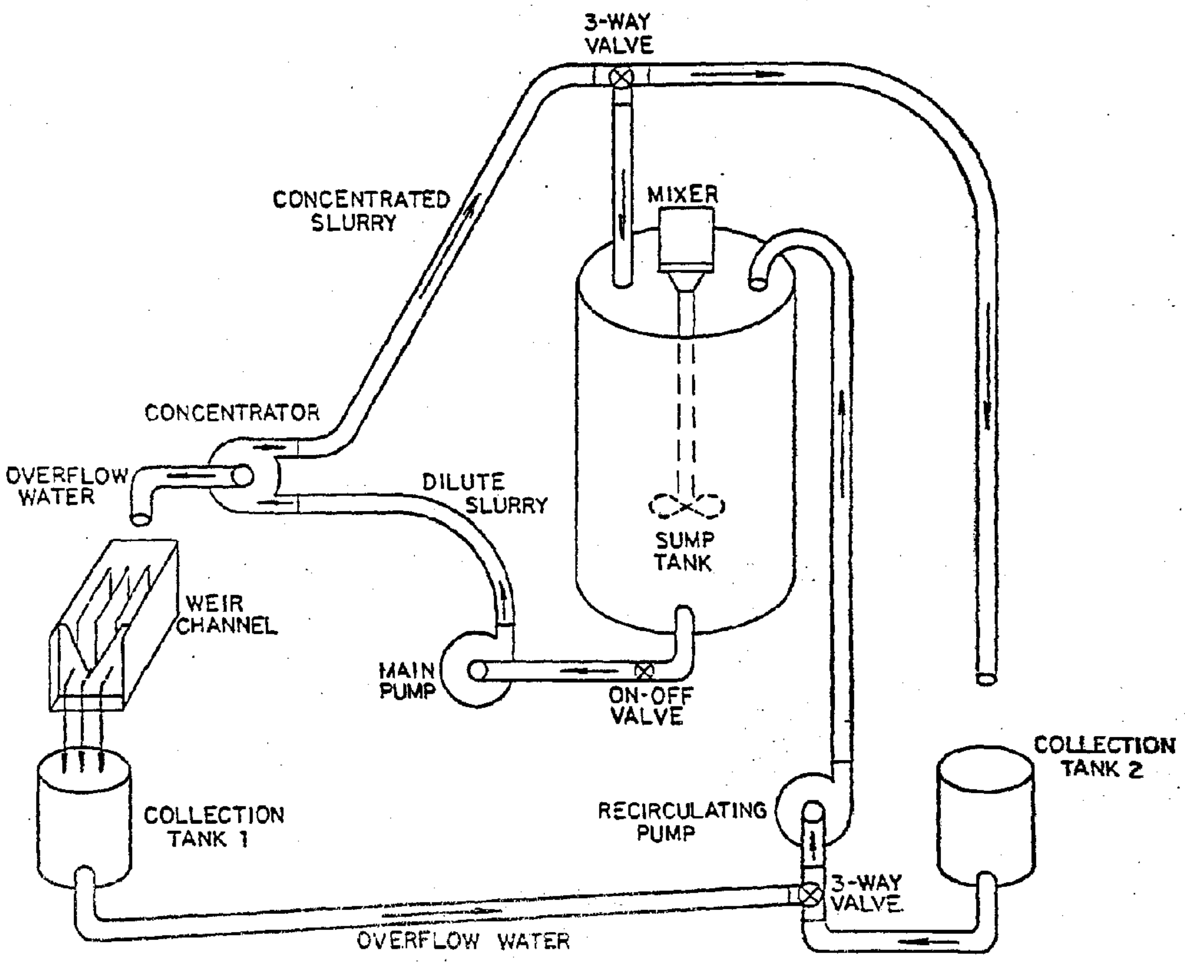


Figure 4-72: Slurry Concentrator Test Loop Schematic

For most testing, coal was simulated by pellets of Delrin, an acetal resin manufactured by DuPont. These pellets were about 1/8 inch diameter (about 3/4 inch full scale) had the same specific gravity (1.41) as most bituminous coal. Delrin was preferred over real coal for testing purposes because of its cleanliness and resistance to breakage.

The goals of the model testing were primarily the optimization of the geometry of the concentrator and secondarily the establishment of the salient performance characteristics of the optimized geometry. Complete and detailed information about the performance of any particular shape under all possible operating conditions was unnecessary and exceeded program resources. The appropriate procedure was, therefore, to collect data only around normal operating points for proposed geometries. A detailed account of the test loop operating procedure, the operating conditions evaluated and the results achieved is given in Appendix E.

4.4.3 Testing of Full-Scale Jet Pump Subassembly

4.4.3.1 Test Facility Planning

In order to carry out the developmental testing of the jet pump injector subassembly it was necessary to have a test facility that provided the following functions.

- Controlled inputs to the jet injector of water under pressure and coal.

- Means for accepting the output of the jet injector which would be a slurry formed by combining the coal and water in the jet pump injector.

- All other facilities, including but not limited to, the required space, utilities, and support personnel necessary to conduct the tests.

Initially, it was planned to establish the necessary surface test facilities at IRRI sufficient to perform developmental testing on the jet pump subassembly. It became evident that problems associated with the handling of large quantities of coal at the site of the Ingersoll-Rand Research Center were formidable. It would be necessary to completely contain all solids and liquids that might otherwise contribute to pollution.

The preparation of such a test loop under this program was considered to be unwarranted since other facilities adaptable to this service were found to be available elsewhere.

With the knowledge and assistance of the Bureau of Mines a suitable slurry test facility constructed under another Bureau of Mines contract by Foster-Miller Inc., was used for testing the jet pump injector subassembly. The test loop is located in Framingham, Massachusetts. All testing reported in this section was performed at this facility.

4.4.3.2 Description of Test Facility

The test loop provides the circulation of slurry through four different test loops of increasing length, the longest one being 1,000 feet. Slurry is formed by the experimental equipment being tested and is injected into the slurry line. After transit through the selected length of slurry line, the slurry is discharged onto a vibrating screen in order to separate the water and solids for subsequent reuse. Continuous circulation of slurry can be maintained over a period of time with the separation and recombination of solids and water being carried out on a continuous basis. Test loop operation is limited only by coal degradation. It has been estimated that approximately 30 passes are required to degrade the coal particle size to the point where it is no longer retained on the vibrating screen and must be replaced by fresh coal.

A recirculating arrangement allowed a limited water supply and a limited coal supply to be cycled through the slurry system on a semi-continuous basis. Coal and water were mixed to form the slurry and a pressure head, generated by the jet injector pumped the slurry to one or more sections of 8-inch diameter cast iron pipe used for a slurry loop. After one time around the test loop, the slurry was delivered to a vibrating screen so that the water and coal fines could be removed, making the lump coal available for the next trip around the loop in the same manner as the first traverse around the test loop.

In order not to have rapid wear on the coal particles that were passing through the slurry line, anthracite coal was used instead of bituminous coal. It was possible for the anthracite coal to be passed around the loop approximately 20 to 30 times before it had degraded to particle sizes no longer considered useful for obtaining test results.

For the jet pump injector testing conducted by

Ingersoll-Rand Research, Inc. major changes and additions to the test facility were required.

First, the total power source for mixing the slurry and for carrying out the injection of the slurry into the slurry line was derived from a water pumping station. The pump used was an Ingersoll-Rand rated at a pressure head of 500 feet and a deliverable flow of 3000 GPM. Water pressure heads could be reduced by throttling the discharge line to obtain a desired flow rate.

The second salient feature added to the test loop was a fast-acting servo system that controlled the incoming water level and the jet injector hopper which could receive both coal and water and also could control the water level in a water reserve tank located adjacent to the hopper used for forming slurry.

All of the components described were sized to represent the unit that would go on the mobile jet pump injector vehicle. Fabrication of the vehicle was not carried out in this program due to a lack of time, money, and an underground test site. It was believed to be far more important to solve the technical problems relating to the functions of the feeder-breaker and the jet injector than to attempt a premature combination of the components.

During the course of the test program, experience dictated that a number of changes to the system were required to achieve an operational jet pump injector subassembly delivering the required performance.

As a case in point, the original plans for the jet injector contemplated control of flow velocity either with water alone or with a coal/water slurry. The only flow measurement that could be made reliably was that of a water filled line without significant coal content to form a slurry. However, a substitute technique was developed.

The successive technique was control of water level in both the reserve water tank and the hopper of the jet injector. During the course of the experimental work, several improvements were made in the technique for maintaining a well-defined water level in the mixing hopper and the reserve tank. This was done despite incoming flow rates as high as 3000 GPM. Substantial baffling was required to achieve this result. The principle baffling technique consisted of introducing water as a sheet flow near the bottom of the water tanks involved. This technique tended to prevent entrainment of air and to prevent the localized effects of high flow rates that would have occurred if the same quantities of water were delivered using a simple inlet jet or orifice to supply the water required.

Overhead views of the jet pump injector subassembly installed in the facility are shown in Figure 4-73. In the view to the left, coal is being delivered to the hopper from the feed belt preparatory to injection into the slurry line. To the right, no coal is being delivered but the water nozzle is operating and maintaining back-pressure on the slurry line. The hydraulic jet is not visible since it is submerged below the water surface that can be seen in the figure. Main nozzle water flow enters the hopper by way of the major line shown in the foreground. Water from the reserve tank, used to assure submerged jet operation at all times, enters through the canted line. This water enters a series of peripheral, baffled conduits and is distributed in sheet form around the walls of the hopper in such a manner to minimize turbulence. The sensitivity of the water level control system requires that adequate attention be given to providing assurance of a tranquil water level. The entry of this overflow water into the hopper can be seen in Figure 4-74.

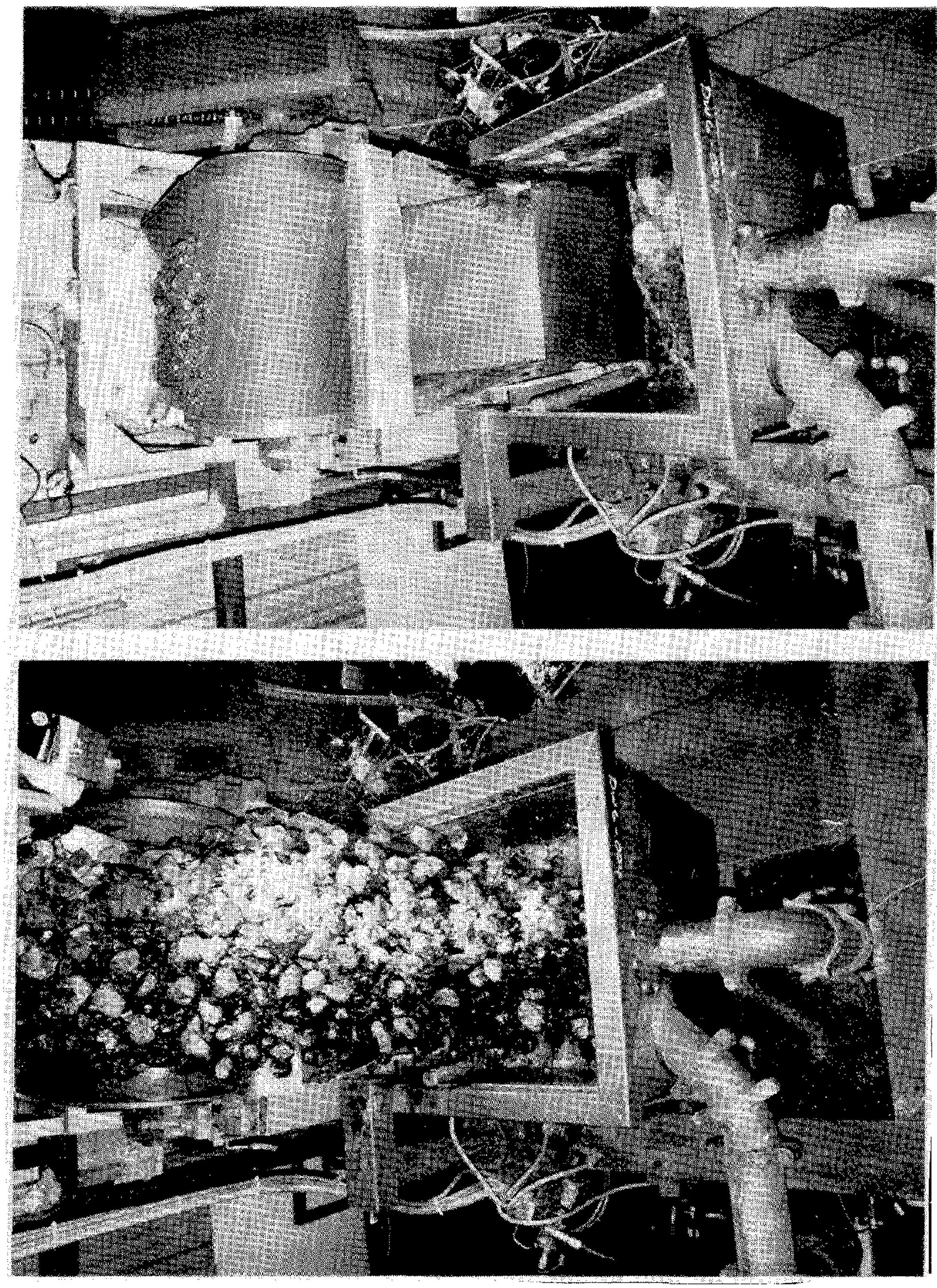
In Figure 4-75 the coal is seen from another view moving along the feed belt as it approached the hopper, followed by free fall from the belt into the hopper.

Finally, Figure 4-76 presents two views of the jet pump injector subassembly installed and in operation in the test system. In the top view, the coal is entering the hopper from the feed belt at top center. High pressure water supply to the jet pump nozzle is from the left. To the right is shown the water reserve tank. In the bottom view, a close up of the reserve tank is shown with its water supply lines and water level control valves.

4.4.3.3 Test Methods and Results

Eighteen (18) test runs were made on the jet pump injector subsystem in the slurry test loop described above. The primary purpose of these tests was to demonstrate reliable, satisfactory operation of the jet pump for the intended service. Specification performance levels of 6.4 TPM coal flow at a slurry weight concentration (at discharge from the concentrator) equal to or greater than 40% were to be demonstrated.

To initiate a test run the water pumping system was activated without coal flow. Water flow through the slurry test loop, including the jet pump injector, was continued



Without Coal Flow

With Coal Flow

Figure 4-73: Overhead Views of the Jet Pump Injector Subassembly During Operation

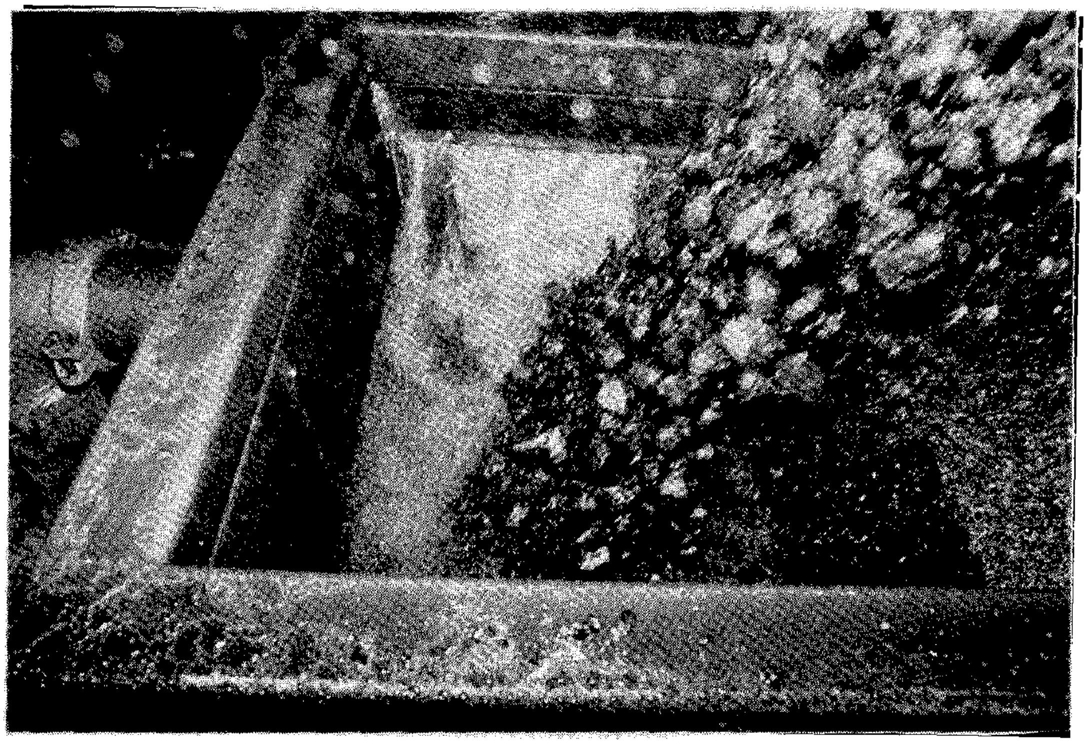


Figure 4-74: View Showing Entry of Overflow Water and Coal Into Hopper

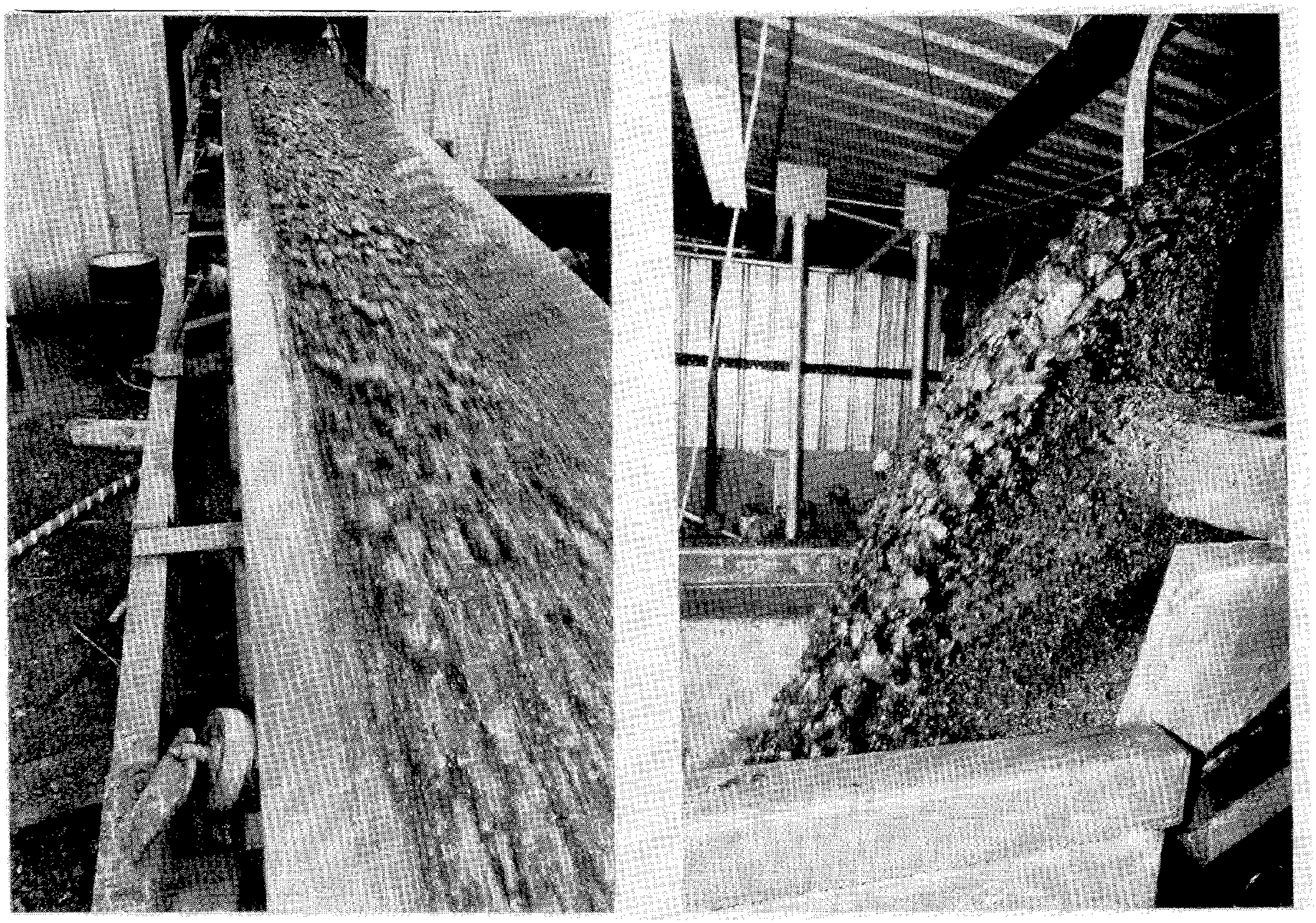
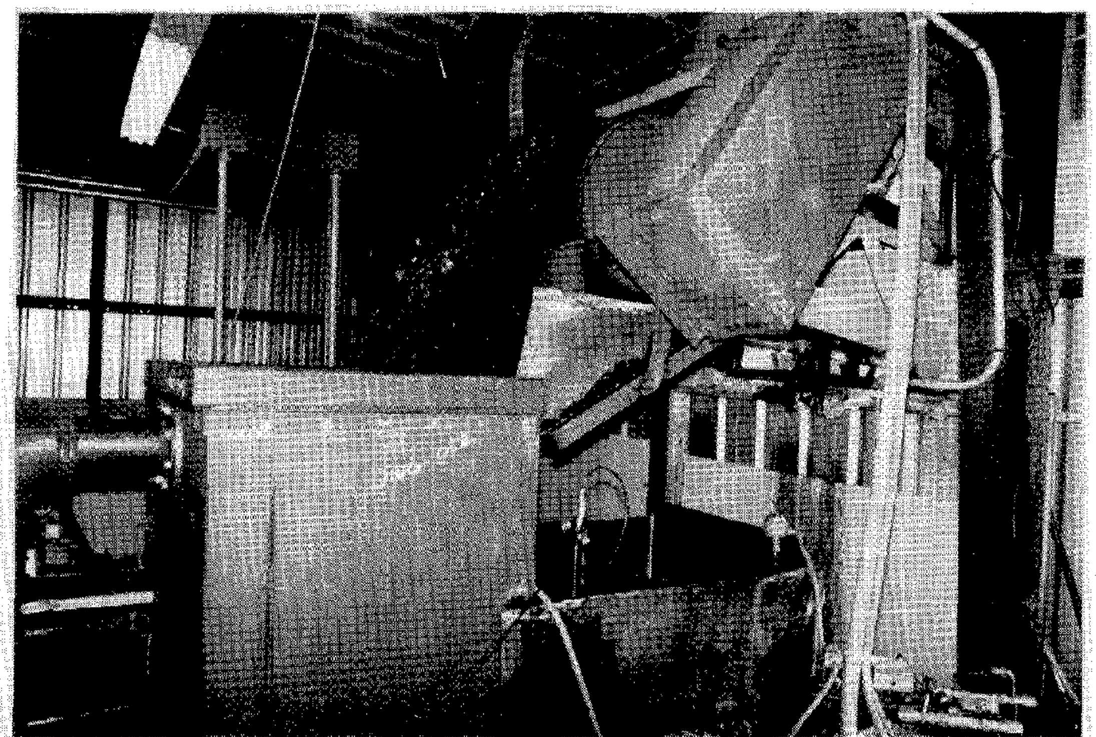


Figure 4-75: Coal on Feed Belt Followed by Entry into Hopper



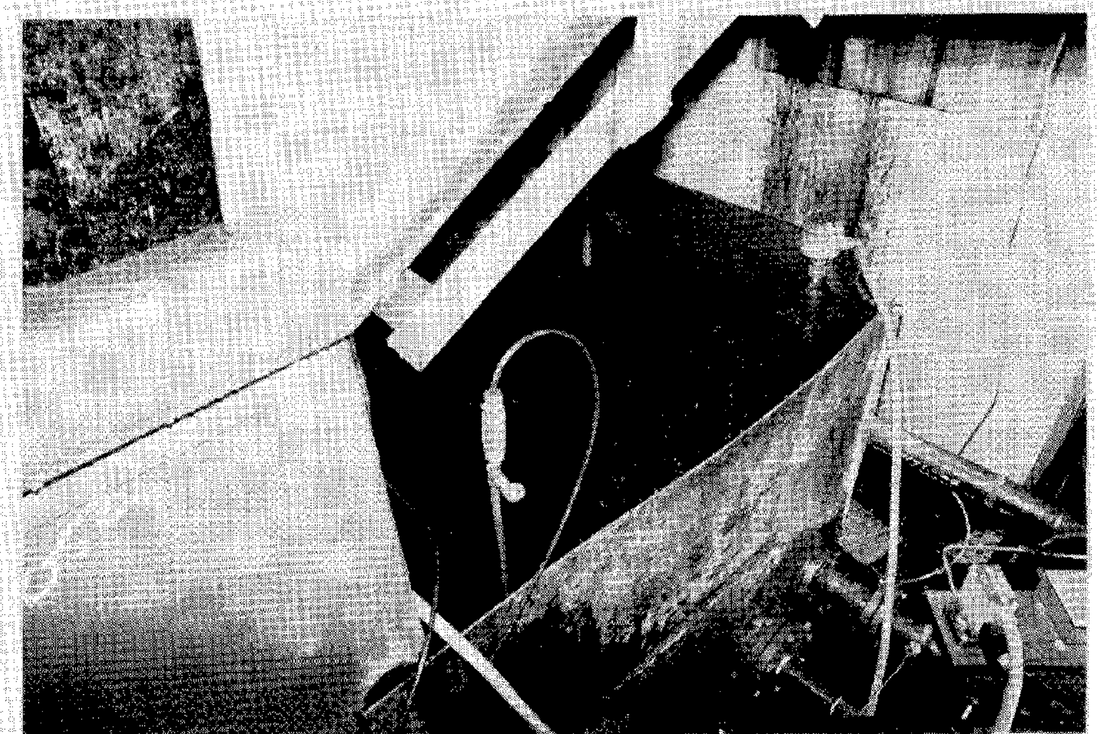


Figure 4-76: Overall Views of the Experimental Jet Pump Injector Subassembly Test Set-up

until steady state conditions were achieved throughout the system at the desired operating point. At that time the coal delivery system was initiated. Coal was then delivered continuously to the hopper at a pre-established rate followed by injection into the slurry line by the jet pump injector. All data were recorded after stabilization of instrumented operating conditions throughout the coal, slurry, and water systems. The coal flow rate was then incrementally increased to the next desired flow level and, following stabilization of flows and instrumentation, data at the new coal flow condition was recorded. This sequence of events was repeated until the point where the maximum flow of coal that could be delivered by the jet pump injector for the set conditions was reached. Upon attainment of the maximum coal flow condition, the test was considered complete. In certain instances testing at a given set of operating conditions was repeated to confirm earlier data or extend operating range.

Performance characteristics of the jet pump were profiled during these tests for two nozzle diameters and seven entrainment lengths. Table 4-7 shows these nine values of inlet parameters as well as the individual test run corresponding to each combination. The actual data from each of the 18 test runs are presented in Appendix C organized in the manner indicated by Table 4-7. Appendix C presents tabulated and/or plotted values of water flow and pressures, slurry concentrations and speeds, and system efficiency, etc. against coal flow rates for each test run. In addition, cross-plots of slurry concentration alone as a function of coal flow are also presented.

For the purpose of presenting a general overview of the test data, a variety of summary plots are presented in this section. Figures 4-77 through 4-81 show the affect of slurry concentration of coal by weight, Cw, on jet pump efficiency. Figures 4-77 and 4-78 show this relationship for a 56 mm diameter nozzle and entrainment lengths of 156, 230 mm respectively. Figures 4-79 through 4-81 show the same affect for the 63 mm diameter nozzle and entrainment lengths of 174, 250, and 285 mm respectively. Within the accuracy prescribed by the data scatter, nozzle diameter appears to have no affect on the concentration-efficiency relationship. On average, a tripling of the coal slurry concentration results in an increase in jet pump efficiency of up to 40%. The efficiency increase reduces as entrainment length is increased.

The affect of nozzle pressure on discharge pressure is illustrated in Figures 4-82 through 4-86. Figures 4-82 and 4-83 show that for a 56 mm diameter nozzle, a given change in discharge pressure requires a 5 fold change in nozzle

TEST DATA SUMMARY

Nozzle Diameter (mm)	Entrainment Length (mm)	Run Number (#)
56	156	2 13 15 16
	230	1 5 6 14 18
	265	17
	280	3 4
63	174	9 10
	250	7 8
	285	11 12

Table 4-7: Test Data Summary

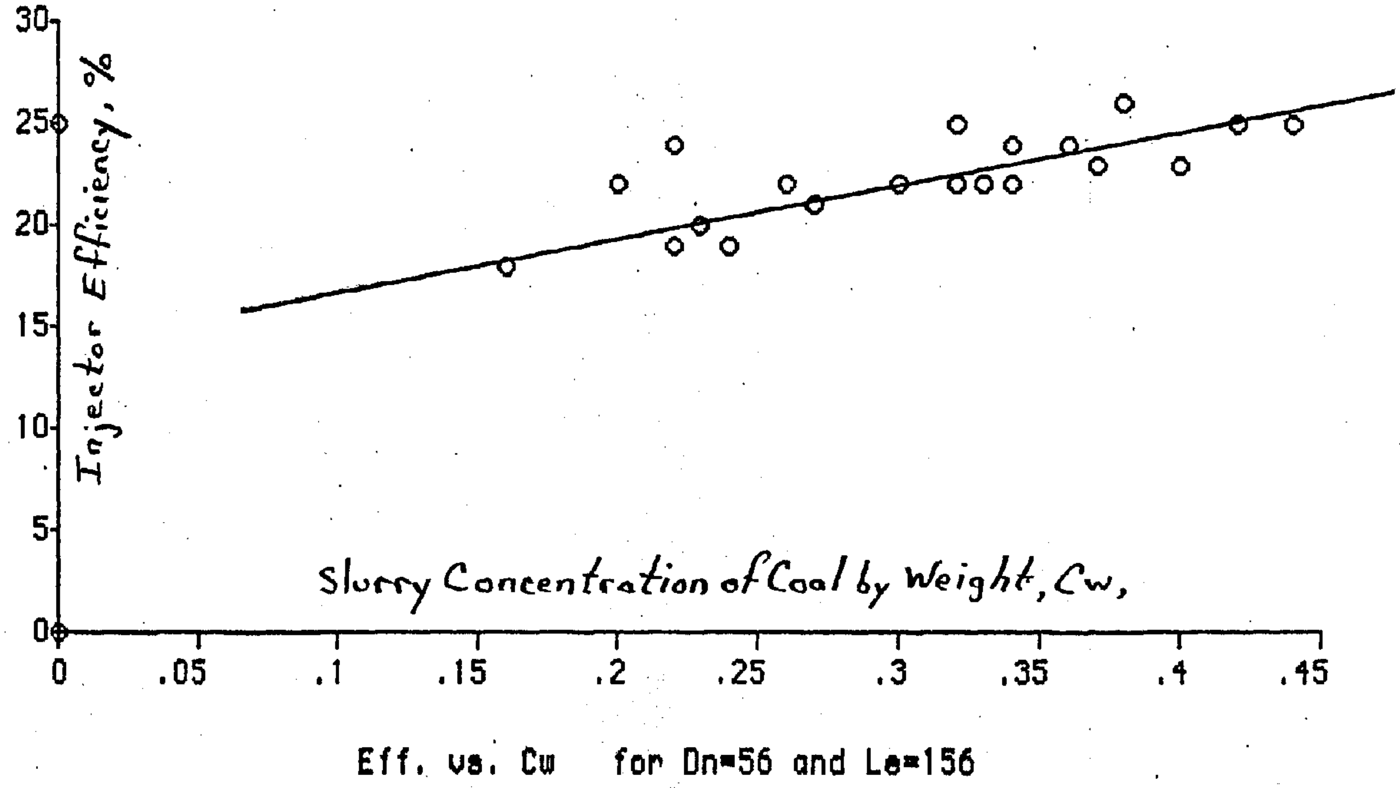
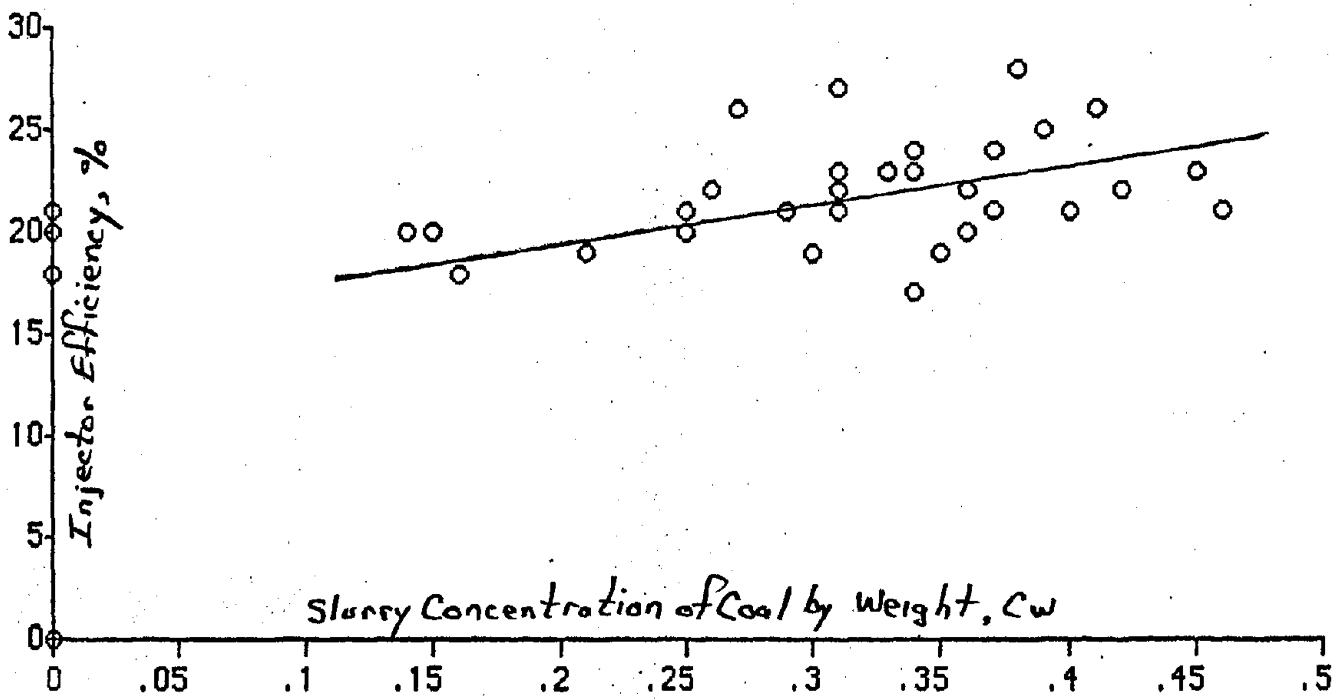
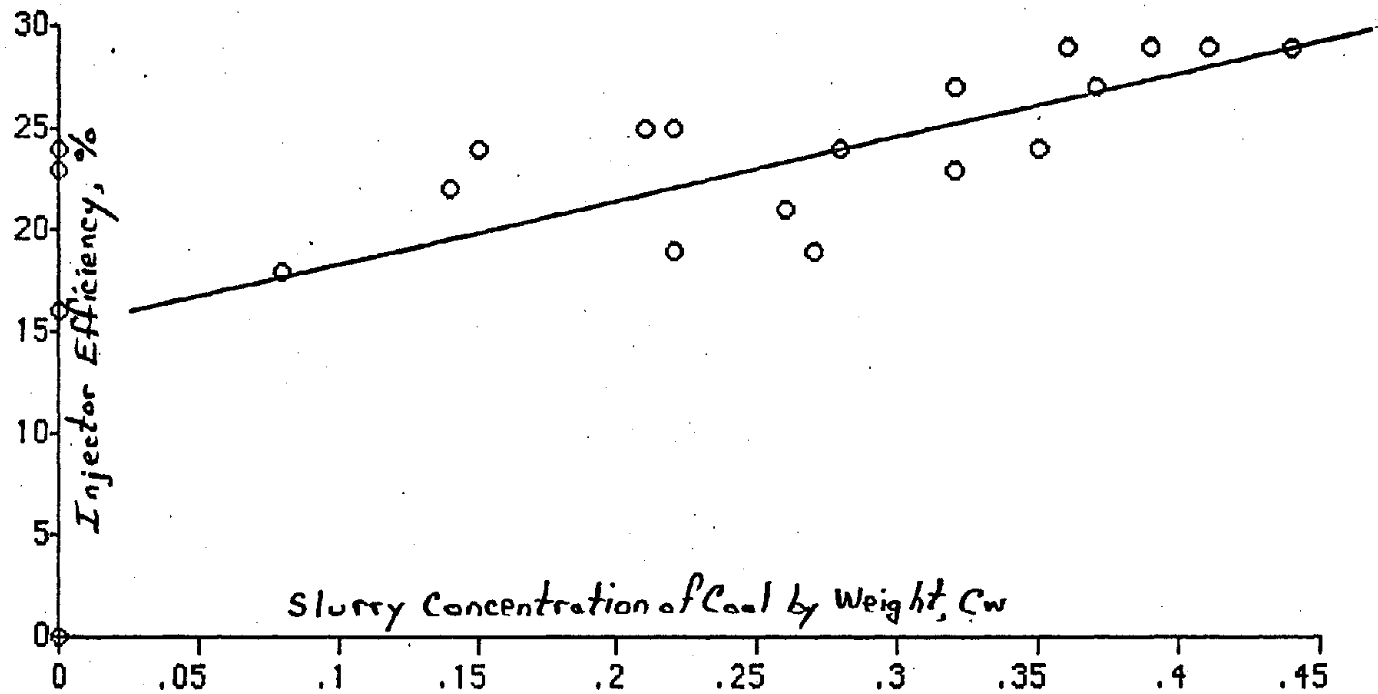


Figure 4-77: Effect of Slurry Concentration on Injector Efficiency ($\emptyset_h$ = 56 mm, L_E = 156 mm)



Concentration Effect on Efficiency

Figure 4-78: Effect of Slurry Concentration on Injector Efficiency ($\varnothing_n$ = 56 mm, L_E = 230 mm)



Slurry Concentration Effect on Efficiency

Figure 4-79: Effect of Slurry Concentration on Injector Efficiency.
($\emptyset_n$ = 63 mm, L_E = 174 mm)

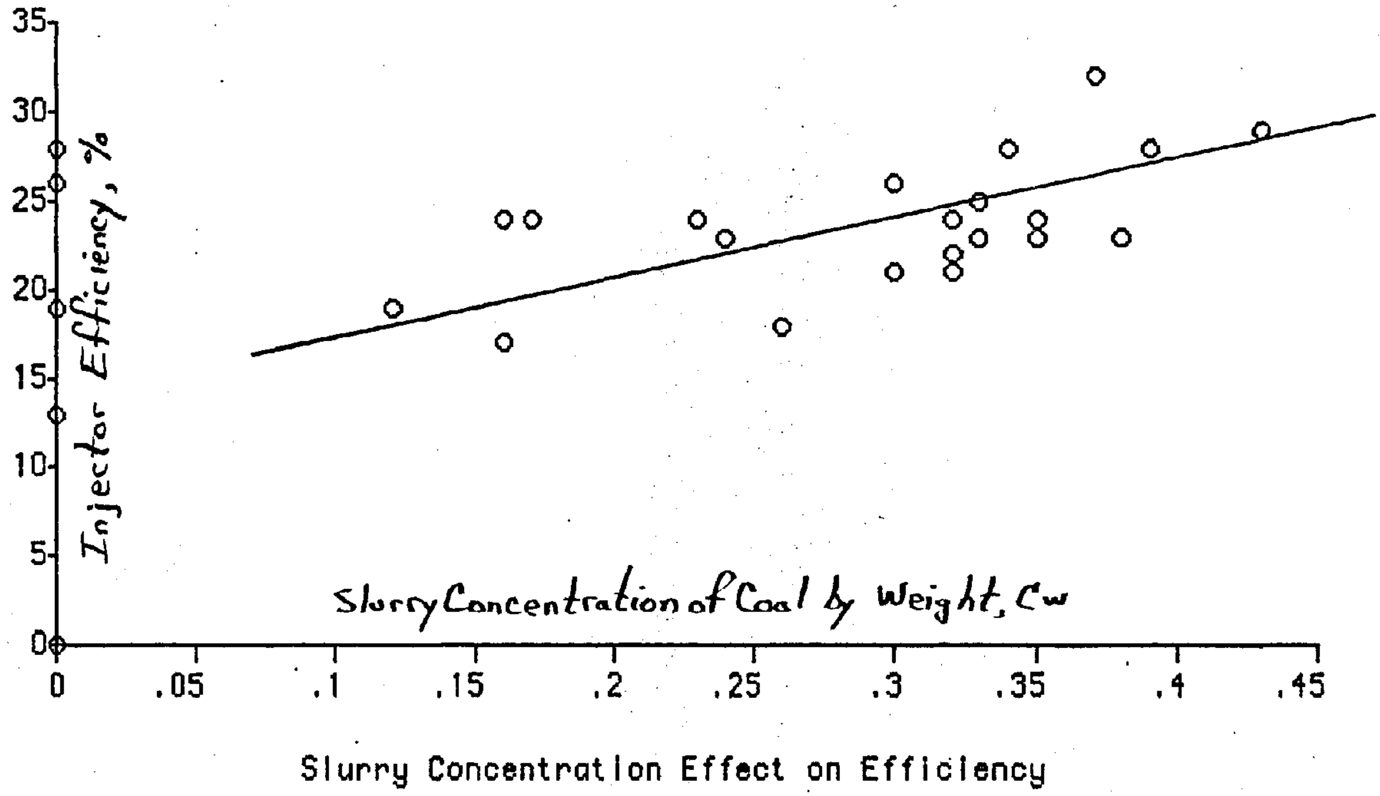
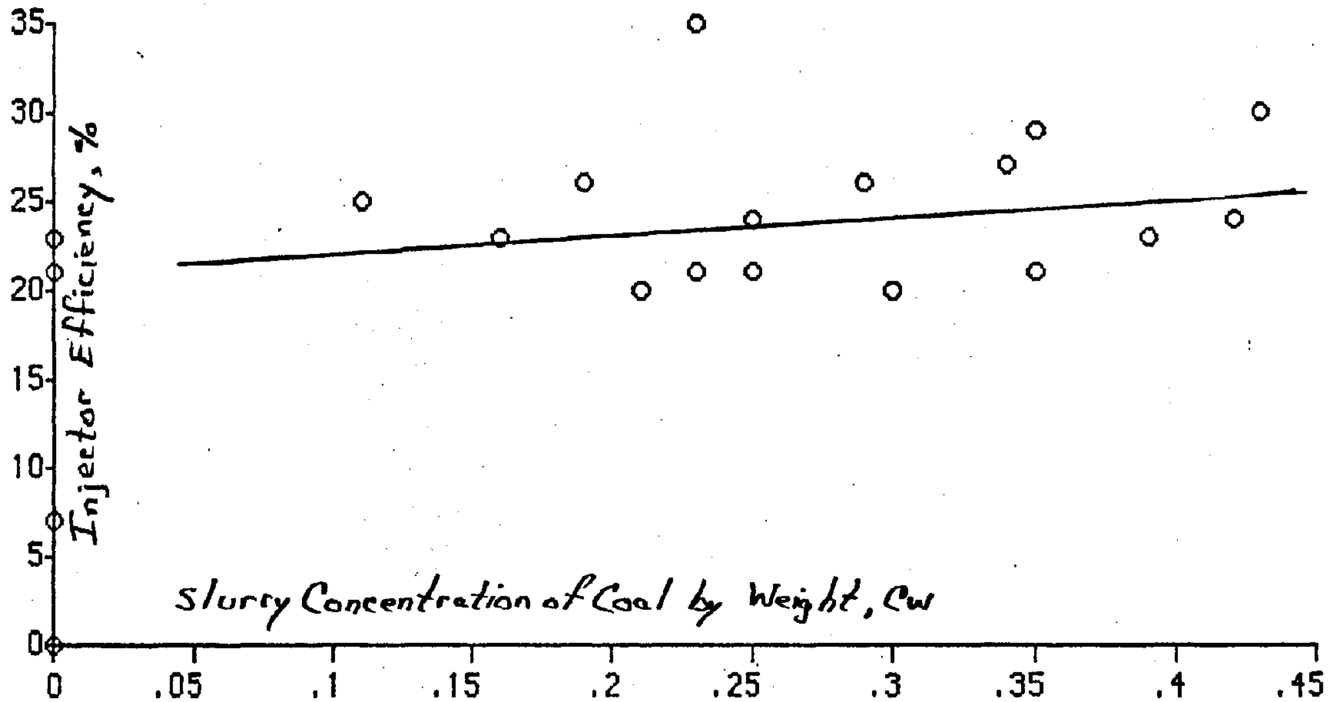


Figure 4-80: Effect of Slurry Concentration on Injector Efficiency.
($\emptyset_n$ = 63 mm, L_E = 250 mm)



Slurry Concentration Effect on Efficiency

Figure 4-81: Effect of Slurry Concentration on Injector Efficiency.
($\emptyset_n$ = 63 mm, L_E = 285 mm)

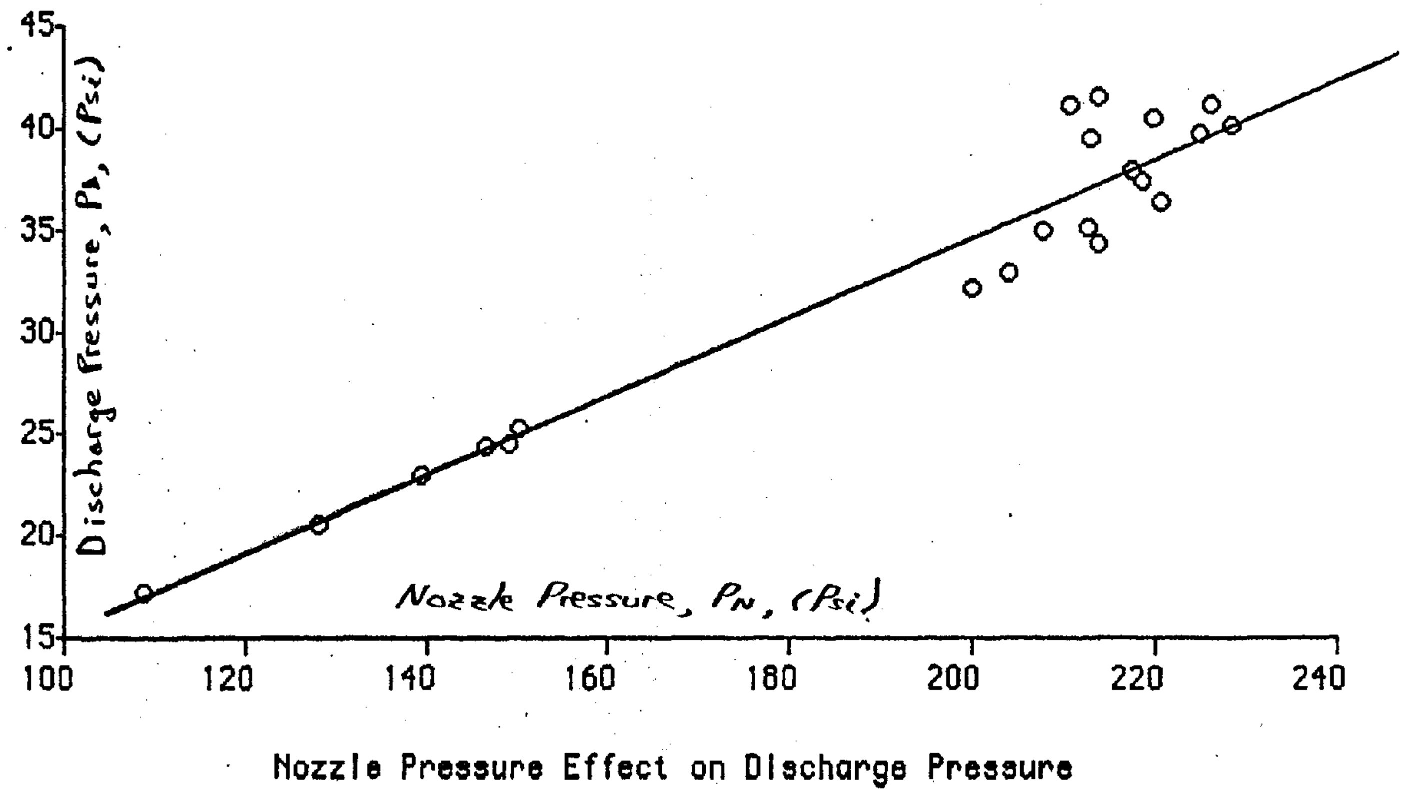
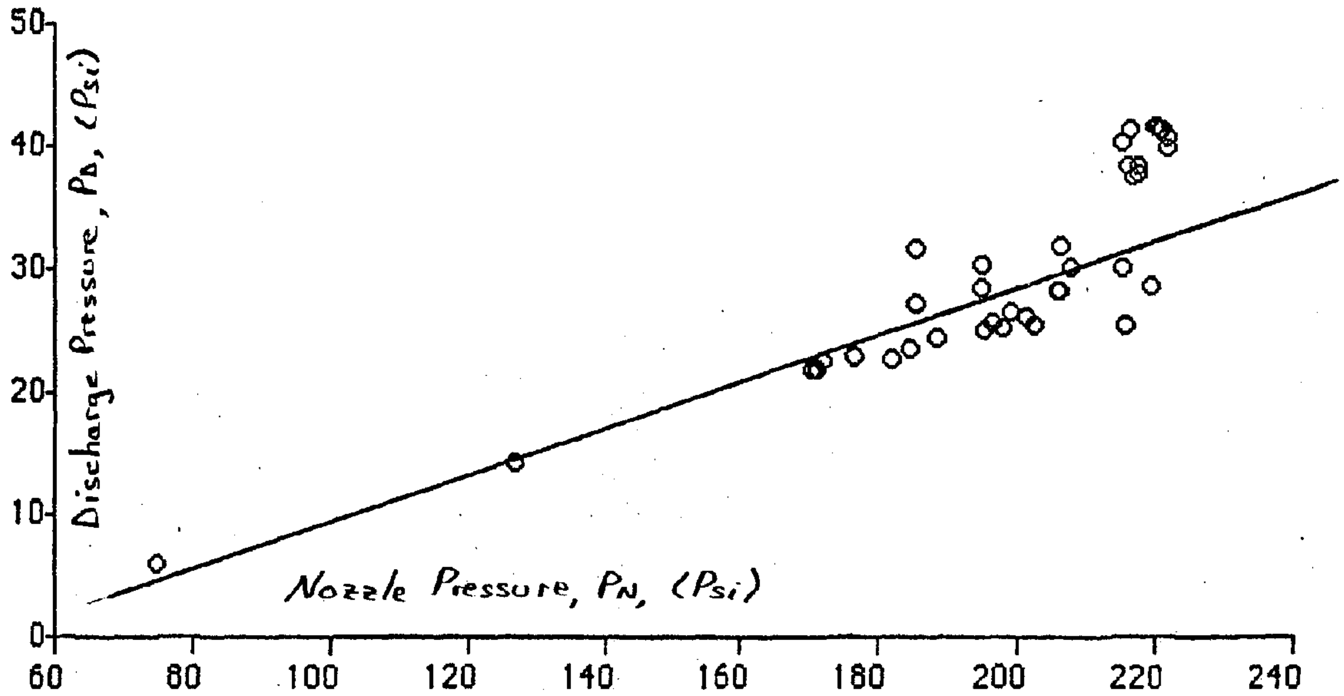
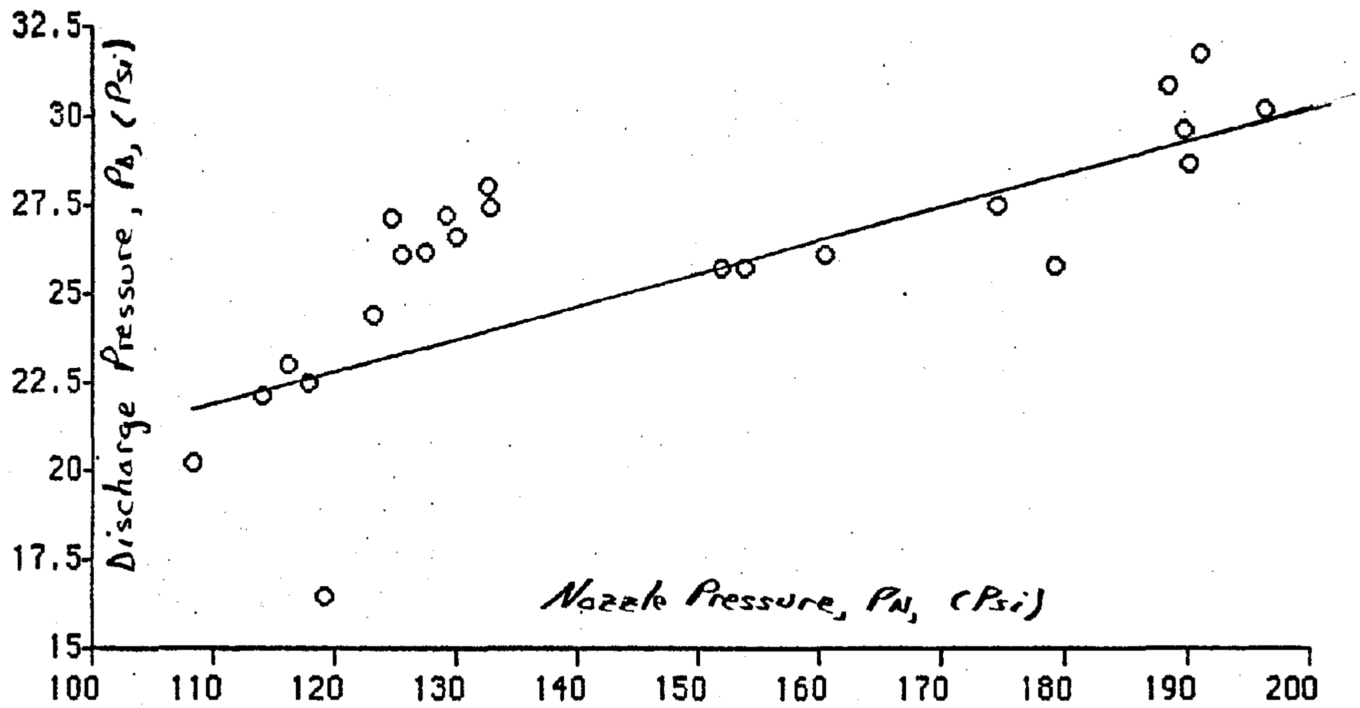


Figure 4-82: Nozzle Pressure Effect on Discharge Pressure. ($\varnothing_n$ = 56 mm, L_E = 156 mm)



Nozzle Pressure Effect on Discharge Pressure

Figure 4-83: Nozzle Pressure Effect on Discharge Pressure.
($\emptyset_n$ = 56 mm, L_E = 230 mm)



Nozzle Pressure Effect on Discharge Pressure

Figure 4-84: Nozzle Pressure Effect on Discharge Pressure.
($\emptyset_n$ = 63 mm, L_E = 174 mm)

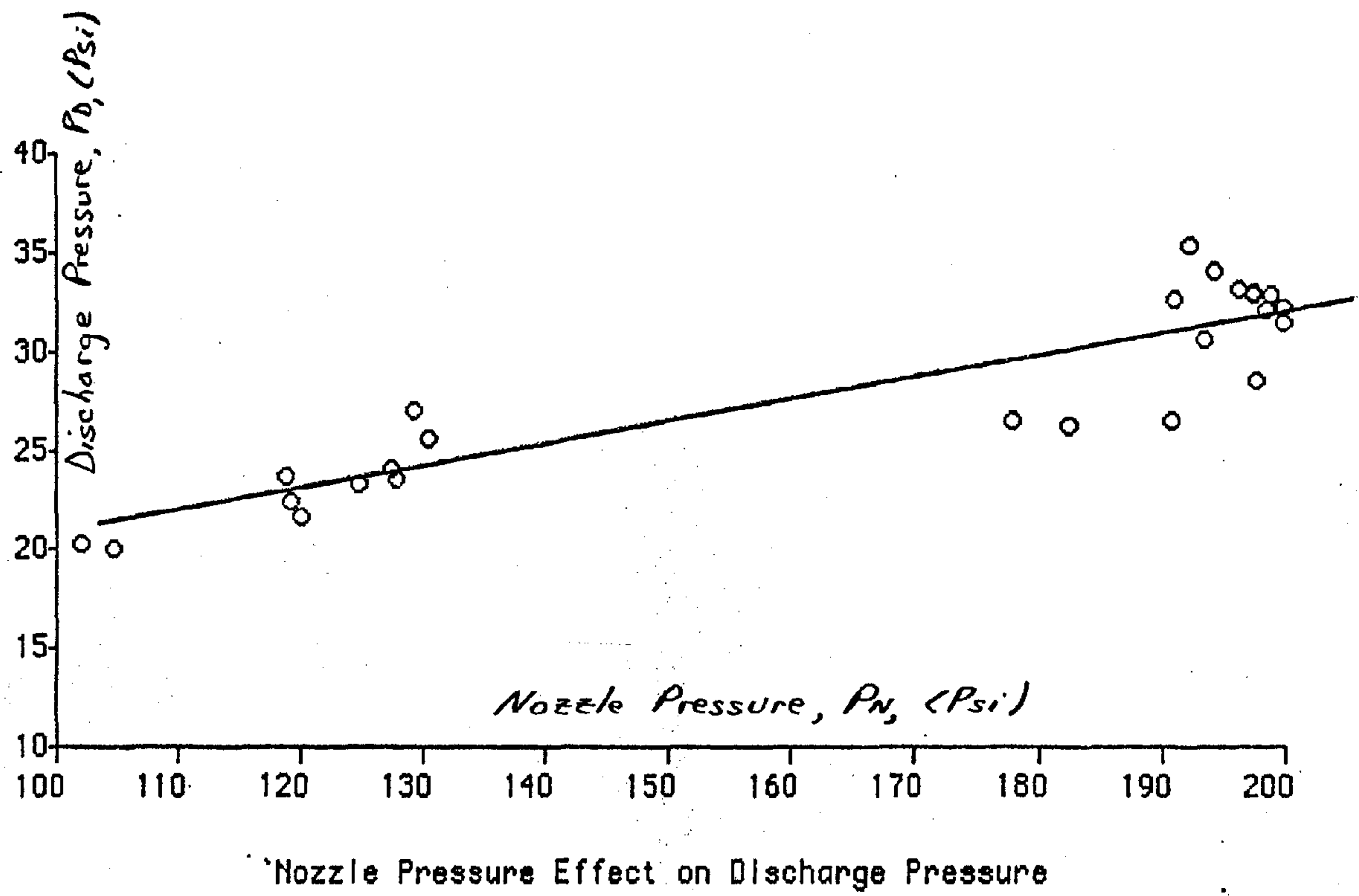


Figure 4-85: Nozzle Pressure Effect on Discharge Pressure.
($\emptyset_n$ = 63 mm, L_E = 250 mm)

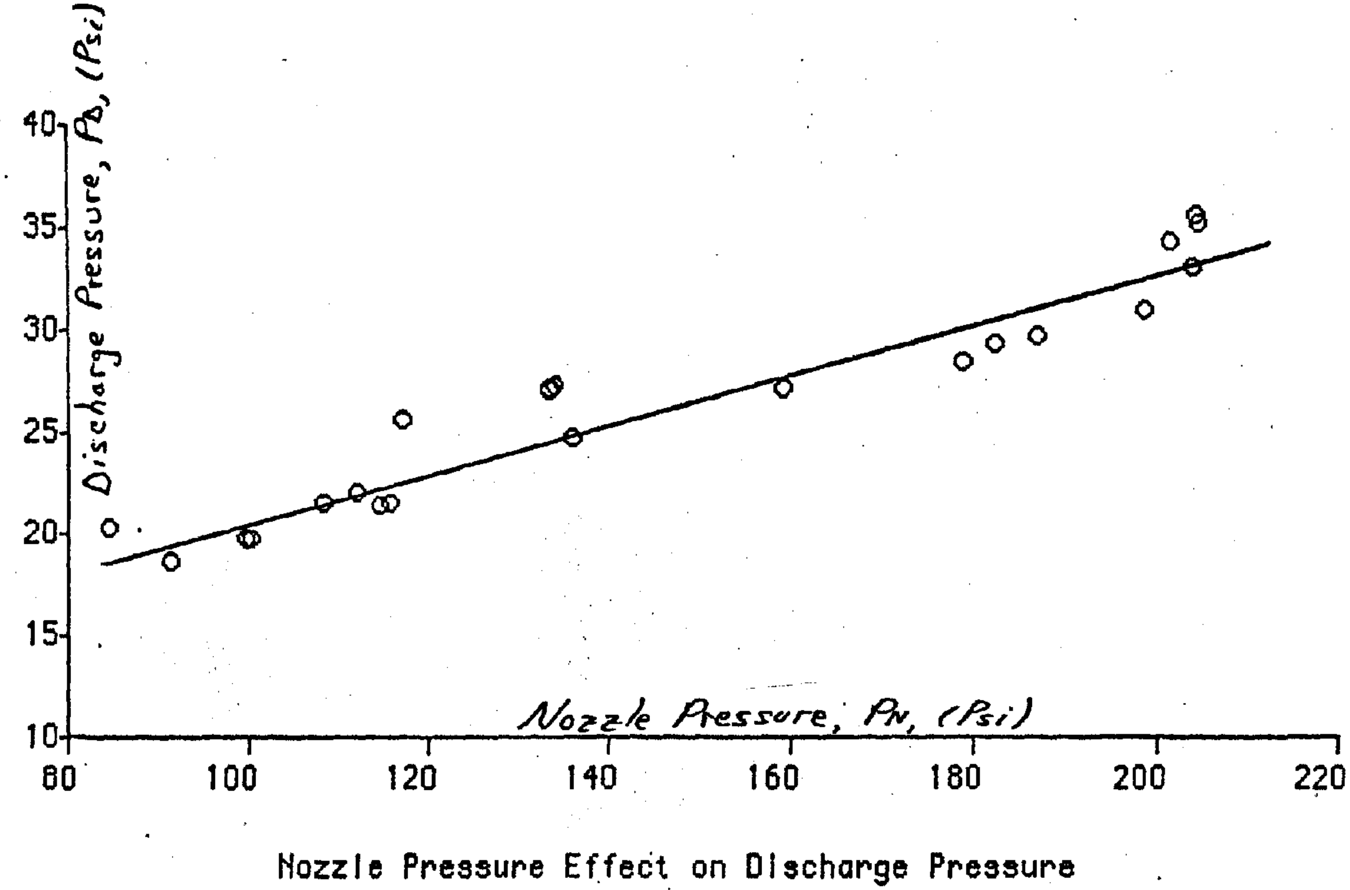


Figure 4-86: Nozzle Pressure Effect on Discharge Pressure.
($\varnothing_n$ = 63 mm, L_E = 285 mm)

pressure. Figures 4-84 through 4-86 show that for a 63 mm diameter nozzle, a given change in discharge pressure requires a ten fold change in nozzle pressure. For each nozzle, the effect of entrainment length is minimal, namely a 60 to 80% increase in entrainment length results in only a 10% increase in the affect of nozzle pressure on discharge pressure.

The one factor prevailing throughout the entire test program was the ability to achieve the design coal flow rate of 6.4 TPM. Equally important was the consistancy of the slurry concentration at that flow rate regardless of the nozzle diameter or entrainment length utilized. The mean slurry concentration of coal by weight at the 6.4 TPM design point was .42 with a standard deviation of less than 10%.

4.4.3.4 Coal Degradation Test

To provide information on the degradation of the coal in the slurry test loop as a function of passes through the system, a specific series of tests were performed in the Foster-Miller Associates 427-foot test loop.

A test was started by filling the hopper with fresh 3-inch top-sized coal. The coal was run through the test system and coal samples taken at different time intervals. Sieve analyses were performed on the coal samples to determine the amount of degradation caused by the test system. This test did not determine the location or component within the test system where the degradation occurred. Rather, it showed that degradation did occur in the system and to what degree. Possible breakage areas included:

- Pipe Line Discontinuities
- Header Box (or Dewatering Screen)
- Dewatering Screen Itself
- Dewatering Screen to Coal Bin Transfer
- Coal Bin to Weigh Belt Transfer
- Weigh Belt to Jet Pump Injector Hopper
- Jet Pump

A detailed discussion of this test series, the sieve analyses performed and an analysis of the results can be found in Appendix F.

4.4.4 Testing of Full Scale Secondary Breaker Subassembly

4.4.4.1 Test Equipment Planning

As a part of the mobile jet pump injector vehicle design study work, it was determined that a two-stage feeder breaker system would be appropriate for this application. For the first, or primary, stage (Item 2, Figure 4-63) an S&S Corporation production Type SE breaker was selected to handle the sizing of ROM material (coal + rock) from 14" to 7" x 0". The design of this breaker was considered to be state-of-the-art and accordingly it was not deemed to be necessary to develop such a breaker under this contract.

However, design of the secondary breaker (Item 4, Figure 4-63) was not so straight forward since space allocations on-board the vehicle substantially constrained the permissible dimensions of the breaker. No commercial unit was identified, suitable for reducing the material discharged from the primary breaker (7" max.) to the 3" x 0" material required for injection into the 8" slurry pipeline by way of the jet pump injector.

Accordingly, the design, fabrication and development testing of an acceptable secondary breaker design was undertaken. At the outset it appeared possible to meet all performance requirements through the use of a 36" wide, double-roll breaker concept. This unit would be capable of processing ROM material (coal + shale) at a higher-than-required rate of 6.4 TPM.

4.4.4.2 Description of Test Apparatus

The test apparatus was a double roll breaker design which incorporated two rollers with picks mounted in each. The rollers rotated towards each other so that the picks converged and meshed. It was the convergence of the picks, called the bite, that performed the breaking. The spacing was such that a piece of material larger than three inches would get caught in the bite and be broken. If the material was less than three inches, it would get caught in the bite pockets, formed between picks, and merely pushed through without further breakage.

Each roller was driven by a hydraulic motor and the two rollers were coupled together by a double row chain. The motors were gear motors with each having a displacement rate of about 24 gpm. The chain was necessary to keep the drums properly timed. If the timing were to change, the spacing of the picks was such that interference of the picks would occur between drums.

For this test the rolls were designed so that two different pick types could be evaluated simultaneously. The first type was called a cutter bit or attack bit. This type of pick is conical in shape and has a pointed carbide insert in the tip. The second type of pick was called a drag bit. This type of pick uses a facial attack of the material compared to the point attack of the cutter bits. The drag bit made it possible to use a larger bite.

The rolls of the test apparatus were divided into two halves - one half with drag bits and one half with cutter bits. A divider plate was placed down the center of the funnel-shaped hopper to isolate each side for testing. The length of the breaker rolls in each pick zone was 12-13 inches. An actual, full scale breaker with one type of pick would measure about 36 inches to match the width of a conveyor.

The secondary breaker subassembly, with a feed hopper fabricated of metal plate, was mounted on the end of an SE-type feeder/breaker as shown in Figure 4-87 (top). A view looking into the breaker receiving hopper below the conveyor in the same figure (bottom) shows material (in this case, shale) entering the attack-pick side of the double roll breaker. The drag-pick section is visible in the left side of this view. Additional plan views of the double roll secondary breaker, looking directly into the bite zone, are shown in Figure 4-88. In the bottom view the open areas that permit properly sized material to pass through without further breakage can be seen.

4.4.4.3 Test Methods and Results

Testing of the experimental secondary breaker assembly was conducted at the Engineering Center of the S&S Corporation, a subsidiary of Ingersoll-Rand Company.

To simulate typical ROM material, a shale from a nearby quarry was used as the feed stock in the development testing of the experimental breaker. This material is used by the S&S Corporation to simulate coal in all test operations.

Under normal operation, the feed conveyor was able to deliver a five-inch thick bed of rock at a rate of 30 fpm or 1.5 ton per minute. To achieve the higher volume of rock required for some of the tests, the conveyor was loaded by hand to a depth that would yield selected throughput. The actual throughput of the breaker was measured by running the breaker for a specific amount of time, then weighing the amount of material that passed through. A special diverter plate moved the material from one side of the conveyor to the other so that most of the material coming up the conveyor would be directed to only one type of pick. About 80% of the

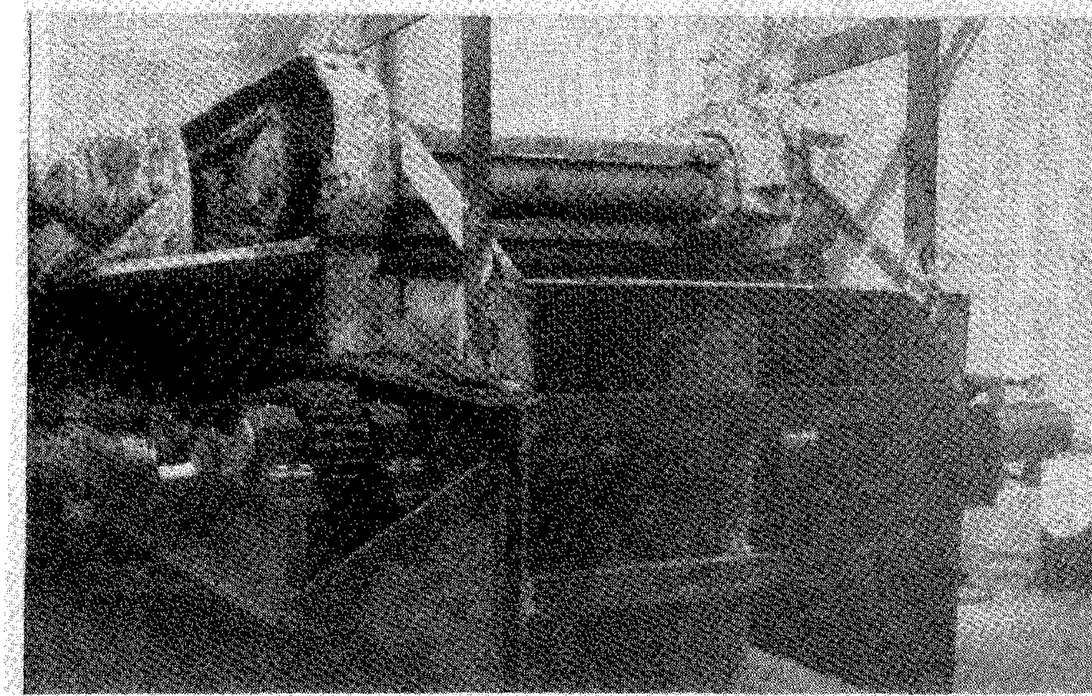


Figure 4-87: Overall Views of the Test Secondary Breaker and Feed Hopper Mounted on SE-Type Feeder Breaker

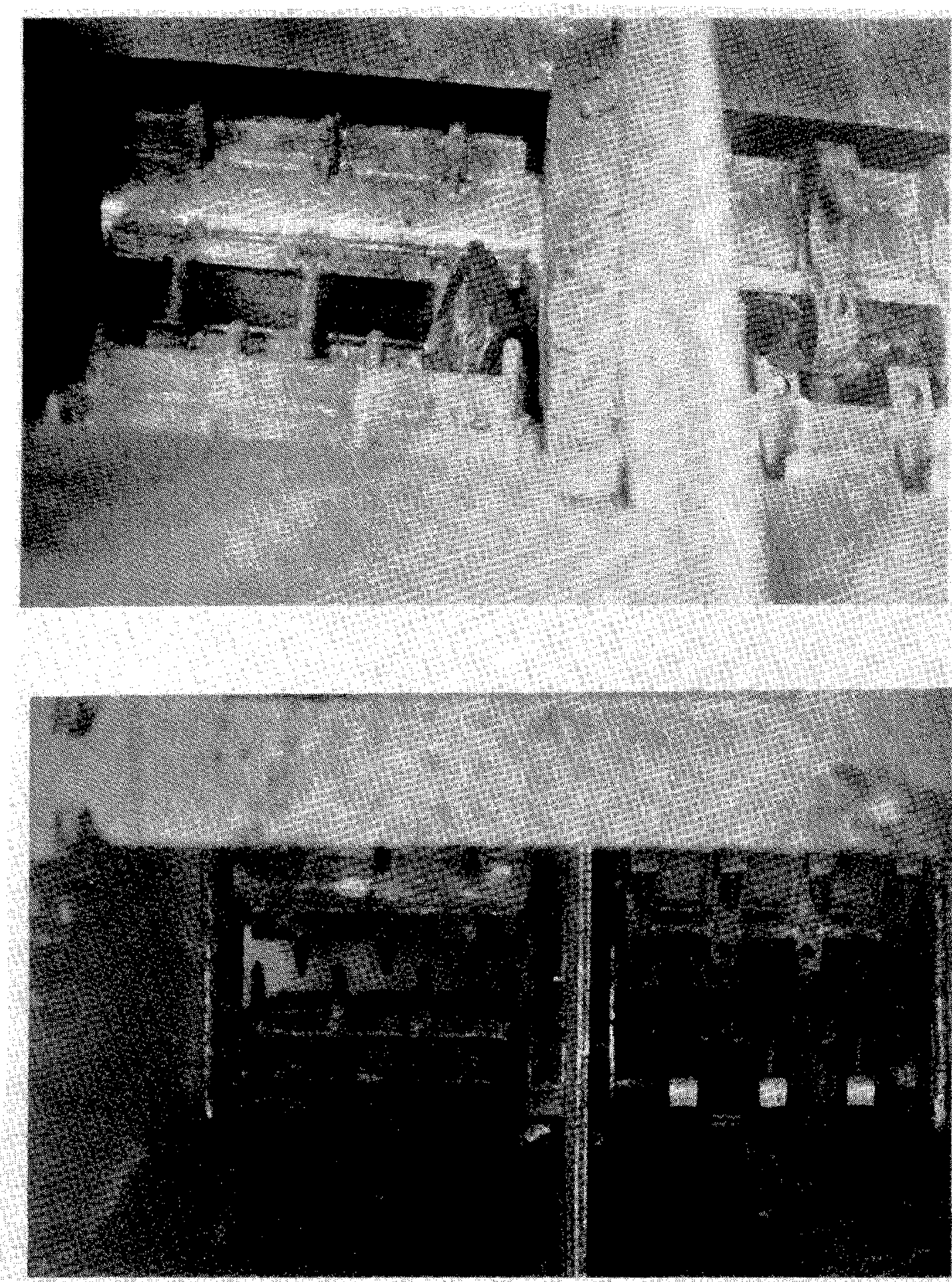


Figure 4-88: Plan Views of Secondary Breaker Double Rolls Looking into Bite

material would be pushed to one side of the conveyor while the other 20%, mostly fines, slid under the diverter, between flights on the conveyor.

From test operating experience it was found that the breaker readily broke incoming shale fragments so long as material was continuously fed from the conveyor. When flow was terminated a single piece positioned at the bite would tend to bounce about in the hopper without being ingested by the rolls.

The testing of this initial concept for a secondary breaker showed that a double roll, small, compact unit yielding a 6.4 TPM throughput of 3" x 0" could be successfully designed. From the tests conducted it appeared that a 36" double roll unit requiring approximately 50 HP would be required. Further, the tests indicated that either the drag type or attack-type bit could be used in the final design.

For additional information on the secondary breaker development programs and results, the reader is referred to Appendix D.

4.4.5 Preparation for Field Demonstration in Coal Preparation Plant

Consideration of many alternative arrangements for testing the complete vehicle led to the conclusion that it would be very difficult and expensive to have a complete demonstration of the equipment without initially carrying out some jet pump evaluation work at a coal preparation plant. The primary consideration was that no other facility would provide for single pass processing of ROM coal over an extended period of time involving several thousand tons of coal.

Acting on this perceived need for participation of a preparation plant, several coal companies that might be able to carry out a program of this kind were approached.

The Pittston Coal Company expressed a strong interest in carrying out this work. Accordingly, planning efforts were undertaken with the Pittston Company to accomplish the engineering planning that would define all of the facilities neded for the subject test.

A plan of this type was completed by the Pittston Company. According to this plan the test facilities were to have been installed at the Moss-III preparation plant. The Pittston planning effort is described below.

The company agreed to provide a test facility at its Moss III preparation plant and this arrangement formed a part of the Phase III plan.

Upon start of the Phase III work a meeting was held to define the test program and to clarify the specific features needed to accommodate the test work at the preparation plant.

In normal use of the Moss III plant, run-of-mine coal is received by rail and truck. Coal is dumped into an underground hopper that feeds the main conveyor belt. Magnetic separators remove tramp iron. Conveyor loading is controlled by varying the speed (as measured by current to the conveyor drive motor). Coal is conveyed to the top of the preparation plant where the flow is divided into four roughly equal parts. It then proceeds through various washing, crushing and beneficiation processes.

For coal to be provided to the jet pump injector, a conveyor already in place at the truck unloading area was to be employed. A magnetic separator would remove tramp iron. Coal input would be regulated by the use of a belt speed control and, if needed, a gate controlling the height and width of the coal feed stream on the conveyor.

The jet injector would be used to feed all or part of the coal to one of the four processing lines. The normal coal processing rate was 400 tons per hour for each of the four sections. Coal not supplied by the jet injector could be diverted from the main conveyor in order to maintain the output of the plant. In order for the main conveyor to be adjusted correctly, a solids flow indication from the jet injector was to be provided to the preparation plant control center.

An emergency shut-down control of the slurry line located in the prep plant control center was to be used so that slurry feed would not continue when equipment receiving the slurry was stopped. In order to leave the slurry line clear of solids, provisions were also to be made to divert the incoming slurry to a holding tank already available in the preparation plant. Flushing of the slurry line with water would continue for as much as three minutes before flow from the jet injector was stopped. According to this plan the first step at the continuation of operation would be to pump out the holding tank and then restart the jet injector.

In order to make the plan fit smoothly into preparation plant operation, it would be necessary for the jet injector to deliver a relatively constant amount of water regardless of the amount of solids delivered. This water flow was to be set at 2,000 GPM plus or minus ten percent. Maintaining constant water flow would, with coal delivered, define the velocity in the slurry line. For example, an increase water flow would not be permitted in order to maintain constant line velocity when coal input decreased.

Although this method of operation would not necessarily be optimum for slurry line testing, it was considered to be a necessary feature to permit coal to be delivered directly to the preparation plant and, thus, to implement one of the important features of the plan which was to provide unlimited, single-pass use of run-of-mine coal in testing the jet injector. This feature of using single-pass, run-of-mine coal at a preparation plant could not be duplicated in any other practical way without substantial additional program cost.

Unfortunately, it was not possible to carry out these plans. Market pressures required that the staff and resources of the Moss-III plant be directed to the basic requirements of their operation.

Withdrawal of the Pittston Company as a potential subcontractor for testing at a preparation plant prompted a search for a replacement. Discussions were held with approximately 15 other major coal companies but it was not possible to find a company willing to undertake the requested work. A number of companies did express an interest, however, and it is believed a willing partner could be found if the program were to be continued at some future date. It is considered, however, that a number of valuable steps were taken to solve the problem of working with a preparation plant.

Although efforts to find another coal operator to participate in this development work was unsuccessful, it is fully expected that coarse coal slurry transport will eventually be carried to a realistic full-scale test and adoption by the coal industry.

5.0 APPLICATIONS

5.1 Potential Applications for the Jet Pump Injector

Potential applications of the Jet Pump Injector will tend to make use of one or more of the desirable qualities of this device. It has been demonstrated that the Jet Pump Injector can control the process of mixing properly sized coal with water to form slurries having concentrations up to 45% by weight. The Jet Pump Injector system can maintain a steady flow of injected coal and water, acting as the first stage for a booster pump that is not well adapted to slurry formation at medium-high concentrations. If the coal inlet flow is continuous, the Jet Injector can act to supply concentrated slurry on a continuous basis.

In circumstances where the coal input is interrupted, or reduced, one of the important virtues of the Jet Pump Injector system immediately becomes apparent. The Jet Pump Injector system can maintain flow in the slurry line in a positive manner. In this particular circumstance, a "cross over" can be substituted immediately for the jet nozzle with the result that the output of the primary water pump will be introduced into the slurry line at the top pressure of 200 psi. Under this condition the flow will be maintained in the slurry line. It will not, however, maintain constant velocity at the same velocity used for pumping slurry. This is best understood if it is realized that the applications for the Jet Injector may include short-haul transport of coarse coal in cases such as moving the coal from the mine face to a preparation plant a few miles away. This situation is quite different from long-haul slurry lines that usually carry very finely divided solids material and progress through the line at velocities very close to the saltation velocity. For such a case, it is imperative to maintain the flow velocity in the line or to shut down the line in a highly controlled manner so that the chances for clogging are minimized.

Again, contrasting this situation with that of the Jet Injector, interruption of solids input will lead to maintaining the flow of solids with a high margin of pressure availability and with the result that, if solids flow is not introduced into the input of the Jet Injector, the result will be clearing the slurry line with water.

In the tests that have been conducted with the Jet Injector, it has been possible to start and stop the solids flow within a very few seconds and yet hold the level constant in the Jet Pump Injector sump.

Examples of potential uses for the Jet Injector have been furnished by coal company personnel concerned with the movement of coal from a mine face to a preparation plant. One very positive case cited was for a slurry line that could be built that would be approximately three miles long, going quite directly from the mine mouth "cross country" to the preparation plant. The conventional alternative would entail construction of a spur rail line some thirty miles long. Unfortunately, this slurry line was not built. Until a number of such installations have been made and proven to be easy to operate and maintain, operators will believe it to be too expensive a risk to explore this new possibility.

Another rather different situation has been cited in which a slurry line has been seriously considered and may be used if the mine in question is opened and put into production. This slurry line installation may be understood with reference to Figure 5-1. In this figure the Jet Injector is used as a first stage to accomplish mixing of coal and water followed by the delivery of a coal concentration in the resulting slurry of approximately 40% by weight. This concentration level is typical of the testing that has already been done with the Jet Injector. The tests conducted to date do not necessarily limit the flow rates or concentrations that may be achieved as this technology is further developed.

Referring to the figure, the Jet Injector output is passed through a centrifugal concentrator that delivers slurry to the inlet stage of a lockhopper type feeder. This feeder concept makes use of a compartmented rotating drum whose segments are alternatively filled with the input material or emptied of the material that has been placed in a segment of the lockhopper by passing high pressure water through that section of the lockhopper. A very attractive feature of this type of feeder is that it is less susceptible to wear. The high pressure water pumps that supply essentially all of the energy used in this system are not exposed to the solids materials being moved and consequently may be designed for optimum performance without the penalty of wear on impeller type members.

Following the diagram shown in Figure 5-1, it can be seen that the lockhopper can be presented with a slurry input at its inlet port at all times. The cyclical nature of having first one and then another portion of the lockhopper aligned to receive coal may make it desirable to use a desurger between the Jet Pump Injector concentrator and the lockhopper feeder, or to utilize a lockhopper feeder with staggered ports to prevent surges.

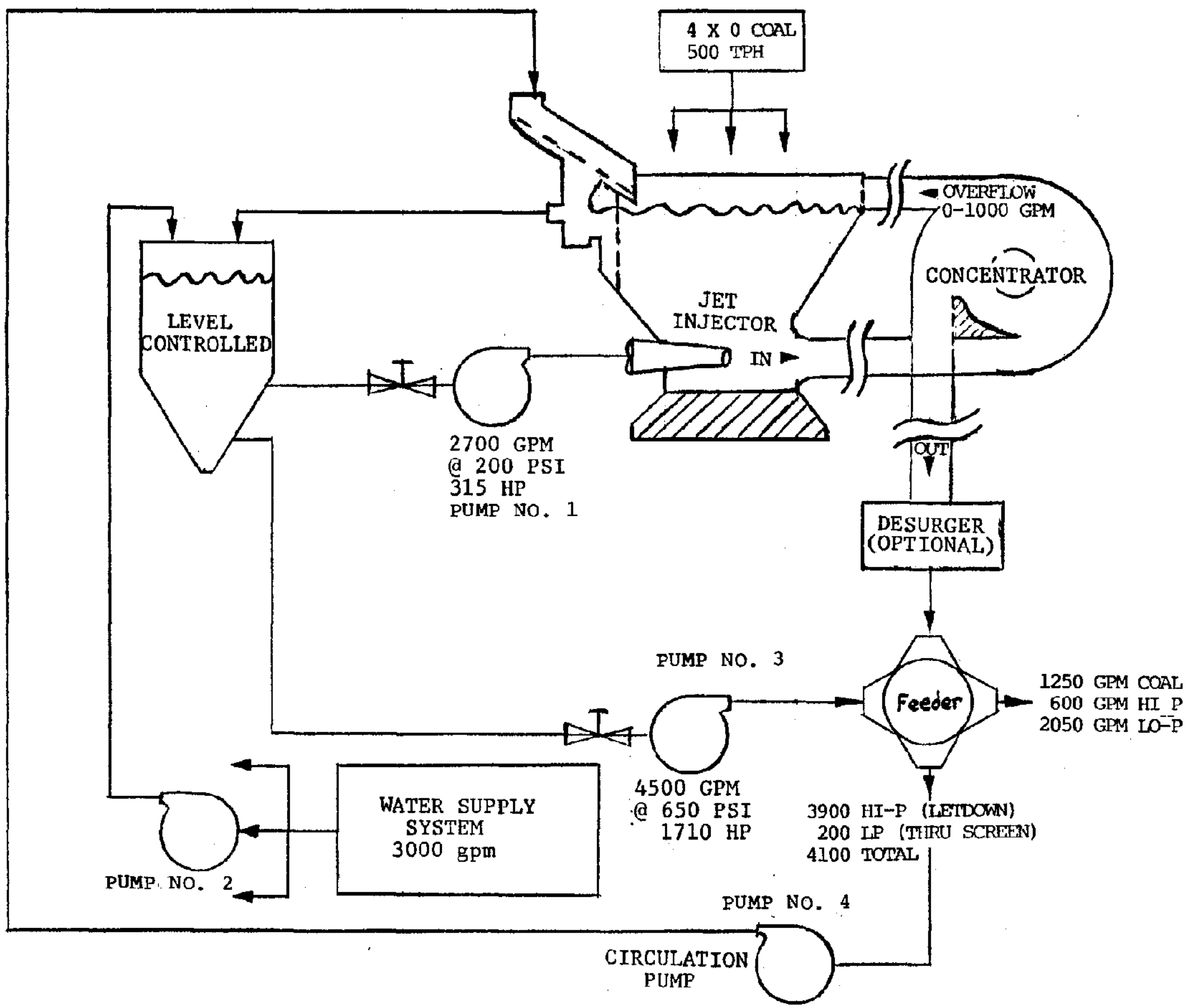


Figure 5-1: Jet Pump Injector Used As First Stage for Coal-Water Mixing in a Slurry Haulage Line

Reviewing the function of the various pumps shown in this diagram, Pump No. 1 is the water supply for the nozzle of the Jet Injector. As labeled, it furnishes 2700 gpm at 200 psi. This represents 315 HP in the fluid flow. Pump No. 2 is identified as the water supply pump operating at very low pressure and at a top flow of 3000 gpm. The function of this pump is to supply a constant level water reservoir which will maintain a flooded inlet for Pump No. 1. Pump No. 3 is a high pressure pump used to accelerate the contents of the lock-hopper. The high pressure water that does not exit with the slurry is returned to the water system by Pump No. 4. This unit is a low pressure circulation pump.

The net effect of the system just described is that the lockhopper and its high pressure water supply as well as the concentration of its delivered slurry can be enhanced by virtue of the fact that the Jet Pump Injector forms the slurry and delivers a relatively high concentration to the input of the lockhopper.

The system just described has been suggested through discussions with industry representatives. Although it has not actually been built, its economics have been assessed as being quite favorable. It is presented as an example of a beneficial application of the Jet Pump Injector with its centrifugal concentrator.

5.2 Design Guide for a Jet Pump Injector

Alhough all of the details of the computer analysis and subscale and full scale test results are documented throughout this report and in previous phase ending reports, they appear more as a compilation of performance facts rather than the development of a progressive design tool. The purpose of this section is to draw upon those previous findings, and within the limits of the available information, provide guidelines for the design of a jet pump injector face haulage system.

The first issue to consider in such a design effort is the selection of an injector configuration. The subscale tests determined that a horizontally oriented injector with a gravity fed entrainment zone is superior to both a vertically oriented injector as well as a horizontally oriented injector with a screw fed entrainment zone, because of its higher developed discharge pressure (Figure 5.2).

The next issue to consider is the size of the injector system. This will be determined as a function of slurry coal

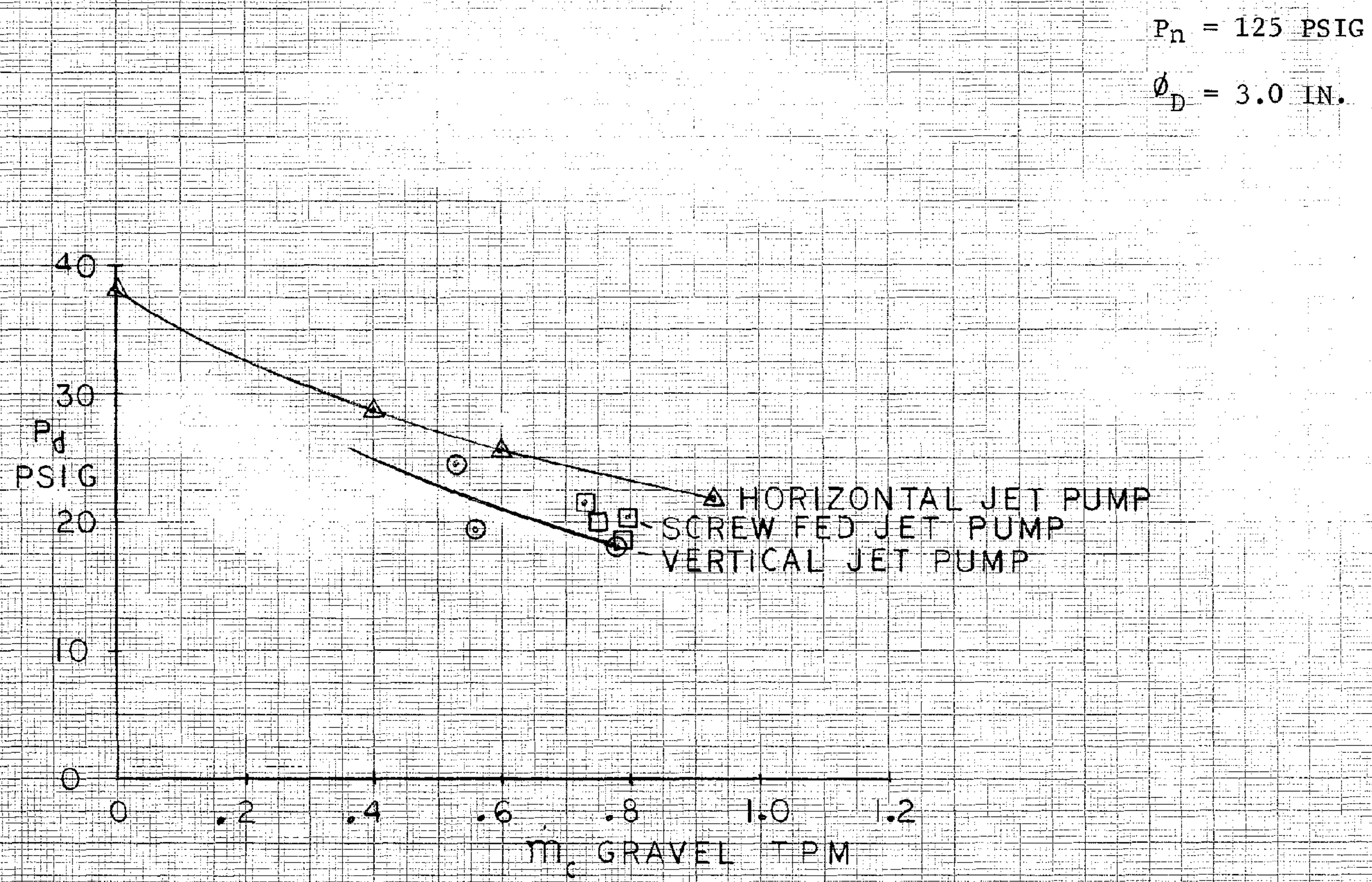


Figure 5.2: COMPARISON OF INJECTOR PERFORMANCES

flow rate, $\dot{M}_c$, as measured in TPM. The first aspects of system size to consider are the water flow rate and nozzle and mixing throat diameters required to pump and ingest at the required slurry tonnage rate. The functional relationship between these system variables and slurry mass flow rate were defined while assessing operating conditions during the prototype injector design effort of Phase II-A (Table 4-3), and are summarized in Figures5.3 through 5.6. These curves provide the basis for interpolating/extrapolating a water flow rate, a hopper flow rate, a nozzle diameter, and a mixing throat diameter for any required coal tonnage rate.

Knowing the required water flow rate will enable the calcuation of pressure losses in the supply line and the determination of centrifugal pump pressure required to maintain 125 psi at the nozzle, for which the data of Table 4-3 were determined. A pump selection can then be made knowing the required pressure and flow. It is desirable to chose a pump with a flat performance characteristic, as is the case with the Ingersoll-Rand Model 8X23SF which was used for the full scale injector test. This flat performance curve provides the capability of responding to limited changes in flow demand without the need for valves or controls at the pump.

It must also be noted that the full scale injector tests varied nozzle pressure between nominally 100 and 200 psi, and were able to achieve 6 TPM flow rate and 40% concentration regardless of the nozzle pressure. The motivation for raising nozzle pressure is the achievement of higher line velocities for longer conveying distances, at the expense of increased abrasive wear.

The issues of slurry line diameter and maximum conveying distances were also addressed while assessing operating conditions during the prototype injector design effort of Phase II-A (Table 4-2), and their functional relationships are summarized in Figures 5.7 and 5.8. These curves provide the basis for selecting the maximum slurry line diameter in order to maintain line speed above the saltation velocity of 10 feet per second, and they indicate the maximum conveying distances corresponding to that line diameter selection.

To increase the maximum potential conveying distance for a given injector geometry, merely increase the discharge pressure proportionately. Figures 4-82 through 4-84 show the relationship between nozzle pressure and discharge pressure for a variety of nozzle diameters and entrainment lengths evaluated during the full scale injector tests. On average, an incremental change in discharge pressure requires a 7 fold change in nozzle pressure.

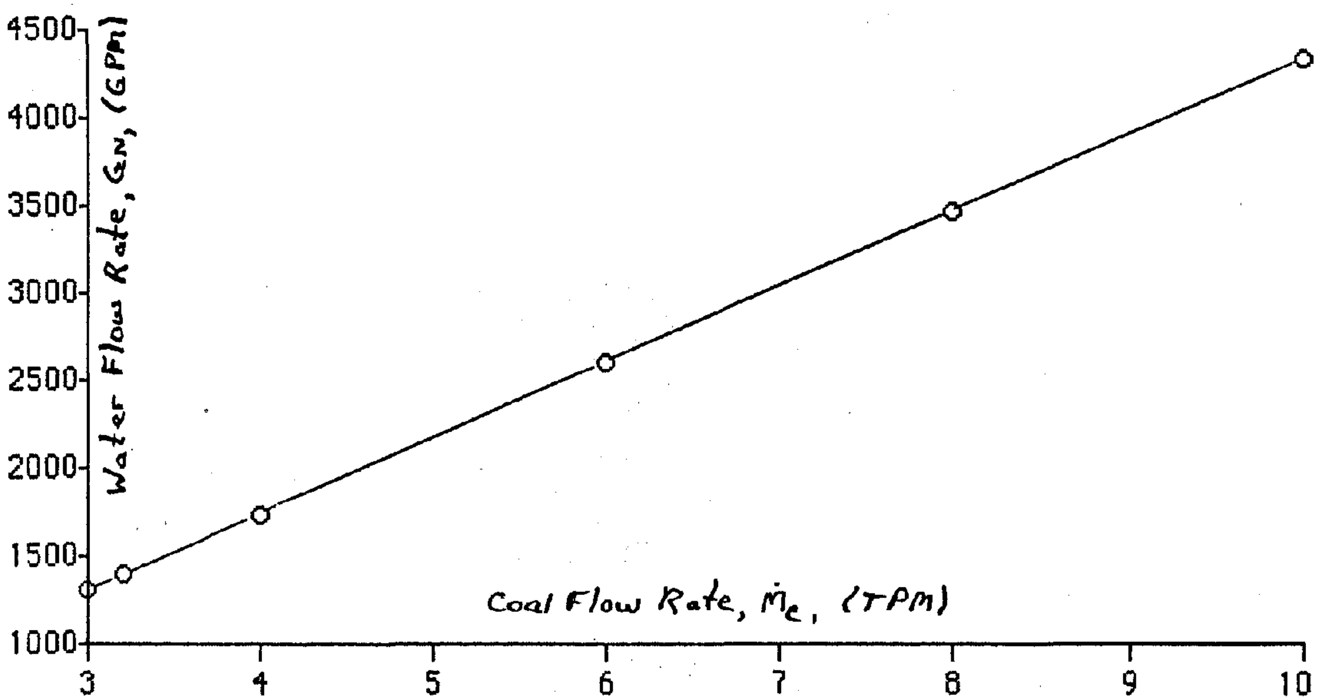


Figure 5.3: Nozzle Water Flow Rates Required for a Range of Coal Tonnage at $C_W = 0.323$

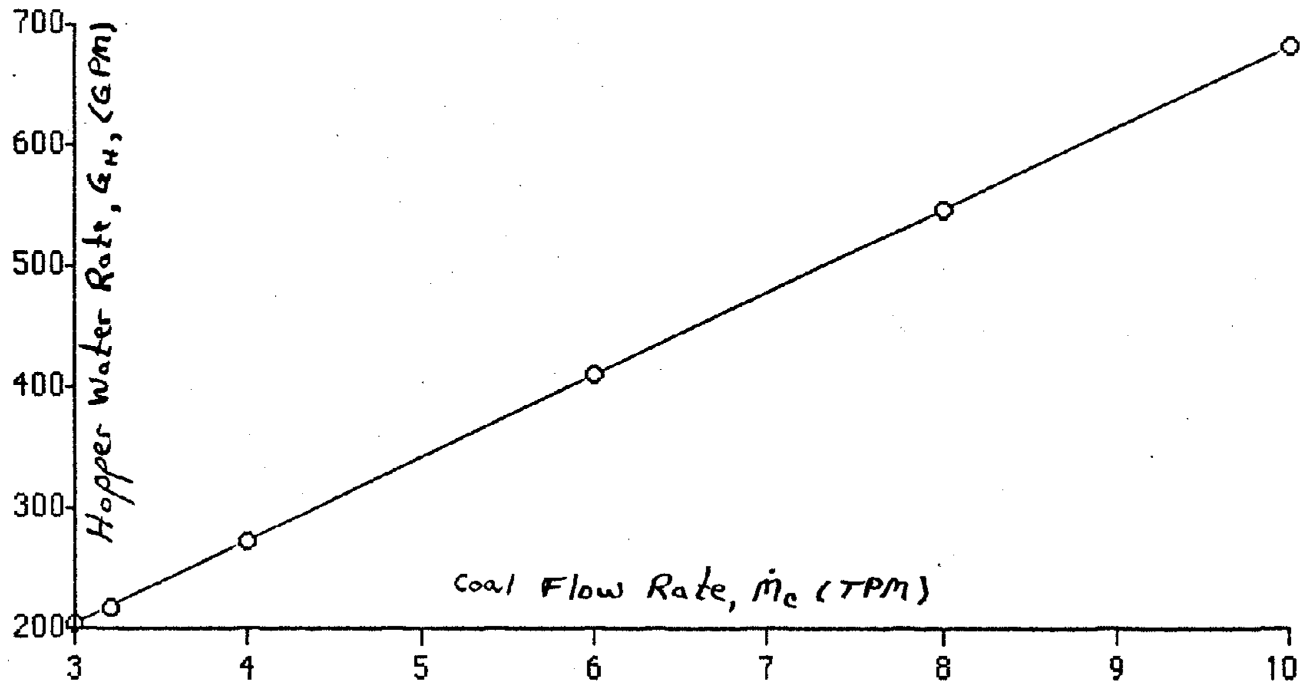


Figure 5.4: Hopper Water Flow Rates Required for a Range of Coal Tonnage at $C_W = 0.323$

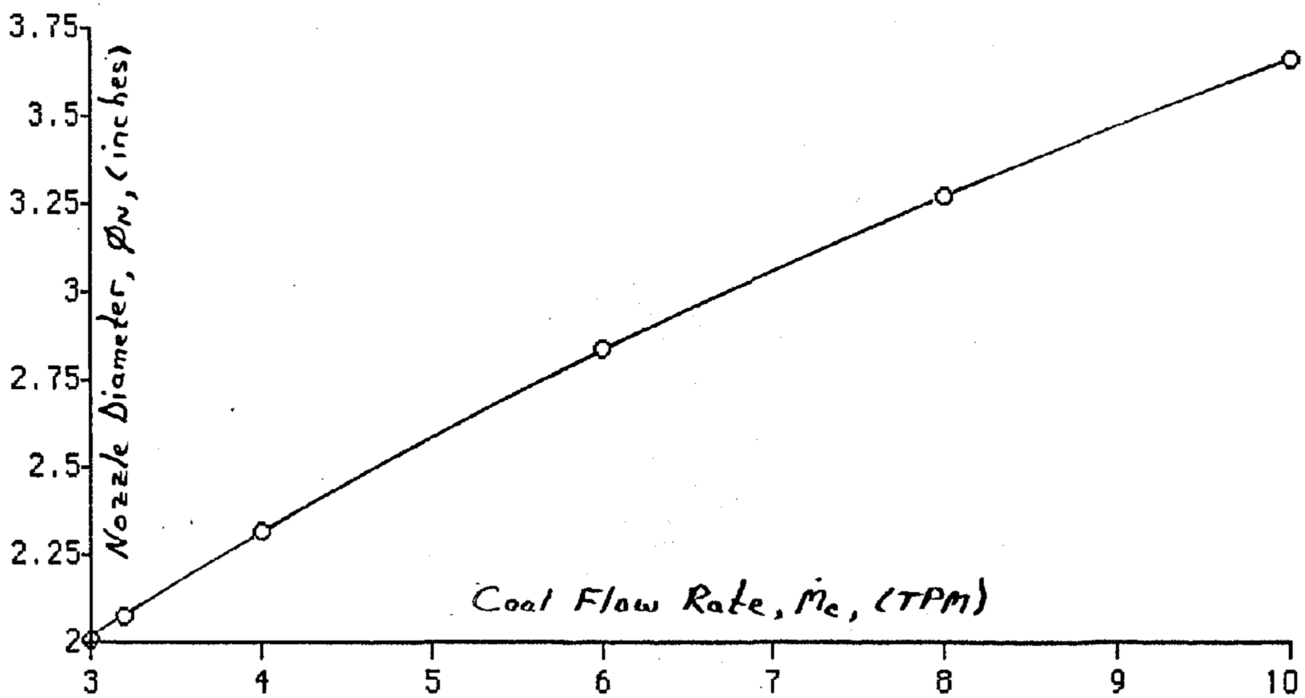


Figure 5.5: Nozzle Diameters Required for a Range of Coal Tonnage at $C_W = 0.323$

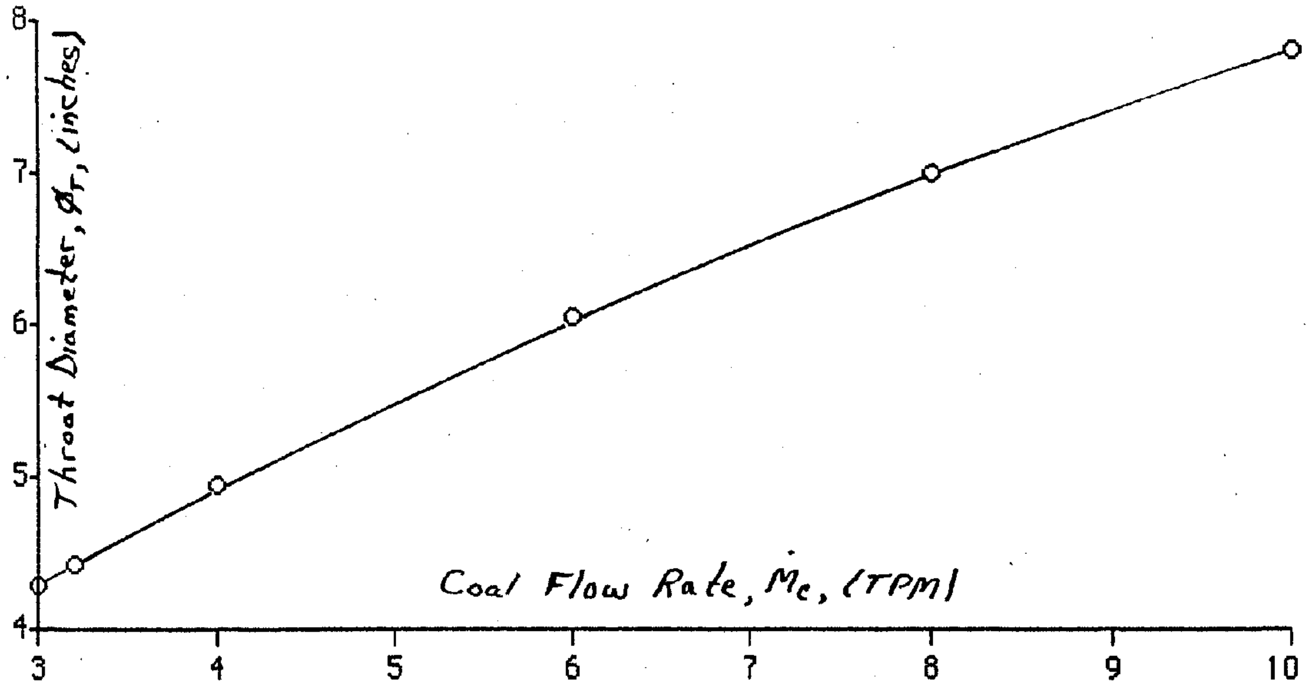


Figure 5.6: Mixing Throat Diameters Required for a Range of Coal Tonnage at $C_W = 0.323$

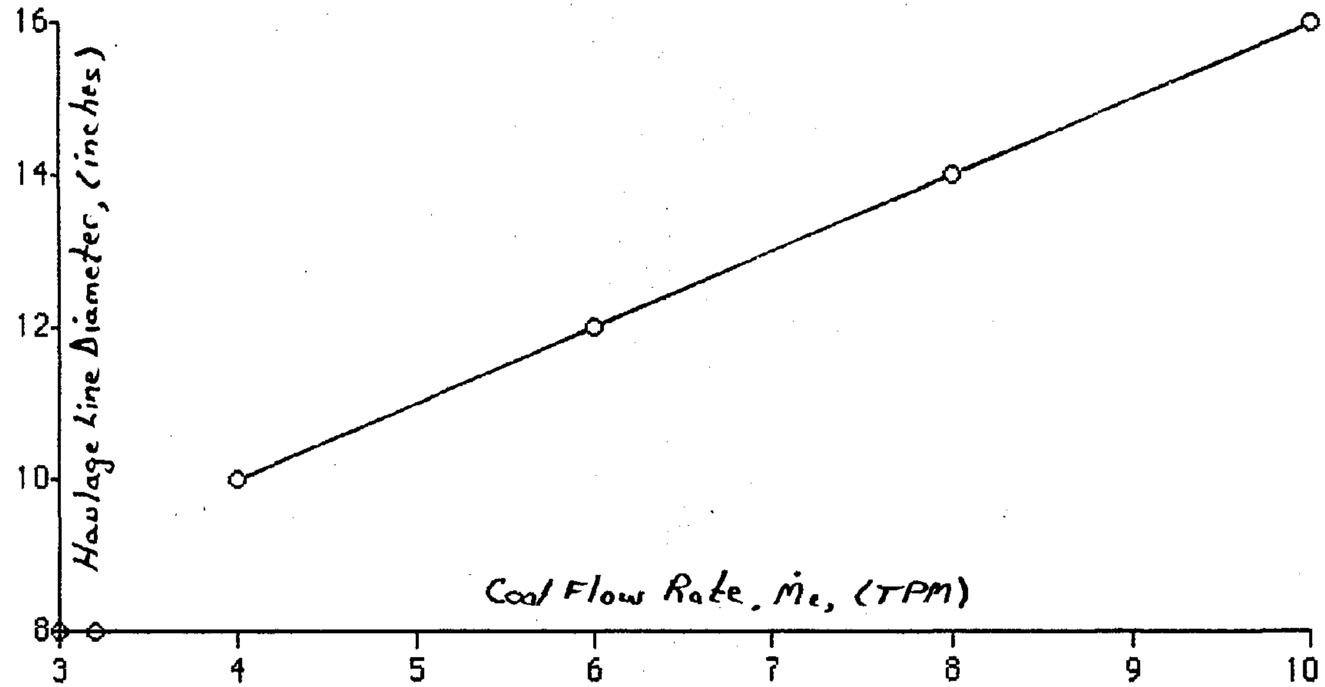


Figure 5.7: Line Diamcters Recommended for a Range of Coal Tonnage at $C_W = 0.323$

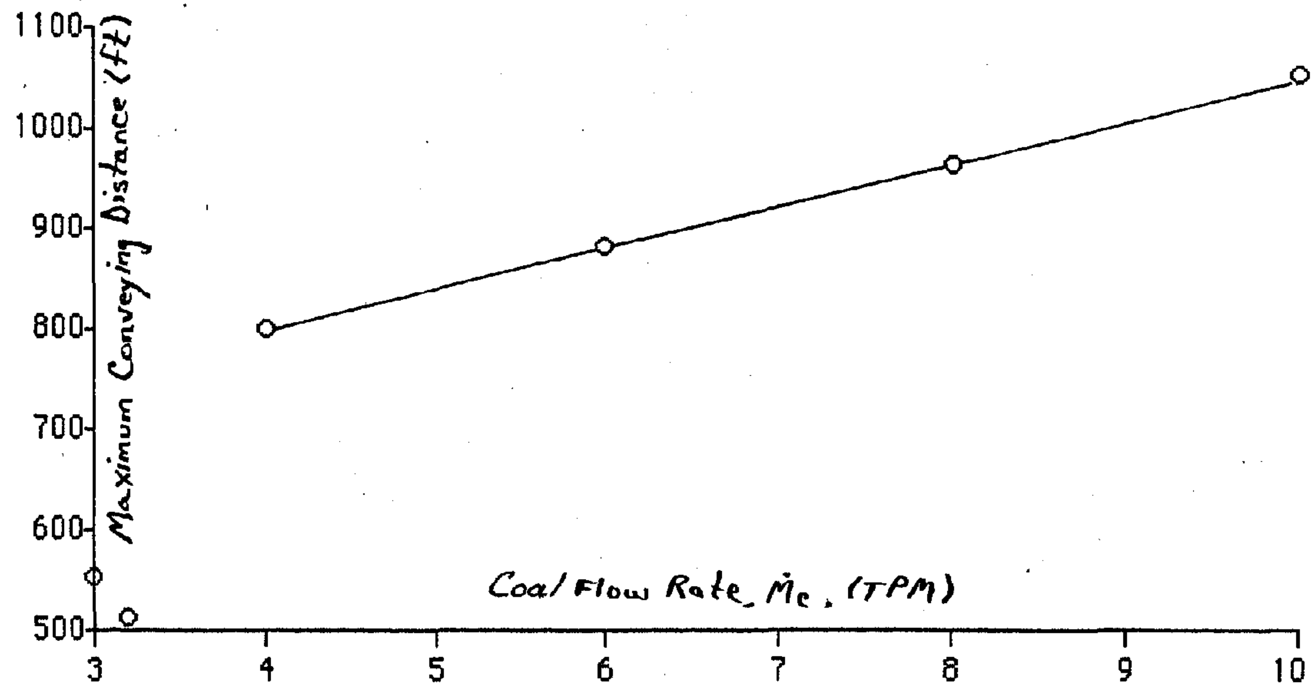


Figure 5.8: Maximum Conveying Distance Based on the Recommended Line Diameters for each Coal Tonnage Rate at $C_w = 0.323$

An outstanding feature of the jet injector developed in this program is its ability to achieve the design coal flow rate and concentration for a variety of component dimensions and operating conditions. This relative insensitivity of performance to certain component dimensions was summarized in Section 3.0. In particular, diffuser angles between 4 1/2 and 10 degrees had little affect on subscale and computer models. The choice of a 14 degree included angle or 7 degree half angle for the full scale injector tests was made on the basis of compactness rather than performance. The lack of influence of diffuser angle on discharge pressure indicates that very little diffusion is actually occurring in the slurry jet in this region, and therefore an even greater diffuser angle could be used without detriment to discharge pressure.

Similarly, the effect of mixing chamber length on discharge pressure was shown to be minimal during subscale tests and computer analysis, and a recommendation was made to avoid unnecessary length. In fact, a mixing length of 3" for a 2" throat diameter was recommended. The large scale injector tests employed a mixing length of nominally 12" for a diameter of nearly 6". Therefore a design rule of thumb is to maintain the ratio of mixing length to throat diameter from between 1.5 to 2.0.

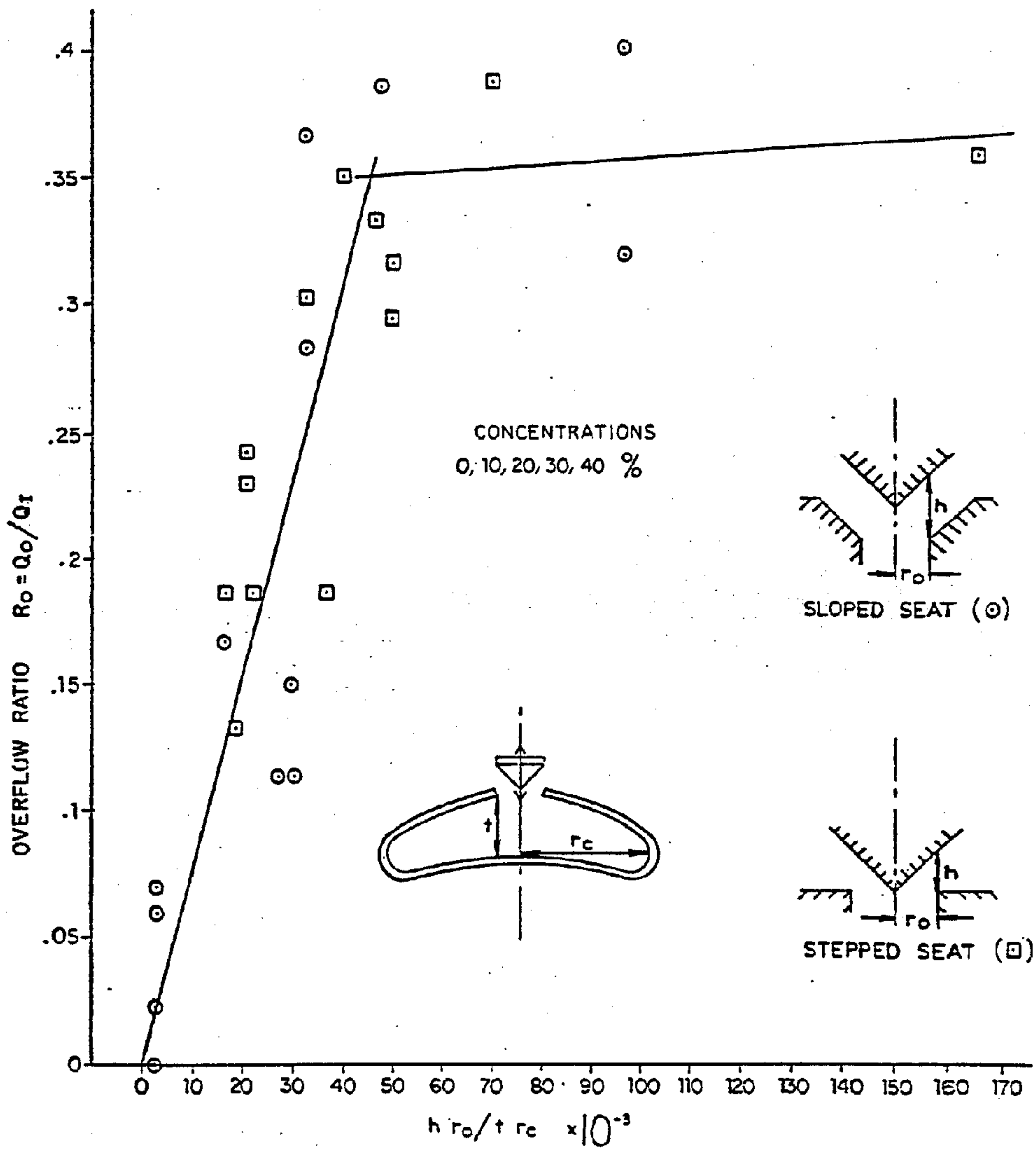
The effect of entrainment length was evaluated both during subscale and full scale injector tests. The results were the same, namely the shorter the better. Excessive entrainment length results in jet degradation. In particular, the full scale tests dictate that entrainment length not exceed 156 mm or nominally 6". This guideline should be followed in future full scale designs.

Regarding guidelines for the remaining components of the jet injector haulage system, there are basically only two other considerations. In general, coal particle size should not exceed 1/3 of the diameter of the slurry pipeline. This therefore determines the downsizing requirements for the primary and secondary breakers feeding the injector.

To define the required volumes of the injector feed hopper and its overflow reserve water tank, it is imperative that the water level control system adopted during the development of this program be utilized. The rapid response of this control system allows the reserve tank volume to be equal to 5 seconds worth of throughflow water. For example, a water flow rate of 2400 GPM would require a reserve tank volume of 200 gallons. The 3 1/2 ton capacity on the feed hopper utilized during full scale tests is considered to be a practical minimum provided that the surge bin volume is identical.

The design of a centrifugal concentrator is governed primarily by 2 guidelines. The first guideline requires that the flow area in the concentrator be no greater than that of the slurry pipeline in order to maintain flow velocities above the saltation speed. The second guideline is aimed at maintaining design concentration performance by insuring that the proper relationship is maintained between overflow and concentrator flow passages: Namely, that the product of overflow passage height and radius be greater than 50,000 times the product of concentrator height and radius as shown in Figure 5-9.

Although several of the design parameters were not varied during full scale injector tests, the fact that those tests clearly demonstrated the ability of the injector to achieve the design throughput and concentration, regardless of operating conditions, lends credence to the belief that the guidelines defined above will serve as an effective procedural tool in designing future jet pump injector face haulage systems.



POPPET-VALVE OVERFLOW DATA

Figure 5.9: Centrifugal Concentrator Overflow Data

APPENDIX A

Bibliography

BIBLIOGRAPHY

1. G. Govatos and I. Zandi, Pumping Solid-Liquid Mixtures by Jet Pump. Symposium Solids Transport in slurries, Part 1. American Institute of Chemical Engineers, Seventh National Meeting, Atlantic City, New Jersey, September, 1971.

2. G. Fish, The Hydraulic Lifting of Heavy Solids Materials by Jet Pump. British Hydromechanics Research Association. BHRA Report RR 100g. May, 1969.

3. G. Fish, The Solids-Handling Jet Pump, Hydrotransport 1, First International Conference on the Hydraulic Transport of Solids in Pipes, BHRA, Cranfield, Bedfore, England, September, 1970.

4. A. W. Wakefield, Practical Solids-Handling Jet Pumps. Symposium on Jet Pumps and Ejectors, Paper 12. London, November, 1972.

5. G. N. Abramovich, The Theory of Turbulent Jets. The MIT Press. Cambridge, Massachusetts.

6. J. M. Link, A. Allen, Jr., and R. R. Faddick, Feasibility of Hydraulic Transportation in Underground Coal Mines, Colorado School of Mines Research Institute for USBM Contact #H0133037, May 30, 1975.

7. R. E. McElvain, High Pressure Pumping, Skillings Mining Review, January 26, 1974.

8. D. K. Mistry, Coal Injector for Coarse Slurry Injector, Phase I Report, TM7514, Ingersoll-Rand Research, Inc. for USBM Contract H0155097, December 15, 1975.

9. Unpublished source.

10. S. Black and J. Rounds, Deep Cutting Continuous Miner Effect of Drum Rotational Speed and Depth of Cut on Airborne Respirable Dust and Specific Energy, TM 7501, Ingersoll-Rand Research, Inc. for USBM Contract H0122039, June 15, 1977.

11. Kao, D.T., Prof. U of Ky. Provided design concept in consultations.

Note: The extensive bibliography compiled for Phase I is on file at the Bureau of Mines' Pittsburgh Research Center.

APPENDIX B

Coal Concentrator Test Data
(Reference Phase IIB)

DEFINITION OF TERMS USED IN DATA TABLE

TERM		DEFINITION
C_W		Fractional mass concentration of coal in slurry at jet pump
C_{WD}		Fractional mass concentration of coal in slurry downstream of concentrator
G_{CON},	CONC. FLOW	Concentrator withdrawal or effluent flow, gpm
G_H,	HOPPER FLOW	Secondary water flow used to flood coal in hopper of jet pump, gpm
G_n		Water flow through jet pump nozzle, gpm
$\dot{m}_c$		Conveyor coal flow rate to jet pump, tons/minute
$\dot{m}_{con}$		Coal flow concentrator effluent, tons/minute
P_c		Pressure at cover of concentrator, PSIG
P_n		Jet pump nozzle pressure, PSIG
P_p		Pressure in slurry pipe one diameter upstream of first concentrator slot, PSIG
ΔP_c, PACE		Pressure drop across the concentrator P_p-P_c as measured with the Pace Eng. Corp pressure transducey PSI
ΔP_c, MANO		Pressure drop across the concentrator P_p-P_c, as measured with the Dwyer inclined mamometer, PSI

COAL CONCENTRATOR DATA

RUN NO.	P_n PSIG	ΔP_c, PACE PSI	ΔP_c, MANO PSI	P_p PSIG	P_c PSIG	CONC. FLOW GPM	HOPPER FLOW GPM	Gn GPM	$\dot{m}_c$ TPM	C_w ORIG SLURRY	Cwd AFTER CONC.	AVE. SLOT VEL. FT/SEC	
1	125	--		16.0	14.0	18.3	--	250	0	0	0	8.2	
2	125	--		20.5	15.5	30.0	--	250	0	0	0	13.4	
3	125	--		23.5	22.5	1.35	--	250	0	0	0	0.6	
4	125	--											
5	80												
6	60	All data result in plugging of slot where P 0											
7	50												
8	40												
9	30												
10	138												
11	125	0.45		22.5	22.0	1.2	--	250	0	0	0	0.5	
12	125	0.80		21.5	22.2	5.2	--	250	0	0	0	2.3	
13	125	0		22.5	22.5	3.0	--	250	0	0	0	1.3	
14	125	1.0		21.5	21.0	4.5	--	250	0	0	0	2.0	
15	125	0.3		21.3	22.0	3.4	--	250	0	0	0	1.5	
16	125	--		22.2	22.4	2.2	--	250	0	0	0	1.0	
17	125	--		23.0	22.0	4.2	--	250	0	0	0	1.9	
18	125	--		23.0	22.3	1.8	--	250	0	0	0	0.8	
19	125	0.5		24.2	23.5	--	--	250	0	0	0		
20	125	.43		23.5	22.7	2.3	120	250	.169	.1625	.1634	1.0	
21	120	.38		22.7	22.2	5.6	107	245	.175	.1681	.1703	2.5	
22	125	.43		24.3	23.7	--	--	250	--				Plugged Concentrator
23	125	.28		24.2	23.5	4.9	118	250	.171	.1623	.1641	2.2	

COAL CONCENTRATOR DATA

RUN NO.	P_r PSIG	ΔP_o, FACE PSI	ΔP_o, MANO PSI	P_p PSIG	P_o PSIG	CONC. FLOW GPM	HOPPER FLOW GPM	Gn GPM	$\dot{m}_o$ TPM	C_w' ORIG. SLURRY	C_{wd} AFTER CONC.	AVE. SLOT VEL. FT/SEC
24	125	.5		25.5	24.8	1.0	104	250	.169	.1691	.1695	0.4
25	125	.7		--	--	--	--	250	0	0	0	plugged concentrator
26	125	.5		23.4	--	--	--	250	0	0	-	1 inch valve plugging
27	125	.5		--	24.2	--	--	250	0	0		1 inch valve plugging
28	136	.5		--	--	--	--	261	0	0	0	
29	136	.1		26.0	25.7	5.8	118	261	.301	.1581	.1601	2.6
30	136	.25		26.2	26.0	5.4	118	261	.297	.1582	.1601	2.4
31	125	0.		23.8	24.6	6.3	99	250	.305	.1690	.1715	2.8
32	125	.15		24.0	24.3	6.6	100	250	.301	.1685	.1711	2.9
33	125	.10		23.8	24.2	6.7	101	250	.301	.1680	.1707	3.0
34	125	.10		24.2	23.8	7.0	103	250	.301	.1671	.1699	3.1
35	125	.10		23.8	24.3	7.0	107	250	.301	.1656	.1683	3.1
36	129	.15		24.6	24.2	3.3	92	254	.301	.1714	.1727	1.5
37	125	.15		24.0	23.5	4.7	92	250	.301	.1724	.1744	2.1
38	126	.10		24.1	23.8	6.5	99	251	.301	.1685	.1711	2.9
39	125	.05		24.2	23.9	6.6	101	250	.301	.1681	.1707	2.9
40	125	.10		24.0	23.5	3.7	89	250	.301	.1741	.1757	1.7
41	125	.20		24.0	23.5	4.0	89.5	250	.301	.1738	.1755	1.8
42	126	.10		24.2	23.8	3.1	99	251	.301	.1698	.1711	1.4
43	125	.15		27.3	27.0	5.2	47	250	.500	.2842	.2878	2.3
44	125	.05		26.0	25.7	5.6		250	.500	--	--	2.5
45	125	0.1		23.1	22.4	19.5		250	0	0	0	Orifice Calibration
46	125	0.13		21.2	21.1	31.2		250	0	0	0	Orifice Calibration

Run 45 marks the beginning of the 18 slot sample.

COAL CONCENTRATOR DATA

RUN NO.	P_n PSIG	ΔP_o, PACE PSI	ΔP_o, MANO PSIG	P_p PSIG	P_o PSIG	CONC. FLOW GPM	HOPPER FLOW GPM	G_n GPM	$\dot{m}_c$ TPM	C_v ORIG. SLURRY	C_{wd} AFTER CONC.	AVE. SLOT VEL. FT/SEC	
47	125	0.1		21.5	21.4	35.7	--					.59	Orifice Calibration
48	125	0.1		21.3	21.4	36.6						.60	Orifice Calibration
49	125	0.05		22.3	22.3	15.9						.26	Orifice Calibration
50	125	0.15		21.5	21.5	25.4						.42	Orifice Calibration
51	125	0.10		21.2	21.3	31.2						.51	Orifice Calibration
52	125	0.05		21.0	21.0	21.7						.36	Orifice Calibration
53	125	0.10		20.8	21.2	13.6						.22	Orifice Calibration
54	125	0.08		21.3	21.3	24.2						.40	Orifice Calibration
55 S	125	0.15		24.3	23.8	46.2t		250	.516	.2698	.3000	.76	
56	125	0.25		23.3	23.7	40.0t		250	.516	.2778	.3052	.66	
57 S	125	0.30		23.3	23.4	17.8t 20.1		250	.516	.2883	.3025	.29	
58	125	0.10		23.1	23.2	40.0t 40.8		250	.516	.2805	.3084	.66	
59	125	0.10		23.2	23.6	31.6t 31.5		250	.516	.2859	.3084	.52	
60 S	125	0.[illegible]		22.4	23.0	33.0t 33.4		250	.500	.2769	.3001	.54	
61	125	0.10		23.0	23.5	40.0t 40.8		250	.500	.2742	.3017	.66	
62	125	-0.05		23.7	23.3	21.7t 23.6		250	.500	.2787	.2935	.36	
63			OVERFILL						.639				
64	125	-0.05		24.2	24.4	21.9t 24.5		250	.554	.3091	.3257	.36	

t, flow timed with bucket and stopwatch

COAL CONCENTRATOR DATA

RUN NO.		P_n PSIG	ΔP_c, PAGE PSI	ΔP_c, MANO PSI	P_p PSIG	P_o PSIG	CONC. FLOW GPM	HOPPER FLOW GPM	G_n GPM	m_c TPM	C_w ORIG SLURRY	C_{wd} AFTER CONC.	AVE. SLOT VEL. FT/SEC	$\dot{m}_{con}$ TPM	
65		127	- .10		23.8	24.0	39.5t 41.2	16.4	252	.542	.2986	.3284	.65		
66		125	0.[illegible]		22.3	22.1	32.6t 34.6	13.4	250	.563	.3148	.3407	.54		
67		125	0.30		24.1	24.1	25.0t 27.8	16.4	250	.568	.3205	.3405	.41		
68		125	- .10		24.4	24.3	17.9t 20.5	18.9	250	.568	.3243	.3388	.30		
69		125	- .10		24.1	23.9	25.4t 30.8	30.0	250	.554	.3066	.3257	.42		
70		125	0		22.5	23.0	39.0t 40.6	13.4	250	.568	.3121	.3427	.64		
71		125	0.05	-.07	21.4	22.0	94.2	83	250	.331	.1948	.2410	1.56		
72	S	125	- .60	-.37	21.0	21.5	101.5	163	250	.568	.2532	.3007	1.68		
73	S	125	- .30	-.19	22.0	22.0	101.5	94	250	.568	.2862	.3484	1.68		
74	S	125	- .40	-.28	21.0	21.0	117	98	250	.568	.2850	.3603	1.93		
75		125		-.07	22.0	23.0	105	69	250	.568	.2944	.3675	1.73	.050	
76		125		-.19	21.4	21.1	127	81	250	.588	.2989	.3823	2.10	--	
77		125		-.19	21.3	21.2	139	94	250	.568	.2868	.3607	2.30	.086	
78		125		-.19	20.5	20.4	148	78	250	.583	.2990	.3922	2.44	.099	
79	S	125		--	21.0	21.2	152	61	250	.621	.3240	.4383	2.51	.104	
80		125		--	21.0	20.2	184	64	250	.627	.3240	.4667	3.04	.153	
81	S	125		-.19	21.3	21.0	140	105	250	.610	.2949	.3682	2.31	.088	plugged - concentrat
82		125		-.19	20.5	20.5	158	74	250	.588	.3034	.4071	2.61	.113	
83		125		-.22	20.4	20.3	161	83	250	.568	.2905	.3862	2.66	.117	

t, flow timed with bucket and stopwatch

COAL CONCENTRATOR DATA

RUN NO.		P_n PSIG	ΔP_o, PACE PSI	ΔP_c, MANO PSIG	P_p PSIG	P_c PSIG	CONC. FLOW GPM	HOPPER FLOW GPM	q_n GPM	$\dot{m}_c$ TPM	C_w ORIG SLURRY	C_{wd} AFTER CONC.	AVE. SLOT VEL. FT/SEC	$\dot{m}_{con}$ TPM	
84		125		--	20.3	20.4	158					Plugged Concentrator			
85		125		-.22	21.0	20.3	157	74	250	.604	.3091	.4147	2.59	.111	
86	S	125		-.15	20.2	20.3	162	74	250	.594	.3056	.4130	2.67	.119	
87		125		--	21.0	20.4	166					Plugged Concentrator			
88		125		--	21.3	20.3	164					Plugged Concentrator			
89		125		-.22	21.0	21.1	152	64	250	.621	.3219	.4337	2.51	.104	
90		125		-.19	20.4	20.3	158	69	250	.604	.3124	.4226	2.61	.113	
91	S	125		-.15	21.1	21.2	158	69	250	.621	.3184	.4309	2.61	.113	
92	S	125		-.17	21.0	20.3	162	64	250	.627	.3240	.4456	2.67	.118	
93		125		-.11	21.2	21.4	149	45	250	.633	.3399	.4670	2.46	.100	
94		125		-.11	20.7	21.5	149	56	250	.621	.3275	.4433	2.46	.100	plugged oc after poir
95a		125		-.13	21.4	22.0	146	45	250	.633	.3399	.4638	2.41	.096	
95b		125		-.11	20.7	21.5	145	37	250	.658	.3549	.4880	2.39	.094	
96		125		--	21.7	21.3	103	45	250	.621	.3356	.4173	1.70	.048	
97		125		-.107	21.6	21.6	126	43	250				2.08	.072	plugged slurry
98	S	125		-.17	21.7	21.6	132	69	250	.621	.3184	.4107	2.18	.078	
99		125		--	--	--	125		250	.616			2.06	.070	plugged slurry
100		125		--	--	--	136	78	250	.616	.3107	.3998	2.24	.083	pump lost suct.
101	S	125		-.15	21.7	21.5	136	81	250	.616	.3087	.3961	2.24	.083	
102	S	125		--	22.3	22.4	125	69	250	.633	.3226	.4105	2.06	.070	
103		125		0	20.5	21.0	125	74	250	.633	.3192	.4044	2.06	.070	

COAL CONCENTRATOR DATA

RUN NO.	P_u PSIG	P_3 PSIG	Q_n GPM	Q_h GPM	Q_{so} GPM	$\dot{m}_o$ #/min	C_w orig. slurry	C_{wd} dwnstrm of conc.	C_{wo} conc. effl.	ΔP_1	ΔP_2	ΔP_3	ΔP_4	ΔP_5	ΔP_6	ΔP_7	Avg. Slot Velocity FPS	Comments
										PRESSURE DROP ACROSS CONCENTRATOR SLOTS, PSI								
104	127	24	252.4	42.2	153.8	Coal Rate	--	--	.06	.77	.083	.11					2.62	
105	125	24.8	250.4	26.7	110.3	Calibra- Was In-	--	--	.051	-.22	-.066	.055					1.88	
106	126	27.4	251.4	22.3	0	correct and not	--	---	0	0	0	0					0	
107	125	24.4	250.4	--	78	repeatable- for these		--	.024	-	-	-					1.33	
108	125	23.7	250.4	23.1	97.3	runs	--	--	.023	-	-	-					1.65	Discharge line pulled
109	125	25.2	250.4	45.3	48.6	897	.242	.299	.018	.132	-.165	-.044					.83	
110	125	24.8	250.4	41.1	93.7	897	.234	.340	--	.165	-.022	.044					1.59	Underflow sample was lost
111	125	24.5	250.4	40.0	116.3	897	.279	.373	.048	.215	.055	.408		ONLY 3 PRESSURE DROPS TAKEN FOR FIRST 20 RUNS			1.98	
112	125	24.2	250.4	37.7	130.0	897	.278	.385	.051	.253	.077	.298					2.21	
113	125	27.4	250.4	26.7	0	823	.262	.262	0	0	0	0					0	
114	126	25.8	251.4	26.7	82.2	1041	.309	.382	.022	.165	-.044	.165					1.40	
115	125	24.2	250.4	25.0	110.3	1041	.323	.415	.080	.309	.088	.143					1.88	
116	125	23.2	250.4	26.7	137.6	1041	.330	.442	.108	.408	.138	.198					2.34	
117	125	23.4	250.4	48.1	147.0	1066	.306	.445	.038	.992	.165	.386					2.50	
118	125	23.2	250.4	51.7	156.0	1066	.310	.449	.060	2.535	.044	.110					2.65	
119	125	23.6	250.4	23.1	147.0	1066	.330	.480	.065	--	--	--					2.50	ΔP Tare Value Lost
120	125	27.2	250.4	13.3	0	842	.277	.277	0	0	0	0					0	
121	125	25.0	250.4	55.0	66.3	842	.250	.292	.033	.242	-.220	.143					1.13	
122	125	24.0	250.4	82.3	110.3	842	.240	.305	.048	--	--	--					1.88	No ΔP Data Taken
123	125	22.5	250.4	93.4	147.0	842	.241	.324	.069	.452	.121	.193					2.50	Sample
124	125	23.0	250.4	64.0	142.4	983	.293	.386	.100	.485	.452	.375	.298	.298	.281	.281	2.42	
125	125	24.6	250.4	37.3	78.0	983	.292	.353	.037	.253	.287	.253	.253	.253	.253	.253	1.33	
126	125	24.0	250.4	44.3	121.9	983	.297	.396	.059	.441	.375	.331	.270	.270	.264	.248	2.07	

COAL CONCENTRATOR DATA

RUN NO.	P_D PSIG	P_s PSIG	Q_D GPM	Q_h GPM	Q_{sc} GPM	$\dot{m}_o$ #/min	C_w orig. slurry	C_{wd} dwnstrm of conc.	C_{wc} conc. effl.	ΔP_1	ΔP_2	ΔP_3	ΔP_4	ΔP_5	ΔP_6	ΔP_7	Avg. Slot Velocity FPS	Comments
										PRESSURE DROP ACROSS CONCENTRATOR SLOTS, PSI								
127	125	--	250.4	42.2	160.4	983	.308	.457	.075*	--	--	--	--	--	--	--	--	* .075 Assumed -
128	125	24.8	250.4	48.1	52.0	898	.265	.299	.027	.077	.132	.110	.121	.121	.138	.138	.88	
129	125	23.8	250.4	44.3	97.3	898	.268	.342	.032	.242	.231	.198	.165	.149	.165	.154	1.65	
130	125	22.9	250.4	42.2	124.7	898	.275	.373	.053	.353	.298	.253	.176	.160	.171	.154	2.12	Discharge line plug
131	125	21.2	250.4	48.1	149.3	1048	.307	.430	.072	.551	.380	.287	.171	.127	.176	.154	2.54	
132	125	21.2	250.4	42.2	158.1	1048	.310	.447	.072	.573	.402	.309	.176	.121	.171	.143	2.69	
133	125	23.1	250.4	18.9	97.3	1048	.315	.407	.027	.193	.171	.165	.127	.110	.143	.143	1.65	
134	125	--	250.4	18.9	160.2	1129	.353	.522	.089*	--	--	--	--	--	--	--	2.73	* Assumed - concentrator plugged pump lost suction
135	125	--	250.4	32.7	166.4	1115	.329	.493	.069*	--	--	--	--	--	--	--	2.83	
136	125	21.0	250.4	37.7	142.4	1109	.333	.449	.101	.496	.353	.287	.176	.160	.154	.143	2.42	
137	125	22.3	250.4	18.9	140.0	1089	.337	.420	.163	--	--	--	--	--	--	--	2.38	
138	125	22.0	250.4	18.9	149.3	1069	.328	.435	.127	.485	.353	.292	.198	.143	.143	.160	2.54	
139	125	22.0	250.4	16.3	152.7	1095	.334	.468	.125	.562	.368	.309	.226	.160	.149	.182	2.60	
140	125	22.0	250.4	16.3	162.3	1095	.347	.484	.127	.634	.485	.369	.281	.231	.209	.215	2.76	
141	125	22.5	250.4	42.2	132.5	972	.294	.386	.090	.391	.314	.276	.204	.138	.182	.187	2.25	Samples
142	125	24.2	250.4	28.3	80.1	972	.294	.354	.051	.121	.138	.143	.132	.083	.094	.105	1.36	Samples
143	125	22.2	250.4	49.0	149.3	972	.293	.396	.099	.512	.419	.369	.237	.176	.198	.198	2.54	Samples

* NOTE: Run 143 marks the beginning of the Variable Area Concentrator.

COAL CONCENTRATOR DATA

RUN NO.	P_n PSIG	P_s PSIG	Q_n GPM	Q_h GPM	Q_{so} GPM	$\dot{m}_c$ #/min	C_w orig. slurry	C_{wd} dwnstrm of conc.	C_{wo} conc. effl.	ΔP_1	ΔP_2	ΔP_3	ΔP_4	ΔP_5	ΔP_6	ΔP_7	Avg. Slot Velocity FPS	Comments
										PRESSURE DROP ACROSS CONCENTRATOR SLOTS, PSI								
166	117	20.8	242.3	*	*	1188	*	*	*	*	*	*	*	*	*	-1.014	*	*Continuation of run #59. All setting constant, except for pressure tap #7 location. Duplicate data not taken.
167	117	20.5	242.3	23.1	53.8	1188	.358	.529	.065	-.077	-.116	-.061	.022	-.033	0	-.171	.80	
168	125	21.3	250.4	37.7	153.8	1173	.339	.473	.087	-.094	-.121	-.039	.116	-.033	-.039	-.237	.80	
169	125	21.8	250.4	29.8	160.2	1212	.351	.498	.092	-.072	-.132	-.033	.121	-.033	-.033	-.237	.84	
170	125	21.7	250.4	18.9	160.2	1271	.370	.532	.089	-.077	-.105	-.028	.116	-.039	-.072	-.308	.84	Concentrator discharge manifold from ports 2 and 3
171	117	20.3	242.3	23.1	142.4	1158	.354	.492	.089	-.061	-.105	-.006	.110	-.039	-.138	-.821	.74	

COAL CONCENTRATOR DATA

RUN NO.	P_D PSIG	P_s PSIG	Q_n GPM	Q_h GPM	Q_{sc} GPM	$\dot{m}_c$ #/min	C_w orig. slurry	C_{wd} dwnstrm of conc.	C_{wc} conc. effl.	ΔP_1	ΔP_2	ΔP_3	ΔP_4	ΔP_5	ΔP_6	ΔP_7	Avg. Slot Velocity FPS	Comments
										PRESSURE DROP ACROSS CONCENTRATOR SLOTS, PSI								
144	125	26.4	250.4	16.3	0	952	.300	.300	0	0	0	0	0	0	0	0	0	Concentrator Closed
145	125	23.3	250.4	32.7	97.3	952	.294	.355	.098	.083	.066	.044	.022	-.088	-1.025	-1.223	.51	Discharge from Upstream Port
146	125	22.6	250.4	55.0	130.0	952	.279	.360	.089	.198	.077	.050	.011	-.088	-.882	-.744	.68	
147	125	22.0	250.4	66.7	153.8	952	.268	.374	.062	.209	.149	.072	.050	-.011	-.369	-.577	.80	
148	125	21.5	250.4	18.9	157.0	1051	.322	.464	.084	.209	.149	.077	.050	-.028	-.512	-.441	.82	
149	125	22.0	250.4	13.3	160.2	1115	.344	.491	.104	.226	.165	.083	.044	-.033	-.386	-.353	.84	Sample
150	125	22.8	250.4	0	166.4	1158	.372	.532	.133	.518	.176	.105	.077	.000	-.154	-2.017	.87	Sample
151	125	23.3	250.4	0	168.5	1173	.372	.530	.135	.435	--	--	--	--	--	--	.88	Ran out of hopper co
152	125	21.8	250.4	23.1	168.5	1158	.356	.499	.131	.463	.298	.099	.077	-.022	-.248	-1.796	.88	More fines this run.
153	125	25.7	250.4	18.9	0	923	.292	.292	0	0	0	0	0	0	0	0	0	Concentrator closed
154	125	22.7	250.4	40.0	120.5	923	.274	.372	.036	.028	.066	.011	.105	.006	-.077	--	.63	Discharge port in center.
155	125	21.5	250.4	89.5	138.8	923	.247	.332	.044	.050	-.094	.028	.127	.006	-.077	--	.72	
156	125	22.9	250.4	62.6	159.2	1024	.286	.404	.065	-.006	-.083	.022	.138	.017	-.077	--	.83	
157	125	22.9	250.4	35.3	165.4	1109	.320	.471	.069	.000	-.061	.017	.138	.006	-.083	--	.86	
158	125	21.8	250.4	18.9	169.5	1143	.346	.510	.094	.033	-.033	.017	.116	.000	-.088	--	.88	
159	117	20.3	242.3	13.3	158.1	1143	.365	.521	.116	-.055	-.044	.017	.143	.017	-.055	-.055	.82	
160	117	24.0	242.3	51.0	0	942	.306	.306	0	0	0	0	0	0	0	0	0	Concentrator closed
161	117	22.5	242.3	49.9	116.3	942	.285	.379	.049	-.077	-.121	-.055	.033	-.011	-.039	--	.61	Discharge port downstream
162	117	21.0	242.3	90.5	172.4	942	.258	.400	.028	-.055	-.110	-.061	.033	-.017	-.033	--	.90	
163	117	21.4	242.3	66.7	140.0	1057	.296	.412	.047	-.077	-.166	-.061	.033	-.017	-.017	--	.73	
164	117	21.3	242.3	18.9	147.0	1143	.345	.517	.041	-.072	-.099	-.055	.033	.022	.017	--	.77	
165	117	20.8	242.3	13.3	152.7	1188	.369	.544	.078	-.110	-.110	-.066	.022	-.039	-.022	-1.576	.80	Concentrator partial plugged, coal level surged.

APPENDIX C

FULL-SCALE JET PUMP INJECTOR SUBASSEMBLY TEST DATA

(Reference Phase III)

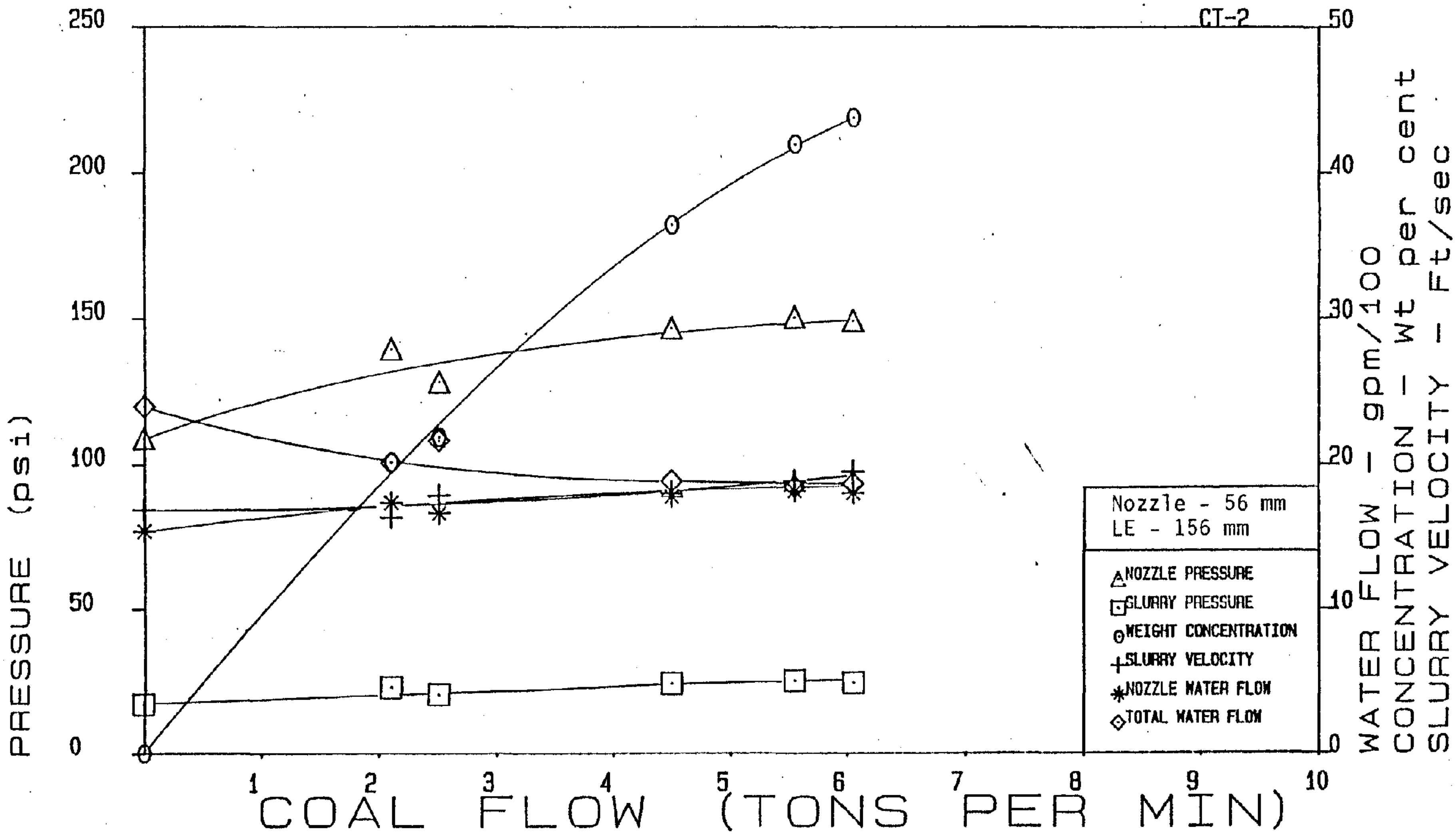
NOMENCLATURE FOR TABULATED TEST DATA

WT, mv	Local cell millivolt reading (for determining coal mass flow)
VEL, Ft/m	Velocity, feet per minute
NOZ, gpm	Jet nozzle flow, gallons per minute
TOTAL, gpm	Jet nozzle flow plus reserve flow, gallons per minute
COAL, TPM	Coal delivered to hopper, tons per minute
CV	Coal concentration by volume
CW	Coal concentration by weight
VEL, Ft/s	Velocity, feet per second
TPM	Tons per minute
QH	Hopper water throughput, gallons per minute
QN	Nozzle water throughput (flow) gallons per minute
PN	Nozzle inlet pressure
PD	Slurry line inlet pressure
PWR OUT	Kinetic energy at inlet to slurry line
PWR IN	Kinetic energy in nozzle

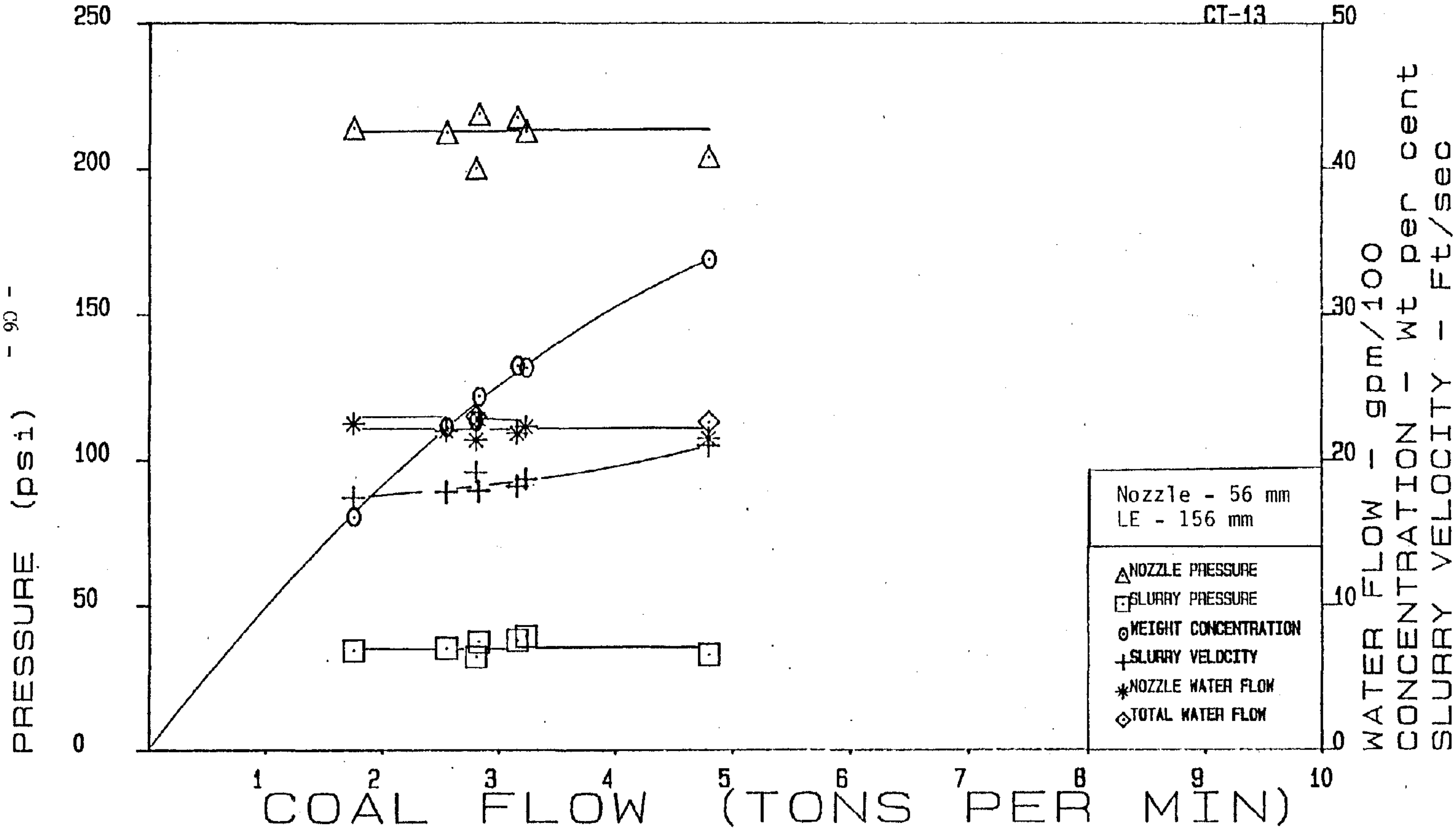
TEST DATA SUMMARY

Nozzle Diameter (mm)	Entrainment Length (mm)	Run Number (#)
56	156	2
		13
		15
		16
	230	1
		5
		6
		14
		18
	265	17
	280	3
		4
63	174	9
		10
	250	7
		8
	285	11
		12

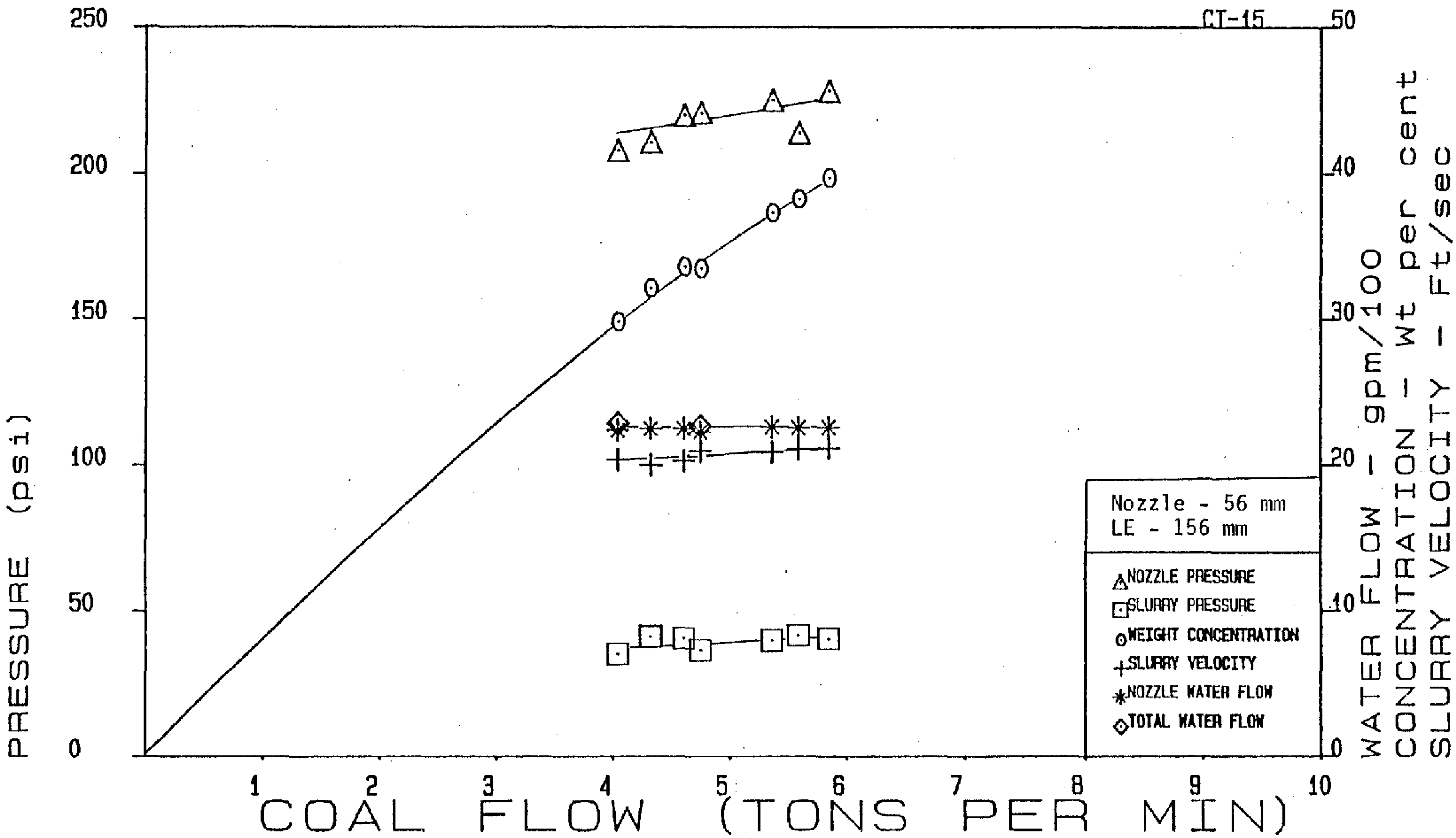
TEST NO.	BELT		WATER		COAL	SLURRY			MASS RATIO	PRESSURE			EFF
	WT mV	VEL Ft/m	NOZ gpm	TOTAL gpm	TPM	CV	CW	VEL Ft/s	240TPM+QH / QN	PN psi	PD psi	PD / PN-PD	PWR OUT / PWR IN
CT-2	0.0	0	1539	2397	0.00	0.00	0.00	16.9	.56	109.0	17.2	.19	25
CT-2	22.5	362	1775	1880	4.50	.26	.36	18.0	.67	146.6	24.3	.20	24
CT-2	21.3	364	1661	2157	2.51	.15	.22	17.8	.66	128.1	20.5	.19	24
CT-2	21.5	278	1736	2004	2.11	.14	.20	16.3	.45	139.4	23.0	.20	22
CT-2	23.8	329	1800	1862	6.05	.33	.44	19.5	.84	149.1	24.5	.20	25
CT-2	23.5	331	1812	1839	5.55	.31	.42	18.8	.75	150.3	25.2	.20	25


CT-2
PRESSURE (psi)
250
200
150
100
50
0
50
40
30
20
10
0
WATER FLOW - gpm/100
CONCENTRATION - Wt per cent
SLURRY VELOCITY - Ft/sec
1
2
3
4
5
6
7
8
9
10
COAL FLOW (TONS PER MIN)
Nozzle - 56 mm
LE - 156 mm
NOZZLE PRESSURE
SLURRY PRESSURE
WEIGHT CONCENTRATION
SLURRY VELOCITY
NOZZLE WATER FLOW
TOTAL WATER FLOW

TEST NO.	BELT WT mV	BELT VEL Ft/m	WATER NOZ gpm	WATER TOTAL gpm	COAL TPM	SLURRY CV	SLURRY CW	SLURRY VEL Ft/s	MASS RATIO 240TPM+QH / QN	PRESSURE PN psi	PRESSURE PD psi	PRESSURE PD / PN-PD	EFF PWR OUT / PWR IN
CT-13	20.9	348	2249	2204	1.76	.11	.16	17.4	.17	213.8	34.3	.19	18
CT-13	21.4	347	2199	2141	2.55	.15	.22	17.7	.25	212.5	35.1	.20	19
CT-13	22.8	348	2149	2260	4.80	.24	.34	21.0	.59	204.1	32.9	.19	22
CT-13	22.3	244	2137	2297	2.81	.15	.23	19.1	.39	200.1	32.2	.19	20
CT-13	21.7	362	2183	2105	3.16	.18	.27	18.1	.31	217.5	38.0	.21	21
CT-13	21.5	362	2281	2109	2.83	.17	.24	17.8	.22	218.6	37.4	.21	19
CT-13	21.6	391	2230	2167	3.24	.18	.26	18.6	.32	212.8	39.5	.23	22

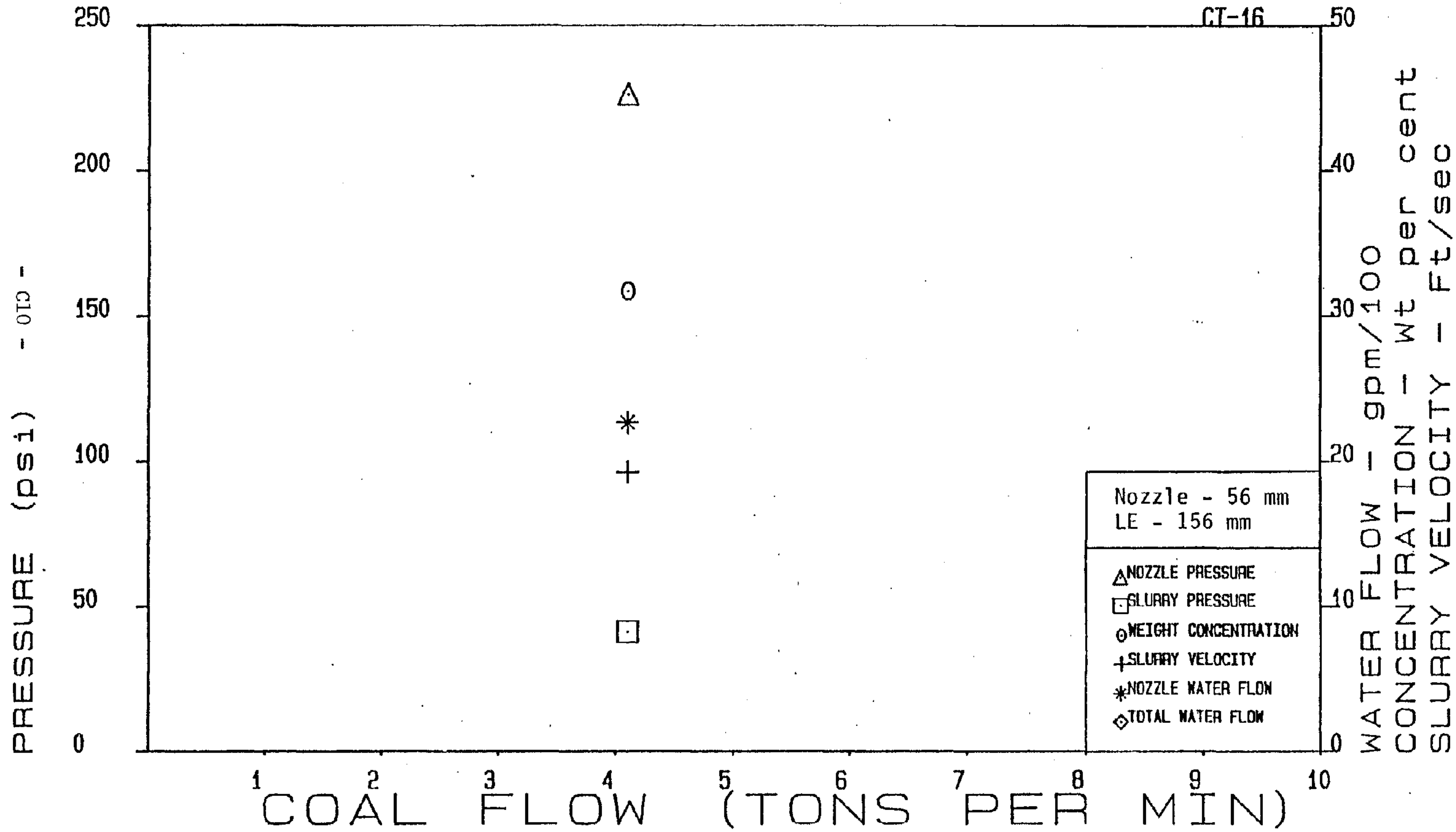

CT-13
PRESSURE (psi)
250
200
150
100
50
0
50
40
30
20
10
0
WATER FLOW – gpm/100
CONCENTRATION – Wt per cent
SLURRY VELOCITY – Ft/sec
1
2
3
4
5
6
7
8
9
10
COAL FLOW (TONS PER MIN)
Nozzle - 56 mm
LE - 156 mm
NOZZLE PRESSURE
SLURRY PRESSURE
WEIGHT CONCENTRATION
SLURRY VELOCITY
NOZZLE WATER FLOW
TOTAL WATER FLOW

TEST NO.	BELT WT mV	BELT VEL Ft/m	WATER NOZ gpm	WATER TOTAL gpm	COAL TPM	SLURRY CV	SLURRY CW	SLURRY VEL Ft/s	MASS RATIO 240TPM+QH / QN	PRESSURE PN psi	PRESSURE PD psi	PRESSURE PD / PN-PD	EFF PWR OUT / PWR IN
CT-15	22.4	338	2237	2284	4.04	.21	.30	20.3	.45	207.7	35.0	.20	22
CT-15	22.3	376	2248	2190	4.32	.23	.32	20.0	.44	210.5	41.1	.24	25
CT-15	23.3	347	2254	2161	5.59	.28	.38	21.1	.55	213.8	41.5	.24	26
CT-15	22.7	356	2226	2265	4.75	.24	.33	20.9	.53	220.7	36.4	.20	22
CT-15	23.4	353	2256	2129	5.85	.29	.40	21.1	.57	228.2	40.1	.21	23
CT-15	24.3	259	2260	2162	5.36	.27	.37	20.9	.53	225.0	39.7	.21	23
CT-15	22.5	371	2248	2186	4.61	.24	.34	20.2	.46	219.8	40.5	.23	24



CT-15
PRESSURE (psi)
250
200
150
100
50
0
1
2
3
4
5
6
7
8
9
10
COAL FLOW (TONS PER MIN)
50
40
30
20
10
0
WATER FLOW - gpm/100
CONCENTRATION - Wt per cent
SLURRY VELOCITY - Ft/sec
Nozzle - 56 mm
LE - 156 mm
NOZZLE PRESSURE
SLURRY PRESSURE
WEIGHT CONCENTRATION
SLURRY VELOCITY
NOZZLE WATER FLOW
TOTAL WATER FLOW

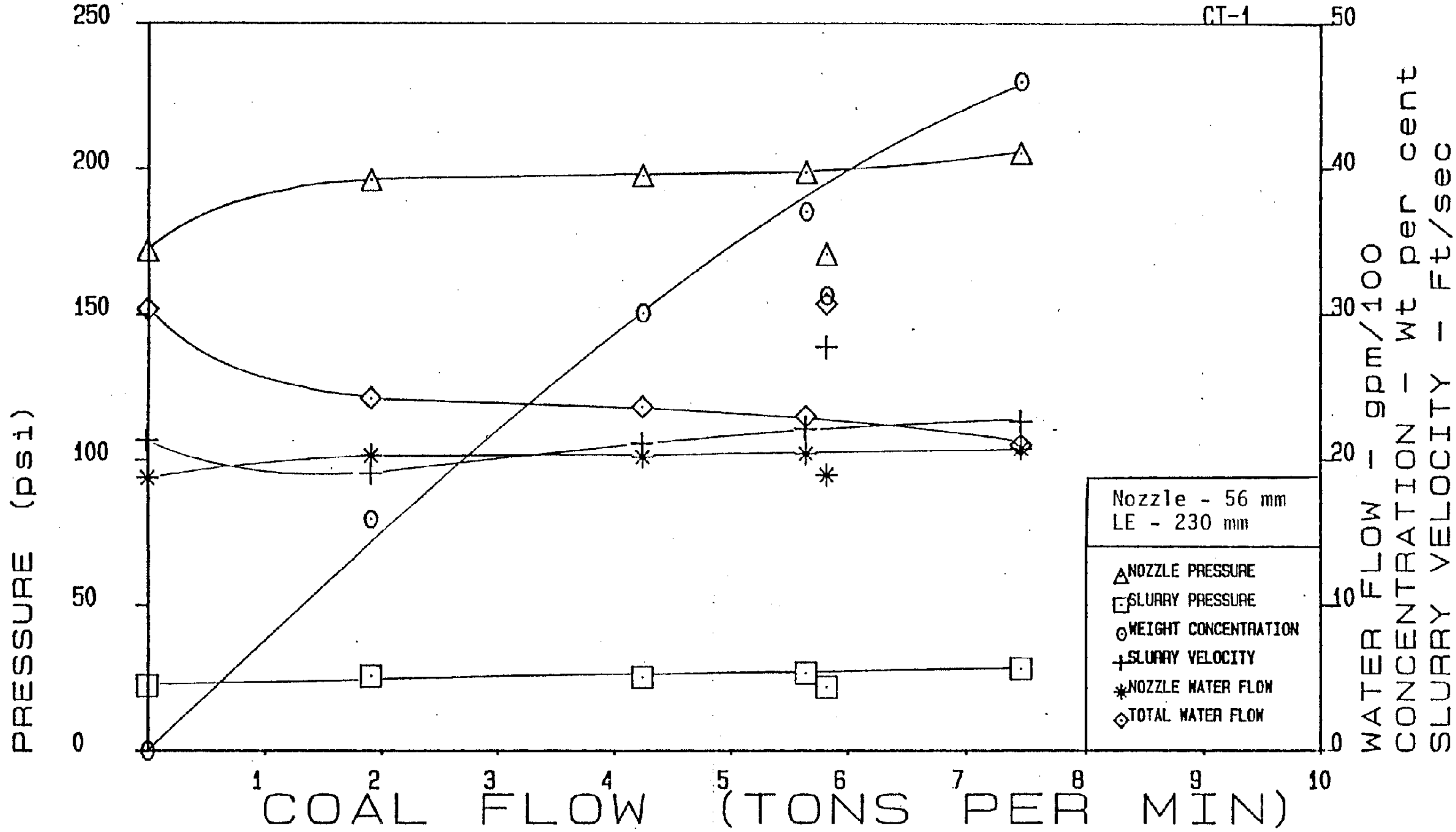
TEST NO.	BELT WT mV	BELT VEL Ft/m	WATER NOZ gpm	WATER TOTAL gpm	COAL TPM	SLURRY CV	SLURRY CW	SLURRY VEL Ft/s	MASS RATIO 240TPM+QH / QN	PRESSURE PN psi	PRESSURE PD psi	PRESSURE PD / PN-PD	EFF PWR OUT / PWR IN
CT-16	22.2	372	2267	2120	4.11	.23	.32	19.2	.37	226.1	41.1	.22	22



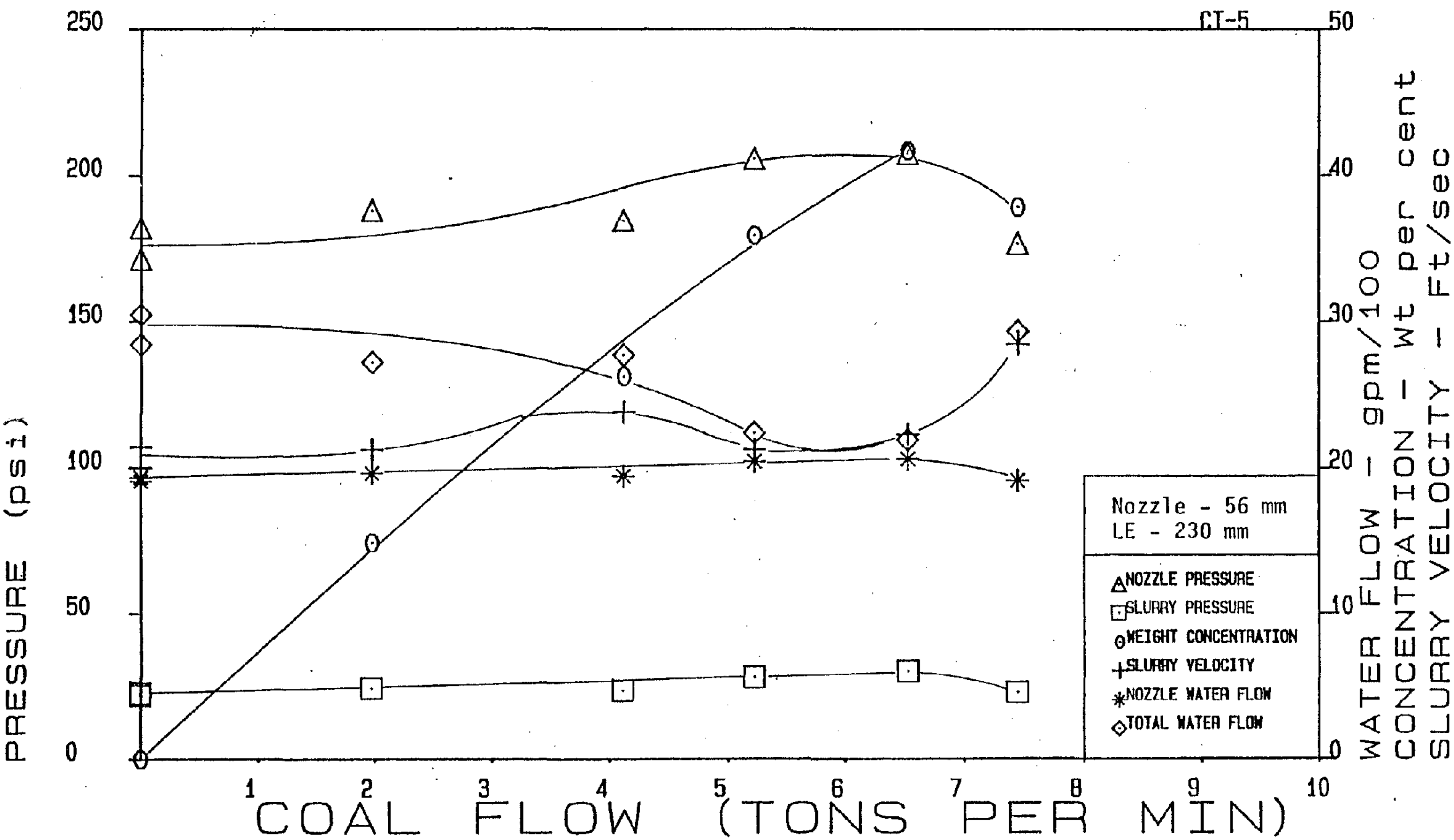
CT-16
Nozzle - 56 mm
LE - 156 mm
NOZZLE PRESSURE
SLURRY PRESSURE
WEIGHT CONCENTRATION
SLURRY VELOCITY
NOZZLE WATER FLOW
TOTAL WATER FLOW
PRESSURE (psi)
WATER FLOW - gpm/100
CONCENTRATION - Wt per cent
SLURRY VELOCITY - Ft/sec
COAL FLOW (TONS PER MIN)
0
50
100
150
200
250
10
20
30
40
50
1
2
3
4
5
6
7
8
9
10

- 010 -

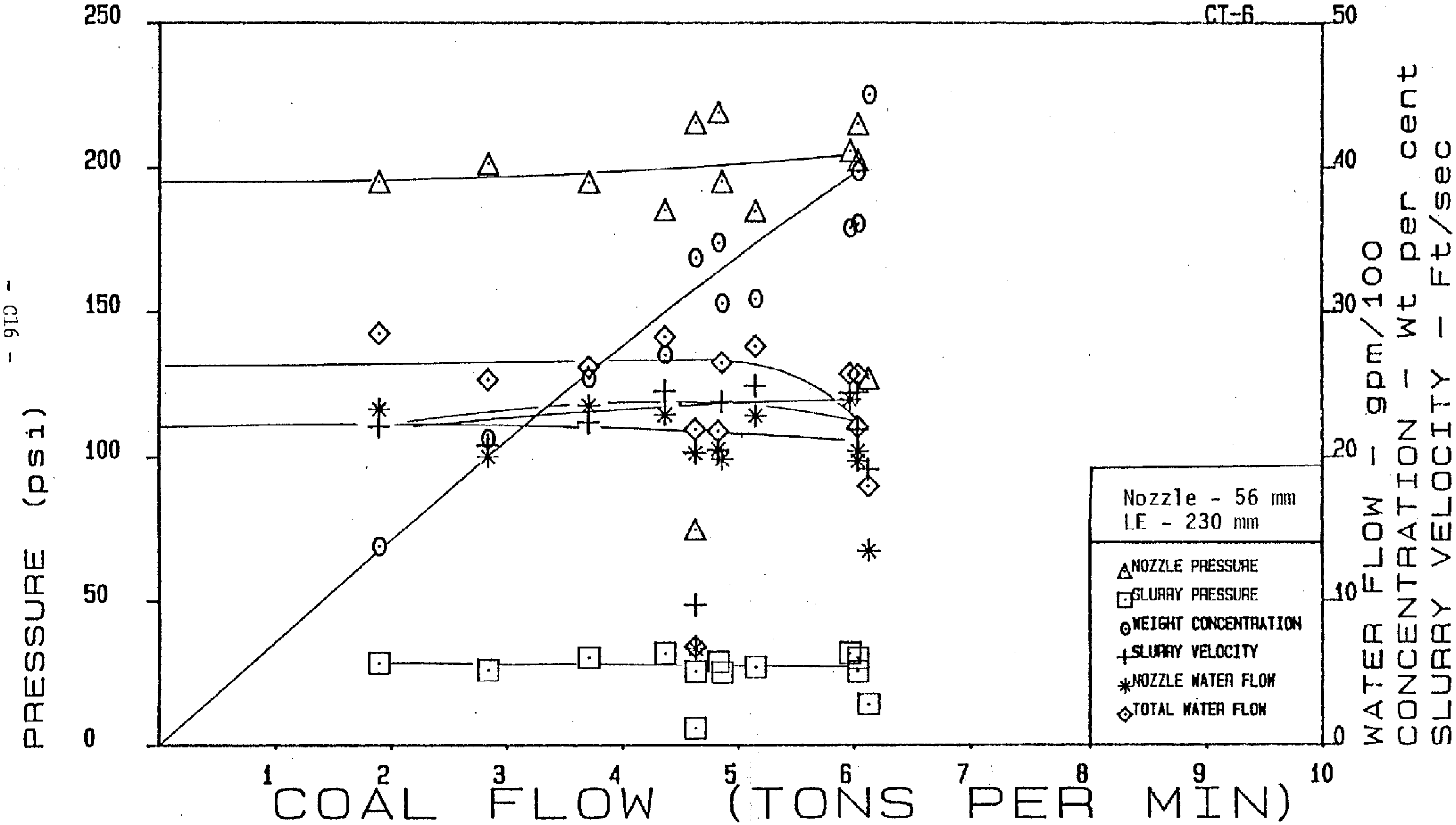
TEST NO.	BELT WT mV	BELT VEL Ft/m	WATER NOZ gpm	WATER TOTAL gpm	COAL TPM	SLURRY CV	SLURRY CW	SLURRY VEL Ft/s	MASS RATIO 240TPM+QH / QN	PRESSURE PN psi	PRESSURE PD psi	PRESSURE PD / PN-PD	EFF PWR OUT / PWR IN
CT-1	0.0	0	1876	3033	0.00	0.00	0.00	21.3	.62	171.9	22.6	.15	21
CT-1	22.9	298	2017	2363	4.24	.21	.30	21.1	.68	197.7	25.4	.15	19
CT-1	21.2	305	2027	2421	1.91	.11	.16	19.0	.42	196.1	25.7	.15	18
CT-1	24.5	261	2040	2297	5.64	.27	.37	22.1	.79	199.0	26.7	.16	21
CT-1	24.3	360	2077	2100	7.46	.35	.46	22.6	.87	205.5	28.3	.16	21
CT-1	23.3	362	1891	3069	5.82	.22	.31	27.7	1.36	170.5	21.9	.15	27


CT-1
PRESSURE (psi)
0
50
100
150
200
250
COAL FLOW (TONS PER MIN)
1
2
3
4
5
6
7
8
9
10
WATER FLOW - gpm/100
CONCENTRATION - Wt per cent
SLURRY VELOCITY - Ft/sec
0
10
20
30
40
50
Nozzle - 56 mm
LE - 230 mm
NOZZLE PRESSURE
SLURRY PRESSURE
WEIGHT CONCENTRATION
SLURRY VELOCITY
NOZZLE WATER FLOW
TOTAL WATER FLOW

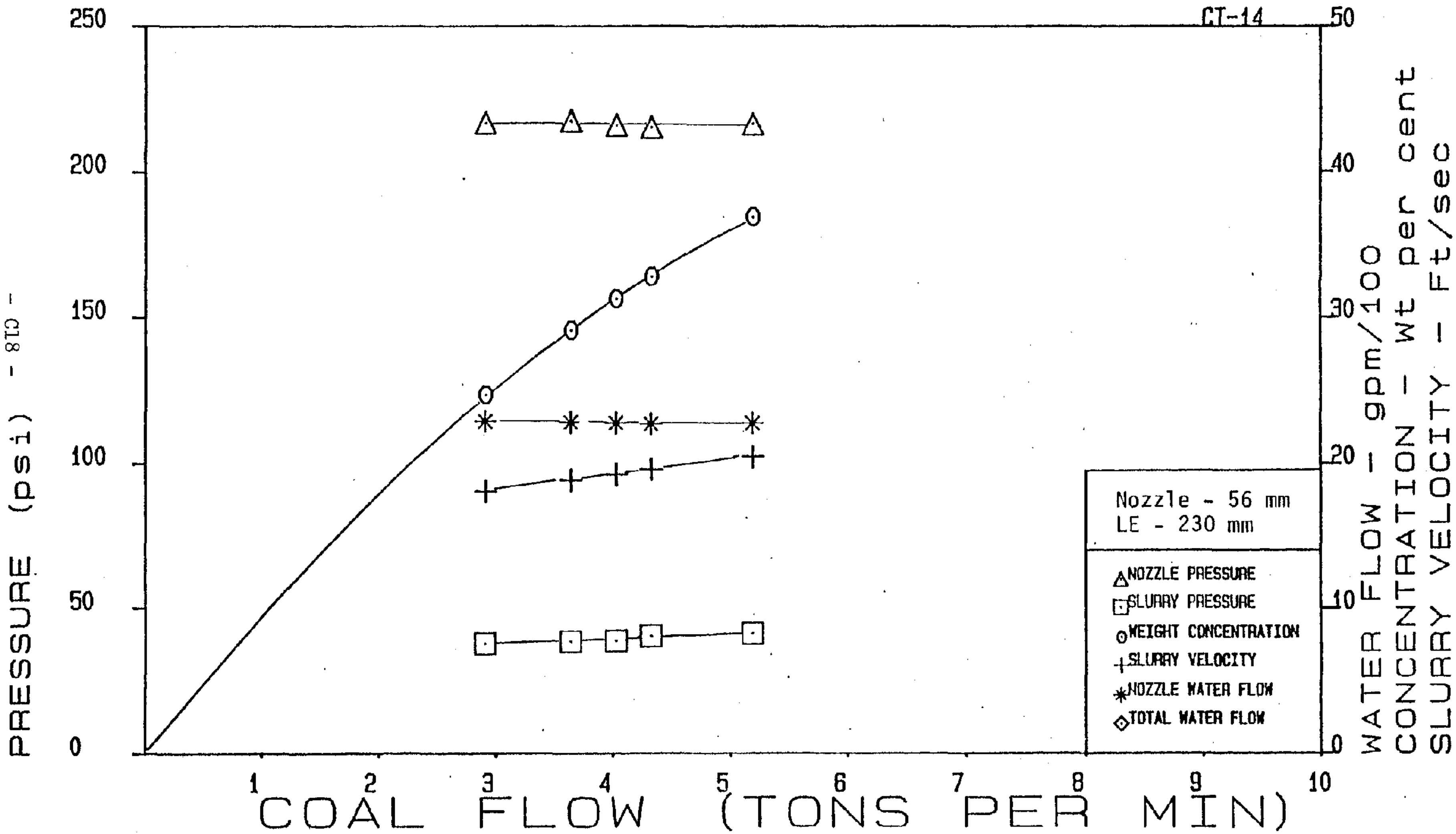
TEST NO.	BELT WT mV	BELT VEL Ft/m	WATER NOZ gpm	WATER TOTAL gpm	COAL TPM	SLURRY CV	SLURRY CW	SLURRY VEL Ft/s	MASS RATIO 240TPM+QH / QN	PRESSURE PN psi	PRESSURE PD psi	PRESSURE PD / PN-PD	EFF PWR OUT / PWR IN
CT-5	0.0	0	1931	2839	0.00	0.00	0.00	20.0	.47	182.0	22.7	.14	18
CT-5	22.3	358	1944	2771	4.12	.18	.26	23.8	.93	184.6	23.6	.15	22
CT-5	21.0	358	1962	2719	1.98	.10	.15	21.2	.63	188.2	24.4	.15	20
CT-5	23.0	355	2044	2237	5.23	.26	.36	21.2	.71	205.9	28.2	.16	20
CT-5	23.8	355	2061	2188	6.53	.31	.42	22.3	.82	207.6	30.2	.17	22
CT-5	24.4	352	1912	2932	7.45	.28	.38	28.5	1.47	176.7	23.0	.15	28
CT-5	0.0	0	1905	3042	0.00	0.00	0.00	21.4	.60	171.2	21.9	.15	20


CT-5
PRESSURE (psi)
0
50
100
150
200
250
WATER FLOW - gpm/100
CONCENTRATION - Wt per cent
SLURRY VELOCITY - Ft/sec
0
10
20
30
40
50
COAL FLOW (TONS PER MIN)
1
2
3
4
5
6
7
8
9
10
Nozzle - 56 mm
LE - 230 mm
NOZZLE PRESSURE
SLURRY PRESSURE
WEIGHT CONCENTRATION
SLURRY VELOCITY
NOZZLE WATER FLOW
TOTAL WATER FLOW

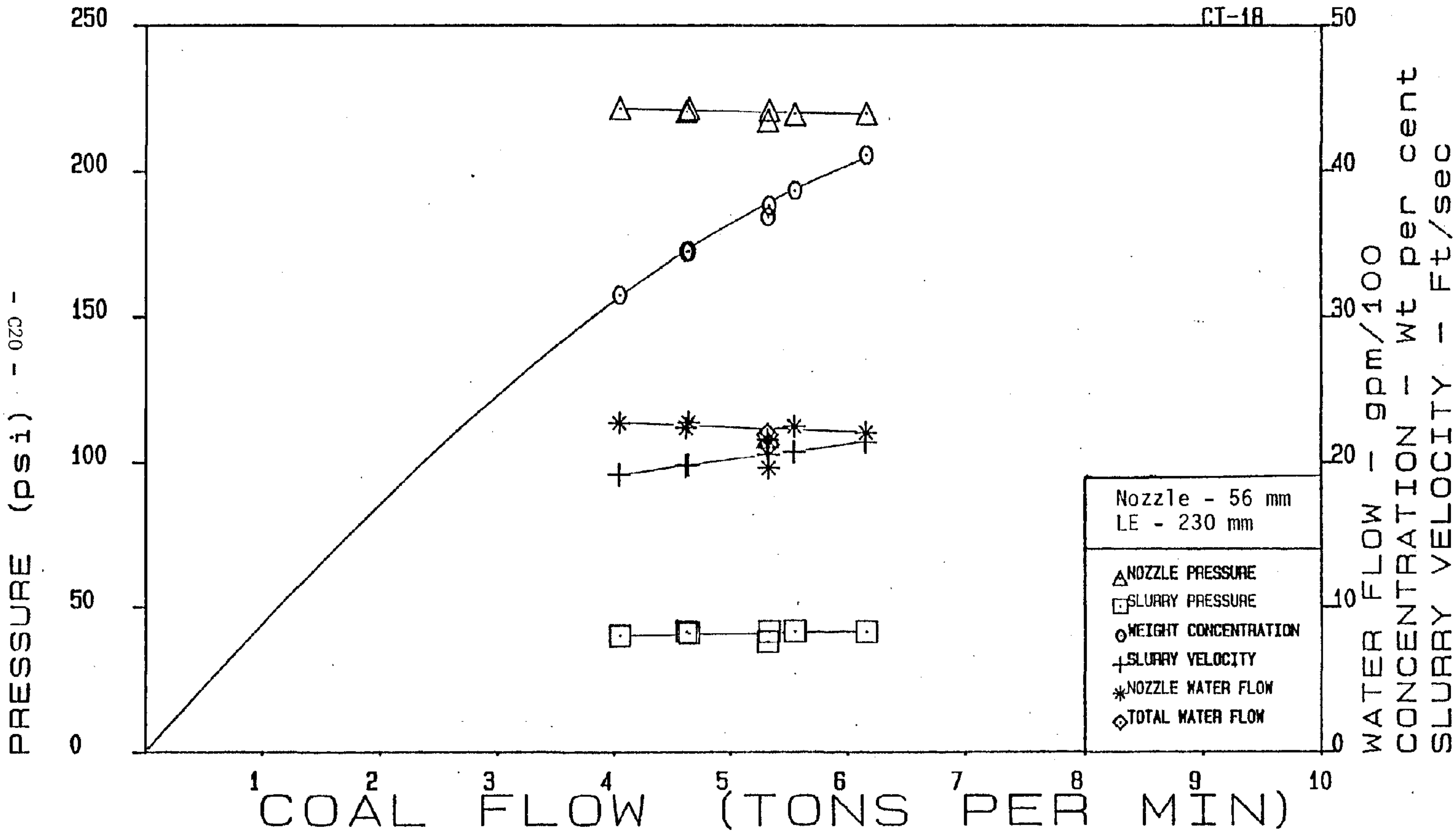
TEST NO.	BELT WT mV	BELT VEL Ft/m	WATER NOZ gpm	WATER TOTAL gpm	COAL TPM	SLURRY CV	SLURRY CW	SLURRY VEL Ft/s	MASS RATIO 240TPM+QH / QN	PRESSURE PN psi	PRESSURE PD psi	PRESSURE PD / PN-PD	EFF PWR OUT / PWR IN
CT-6	22.9	341	1979	2647	4.86	.22	.31	23.7	.93	195.0	25.2	.15	22
CT-6	21.6	343	1998	2528	2.84	.14	.21	20.8	.61	201.3	26.1	.15	19
CT-6	21.2	295	2326	2851	1.90	.09	.14	22.0	.42	194.9	28.5	.17	20
CT-6	23.5	351	2393	2571	5.97	.26	.36	24.4	.67	205.9	31.9	.18	22
CT-6	23.6	351	1345	1794	6.14	.34	.45	19.1	1.43	126.9	14.2	.13	23
CT-6	22.1	351	2350	2619	3.71	.18	.25	22.3	.49	194.9	30.4	.18	21
CT-6	22.5	352	2287	2827	4.37	.19	.27	24.5	.70	185.3	31.8	.21	26
CT-6	23.0	350	2279	2765	5.15	.22	.31	24.9	.76	185.1	27.2	.17	23
CT-6	22.8	350	2049	2172	4.83	.25	.35	20.4	.63	219.1	28.8	.15	19
CT-6	23.5	355	1969	2564	6.04	.26	.36	24.4	1.04	202.4	25.6	.14	22
CT-6	23.5	355	2035	2201	6.04	.29	.40	21.8	.79	215.1	30.2	.16	21
CT-6	22.5	373	2021	2184	4.63	.24	.34	20.2	.63	215.5	25.5	.13	17
CT-6	22.5	373	669	679	4.63	.51	.62	9.7	1.68	74.6	5.9	.09	16


CT-6
PRESSURE (psi)
WATER FLOW - gpm/100
CONCENTRATION - Wt per cent
SLURRY VELOCITY - Ft/sec
COAL FLOW (TONS PER MIN)
Nozzle - 56 mm
LE - 230 mm
NOZZLE PRESSURE
SLURRY PRESSURE
WEIGHT CONCENTRATION
SLURRY VELOCITY
NOZZLE WATER FLOW
TOTAL WATER FLOW

TEST NO.	BELT WT mV	BELT VEL Ft/m	WATER NOZ gpm	WATER TOTAL gpm	COAL TPM	SLURRY CV	SLURRY CW	SLURRY VEL Ft/s	MASS RATIO 240TPM+QH / QN	PRESSURE PN psi	PRESSURE PD psi	PRESSURE PD / PN-PD	EFF PWR OUT / PWR IN
CT-14	21.8	396	2276	2127	3.64	.20	.29	18.8	.32	217.4	38.5	.21	21
CT-14	22.6	226	2288	2127	2.91	.17	.25	18.0	.24	216.6	37.7	.21	20
CT-14	22.9	364	2275	2129	5.19	.27	.37	20.4	.48	216.3	41.4	.24	24
CT-14	22.4	337	2275	2127	4.03	.22	.31	19.2	.36	215.9	38.6	.22	21
CT-14	22.6	336	2272	2128	4.33	.23	.33	19.5	.39	215.2	40.4	.23	23

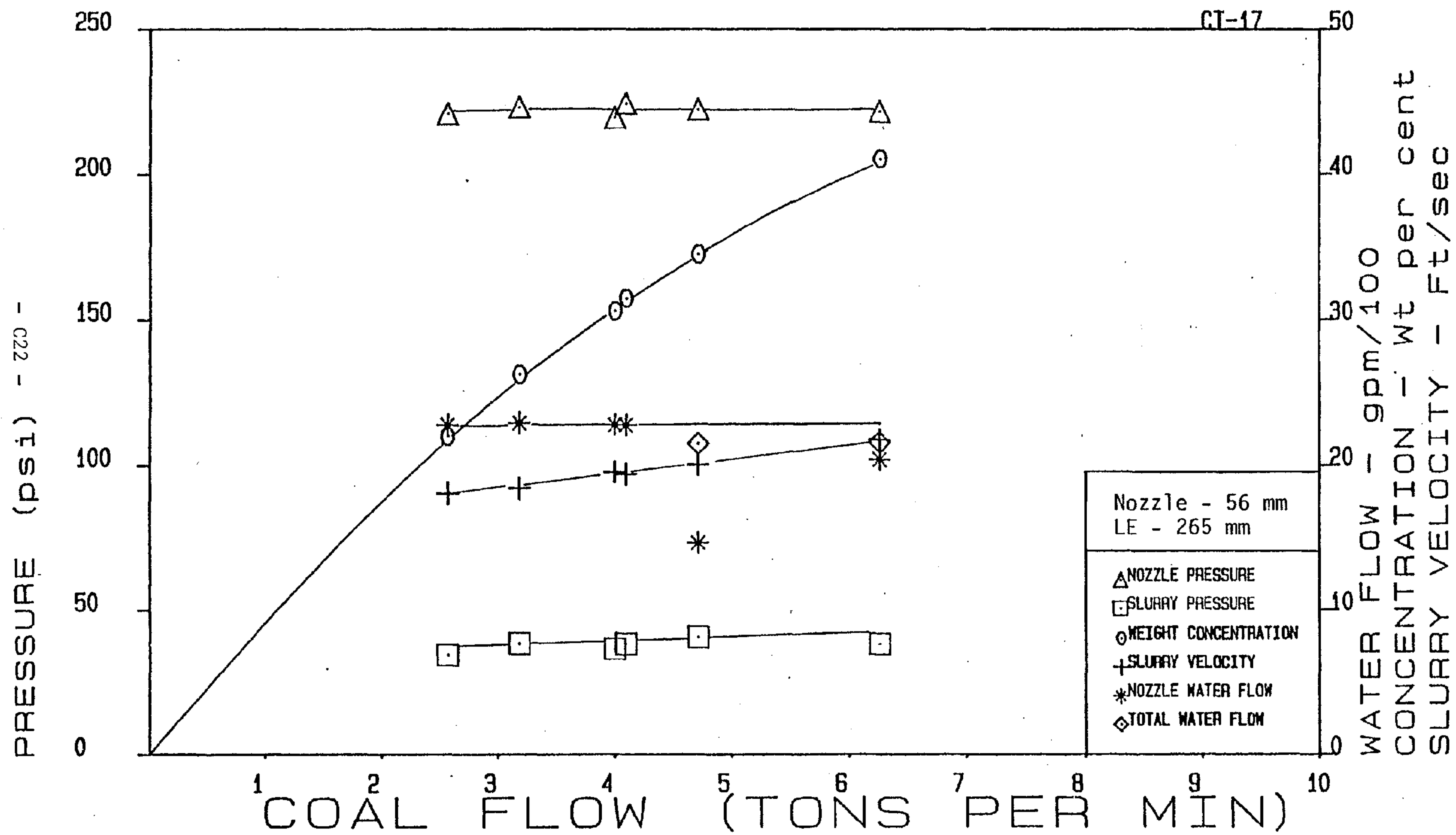

CT-14
250
200
150
100
50
0
50
40
30
20
10
0
PRESSURE (psi)
WATER FLOW - gpm/100
CONCENTRATION - Wt per cent
SLURRY VELOCITY - Ft/sec
1
2
3
4
5
6
7
8
9
10
COAL FLOW (TONS PER MIN)
Nozzle - 56 mm
LE - 230 mm
NOZZLE PRESSURE
SLURRY PRESSURE
WEIGHT CONCENTRATION
SLURRY VELOCITY
NOZZLE WATER FLOW
TOTAL WATER FLOW

TEST	BELT		WATER		COAL	SLURRY			MASS RATIO	PRESSURE			EFF
NO.	WT mV	VEL Ft/m	NOZ gpm	TOTAL gpm	TPM	CV	CW	VEL Ft/s	240TPM+QH / QN	PN psi	PD psi	PD / PN-PD	PWR OUT / PWR IN
CT-18	23.2	340	2151	2186	5.32	.27	.37	21.0	.61	217.2	37.9	.21	24
CT-18	22.5	374	2271	2119	4.65	.25	.34	19.8	.42	221.4	40.9	.23	23
CT-18	22.0	400	2268	2115	4.05	.22	.31	19.1	.36	221.5	40.0	.22	22
CT-18	23.3	331	1957	2117	5.33	.27	.38	20.5	.74	220.8	41.4	.23	28
CT-18	22.8	335	2235	2112	4.62	.25	.34	19.7	.44	220.4	41.5	.23	24
CT-18	23.4	335	2244	2111	5.55	.28	.39	20.7	.53	219.7	41.6	.23	25
CT-18	23.4	372	2204	2116	6.16	.30	.41	21.4	.63	219.9	41.6	.23	26

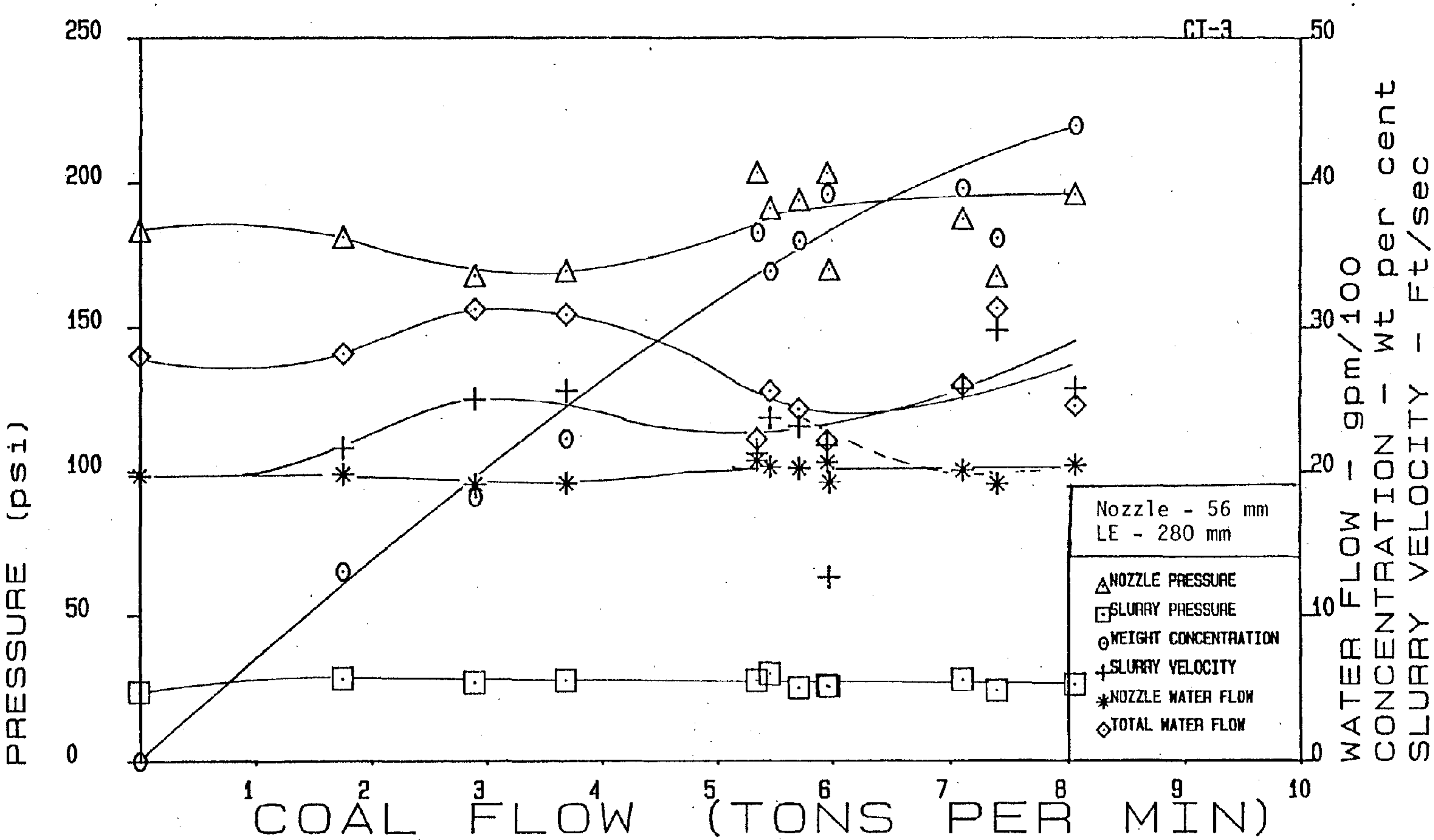


CT-18
PRESSURE (psi) - C20 -
COAL FLOW (TONS PER MIN)
WATER FLOW - gpm/100
CONCENTRATION - Wt per cent
SLURRY VELOCITY - Ft/sec
Nozzle - 56 mm
LE - 230 mm
NOZZLE PRESSURE
SLURRY PRESSURE
WEIGHT CONCENTRATION
SLURRY VELOCITY
NOZZLE WATER FLOW
TOTAL WATER FLOW
0
50
100
150
200
250
10
20
30
40
50
1
2
3
4
5
6
7
8
9
10

TEST NO.	BELT WT mV	BELT VEL Ft/m	WATER NOZ gpm	WATER TOTAL gpm	COAL TPM	SLURRY CV	SLURRY CW	SLURRY VEL Ft/s	MASS RATIO $\frac{240TPM+QH}{QN}$	PRESSURE PN psi	PRESSURE PD psi	PRESSURE $\frac{PD}{PN-PD}$	EFF $\frac{PWR\ OUT}{PWR\ IN}$
CT-17	22.2	371	2277	2139	4.10	.22	.31	19.4	.37	224.3	38.2	.21	21
CT-17	21.3	372	2278	2185	2.57	.15	.22	18.1	.23	220.8	34.3	.18	18
CT-17	22.6	247	2292	2141	3.18	.18	.26	18.4	.27	223.1	38.3	.21	20
CT-17	23.5	368	2037	2155	6.26	.30	.41	21.8	.80	221.5	38.1	.21	26
CT-17	23.5	235	2279	2177	4.00	.22	.31	19.5	.38	219.5	36.4	.20	20
CT-17	22.8	366	1459	2147	4.71	.25	.35	20.1	1.25	222.3	40.6	.22	36

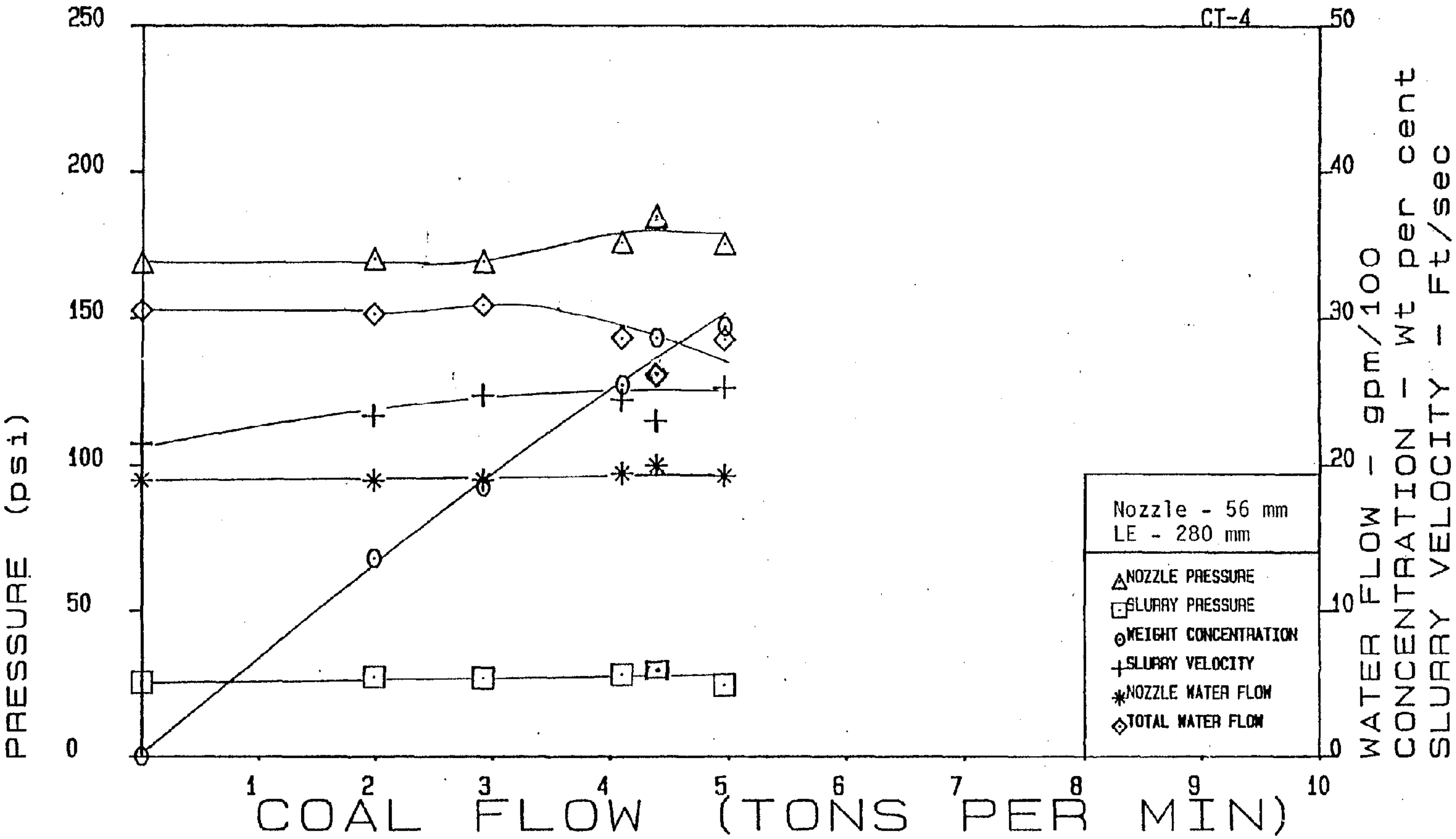

CT-17
PRESSURE (psi)
250
200
150
100
50
0
WATER FLOW - gpm/100
CONCENTRATION - Wt per cent
SLURRY VELOCITY - Ft/sec
50
40
30
20
10
0
1
2
3
4
5
6
7
8
9
10
COAL FLOW (TONS PER MIN)
Nozzle - 56 mm
LE - 265 mm
NOZZLE PRESSURE
SLURRY PRESSURE
WEIGHT CONCENTRATION
SLURRY VELOCITY
NOZZLE WATER FLOW
TOTAL WATER FLOW

TEST	BELT		WATER		COAL	SLURRY			MASS RATIO	PRESSURE			EFF
NO.	WT mV	VEL Ft/m	NOZ gpm	TOTAL gpm	TPM	CV	CW	VEL Ft/s	240TPM+QH / QN	PN psi	PD psi	PD / PN-PD	PWR OUT / PWR IN
CT-3	0.0	0	1971	2796	0.00	0.00	0.00	19.7	.42	183.3	24.0	.15	19
CT-3	23.3	332	2079	2222	5.35	.27	.37	21.3	.69	203.8	27.9	.16	20
CT-3	25.2	325	2046	2459	8.06	.33	.44	25.8	1.15	196.3	26.5	.16	24
CT-3	24.5	329	2009	2597	7.10	.29	.40	25.7	1.14	187.5	27.9	.18	27
CT-3	22.2	334	1919	3086	3.69	.15	.22	25.6	1.07	169.3	27.9	.20	31
CT-3	22.6	225	1912	3121	2.90	.12	.18	25.0	1.00	167.7	27.2	.19	30
CT-3	21.4	238	1981	2811	1.75	.09	.13	21.6	.63	181.1	28.5	.19	24
CT-3	23.3	339	2036	2555	5.46	.24	.34	23.7	.90	191.2	30.2	.19	26
CT-3	23.0	388	2025	2434	5.71	.26	.36	23.1	.88	194.3	25.3	.15	21
CT-3	23.7	332	2067	2212	5.96	.29	.39	21.8	.76	203.5	26.0	.15	19
CT-3	24.5	342	1917	3130	7.39	.26	.36	29.8	1.56	167.5	24.3	.17	32
CT-3	23.7	333	1927	2820	5.97	.50	.62	12.6	.21	169.9	25.7	.18	14

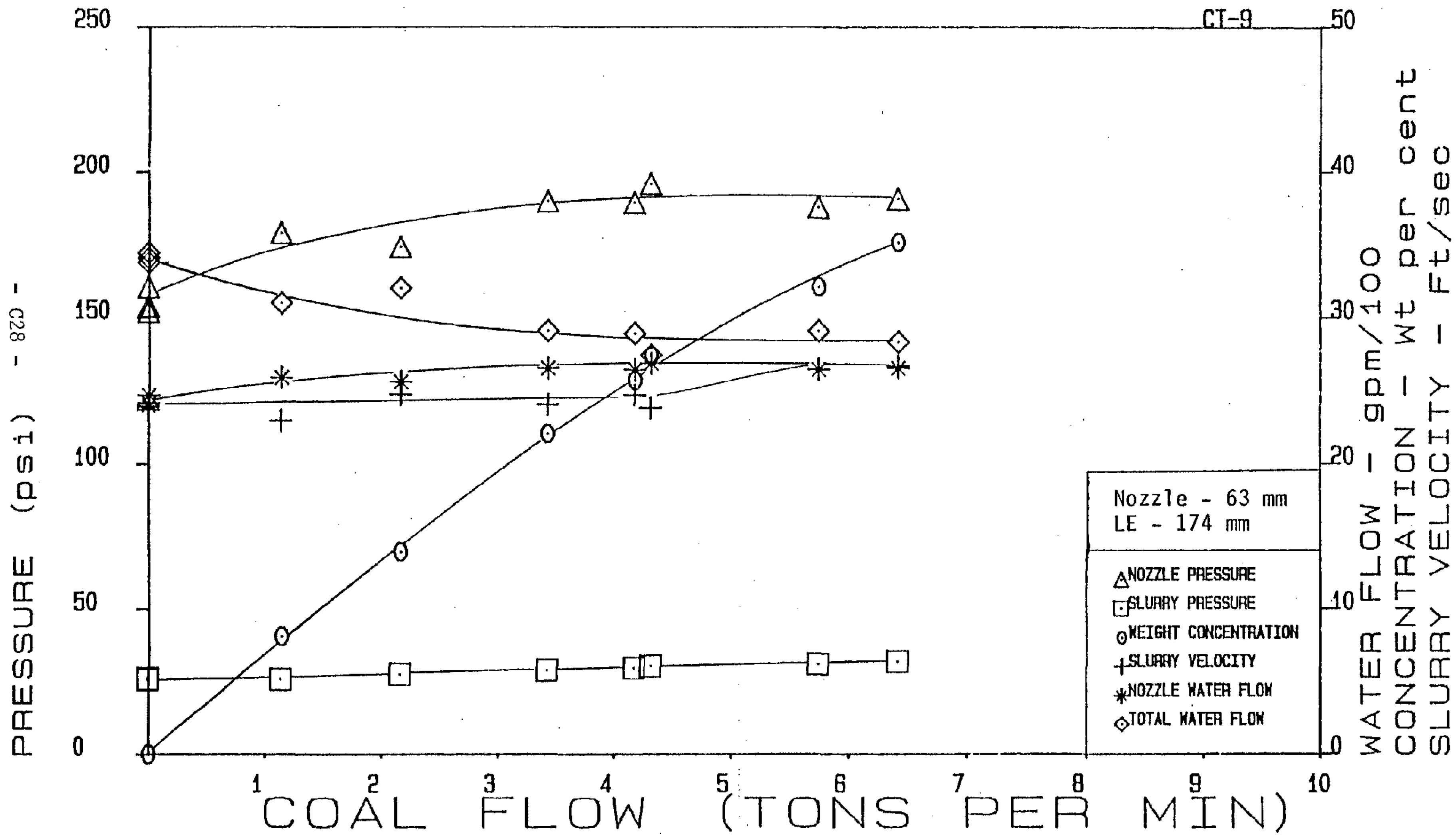


CT-3
PRESSURE (psi)
COAL FLOW (TONS PER MIN)
WATER FLOW - gpm/100
CONCENTRATION - wt per cent
SLURRY VELOCITY - Ft/sec
Nozzle - 56 mm
LE - 280 mm
NOZZLE PRESSURE
SLURRY PRESSURE
WEIGHT CONCENTRATION
SLURRY VELOCITY
NOZZLE WATER FLOW
TOTAL WATER FLOW

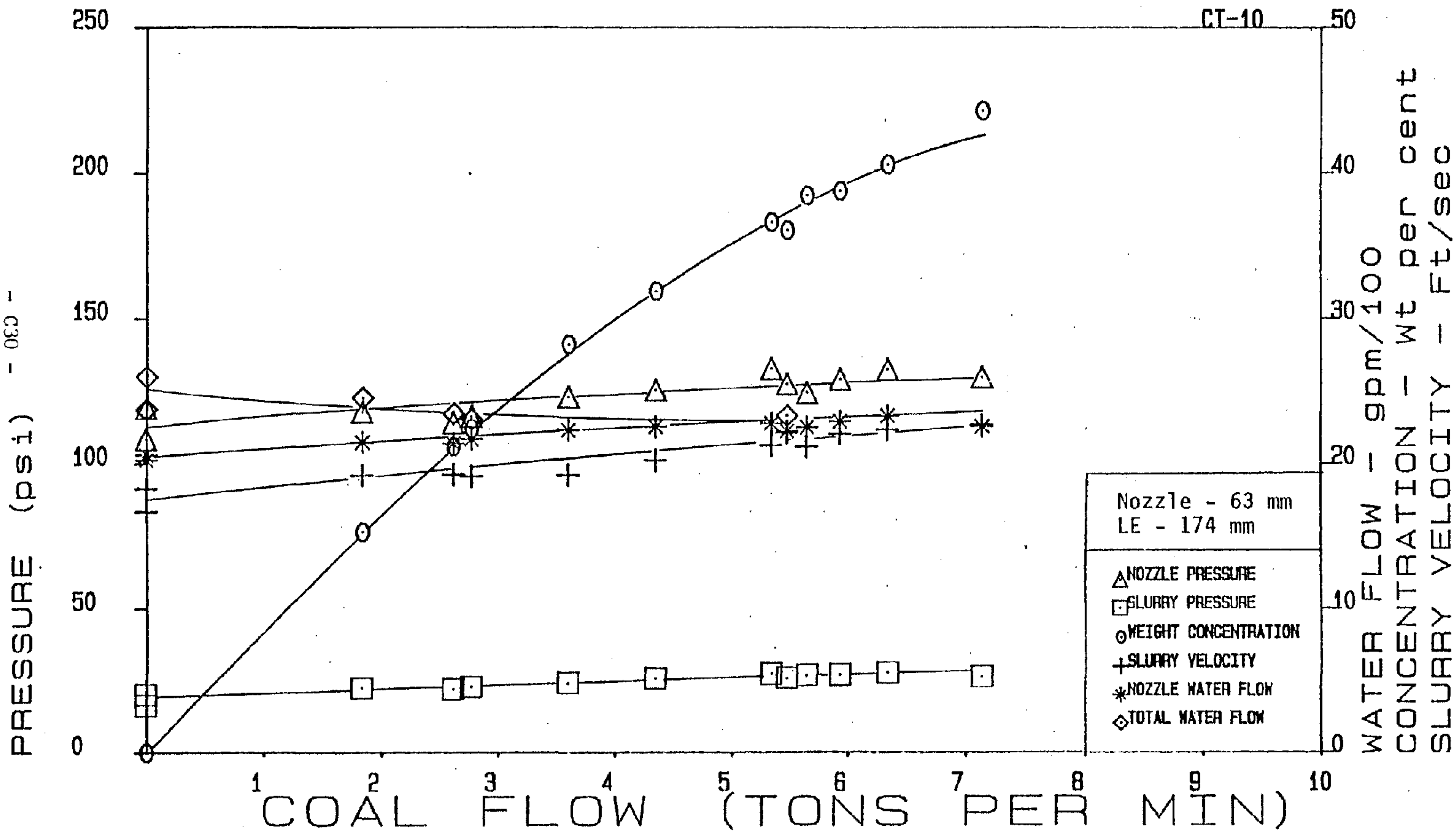
TEST NO.	BELT WT mV	BELT VEL Ft/m	WATER NOZ gpm	WATER TOTAL gpm	COAL TPM	SLURRY CV	SLURRY CW	SLURRY VEL Ft/s	MASS RATIO $\frac{240TPM+QH}{QN}$	PRESSURE PN psi	PRESSURE PD psi	PRESSURE $\frac{PD}{PN-PD}$	EFF $\frac{PWR\ OUT}{PWR\ IN}$
CT-4	21.9	302	1894	3085	2.92	.12	.18	24.8	1.00	169.3	26.8	.19	29
CT-4	23.1	327	1924	2854	4.96	.21	.29	25.3	1.10	175.5	24.6	.16	26
CT-4	0.0	0	1897	3049	0.00	0.00	0.00	21.4	.61	169.0	25.4	.18	24
CT-4	21.1	332	1894	3026	1.99	.09	.14	23.4	.85	170.3	27.3	.19	28
CT-4	22.5	330	1943	2869	4.10	.18	.26	24.5	.98	176.1	28.3	.19	29
CT-4	22.7	329	1996	2616	4.39	.20	.29	23.0	.84	185.0	29.0	.19	26
CT-4			1898	2470				22.0		166.8	27.0	.19	27
CT-4			1730	1868				17.8		133.4	23.5	.21	26
CT-4			1474	1549				15.5		91.0	20.8	.30	34


CT-4
250
200
150
100
50
0
PRESSURE (psi)
50
40
30
20
10
0
WATER FLOW - gpm/100
CONCENTRATION - wt per cent
SLURRY VELOCITY - Ft/sec
1
2
3
4
5
6
7
8
9
10
COAL FLOW (TONS PER MIN)
Nozzle - 56 mm
LE - 280 mm
NOZZLE PRESSURE
SLURRY PRESSURE
WEIGHT CONCENTRATION
SLURRY VELOCITY
NOZZLE WATER FLOW
TOTAL WATER FLOW

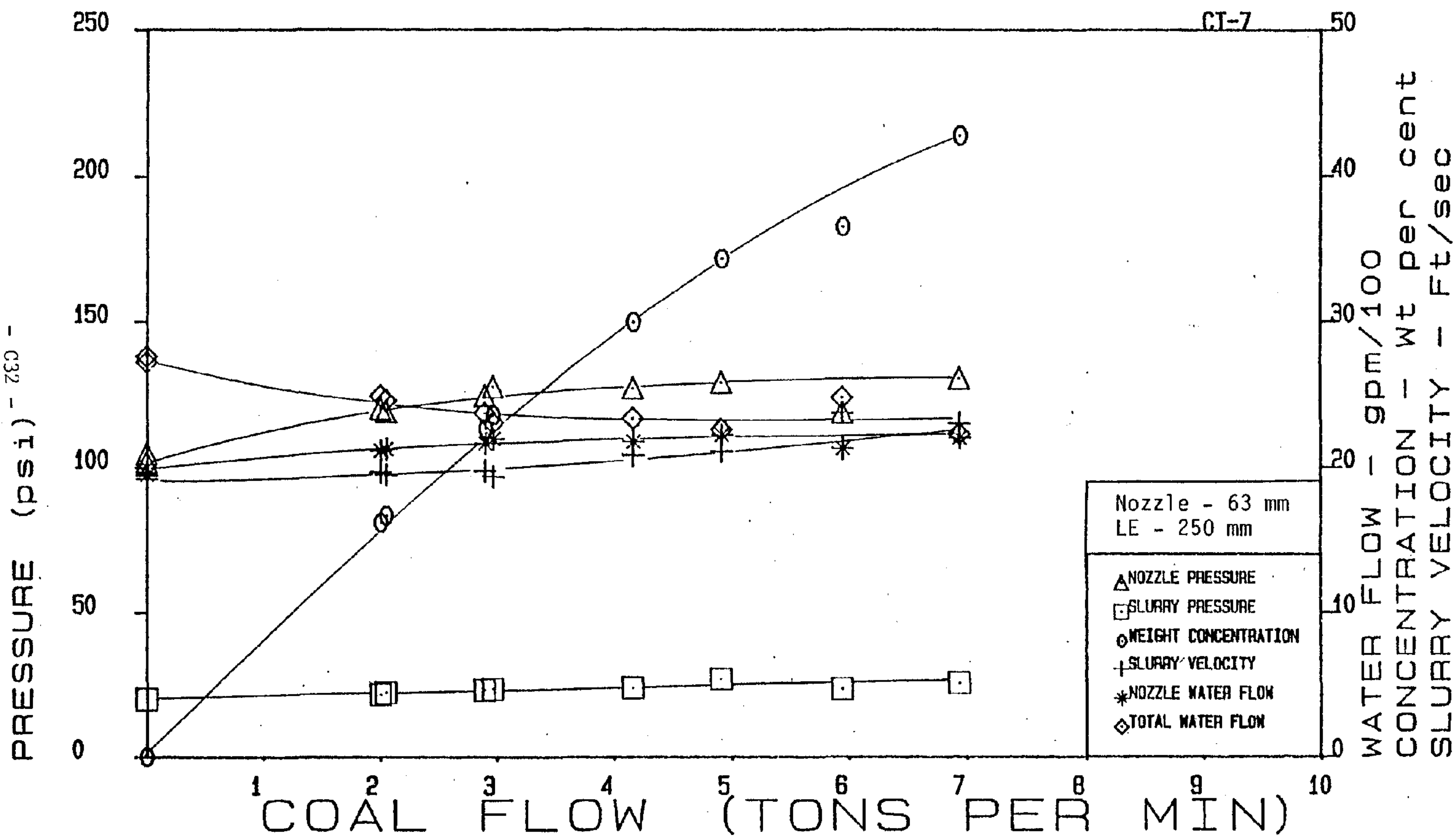
TEST NO.	BELT WT mV	BELT VEL Ft/m	WATER NOZ gpm	WATER TOTAL gpm	COAL TPM	SLURRY CV	SLURRY CW	SLURRY VEL Ft/s	MASS RATIO 240TPM+QH / QN	PRESSURE PN psi	PRESSURE PD psi	PRESSURE PD / PN-PD	EFF PWR OUT / PWR IN
CT-9	0.0	0	2474	3442	0.00	0.00	0.00	24.2	.39	160.5	26.1	.19	23
CT-9	21.0	207	2594	3098	1.14	.05	.08	23.0	.30	179.2	25.8	.17	18
CT-9	21.1	362	2566	3205	2.16	.09	.14	24.8	.45	174.5	27.5	.19	22
CT-9	23.3	357	2648	2910	5.75	.23	.32	26.5	.62	188.2	30.9	.20	23
CT-9	22.4	287	2657	2910	3.43	.15	.22	24.1	.41	189.9	28.7	.18	19
CT-9	22.1	395	2648	2892	4.18	.18	.26	24.7	.47	189.6	29.6	.18	21
CT-9	22.4	361	2692	2745	4.32	.19	.27	23.8	.40	196.2	30.2	.18	19
CT-9	23.7	358	2658	2835	6.42	.25	.35	26.7	.65	190.9	31.8	.20	24
CT-9	0.0	0	2401	3407	0.00	0.00	0.00	24.0	.42	152.0	25.7	.20	24
CT-9	0.0	0	2419	3376	0.00	0.00	0.00	23.7	.40	153.9	25.7	.20	23


CT-9
Nozzle - 63 mm
LE - 174 mm
NOZZLE PRESSURE
SLURRY PRESSURE
WEIGHT CONCENTRATION
SLURRY VELOCITY
NOZZLE WATER FLOW
TOTAL WATER FLOW
PRESSURE (psi)
COAL FLOW (TONS PER MIN)
WATER FLOW - gpm/100
CONCENTRATION - Wt per cent
SLURRY VELOCITY - Ft/sec
0
50
100
150
200
250
1
2
3
4
5
6
7
8
9
10
10
20
30
40
50

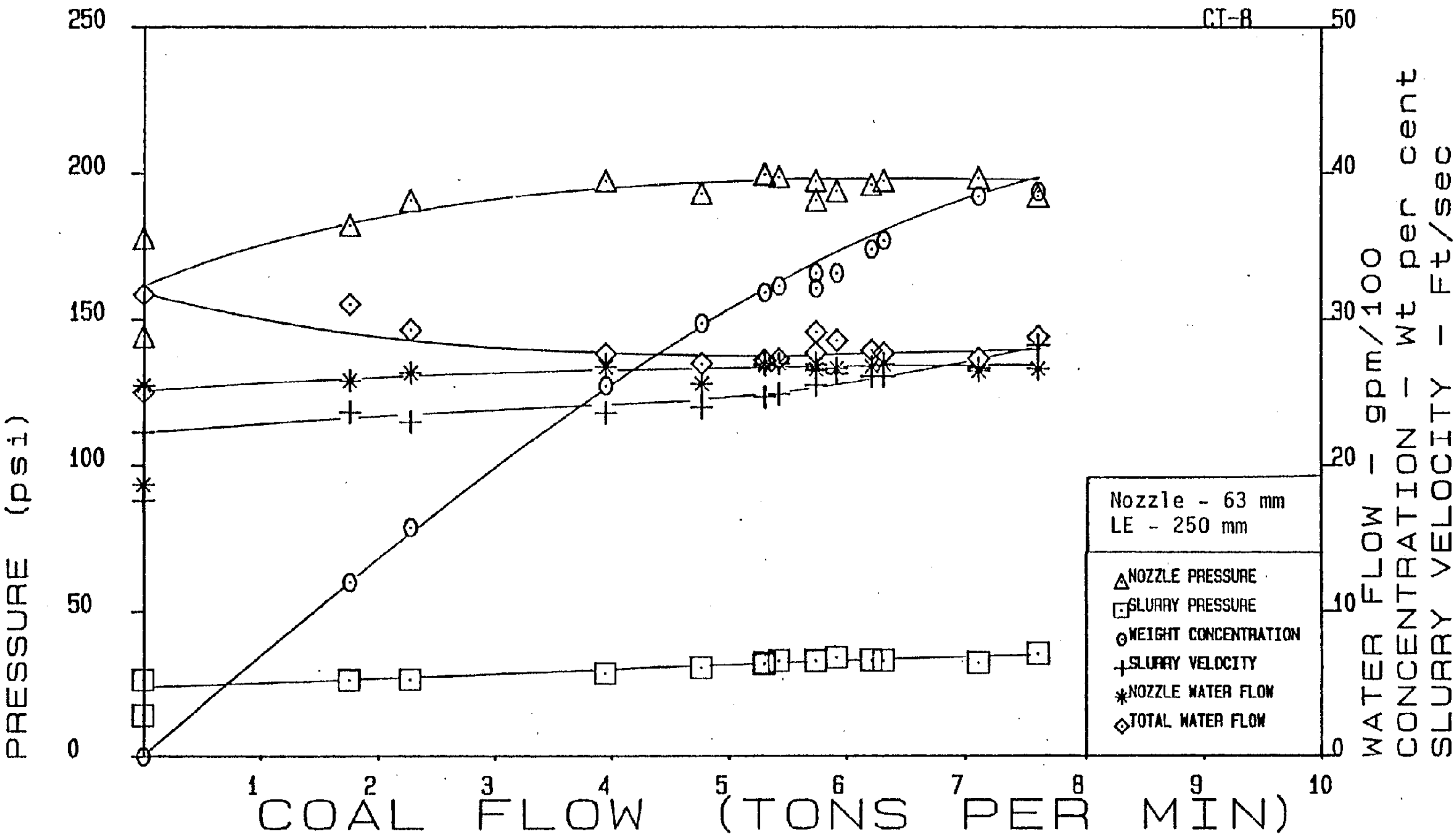
TEST NO.	BELT WT mV	BELT VEL Ft/m	WATER NOZ gpm	WATER TOTAL gpm	COAL TPM	SLURRY CV	SLURRY CW	SLURRY VEL Ft/s	MASS RATIO 240TPM+QH / QN	PRESSURE PN psi	PRESSURE PD psi	PRESSURE PD / PN-PD	EFF PWR OUT / PWR IN
CT-10	0.0	0	2060	2600	0.00	0.00	0.00	18.3	.26	108.4	20.2	.23	24
CT-10	23.1	361	2224	2328	5.48	.26	.36	22.1	.64	127.5	26.2	.26	29
CT-10	20.9	364	2147	2452	1.84	.10	.15	19.2	.35	117.8	22.5	.24	24
CT-10	0.0	0	2024	2371	0.00	0.00	0.00	16.7	.17	119.2	16.5	.16	16
CT-10	22.4	364	2256	2227	4.35	.23	.32	20.2	.45	125.6	26.1	.26	27
CT-10	21.4	356	2137	2344	2.62	.14	.21	19.2	.39	114.1	22.1	.24	25
CT-10	21.5	355	2170	2305	2.78	.15	.22	19.1	.37	116.2	23.0	.25	25
CT-10	22.0	356	2234	2196	3.60	.20	.28	19.2	.37	123.2	24.4	.25	24
CT-10	23.3	351	2251	2166	5.65	.28	.39	21.2	.56	124.7	27.1	.28	29
CT-10	24.3	345	2256	2157	7.14	.33	.44	22.7	.72	130.0	26.6	.26	29
CT-10	23.1	352	2275	2220	5.34	.27	.37	21.2	.54	132.7	27.4	.26	27
CT-10	22.9	445	2330	2229	6.35	.30	.41	22.4	.61	132.5	28.0	.27	29
CT-10	22.7	445	2292	2250	5.94	.28	.39	22.1	.60	129.2	27.2	.27	29


CT-10
PRESSURE (psi)
COAL FLOW (TONS PER MIN)
WATER FLOW - gpm/100
CONCENTRATION - Wt per cent
SLURRY VELOCITY - Ft/sec
Nozzle - 63 mm
LE - 174 mm
NOZZLE PRESSURE
SLURRY PRESSURE
WEIGHT CONCENTRATION
SLURRY VELOCITY
NOZZLE WATER FLOW
TOTAL WATER FLOW

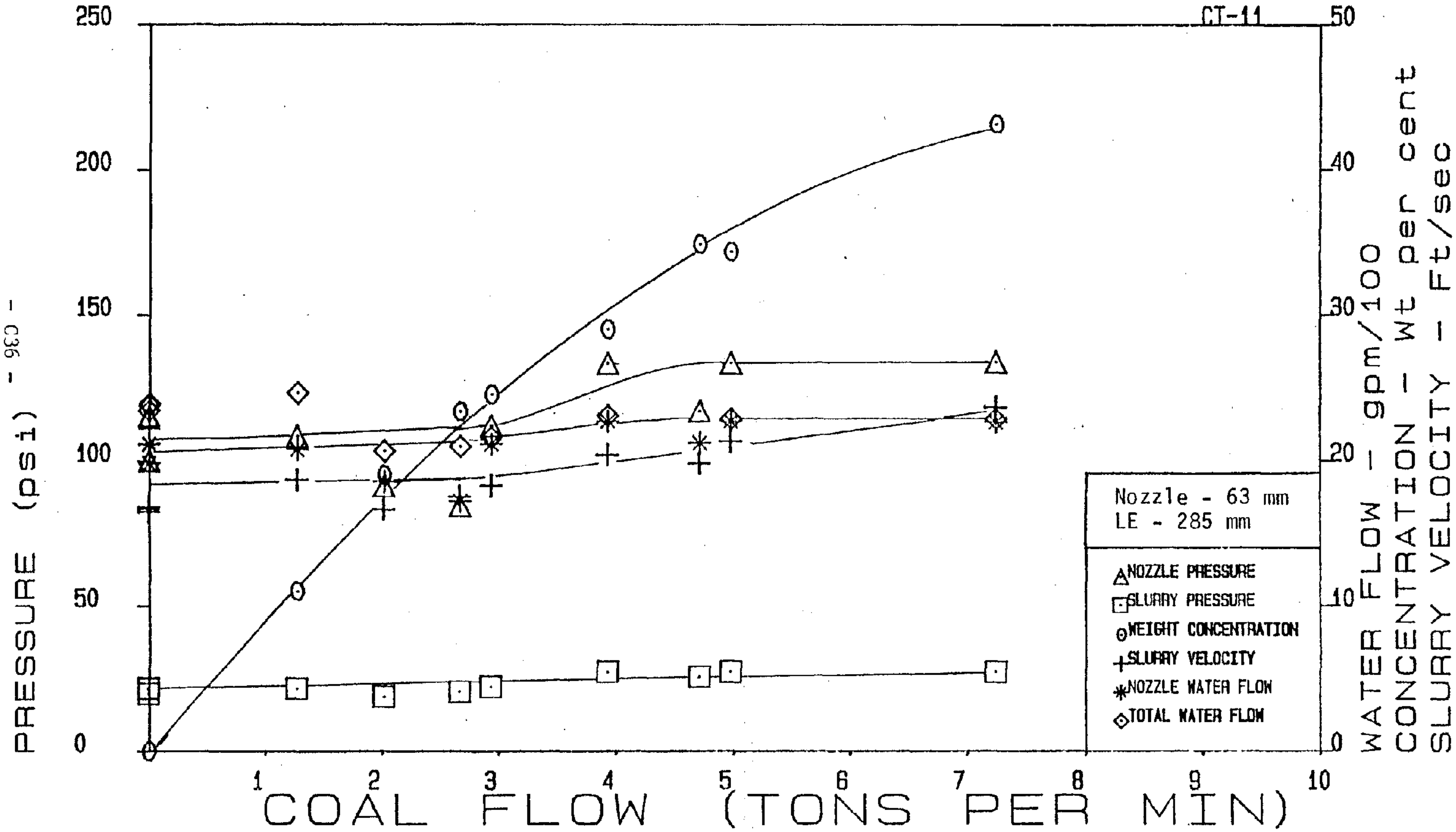
TEST NO.	BELT WT mV	BELT VEL Ft/m	WATER NOZ gpm	WATER TOTAL gpm	COAL TPM	SLURRY CV	SLURRY CW	SLURRY VEL Ft/s	MASS RATIO 240TPM+QH / QN	PRESSURE PN psi	PRESSURE PD psi	PRESSURE PD / PN-PD	EFF PWR OUT / PWR IN
CT-7	0.0	0	1948	2763	0.00	0.00	0.00	19.4	.42	102.0	20.3	.25	28
CT-7	23.4	359	2128	2478	5.95	.26	.37	23.7	.83	118.7	23.7	.25	32
CT-7	22.8	356	2220	2256	4.91	.25	.34	21.0	.55	129.2	27.0	.26	28
CT-7	0.0	0	1999	2723	0.00	0.00	0.00	19.1	.36	104.6	20.0	.24	26
CT-7	21.6	358	2195	2304	2.96	.16	.24	19.3	.37	127.7	23.6	.23	23
CT-7	22.0	286	2164	2371	2.89	.15	.23	19.7	.42	124.7	23.3	.23	24
CT-7	21.3	297	2124	2458	2.05	.11	.17	19.4	.39	119.1	22.4	.23	24
CT-7	21.1	335	2112	2489	2.00	.11	.16	19.6	.41	120.1	21.7	.22	24
CT-7	22.5	335	2175	2333	4.16	.21	.30	20.8	.53	127.3	24.1	.23	26
CT-7	24.4	328	2202	2228	6.94	.32	.43	23.0	.77	130.5	25.7	.25	29


CT-7
PRESSURE (psi)
250
200
150
100
50
0
WATER FLOW - gpm/100
CONCENTRATION - Wt per cent
SLURRY VELOCITY - Ft/sec
50
40
30
20
10
0
Nozzle - 63 mm
LE - 250 mm
NOZZLE PRESSURE
SLURRY PRESSURE
WEIGHT CONCENTRATION
SLURRY VELOCITY
NOZZLE WATER FLOW
TOTAL WATER FLOW
1
2
3
4
5
6
7
8
9
10
COAL FLOW (TONS PER MIN)

TEST NO.	BELT		WATER		COAL	SLURRY			MASS RATIO	PRESSURE			EFF
	WT mV	VEL Ft/m	NOZ gpm	TOTAL gpm	TPM	CV	CW	VEL Ft/s	240TPM+QH / QN	PN psi	PD psi	PD / PN-PD	PWR OUT / PWR IN
CT-8	0.0	0	1868	2501	0.00	0.00	0.00	17.6	.34	144.4	14.3	.11	13
CT-8	22.6	370	2563	2699	4.77	.21	.30	24.0	.50	193.3	30.7	.19	21
CT-8	24.0	368	2650	2732	7.11	.28	.38	26.7	.68	198.4	32.2	.19	23
CT-8	21.4	309	2635	2925	2.27	.10	.16	23.0	.32	190.6	26.5	.16	17
CT-8	21.0	318	2582	3100	1.76	.08	.12	23.6	.36	182.3	26.3	.17	19
CT-8	22.8	286	2678	2767	3.95	.18	.26	23.6	.39	197.5	28.6	.17	18
CT-8	22.9	372	2689	2718	5.30	.23	.32	24.7	.48	199.8	32.3	.19	21
CT-8	22.4	443	2697	2721	5.30	.23	.32	24.7	.48	199.8	31.6	.19	21
CT-8	0.0	0	2551	3168	0.00	0.00	0.00	22.3	.24	177.8	26.5	.18	19
CT-8	23.3	386	2680	2788	6.21	.25	.35	26.2	.60	196.1	33.2	.20	24
CT-8	23.1	378	2692	2771	5.74	.24	.33	25.5	.54	197.4	33.1	.20	23
CT-8	23.0	402	2666	2857	5.92	.24	.33	26.3	.60	194.0	34.1	.21	25
CT-8	22.6	421	2698	2728	5.42	.23	.32	24.9	.49	198.8	32.9	.20	22
CT-8	23.4	459	2664	2883	7.60	.28	.39	28.3	.77	192.1	35.4	.23	28
CT-8	23.0	390	2659	2912	5.74	.23	.32	26.5	.61	190.8	32.7	.21	24
CT-8	23.2	404	2692	2766	6.32	.26	.35	26.1	.59	197.3	32.9	.20	23

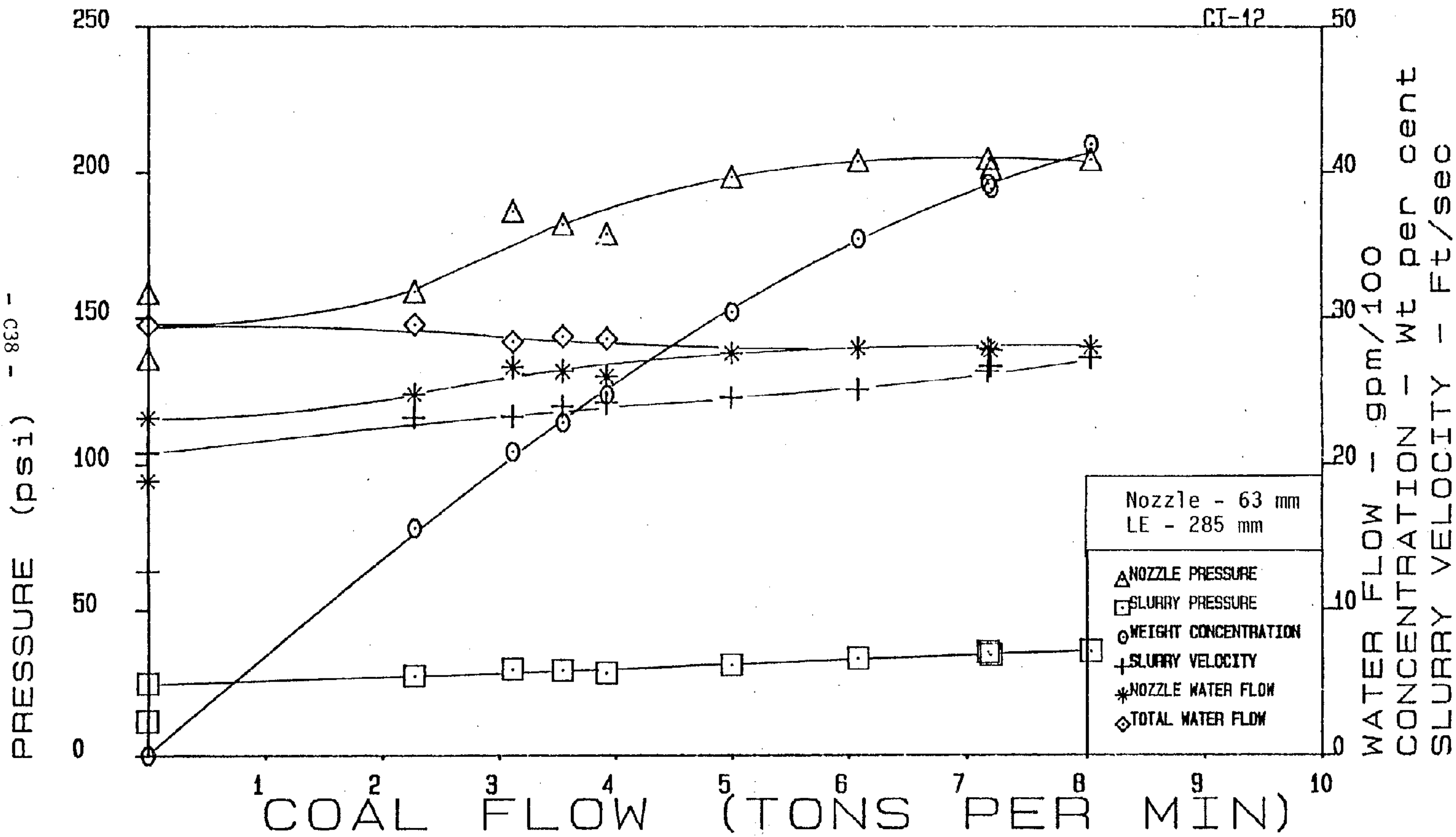

CT-8
Nozzle - 63 mm
LE - 250 mm
NOZZLE PRESSURE
SLURRY PRESSURE
WEIGHT CONCENTRATION
SLURRY VELOCITY
NOZZLE WATER FLOW
TOTAL WATER FLOW
PRESSURE (psi)
COAL FLOW (TONS PER MIN)
WATER FLOW - gpm/100
CONCENTRATION - Wt per cent
SLURRY VELOCITY - Ft/sec
250
200
150
100
50
0
1
2
3
4
5
6
7
8
9
10
50
40
30
20
10
0

TEST	BELT		WATER		COAL	SLURRY			MASS RATIO	PRESSURE			EFF
NO.	WT mV	VEL Ft/m	NOZ gpm	TOTAL gpm	TPM	CV	CW	VEL Ft/s	240TPM+QH / QN	PN psi	PD psi	PD / PN-PD	PWR OUT / PWR IN
CT-11	0.0	0	1981	2345	0.00	0.00	0.00	16.5	.18	100.4	19.7	.24	23
CT-11	0.0	0	1986	2349	0.00	0.00	0.00	16.5	.18	99.7	19.7	.25	23
CT-11	22.1	446	2124	2113	4.72	.25	.35	19.8	.53	117.2	25.6	.28	29
CT-11	22.6	228	2115	2162	2.94	.17	.25	18.3	.36	112.1	22.0	.24	24
CT-11	21.0	366	1857	2065	2.02	.13	.19	16.7	.37	91.4	18.6	.25	26
CT-11	21.0	231	2079	2466	1.28	.07	.11	18.7	.33	108.4	21.5	.25	25
CT-11	21.4	363	1724	2097	2.67	.16	.23	17.6	.59	84.5	20.3	.32	35
CT-11	0.0	0	2114	2395	0.00	0.00	0.00	16.8	.13	114.7	21.4	.23	21
CT-11	24.2	358	2269	2289	7.25	.32	.43	23.7	.78	133.9	27.3	.26	30
CT-11	22.8	361	2273	2288	4.98	.25	.34	21.3	.53	133.5	27.2	.26	27
CT-11	22.8	285	2271	2310	3.93	.20	.29	20.4	.43	133.4	27.1	.26	26
CT-11	0.0	0	2120	2379	0.00	0.00	0.00	16.7	.12	115.6	21.6	.23	21

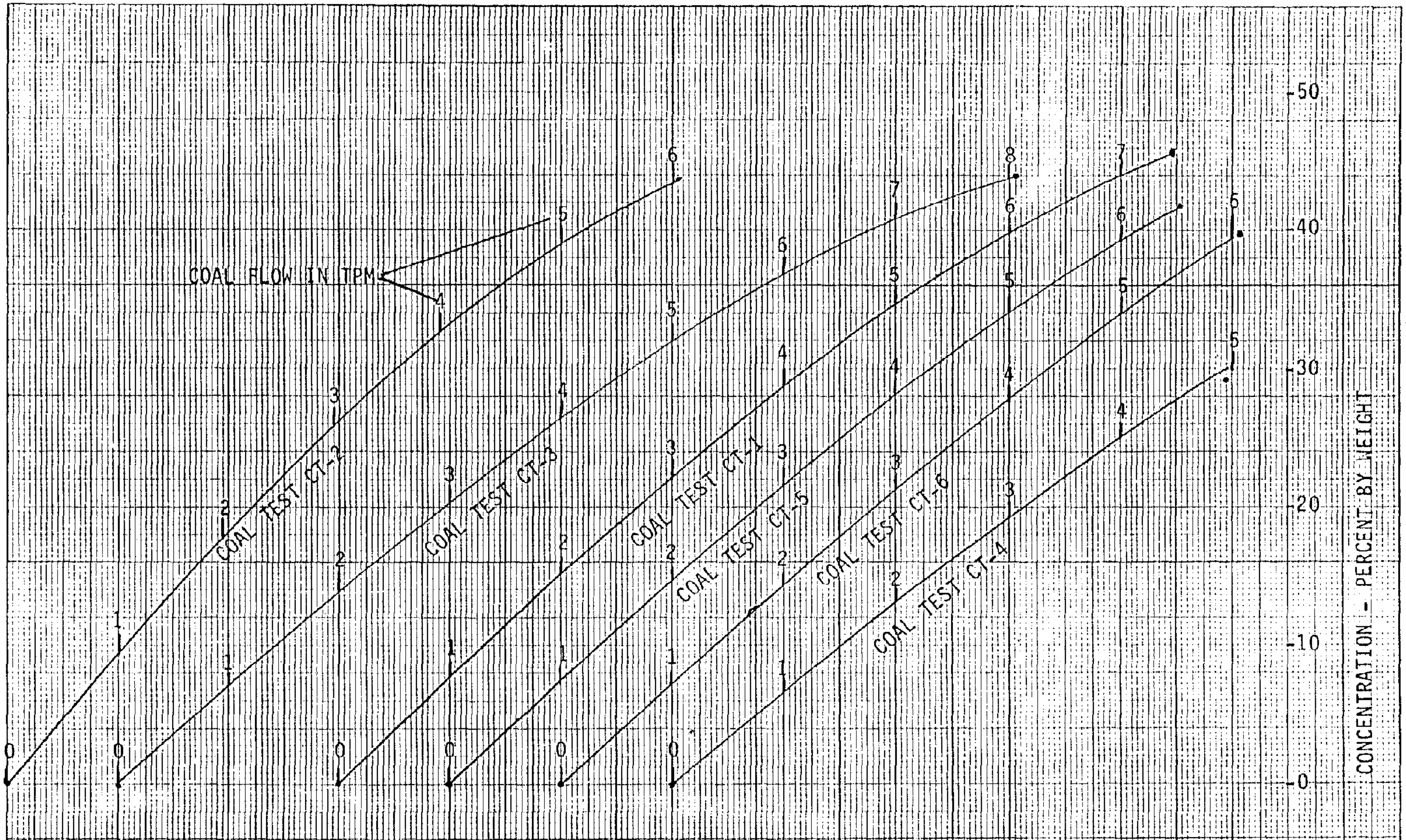


CT-11
PRESSURE (psi)
250
200
150
100
50
0
1
2
3
4
5
6
7
8
9
10
COAL FLOW (TONS PER MIN)
50
40
30
20
10
0
WATER FLOW - gpm/100
CONCENTRATION - Wt per cent
SLURRY VELOCITY - Ft/sec
Nozzle - 63 mm
LE - 285 mm
NOZZLE PRESSURE
SLURRY PRESSURE
WEIGHT CONCENTRATION
SLURRY VELOCITY
NOZZLE WATER FLOW
TOTAL WATER FLOW

TEST NO.	BELT WT mV	BELT VEL Ft/m	WATER NOZ gpm	WATER TOTAL gpm	COAL TPM	SLURRY CV	SLURRY CW	SLURRY VEL Ft/s	MASS RATIO 240TPM+QH / QN	PRESSURE PN psi	PRESSURE PD psi	PRESSURE PD / PN-PD	EFF PWR OUT / PWR IN
CT-12	0.0	0	1886	1799	0.00	0.00	0.00	12.6	-.05	158.6	11.6	.08	7
CT-12	22.1	371	2604	2857	3.93	.17	.25	24.2	.46	178.9	28.4	.19	21
CT-12	22.3	271	2665	2838	3.12	.14	.21	23.2	.35	186.8	29.7	.19	20
CT-12	21.5	291	2481	2955	2.28	.10	.16	23.2	.41	159.1	27.2	.21	23
CT-12	23.0	241	2640	2875	3.55	.16	.23	24.0	.41	182.2	29.3	.19	21
CT-12	22.6	388	2760	2741	5.00	.21	.30	24.5	.43	198.6	31.0	.19	20
CT-12	23.9	382	2779	2713	7.20	.28	.39	26.7	.60	201.4	34.3	.21	23
CT-12	23.3	446	2793	2669	7.18	.29	.39	26.3	.57	204.7	35.2	.21	23
CT-12	23.7	448	2804	2670	8.04	.31	.42	27.2	.64	204.2	35.6	.21	24
CT-12	0.0	0	2319	2949	0.00	0.00	0.00	20.7	.27	136.1	24.7	.22	23
CT-12	23.2	389	2796	2656	6.08	.26	.35	25.1	.47	204.0	33.1	.19	21

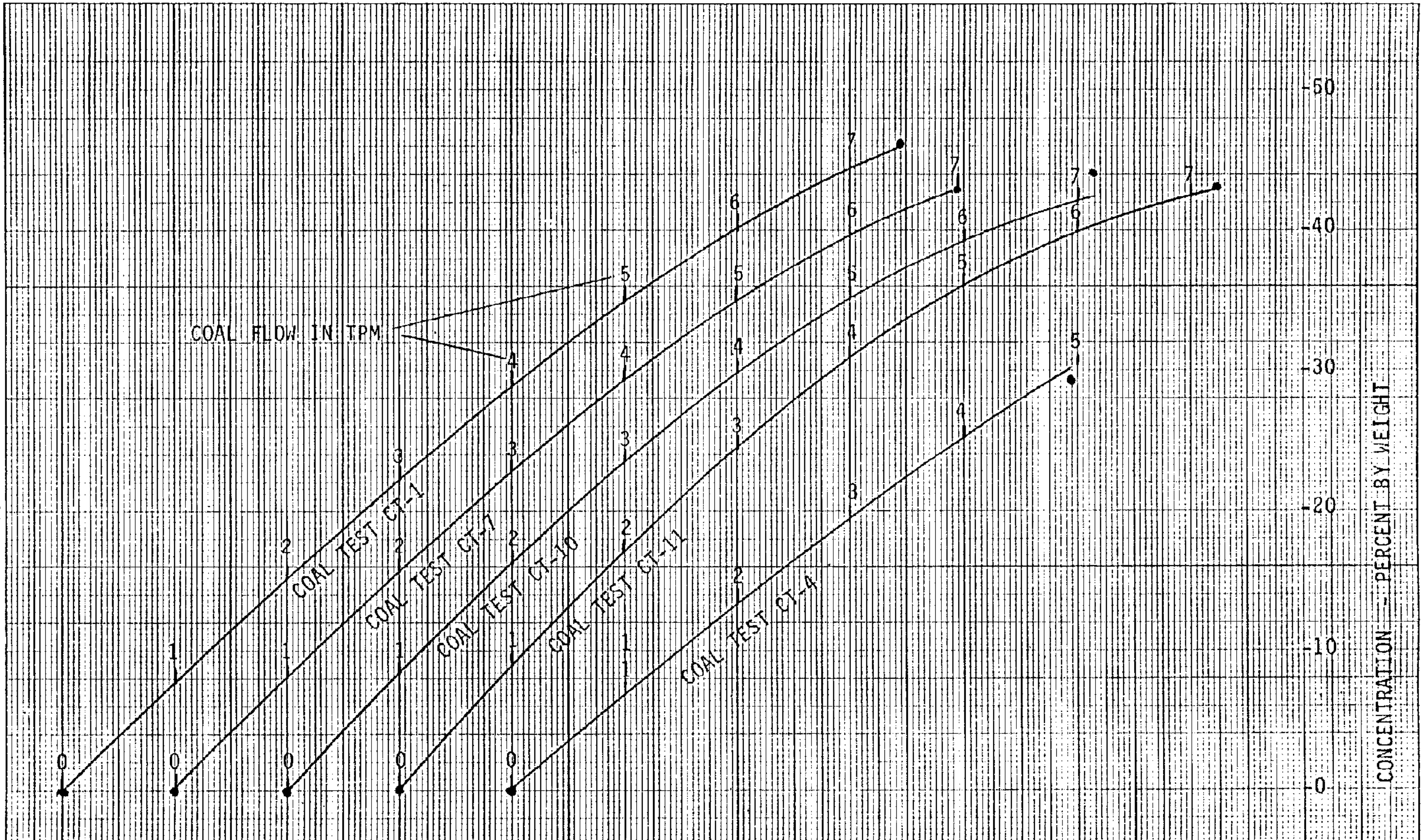

CI-12
PRESSURE (psi)
250
200
150
100
50
0
COAL FLOW (TONS PER MIN)
1
2
3
4
5
6
7
8
9
10
50
40
30
20
10
0
WATER FLOW - gpm/100
CONCENTRATION - Wt per cent
SLURRY VELOCITY - Ft/sec
Nozzle - 63 mm
LE - 285 mm
NOZZLE PRESSURE
SLURRY PRESSURE
WEIGHT CONCENTRATION
SLURRY VELOCITY
NOZZLE WATER FLOW
TOTAL WATER FLOW

- C38 -



SLURRY CONCENTRATION AS A FUNCTION OF COAL FLOW
FOR COAL TESTS CT-2, CT-3, CT-1, CT-5, CT-6, CT-4

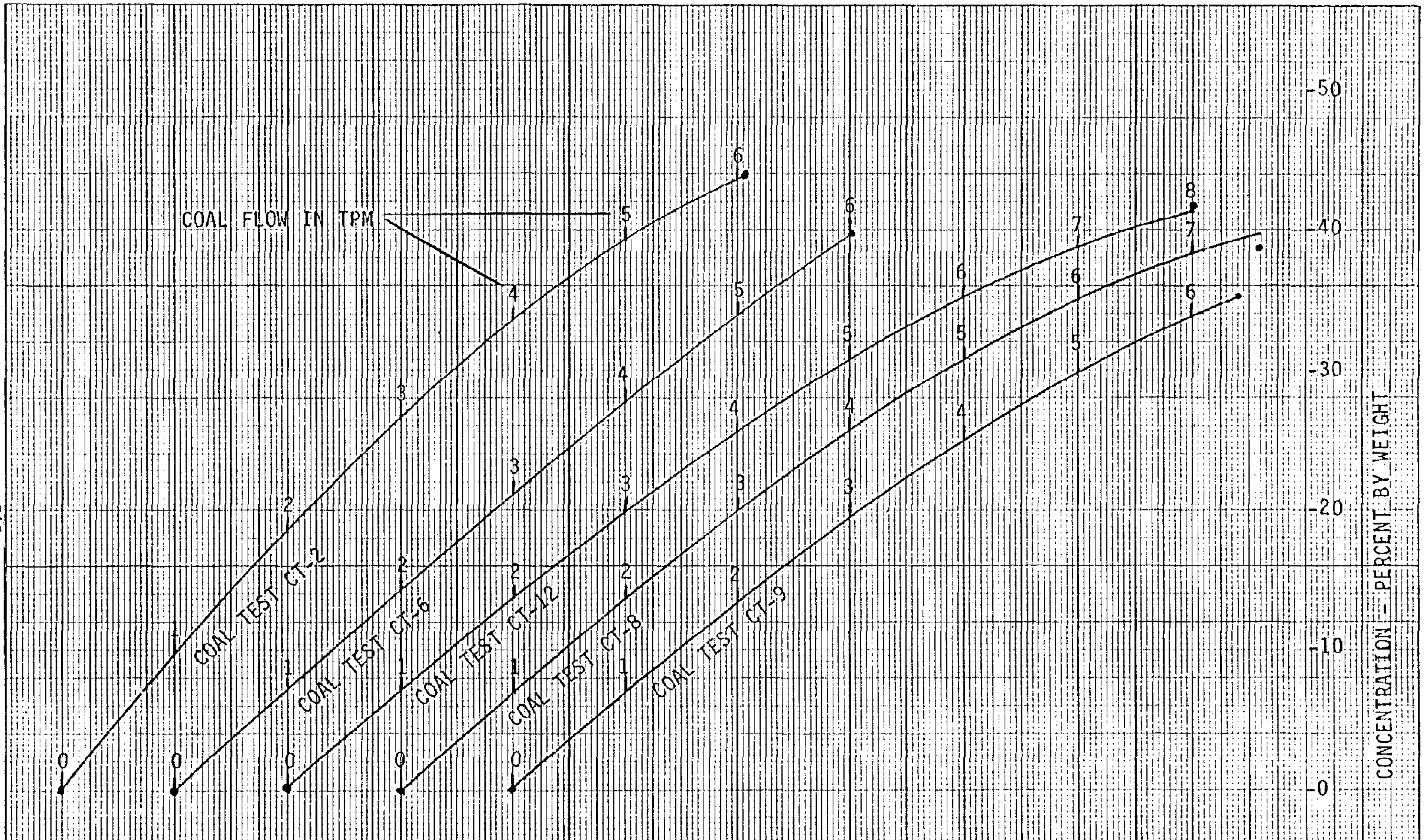
Coal Flow In TPM is shown for Each Test
Data Points are shown at the Start and End of Each Test
For Added Data, Refer to Figures and Tables for Individual Tests



SLURRY CONCENTRATION AS A FUNCTION OF COAL FLOW
FOR COAL TESTS CT-1, CT-7, CT-10, CT-11, CT-14

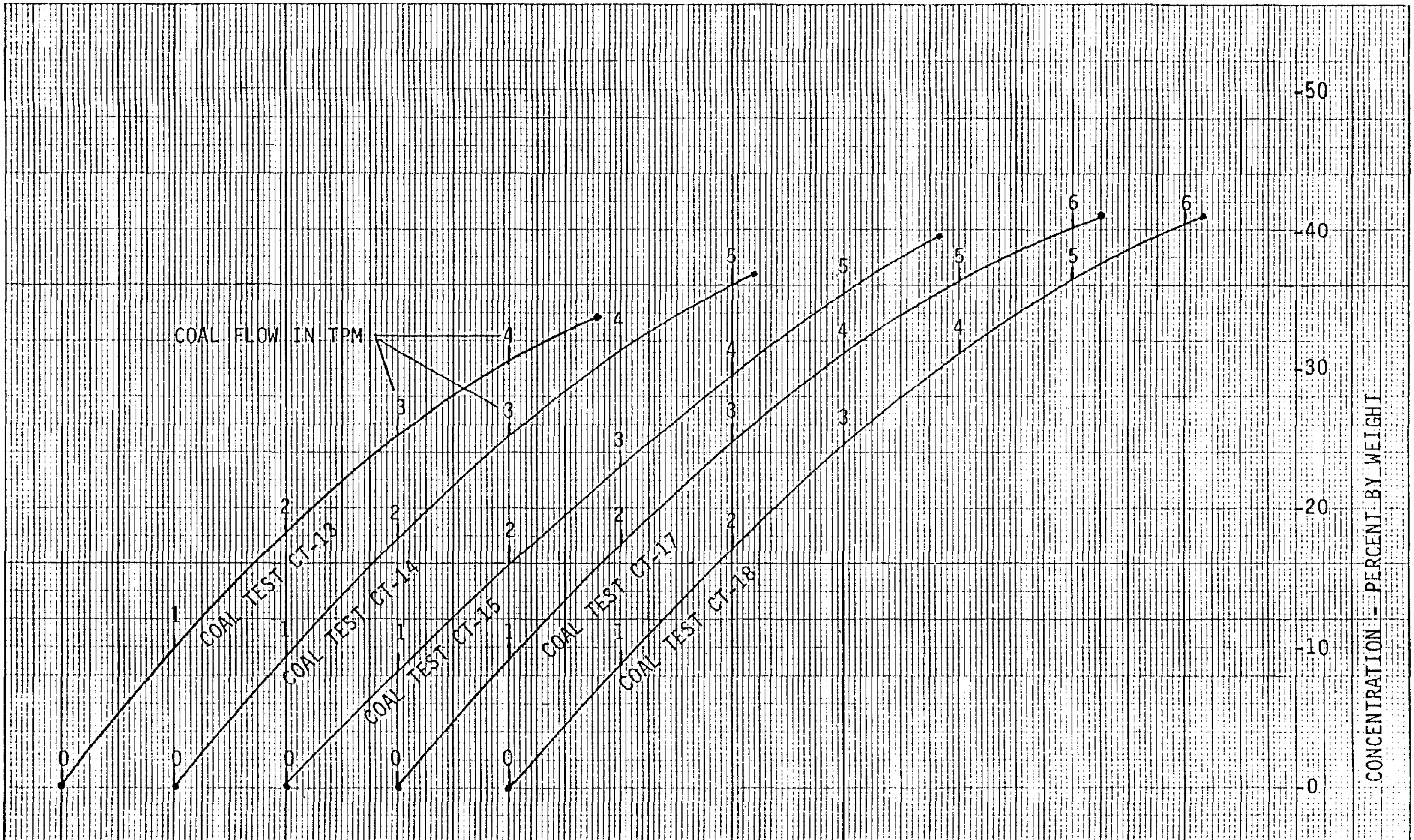
Coal Flow in TPM is Shown for Each Test

Data Points are shown for the Start and End of Each Test
For Added Data, Refer to Figures and Tables for Individual Tests



SLURRY CONCENTRATION AS A FUNCTION OF COAL FLOW
FOR COAL TESTS CT-2, CT-6, CT-12, CT-8, CT-9

Coal Flow In TPM is shown for Each Test
Data Points are shown at the Start and End of Each Test
For Added Data, Refer to Figures and Tables for Individual TEsts



SLURRY CONCENTRATION AS A FUNCTION OF COAL FLOW
FOR COAL TESTS CT-13, CT-14, CT-15, CT-17, CT-18

Coal Flow In TPM is shown for Each Test
Data Points are shown at the Start and End of Each Test
For Added Data, Refer to Figures and Tables for Individual Tests

APPENDIX D

REPORT OF TESTING RESULTS OF
A DOUBLE ROLL BREAKER DESIGN
AT S&S ENGINEERING CENTER

BY

R. DICKOL

(Reference Phase III)

<u>TABLE OF CONTENTS</u>

PURPOSE

The object of the tasks undertaken was to design, build, and test a double roll sizing breaker so that coal from a mine face could be broken to 3" x 0" for injection into a 9" slurry pipe line.

This breaker would have to take material being delivered from a standard SE breaker. The size of this material could range from as large as slabs 2-3 inches thick and 12 inches across.

The breaker would have to pass the sized material at a rate of 6.4 TPM, with all of it being top-sized to 3".

The breaker would have to be compact enough to fit in a modified machine in addition to the Jet Pump module at the rear of the vehicle.

TEST APPARATUS

The test apparatus is a double roll breaker design which incorporates two rollers with picks mounted in each (see fig. 1). The rollers rotate towards each other so the picks converge and mesh. It's this convergence of the picks called the bite which does the breaking. The spacing is such that a piece of material larger than three inches will get caught in the bite and be broken If the material is less than three inches, it will get caught in the bite pockets formed between picks, and pushed through.

Each roller was driven by a hydraulic motor and the two rollers were coupled together by a double row chain. The motors were gear motors with each having a displacement rate of about 24 gpm. The chain was necessary to keep the drums properly timed. If the timing changed, the spacing of the picks is such that interference of picks would occur between drums.

For this test, two different type picks were being evaluated. The first type is called a cutter bit or attack bit. This type pick is conical in shape and has a pointed carbide insert in the tip (see figure 2).

The second type pick is called a drag bit. This type pick uses more of a facial attack of the material compared to the point attack of the cutter bits. The drag bit made a larger bite possible. These picks are used on some long-wall mining machines

The test apparatus was divided in two with drag bits on half and cutter bits on half. A divider plate was placed down the center of the funnel shaped hopper to isolate each side in later testing. The actual length of breaker of each type pick was 12-13 inches for each side. An actual full scale breaker with one type pick would measure about 36 inches to match the width of a conveyor.

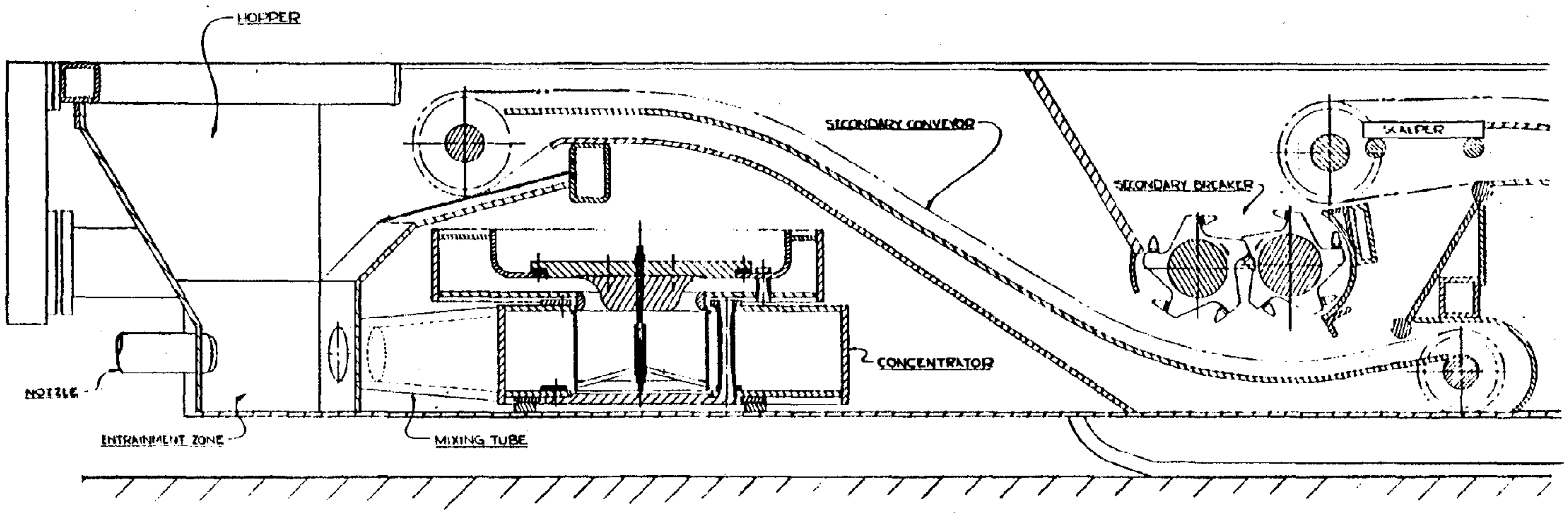
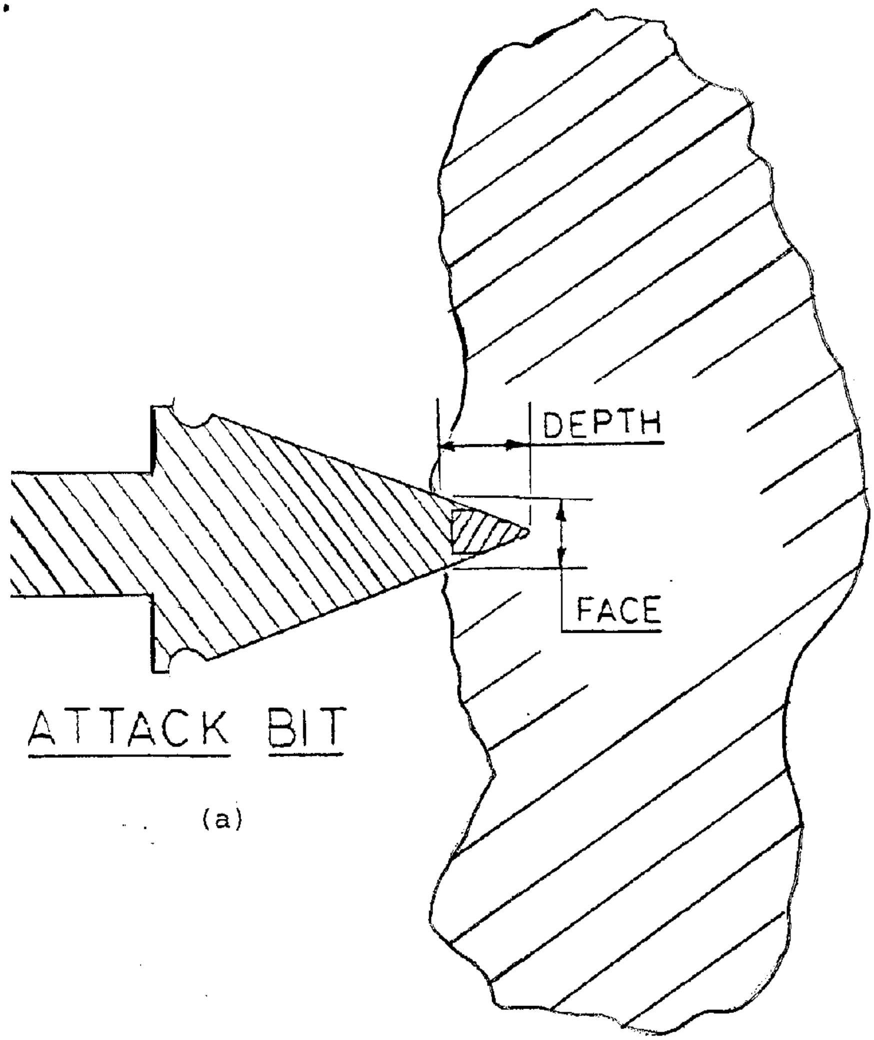
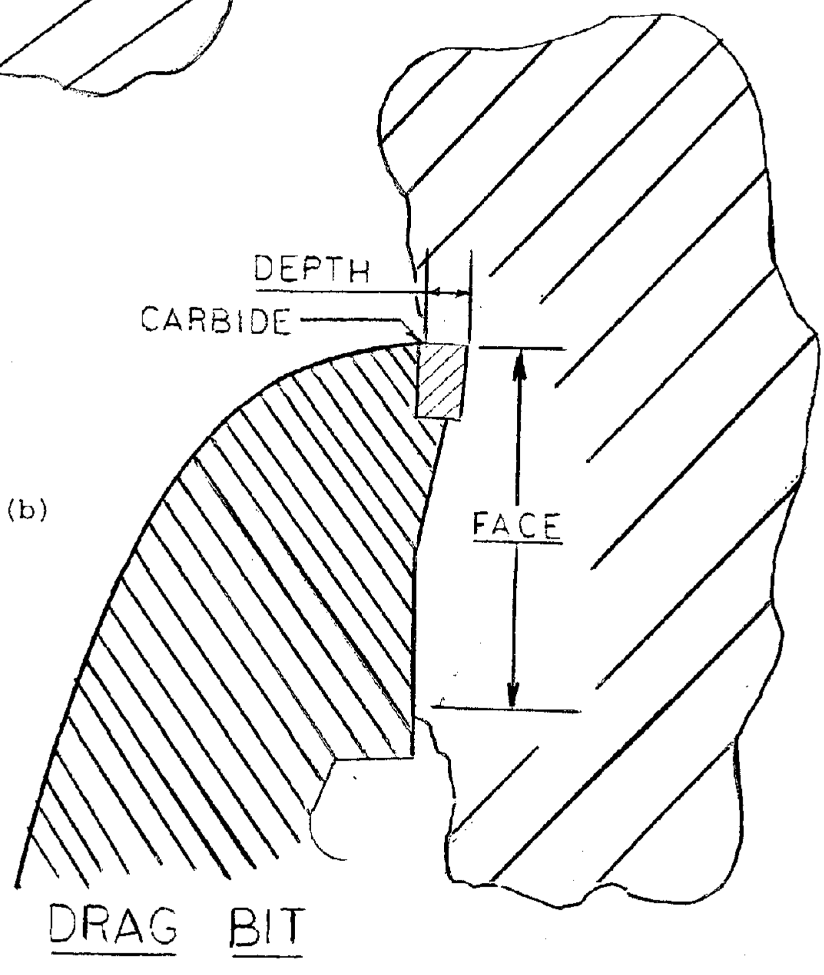


Figure 1: Typical Arrangement of Double Roll Secondary Breaker



(a)

FIGURE 2



(b)

The breaker with a metal plate hopper was mounted on the end of an SE feeder/breaker. The on board breaker was a single drum type with 4 rows of picks. This breaker was capable of breaking material down to a size of two inches thick and 12-18 inches in diameter. This fact must be considered throughout the tests.

Though initial runs were not instrumented, later runs were with hydraulic pressure and shaft rpm being recorded on a Gould two channel pen recorder. On two runs, two pressure readings were recorded, one for each motor. On subsequent runs, one pressure was used and the shaft rpm added.

The throughput of the breaker was measured by running the breaker for a specific amount of time, then weighing the amount of material that passed through.

Under normal operation, the conveyor was only able to deliver a five inch thick bed of rock at a rate of 30 fpm or 1.5 ton per minute. To achieve the high volume of rock required for some of the tests, the conveyor was loaded by hand to a depth that would yield the chosen throughput.

For the last tests, a special diverter plate was fabricated to move the material from one side of the conveyor to the other so that most of the material coming up the conveyor would be directed to only one type of pick . About 80% of the material would be pushed to one side of the conveyor while the other 20%, mostly fines, slid under the diverter, between flights on the conveyor.

The rock being tested was a shale taken from a quarry near the S & S Engineering Center. This rock is continually used by S & S to simulate coal in all their testing.

OBSERVATIONS

Initially, material was just hand fed to both sides of the breaker. The breaker had no trouble breaking the pieces but without additional material falling on top, a single piece had a tendency to bounce around in the hopper on top of the breaker drums and at times would be thrown out of the hopper. As long as material was fed from the conveyor, very little would be thrown out of the hopper.

There were three different observations which the two type picks compared dramatically. 1) Sizing, 2) Stalling, 3) Bridging.

The first comparison, sizing, differed some, but without extensive sieve analysis, final conclusions could not be made.

The drag bits had a larger size throughput, than the attack bits. This was attributed to less restrictions between the picks. The drag picks were welded directly to the shaft, where the attack picks were attached to a large holder which created many more restrictions. Whether it was from a fault in fabrication or just the effect of the extra body on the attack picks, it was obvious that the spacing between drag picks was larger than the attack picks.

In samples taken by hand, approximately 17% by weight of the sample from the drag bits was over 3 inches, where only 7% was over for the attack picks.

It is also possible that the construction of the test breaker added to the excessively large pieces. Spaces were found at the ends of the shafts that large pieces could slip through. These would be closed off on a production model.

The second and third observations were inversely related from the two types of picks.

The drag picks had a tendency to stall the breaker. If a large (thick) piece entered the bite of the drag picks it was possible for the carbide of the bit to sink into the rock until the pick body hit the rock thus stalling it. This was due to the breaking force being distributed over a large area (see figure 2b). The attack picks did not have this problem because these picks don't have a large distributed area after penetration of the carbide, allowing penetration to continue and breakage to occur.

Stalling was observed only on pieces that were determined larger than the new primary breaker would allow through.

Bridging occured on the attack picks when large slabs would fall in the hopper and block the entrance to the breaker. This never happened with the drag picks which shows the effect of the larger bite of the drag picks. The attack pick did not have a large enough bite to grab the large pieces and break them.

A secondary cause of bridging was possibly due to the size of the funnel shaped hopper which allowed the large pieces to bridge the sloping hopper walls before contacting the breaker.

During tests where the highest volume of rock was run, one could look into the breaker and continue to see the drag bits while material going to the attack picks would tend to pile up some. This was possibly due again to the larger spacing of the drag picks allowing more material through. Both type picks were able to keep up with the feed rate over 6.4 TPM.

There was a high power draw by the breaker with no load, or 15 horse power. This was probably due to a very poor machining process in the fabrication of the test rig. During initial start-up, it was noticed that one of the bearings was badly over heating. This was due to some of the tolerances being out of spec up to 10 times in some cases. The splines holding the end of the shaft at the overheated bearing were off center as much as 1/32 of an inch. Some of the clearances were opened up to accomplish testing but throughout the evaluation of data, the 15 HP drag must be realized as a contributing factor in the total horsepower draw.

RESULTS

Table 1 lists all runs made during tests of the secondary breaker at S & S Engineering Center during 5/81.

Pick Type	Weight Material Tons	Time Secs	Diverter Plate Correction %	Through-put for Specified Picks TPM	Through-put Scaled for 36" Breaker TPM	Actual HP For Specified Picks +15HP Friction	HP Scaled for 36" Breaker	HP TPM	Comments
Drag & Attack	-	-	-	1.0	1.4	18.2	19.4	13.9	Diverter plate not installed
"	-	-	-	1.2	1.7	22.4	25.3	14.9	Throughput estimate by bed thickness and speed
"	1.3	20.6	-	3.6	5.0	30.1	35.9	7.2	Material stacked 12"-14" deep
Drag	1.4	44	80	1.5	4.48	23.8	40.3	9.0	
Drag	1.3	29	90	2.7	8.07	23.8	40.3	5.0	Conveyor stacked
Attack	1.2	26	90	2.7	8.22	27.3	50.4	6.1	
Drag	.9	34	80	1.6	4.38	19.6	28.3	6.5	Standard SE breaker output
Attack	1.2	62	80	1.2	3.27	21.8	34.6	10.6	
Attack	1.2	80	80	.91	2.53	25.1	44.1	17.4	
Drag	1.7	101	80	1.0	3.02	16.4	19.0	6.3	

TABLE 1: SECONDARY BREAKER TESTS

The actual rates & horsepowers were taken from a strip-chart recorder output which was averaged for each particular run. Each run consists of two parts because in the approximate time of 30 seconds, the chute under the breaker would fill up, requiring clearing. The chute would be cleared and the second half of a run would be made.

The actual throughput rate was adjusted in most cases by 20% due to some material slipping under the diverter plate. For the 2 highest volume runs, the adjustment was only 10% due to the conveyor being carefully loaded by hand.

Also reflected on this table are results for throughput and horsepower interpolated up for a 36" long breaker from the 12"-13" test breaker length.

The following two formulas are the basis for the scaled figures:

$$\text{scaled throughput} = \frac{(\text{actual throughput} \times \overset{(.90)}{.80} \times 36'')}{\text{Test breaker length}}$$

$$\text{scaled HP} = \frac{(\text{single side HP} - 15^{*})}{\text{Test breaker length}} \times 36 + 15^{*}$$

*Equals the 15 HP drag due to excessive friction in test apparatus.

Even with the limited number of data points obtained, a trend appears comparing drag picks with attack picks. This can be seen on Chart 1, which extrapolates all points to a 36 inch long breaker. It's clear from the graph that for the same throughput, less horsepower is required to break the material with the drag picks.

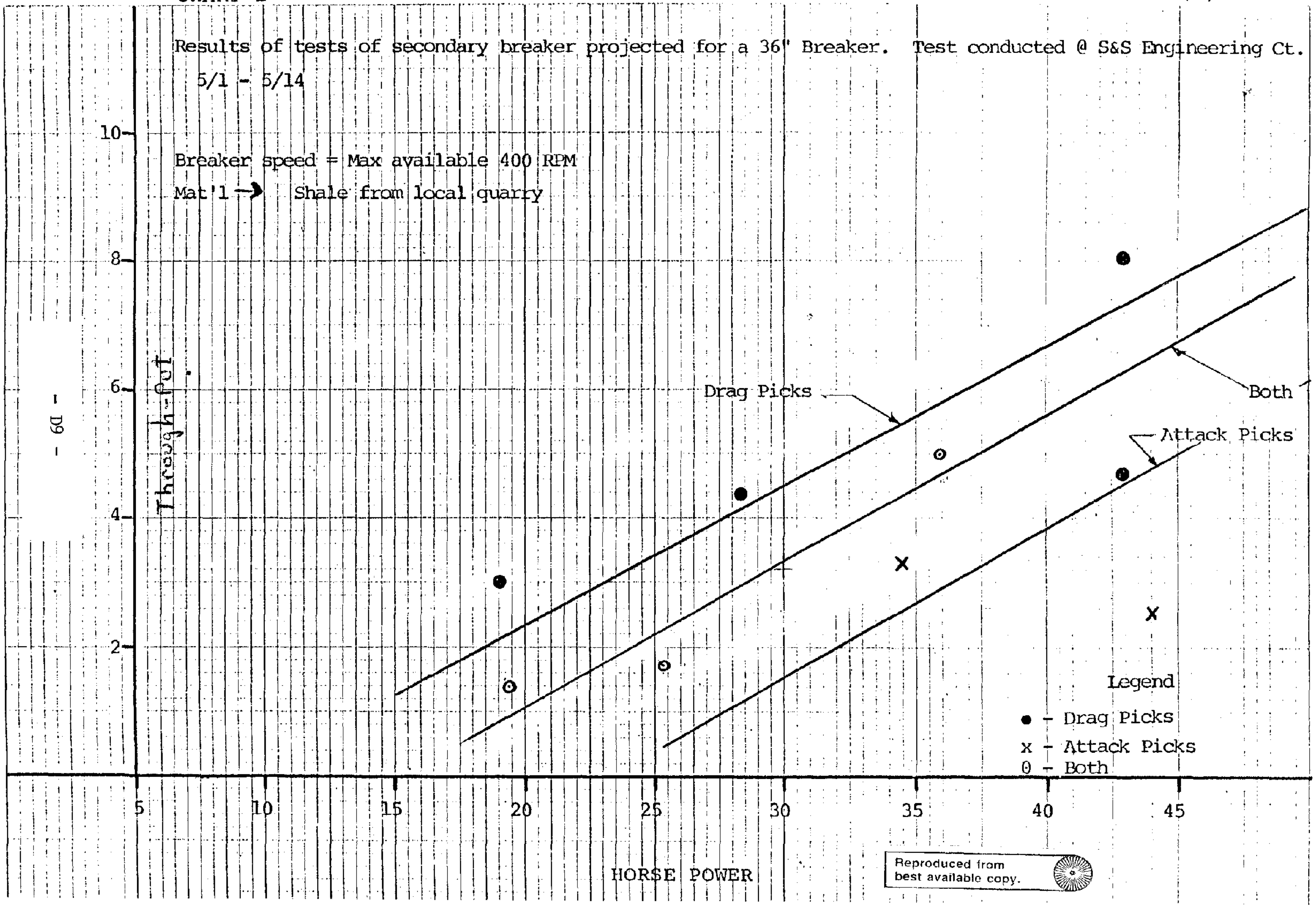
The most likely explanation for this leads back to the spacing of the picks, and observations of the output. Though there was a slightly larger percentage of material passing through the drag bits, it was also obvious that the remaining material was larger in size over all. This leads to the conclusion that less breakage was done for the same volume of material on the drag pick side than on the attack pick side.

CONCLUSIONS

The testing of the double roll secondary breaker gave the first indication that a small, compact, sizing breaker could be designed to have a throughput of at least 6.4 TPM.

The effect of the angle of attack was realized on the drag bits by this type bit stalling on large pieces. By slightly changing the angle of the drag picks, it would be likely that more carbide

CHART 1 RAD 6/4/81



would penetrate the rock before the bit body would bottom out, thus reducing the chances of stalling.

The drag bits had an advantage over the attack bits in that because of the design, drag bits would make a larger bite. This larger bite would facilitate the breakage of the large slab pieces that tend to bridge over the attack picks.

Though there was a larger quantity of oversized pieces put out by the drag bits, they demonstrated the capability to better size the material without over breakage. A more discrete sizing with drag picks could be accomplished by closing down some of the excessive open space in the test rig.

The hopper design was found to be important in that if the hopper opening above the breaker drum was too narrow, bridging would also occur.

In general, this breaker successfully showed that a double roll breaker could pass the complete 6.4 TPM of material without the use of a scalper. Both drag type and attack type picks did the job with different degrees of difficulty. Also to do the job, only about 50 HP is required which is quite reasonable for a breaker of this size and thoughput.

FURTHER TESTING

Testing should be continued in the following areas:

-- Instead of a straight line location of the picks on the shaft, spiral their position. This might increase the concentrated stress delivered by each pick.

-- Test various breaker speeds. This could not be done because of lack of available power.

-- A full scale breaker (36 inches long) should be tested to confirm data. This would require 2 or more pairs of drums each with a different type pick.

-- The spacing should be corrected to improve sizing. The drag bits spacing should be closed down 1-2 inches between drums while opening the attack picks might reduce over breakage, reduce horsepower draw and increase bite.

-- Rotating each drum at a different speed (example -2:1 ratio), might increase the breaker capabilities but the timing of the shafts becomes much more critical.

-- The angle of attack of the drag bits should be changed. This will probably reduce the chance of stalling considerably.

<u>APPENDIX E</u>

Sub-Scale Centrifugal Slurry Concentrator

(Reference Phase III)

I. Introduction

A centrifugal slurry concentrator is under development at Ingersoll-Rand Research, Inc. (IRRI), Princeton, N.J. under a United States Bureau of Mines (BOM) contract to design a jet pump injector (JPI) vehicle for hydraulic face haulage of coal or, more generally, for coarse slurry transport. The concentrator serves to increase concentration of the coarse slurry being handled in order to improve line performance characteristics. The device is geometrically simple and has no moving parts. Testing of a scale model at IRRI shows that the device easily meets its design performance specifications. The following is a description of the centrifugal concentrator, a theoretical analysis of it, and a discussion of the results and significance of the scale model tests.

II. Objectives

Within the range of interest, increasing the slurry concentration is desirable because the attendant decrease in volumetric flow rate and mean velocity through a line of given size reduces pressure drop, reduces wear rates, and reduces the power requirements of the pumping system. Higher slurry concentrations also mean easier dewatering and other processing at the preparation facility.

The BOM contract referred to establishes the maximum slurry concentration as at least 40% by weight, and preferably 50%. These concentrations are needed in order to pump specified maximum tonnage rates in the specified 8-inch slurry line. 40 to 50% concentration is desired in a coarse coal slurry line. A concentrator, downstream of the jet pump, that could extract excess water and bring concentration up into the desired 40-50% range would be highly desirable for use on the JPI.

The operating conditions that the concentrator would have to face in the jet pump injector system are:

A. Coal flow rate = 6.4 tons per minute. This would be run-of-mine Appalachian coal with a large fraction of included dense refuse. Its average specific gravity would be about 1.8.

B. Maximum solid particle size = 3 inches, although occasional somewhat-larger particles may be encountered and should not present difficulties.

C. Inlet concentration $C_I \leq 30\%$ solids weight.
D. Total inlet water flow rate = 2000 gallons per minute supplied by the jet pump and exiting through the discharge + 1600 GPM recirculating between the overflow and the inlet hopper = 3600 GPM total.

The performance characteristics desired of the concentrator are:

A. Discharge concentration C_D = 40 to 50% by weight.
B. Discharge pressure $P_D \geq 0.95$ x inlet pressure P_I.
C. Steady-state solids concentration in the overflow should be low enough so as not to cause operational difficulties in the overflow system.
D. There should be little likelihood of plugging the concentrator with solids unless the discharge line is already plugged In other words, the device should not add any operational problems to the system that do not already exist.

III. Concentrator Design

The currently-contemplated configuration of the cyclonic concentrator is shown in Figure 1. The concentrator proper is roughly disk-shaped with tangential inlet and discharge ports and a central overflow port. The tangential inlet creates a cyclonic flow pattern in the concentrator. The density difference between the coal and water in the incoming dilute slurry and the centrifuga effect of the swirling flow tend to drive the denser solids toward the periphery of the concentrator. Simultaneously, any air and light solids that are entrained in the inlet flow tend to migrate toward the center and eventually exit through the overflow port.

The concentrator walls are vertical so that the flow may reach the maximum radius everywhere, for better concentrator performance. The section between tangential ports is of a smaller radius than the main part to more efficiently redirect flow that does not exit through the discharge port on the first pass.

The overflow stream passes from the concentrator into a volute whose purpose is to slow the flow down and recover pressure. This is done because the overflow stream is intended to be used for the following functions on the JPI vehicle:

A. Augmentation of the jet pump's performance by passing most of the overflow water at high velocity through auxiliary nozzles surrounding the main nozzle,
B. Flushing the hopper to help maximize the coal flow rate into the entrainment zone, and

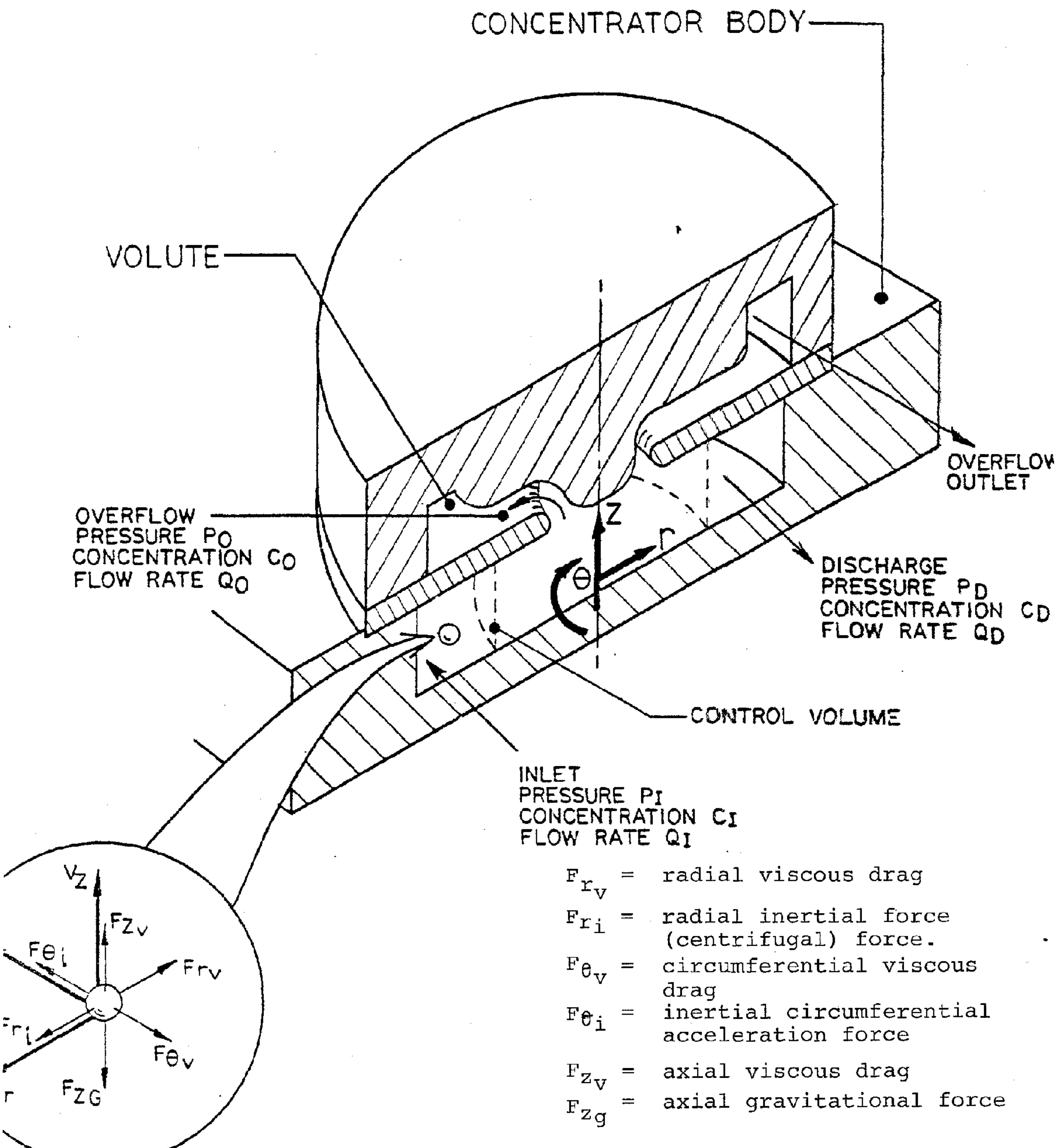


FIGURE I

CYCLONIC SLURRY CONCENTRATOR

C. Maintaining a minimum water level in the hopper to ensure that the jet is completely submerged and hence keep its performance at optimum.

Valves on the outlet of the volute control the fraction of the total flow that is allowed to escape through the overflow port and recirculate through the system, and so also controls the ratio of discharge to inlet concentrations. It is necessary to reach a compromise between optimum concentration increase and optimum overflow volumetric flow rate in order to maintain the desired balance between overflow water supply and secondary water demand. This is necessary for purposes of water level control in the hopper.

A cylindrical screen serves as a filter to keep large solids from clogging the overflow passage. It may be possible for pieces of wood to get into the system and these would migrate toward the center. Also, large particles of heavy solids tend toward the center on startup as the concentrator fills.

IV. Theoretical Analysis

An accurate analytical description of the physical principles governing the operation of the centrifugal concentrator is unfortunately impossible due to the heavy turbulence of the flow pattern, its nonhomogeneity, the wide variation in size and shape of the solid particles, variability of solids concentrations, and interactions between solids. However, an approximate description can be given if these considerations are not taken into account. This analysis will ignore these difficulties.

A. Particle Separation

The criterion determining whether any given particle will migrate toward the periphery of the concentrator (separate), toward the center (entrain), or toward neither, staying at a constant radius, is simply the radial force balance on the particle (please refer to Figure 1). The two opposing forces acting radially are inertia and viscous drag. The tangential and vertical forces have no first-order effect on particle separation.

If the inertial force on the particle is greater than the viscous drag resisting it, the particle will move outward. Mathematically, if

$$\frac{F_{r_I}}{F_{r_V}} > 1 \qquad (1)$$

where F_{r_I} = radial inertial (centrifugal) force

F_{r_V} = radial viscous drag force

then the particle will separate as desired. However, if

$$\frac{F_{r_I}}{F_{r_V}} < 1 \qquad (2)$$

then the particle will entrain in the overflow stream.

Using a cylindrical coordinate system as defined in Figure 1, radial inertial force is expressed as

$$F_{r_I} = \frac{m_p V_\theta^2}{r} = \frac{\rho_p \nu_p V_\theta^2}{r} \qquad (3)$$

where m_p = particle mass

V_θ = tangential velocity component

r = radius from concentrator center to particle center of mass

ρ_p = particle density

ν_p = particle volume

and radial viscous drag force is expressed as

$$F_{r_V} = \tfrac{1}{2} \rho_w V_r^2 a_{p_r} C_{D_r} \qquad (4)$$

where ρ_w = water density

V_r = radial velocity

a_{p_r} = particle frontal area as seen radially

C_{D_r} = particle drag coefficient as seen radially

The radial force ratio then becomes

$$\frac{F_{r_I}}{F_{r_v}} = \frac{2\ \rho_p\ V_\theta^2\ \nu_p}{C_{D_r}\ \rho_w\ V_r^2\ a_{p_r}\ \ r} \tag{5}$$

The criterion for particle separation is therefore

$$\frac{2\ G\ V_\theta^2\ \nu_p}{C_{D_r}\ V_r^2\ a_{p_r}\ r} > 1 \tag{6}$$

where $G = \rho_p/\rho_w$, the particle's specific gravity

Or, more simply, a particle will go to the discharge outlet as desired if

$$\frac{2\ G\ \nu_p}{C_{D_r}\ a_{p_r}} > \left(\frac{V_r}{V_\theta}\right)^2 r \tag{7}$$

Radial velocity V_r is determined by overflow opening and concentrator geometry. Tangential velocity V_θ is set by the injection system. Drag coefficient C_{D_r} depends on particle shape and Reynolds number, as well as orientation.

The radial velocity component is caused by the presence of the overflow opening and the fact that the pressure in the concentrator is everywhere above atmospheric. The flow that hence passes through the overflow port draws fluid in from the periphery, along with any air, light solids, and high-drag dense solids that are included. This inflow results in a radial pressure gradient, with static pressure decreasing with radius. While a formula for this radial gradient is underivable for the reasons listed at the beginning of this analysis, experimental data indicate that its form resembles that shown in Figure 2. While the gradient is approximately linear for large radii, it becomes nonuniform and highly geometry-dependent near the center and under the port. This means that care must be taken in the design of the port if energy recovery is to be maximized.

Constricting the overflow opening reduces overflow ratio ($R_O = Q_O/Q_I$) along with radial velocity V_r and tends to level out the pressure gradient. Entirely closing the opening entirely eliminates the radial velocity component and entirely flattens the radial pressure gradient. When this happens, all solids denser than water migrate outwards since there no longer is a radial velocity component trying to carry them inwards.

Radial velocity may be estimated as follows: Referring once again to figure 1, let us define a cylindrical control volume centered on the concentrator axis. Let us assume a constant concentrator thickness and a zero vertical velocity component. Knowing the volumetric flow rate into the center and the relation between radius and area of the control volume will yield a relation between radius and velocity.

Available flow area into the control volume is

$$A_r = 2\Pi rt \tag{8}$$

where A_r = area at radius r

r = radius of control volume

t = thickness of concentrator

Overflow volumetric flow rate is

$$Q_o = R_o \, Q_I \tag{9}$$

where Q_o = overflow rate

R_o = overflow ratio

Q_I = inlet flow rate

Hence radial velocity is

$$V_r = \frac{Q_o}{A_r} = \frac{R_o \quad Q_I}{2\Pi \ rt} \tag{10}$$

Under normal operating conditions, R_o is about 0.3 and Q_I is about 4200 gal/min (water plus coal). The concentrator design intended for the JPI vehicle is 8 inches thick. Thus

$$V_r = \frac{(0.3)(4200 \text{ gal min}^{-1})}{(2\pi r)(8 \text{ in})} \times \frac{\text{ft}^3}{7.481 \text{ gal}} \times \frac{\text{min}}{60 \text{ sec}} \times \frac{1728 \text{ in}^3}{\text{ft}^3}$$

$$V_r = \frac{1 \text{ in}}{r} \times 96 \frac{\text{in}}{\text{sec}} \tag{11}$$

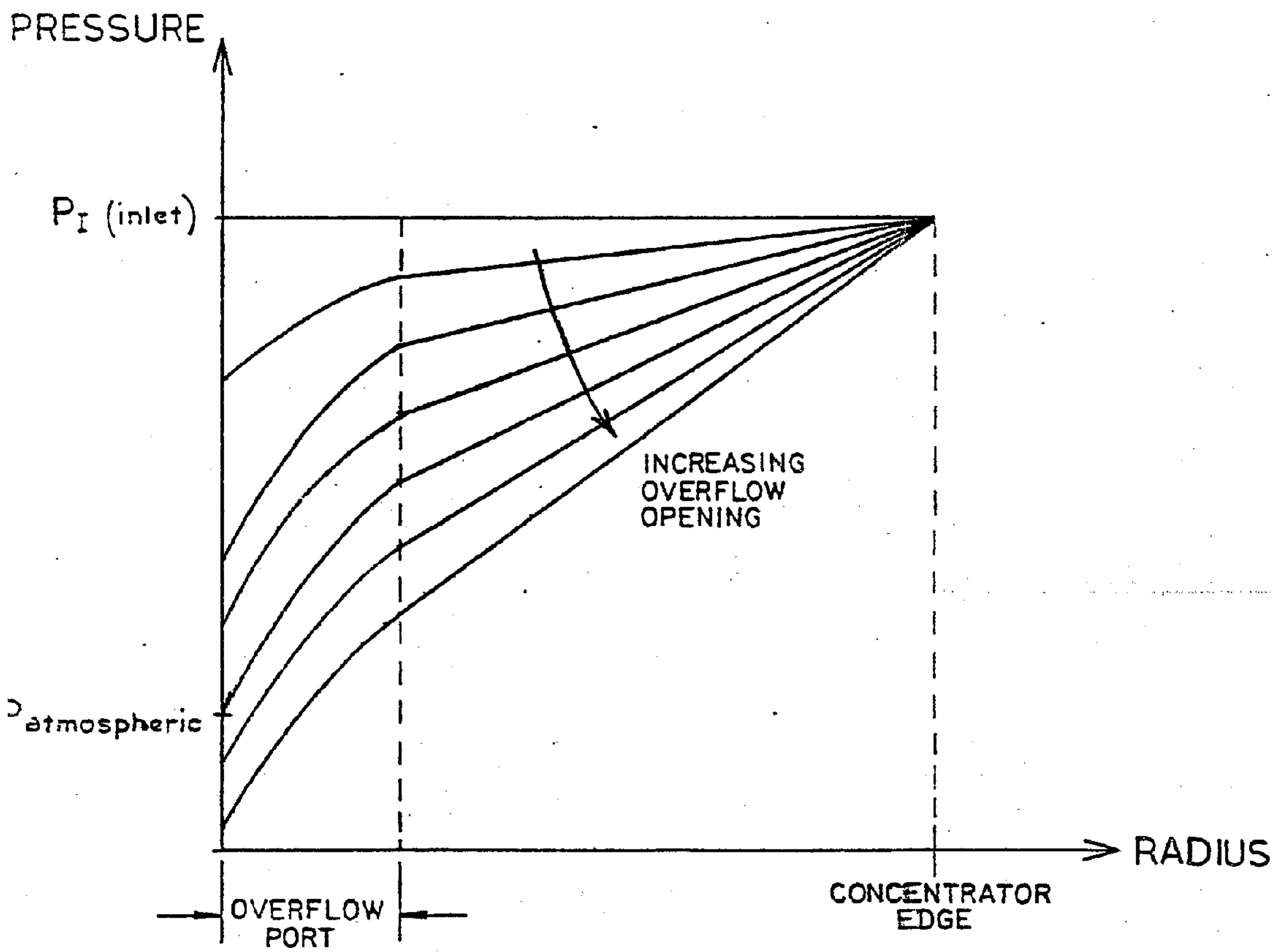


FIGURE 2

QUALITATIVE RADIAL PRESSURE GRADIENT

Also, average tangential velocity is easily found from inlet port cross-sectional area and volumetric flow rate.

$$V_\theta = \frac{Q_I}{A_I} = \frac{4200 \text{ gal min}^{-1}}{\pi\ (8 \text{ in}/2)^2} \times \frac{\text{ft}^3}{7.481 \text{ gal}} \times \frac{\text{min}}{60 \text{ sec}} \times \frac{1728 \text{ in}^3}{\text{ft}^3}$$

$$V_\theta = 322\ \frac{\text{in}}{\text{sec}}$$

****

Referring again to equation 7, note that increased concentrator radius increases likelihood of particle separation. Even though inertia decreases inversely with radius, V_r^2 decreases with the square of radius. Concentrator radius should therefore be as large as possible.

Noting that the ratio of volume to frontal area for a given shape increases linearly with its size, it become apparent that there will be a critical size above which a particle is likely to separate and below which it is likely to entrain. Equation 7 allows us to estimate that size:

Let us assume that all particles are spherical and then solve for their critical diameter d_p. Since separation depends strongly on drag coefficient which in turn depends irregularly upon diameter, the solution must be iterative, using the known relation between C_D and Re for a sphere. This relation does not change drastically for other shapes, so a size estimate based on the sphere assumption should yield sufficient accuracy.

Using the velocity values estimated earlier and an assumed radius of 18 inches, equation 7 becomes:

$$\frac{2\ G\ v_p}{C_{D_r}\ a_{p_r}} > \left(\frac{V_r}{V_\theta}\right)^2 r \tag{7}$$

$$\frac{2 \times 1.4 \times \frac{4}{3}\pi (d_p/2)^3}{C_{D_r} \times (\pi\ (d_p/2)^2)} > \left(\frac{96/18}{322}\right)^2 (18 \text{ in.}) \tag{12}$$

$$d_p > .0026 \text{ inch} \times C_{D_r} \tag{13}$$

A few iterations referring to the graph of C_D vs. Re give the result: $d_p > \backsim .003$ inch (or plus 200 mesh). Particles of coal or dense refuse larger than roughly .003 inch should separate as desired. Since at least 98% of the total solids weight the concentrator will encounter is comprised of larger particles, particle entrainment in the overflow stream should present no noticeable difficulties.

B. <u>Concentration Increase</u>

The relationship between overflow ratio and concentration increase may be determined as follows: Assuming that all solids entering the concentrator leave it through the discharge port,

$$C_I \ \dot{m}_I = C_D \ \dot{m}_D \tag{14}$$

where $\dot{m}_I$ = total mass flow through inlet

$\dot{m}_D$ = " " " " discharge

$$C_D = \left(\frac{\dot{m}_I}{\dot{m}_D}\right) C_I \tag{15}$$

$$C_D = C_I/(1-R_o) \tag{16}$$

This means that increasing concentration from 30% to 43% requires that R_o = 30%.

C. <u>Model Scaling</u>

Extrapolation of data obtained from a model to establish the performance of a device of different size requires application of the proper scaling laws to the model. Ratios of the dominant forces must be the same in both the model and the full-sized prototype in order to ensure transferability of data obtained from the model.

In the concentrator, six forces act on any given solid particle: radial inertia and viscous drag (F_{r_I} and F_{r_V}), gravity and vertical viscous drag (F_{z_G} and F_{z_V}) and tangential inertia and viscous drag (F_{θ_I} and F_{θ_V}).

The tangential forces F_{θ_I} and F_{θ_V} can be ignored if it is assumed that there is no "slip" (velocity difference) between the solid and liquid components of the slurry. In this case, their ratio is unity for both model and prototype, no matter what scale is used.

The magnitudes of the radial forces are found from the following equations:

Radial inertia (centrifugal force)

$$F_{r_I} = \frac{\rho_p \, v_p \, V_\theta^2}{r} \tag{3}$$

Radial viscous drag

$$F_{r_V} = \tfrac{1}{2} \, \rho_w \, V_r^2 \, a_{p_r} \, C_{D_r} \tag{4}$$

The magnitudes of the vertical forces are found from these equations:

Gravity

$$F_{z_G} = m_p g = \rho_p \, v_p \, g \tag{17}$$

where g = gravitational acceleration = 32.2 ft/sec^2

Vertical viscous drag

$$F_{z_V} = \tfrac{1}{2} \, \rho_w \, V_z^2 \, a_{p_z} \, C_{D_z} \tag{18}$$

where V_z = vertical velocity

a_{p_z} = particle frontal area as seen vertically

C_{D_z} = particle drag coefficient as seen vertically

For the sake of the order-of-magnitude analysis, it is assumed that particles are spherical. The analysis is performed for two different assumed sizes, one larger and one smaller than the critical diameter of 0.003 inch, say 0.001 inch and 0.1 inch.

Finding which of two forces is dominant is simply a matter of finding their ratio. Comparison of gravity and radial viscous drag forces yields

$$\frac{F_{z_G}}{F_{r_V}} = \frac{\rho_p v_p \, g}{\tfrac{1}{2} \rho_w \, V_r^2 \, a_{p_r} \, C_{D_r}} \tag{19}$$

For a 0.001-inch sphere of bituminous coal,

$$\frac{F_{z_G}}{F_{r_V}} = \frac{1.4 \times \frac{4}{3} \pi (0.0005 \text{ in})^3 \times 32.2 \times 12 \text{ in sec}^{-2}}{\frac{1}{2} \times \left(\frac{96}{18} \text{ in sec}^{-1}\right)^2 \times \pi (0.0005 \text{ in})^2 \times 1.5} \cong .017 \tag{20}$$

For extremely fine particles, viscous effects will dominate gravitational effects. Now, for a 0.1-inch sphere,

$$\frac{F_{z_G}}{F_{r_V}} = \frac{1.4 \times \frac{4}{3} \pi (0.05 \text{ in})^3 \times 32.2 \times 12 \text{ in sec}^{-2}}{\frac{1}{2} \times \left(\frac{96}{18} \text{ in sec}^{-1}\right)^2 \times \pi (0.05 \text{ in})^2 \times 0.4} \cong 6.3 \tag{21}$$

For coarser particles, gravitational effects rather than viscous effects will predominate. Note that the relative dominance of these forces depends on particle size. Since there is a wide range of sizes in the mix, the forces that dominate for the majority of cases must be assumed to dominate for all cases in order for modeling to be possible. Since roughly 98% of the coal weight is comprised of particles larger than the critical size of 0.003 inch, gravity will be more important than viscosity for nearly all particles. Note that gravity is increasingly dominant over viscosity with increasing particle size.

Now, let us perform the same operation on inertia and gravity.

$$\frac{F_{r_I}}{F_{z_G}} = \frac{\rho_p \, v_p \, V_\theta^2}{r \, \rho_p \, v_p g} = \frac{V_\theta^2}{rg} \tag{22}$$

This ratio, the square of the Froude number, is independent of particle properties.

$$\frac{F_{r_I}}{F_{z_G}} = \frac{(322 \text{ in sec}^{-1})^2}{18 \text{ in} \times 32.2 \times 12 \text{ in sec}^{-2}} \cong 15 \tag{23}$$

Therefore inertia will in turn dominate gravity for all particles near the concentrator periphery.

The remaining possibly-significant force, vertical viscous drag, is highly dependent on the vertical velocity component (V_z) of the fluid. Since the flow pattern is highly turbulent, it is impossible to even estimate a typical value for vertical velocity. However, vertical viscous drag will be only a fraction of gravitational force and so may be dismissed.

Since it has been determined that inertia and gravity are the dominant forces establishing the flow regime in the concentrator, proper scaling means holding their ratio, the so-called Froude numbe (Fr), constant between model and prototype. Since this dimensionle group is defined as

$$Fr = \frac{V}{\sqrt{gL}} \tag{24}$$

where V = velocity of fluid relative to particle
g = gravitational acceleration
L = characteristic length dimension

it is necessary that

$$\left.\frac{V_\theta}{\sqrt{gL}}\right|_{model} = \left.\frac{V_\theta}{\sqrt{gL}}\right|_{prototype} \tag{25}$$

(tangential velocity V_θ is the controllable velocity here), or simply

$$V_\theta\Big|_{model} = \sqrt{\frac{L|_{model}}{L|_{prototype}}} \times V_\theta\Big|_{prototype} \tag{26}$$

Model velocities must be lower than prototype velocities by the square root of the geometric scale factor for data to be transferable. Since volumetric flow rates would have to be reduced by the square of the scale factor even if velocity were to be held con flow rate must be reduced by the $\frac{5}{2}$ power of the scale factor. For a model scale of 1/6.4, a volumetric flow rate of Q_I = (4200 GP $(6.4)^{5/2}$ = 41 GPM must be held.

V. Test Apparatus

A 1-to-6.4 scale model of a domed-disk cyclonic concentrator has been built and installed in a small-scale test loop at Ingersoll-Ranc Research, Inc. The loop and concentrator are illustrated schematically in Figure 3. Note: The scale factor of 1 to 6.4 was selected t allow 1 1/4 inch inside diameter hose to be used to represent the 8 inch ID inlet and discharge pipes of the prototype. There is no relation to the design coal flow rate of 6.4 tons per minute.

The model concentrator, made for transparent acrylic for visibility, was supplied with slurry from a 4 ft diameter by 5 ft high s tank which normally contained 100 to 200 gallons of liquid. A rough homogeneous solids concentration was maintained in the tank by a pro peller-type mixer. An electrically-powered centrifugal pump can deliver slurry from the sump tank to the concentrator at flow rates up 100 gallons per minute or at pressures up to 50 psi. A positive-dis

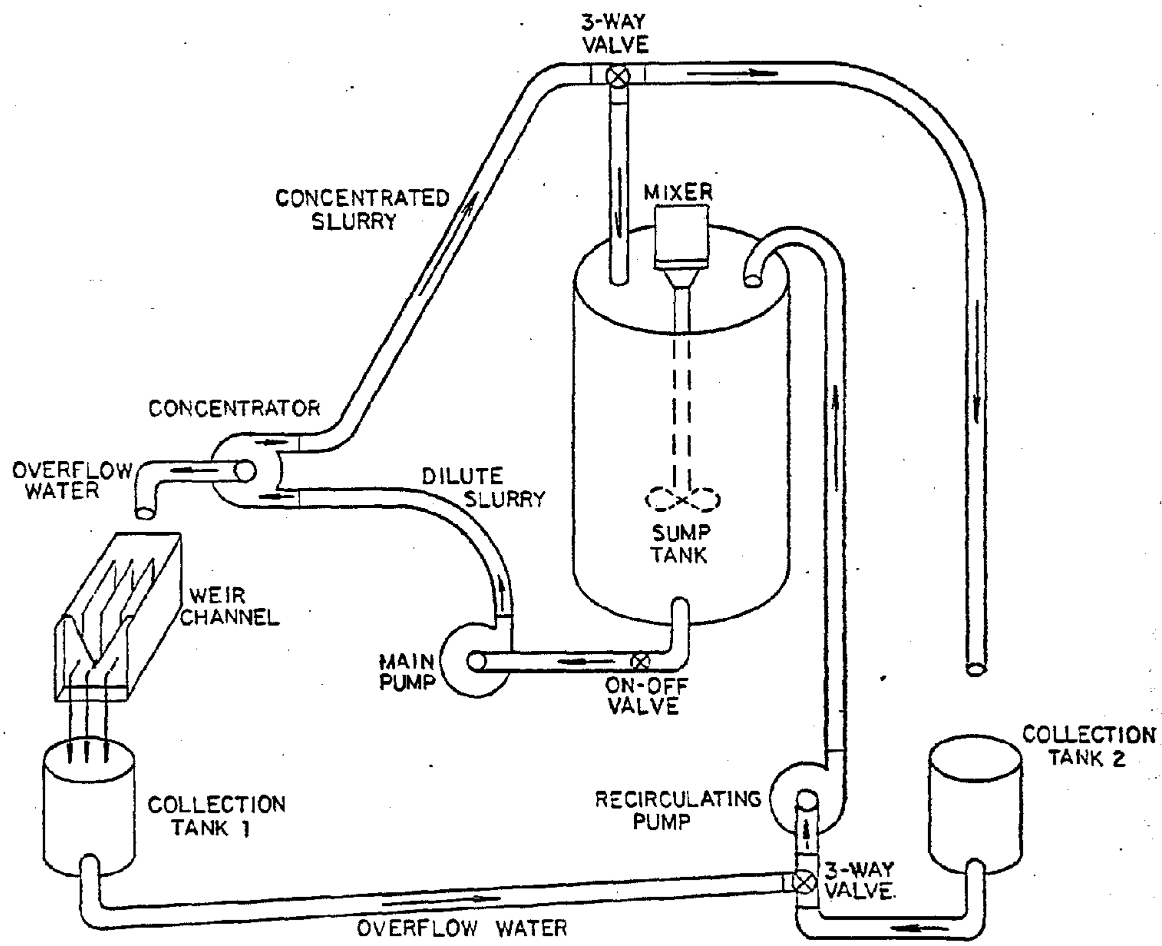


FIGURE 3
TEST LOOP SCHEMATIC

placement pump would be desirable in order to allow flow rate and pressure to be set independently, but a positive-displacement pump that is both small enough and able to handle coarse slurry is unavailable.

Inlet and discharge streams flow through transparent, flexible polyvinyl chloride hose. The hose is squeezed with C-clamps to set back pressure, which like all pressures is measured with Bourdon-tube type pressure gauges. The overflow stream goes from the concentrator into a box fabricated from transparent acrylic sheet which in turn empties into a weir channel for volumetric flow measurement. The weir is necessary because the overflow stream is by then open-channel (at atmospheric pressure) and conventional flow gauges are unusable. For a V-notch weir such as is used here, flow rate is proportional to the 5/2 power of the height of the water level above the notch bottom. The weir channel used here has a capacity of 50 GPM. It empties into a collection drum whose contents are returned to the sump tank by a recirculating pump.

Measurement of the flow rate of the discharge stream is not nearly so straight-forward, however. Since the discharge contains a high concentration of coarse solids, the only way to measure volume flow accurately is with a collection vessel and stopwatch. A second drum is therefore included in the apparatus. This drum is calibrated on the inside with paint marks every 5 gallons. A three-way valve at the end of the discharge hose allows flow to be directed either immediately into the sump tank or into the collection drum. To measure discharge flow rates the flow is directed into the collection drum and the time to fill it to a preselected volume mark is measured with a stopwatch. Simple arithmetic yields flow rate. The same recirculating pump that empties the overflow drum returns the collected slurry to the sump pump.

For most testing, coal is simulated by pellets of Delrin, an acetal resin manufactured by DuPont. These pellets are about 1/8 inch diameter (about 3/4 inch full scale) and have the same specific gravity (1.41) as most bituminous coal. Delrin is preferred over real coal for testing purposes because of its cleanliness and resistance to breakage. However, later testing may require a mixture of particle sizes and, an appropriate polymeric material unavailable at reasonable cost, where sized real coal would be used instead.

VI. Test Procedures

The goals of the model testing are, primarily, optimization of the geometry of the concentrator, and, secondarily, establishment of the salient performance characteristics of the optimized geometry. Complete and detailed information about the performance of any particular shape under all possible operating conditions is unnecessary and exceeds our limitations on time and resources. The appropriate procedure is therefore to collect data only around normal operating points for proposed geometries. The following outline describes the testing protocol used on the scale model at IRRI.

A. Determine geometry to be tested through engineering judgment and information from previous tests. Model and install.

B. Select solids to be used, either single-sized Delrin or mixed-size real coal.

C. Set inlet concentration. Values of 0, 10, 20, 30 and 40% are thought to be sufficiently close but broad-ranged.

D. Set openings and back pressures on the three ports, accomplishing this with C-clamps on the inlet and discharge hoses and with whatever method is provided on the overflow.

E. Set inlet flow rate Q_I. For this case, using a velocity scaled down by 1/ 6.4, the correct model inlet flow rate would be 41 GPM (total water plus solids).

F. Collect data: important pressures, flow rates and concentrations at the three ports, and pressures at selected intermediate points.

G. Correlate results, yielding pressure-flow relationships for all three ports at all test concentrations. Based on need for added information, design new geometry and return to step B.

VII. Preliminary Results and Discussion

A 1/6.4 scale model of a domed full-disk concentrator (see Figure 4) has already undergone testing at IRRI with several different overflow-port configurations. While the testing process could be extended, enough information has been obtained to allow a high degree of confidence in the viability of the concept. The scale model easily meets all of its design performance characteristics, and developments described later should bring improvement even on current performance.

The cyclonic concentrator yields substantial and controllable increases in concentration over a much wider range of flow rates, velocities, and concentration levels than will be encountered in practice. No solid particles (only Delrin has so far been used) whatsoever entrain in the overflow except at startup and at very low velocities. It has not been possible to plug the concentrator with solids without first blocking the discharge line. Even then, the overflow port must be nearly closed or solids will simply exit through it instead. Discharge pressure approximately equals inlet pressure within gauge accuracy in all modes of operation, and frequent pressure <u>rise</u> of 1 to 2 psi may be observed. The rise is attributable to the fact that discharge velocities are lower than inlet velocities.

Different overflow geometries have been built, tested, and discarded. These include a simple hose inserted into the top center of the device, the sliding plates depicted in Figure 4, and a conical poppet-type valve.

Figure 5 shows the performance of the domed-disk concentrator with a hose leading from the overflow port installed. In this case, overflow back pressure could be controlled by squeezing the hose. (In all other geometries tested, the overflow port has opened directly to atmospheric pressure). The presence of overflow back pressure causes an interesting phenomenon at midrange settings. If overflow pressure is allowed to drop below the vapor pressure of water, the water near the center of the device cavitates and forms a vortex. The vortex is enlarged somewhat by dissolved air coming out of solution, and is not exactly at the concentrator's geometric center because of the device's asymmetry. On the model, the vortex would extend for several feet down the overflow hose.

Numerical data on this version were taken only for water-only operation, since a modification was decided upon midway in the testing. This next version, the one with the sliding plates shown in Figure 4, was rapidly abandoned when severe leakage and actuator power problems were recognized.

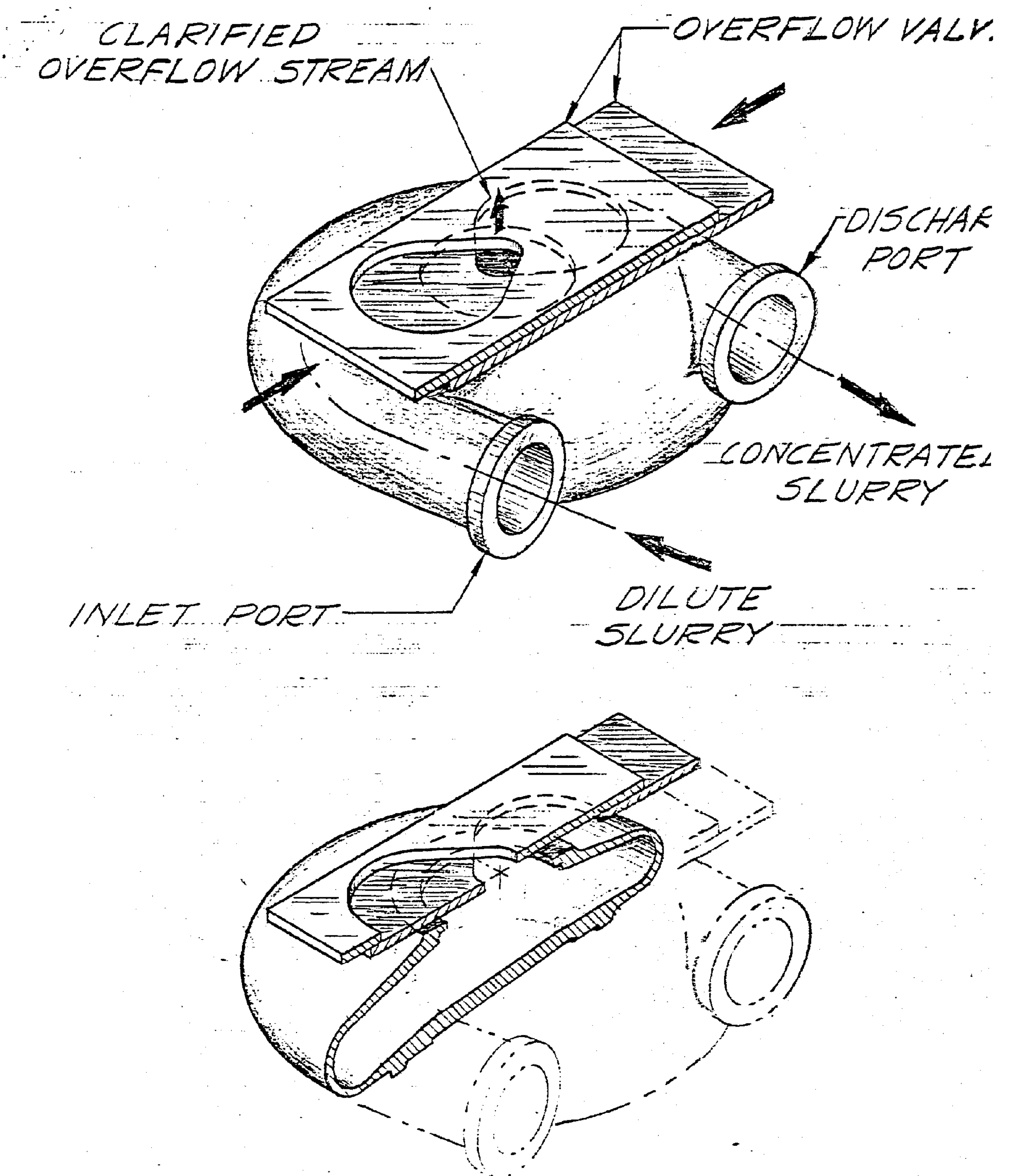


FIGURE 4
CENTRIFUGAL CONCENTRATOR

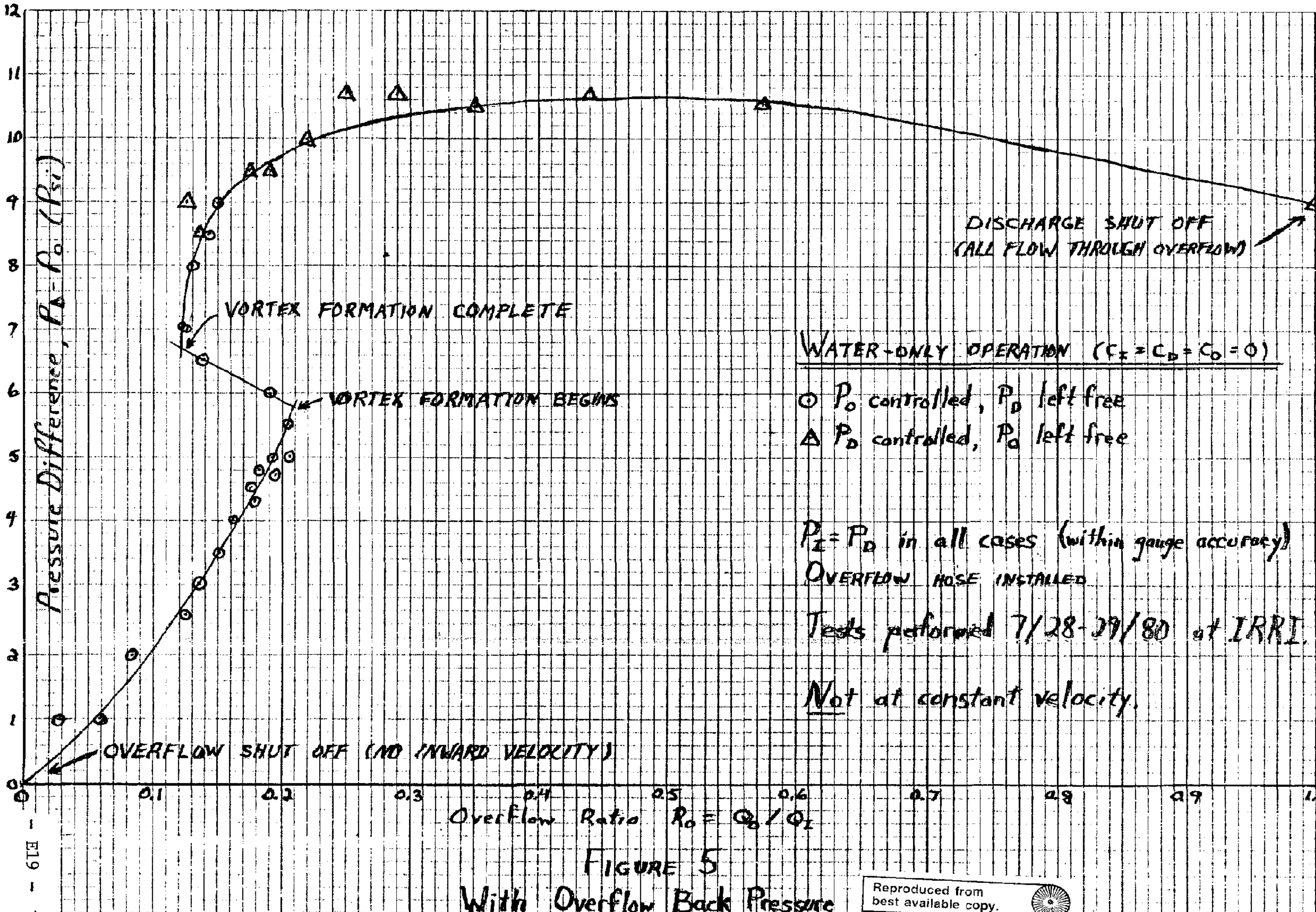


FIGURE 5
With Overflow Back Pressure

When overflow energy recovery maximization was decided to be desirable, the conical poppet-valve design shown in Figure 6 was modeled. Data from it are shown in nondimensional form in Figure 6. Inlet concentrations of 0,10,20,30, and 40 percent by weight were used but did not affect the performance of the overflow port. Both sloped and stepped seats were used, as were overflow radii of r_O = 0.25, 0.50, and 0.75 inch. Overflow ratio R_O was fairly linear with hr_O/tr_C up to about R_O = 0.35, after which further increase in overflow ratio was difficult to attain. For the jet-pump injector application intended, R_O = 0.3 is the desired operating point so the apparent upper limit on R_O will not be a problem.

An outline view of the main concentrator body which is planned for use on the JPI vehicle is shown in Figure 7. The inlet opens into the concentrator at a tangent point rather than at the edge, as previously tested. The discharge line also comes off from a tangent. The floor and ceiling are flat and the walls are vertical for ease of fabrication. The leading edge of the discharge port is broadly rounded rather than sharp so that the flow can select its own separation point without causing cavitation. While a model of this shape is still undergoing fabrication as of this writing, the domed-disk model has been modified by filling in the section between ports and has been operated. Judging from visual inspection, the device is in fact improved by the change. Nearly all solids make only one pass, around the back side of the concentrator, instead of only about 75% as it appeared on the Full-disk model. The new model, with a volute installed, will give more definite information. Another reason for the change is the decrease in space requirements it allows.

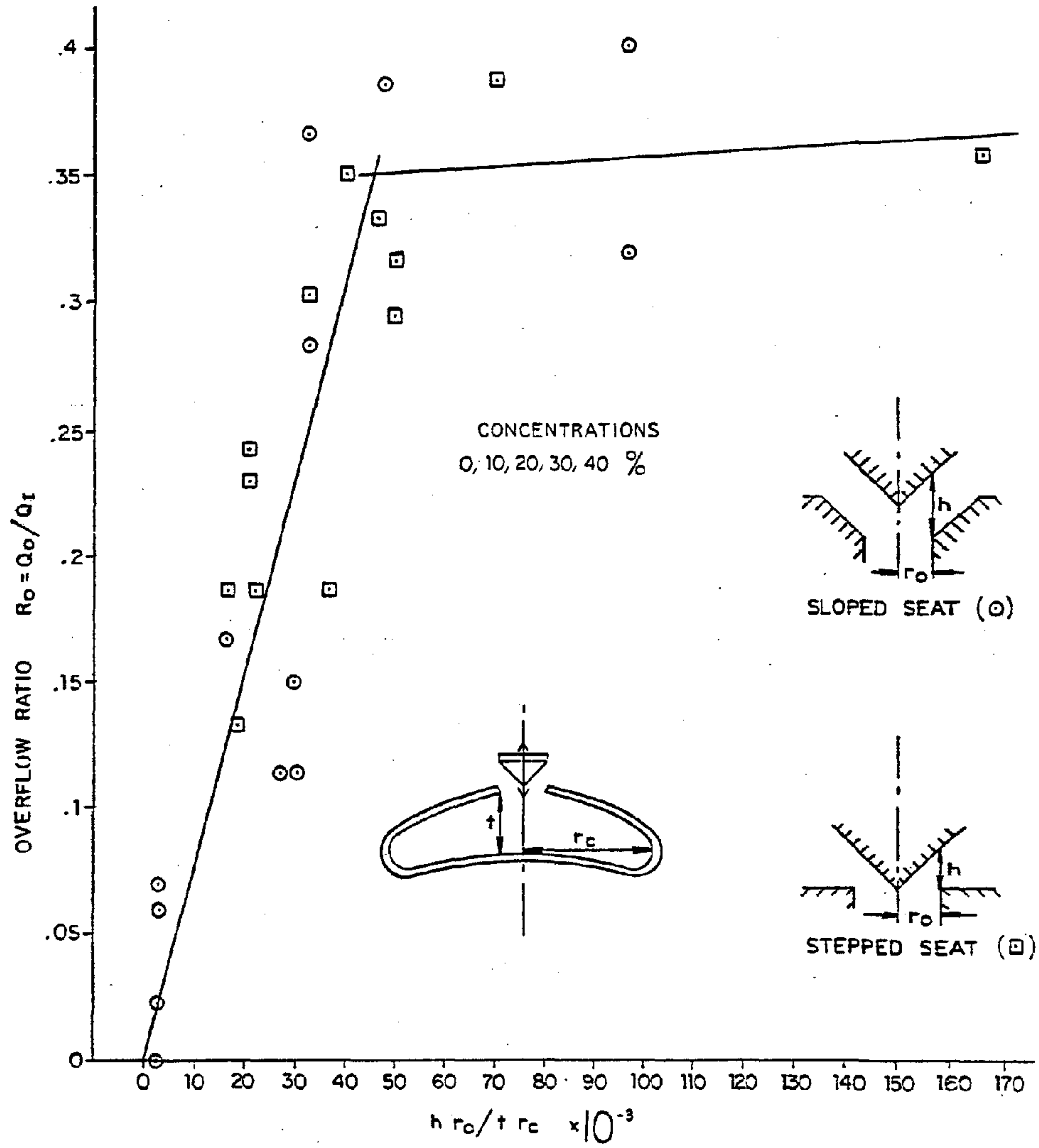


FIGURE 6

POPPET-VALVE OVERFLOW DATA

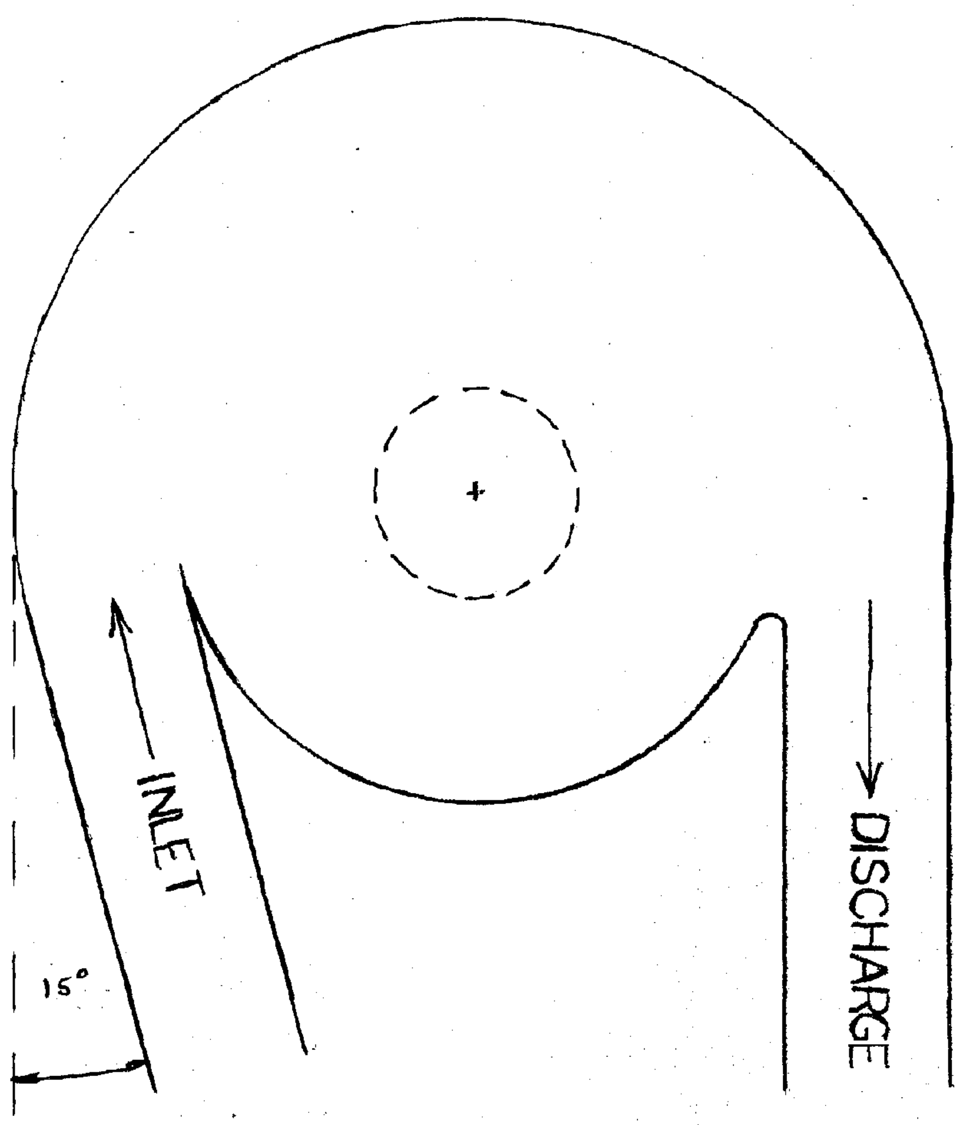


FIGURE 7
PROPOSED GEOMETRY

APPENDIX F

COAL DEGRADATION TEST

(Reference Phase III)

COAL DEGRADATION TEST

1.0 Objective of Test

To determine the degree and rate of coal degradation associated with repeated passes through the test system.

2.0 Test Method

Coal degradation tests were run using the Foster-Miller Associates' 427-foot test loop. The test was started by filling the hopper with fresh 3-inch top-sized coal. The coal was run through the test system and coal samples taken at different time intervals. Sieve analyses were performed on the coal samples to determine the amount of degradation caused by the test system. This test did not determine the location or component within the test system where the degradation occurred. Rather it showed that degradation did occur in the system and to what degree. Possible breakage areas included:

- Pipe line: In addition to breakage that would occur in a straight, smooth pipe, there is added breakage due to various discontinuities such as wyes, valves and so on. Line-induced breakage was not a major factor.

- Header box on dewatering screen - The most severe breakage probably occurred at the header box on the dewatering screen. In this case, the slurry traveling at a velocity in the range of 20 ft. per second, impinged directly on a 45-degree distributor plate. It was noted that the full slurry stream underwent this impact continuously, so that material circulating around the loop was exposed to this source of breakage many times.

- Dewatering Screen: While the dewatering screens involved some wear on the material, it did not involve high velocities and probably is a small factor.

- Transfer from dewatering screen to coal bin: Transfer to the coal bin may involve a vertical drop of as much as 10 feet depending on the fill level and flow patterns. Impact velocities were as high as 25 feet per second. This was potentially a severe cause of breakage.

- Transfer to the weigh belt from the coal bin:

Transfer to the weigh belt was believed to be relatively mild, only involving the grinding that may have occurred in moving the belt under the hopper and the hopper gate.

- Transfer from the weigh belt to the Jet Injector: This transfer involved a vertical drop of 4 to 6 feet, and sometimes involved impact on the inside of the front wall of the hopper. At times, a "shower" of small coal particles resulted from fracture of large pieces hitting the hopper wall. This factor was mainly associated with a high weigh-belt speed and could be controlled.

- Jet Injector: The Jet Injector itself was capable of causing relatively high impact velocities in the hopper and mixing tube as particles were accelerated to the final average value of 20 feet per second. It would be desirable to evaluate this component of breakage separately from all the others recounted above.

Were the facility to be redesigned to improve measurement of coal degradation, a number of changes would be required. First, it would be necessary to define the entire sample that was being circulated repeatedly through the Jet Injector and to arrange to have all portions of the sample see the same number of cycles through the test loop.

As one method, a loaded conveyor system could contain the entire sample to be tested. Sieve analysis would be required immediately before material was introduced into the injector and immediately after it exited the injector.

Brief consideration of a device that would not harm the sample makes the following scheme seem attractive.

A diverter could be installed at each sample location so that slurry could be diverted into a sample conduit or could be returned to the main slurry line. A rectangular conduit is suggested because of the simplicity of design of the diverting gate and the fact that it would be possible to use a thin, blade-like diverting member.

In order to decelerate slurry that had been diverted into the sample line, the most gentle means probably would be to allow the material to declerate in a free trajectory such that the acceleration of gravity would reduce the velocity to a low energy value. As an example, slurry moving 20 feet per second could be declerated in a free trajectory having a 6.25-foot vertical travel, a quite modest value in terms of equipment construction. If the material were caught near the top of such a trajectory, say one foot below the 6.25-foot level, most of its kinetic energy would have been converted to potential energy and little further degradation would be expected in the final deceleration and collection of the sample.

While development of an improved sampling device was not carried out, a basically attractive design is suggested for future use.

3.0 Degradation Test Results

Tables 1 through 4 list the results of sieve analysis on the degradation tests. Corresponding Figures 1 through 4 show the data in standard gradation curves of the per cent finer and per cent coarsere for sieve sizes of 3-inch, 2-inch, 1-inch, 1/2-inch, #3 sieve, #8 sieve and pan.

Figure 5 presents the same data replotted to show Per Cent Weight Retained as a function of cumulative weight of anthracite passed through the system.

4.0 Analysis of Results

It appears that very little material over 3-inches was present (as expected). Over 90 per cent of the original material was held on the 2-inch and 1-inch screens. As the test continued, the +2-inch material was reduced smoothly to about half its initial weight in 64 tons of throughput (roughly 5-8 passes) through the system.

In the same interval, +1-inch material increased by a factor of 2, indicating conversion of part of the 2-inch material to 1-inch material.

The third curve represents the balance of the material - distributed among 1/2-inch, #3, #8 and pan. It started at about 4 per cent by weight and increased smoothly to some 24 per cent at the 64-ton mark.

The same trends continued to the end of the test at 141 tons throughput - 10 to 15 passes. At this level, most of the material had been reduced to 1/2-inch or less.

5/20/82

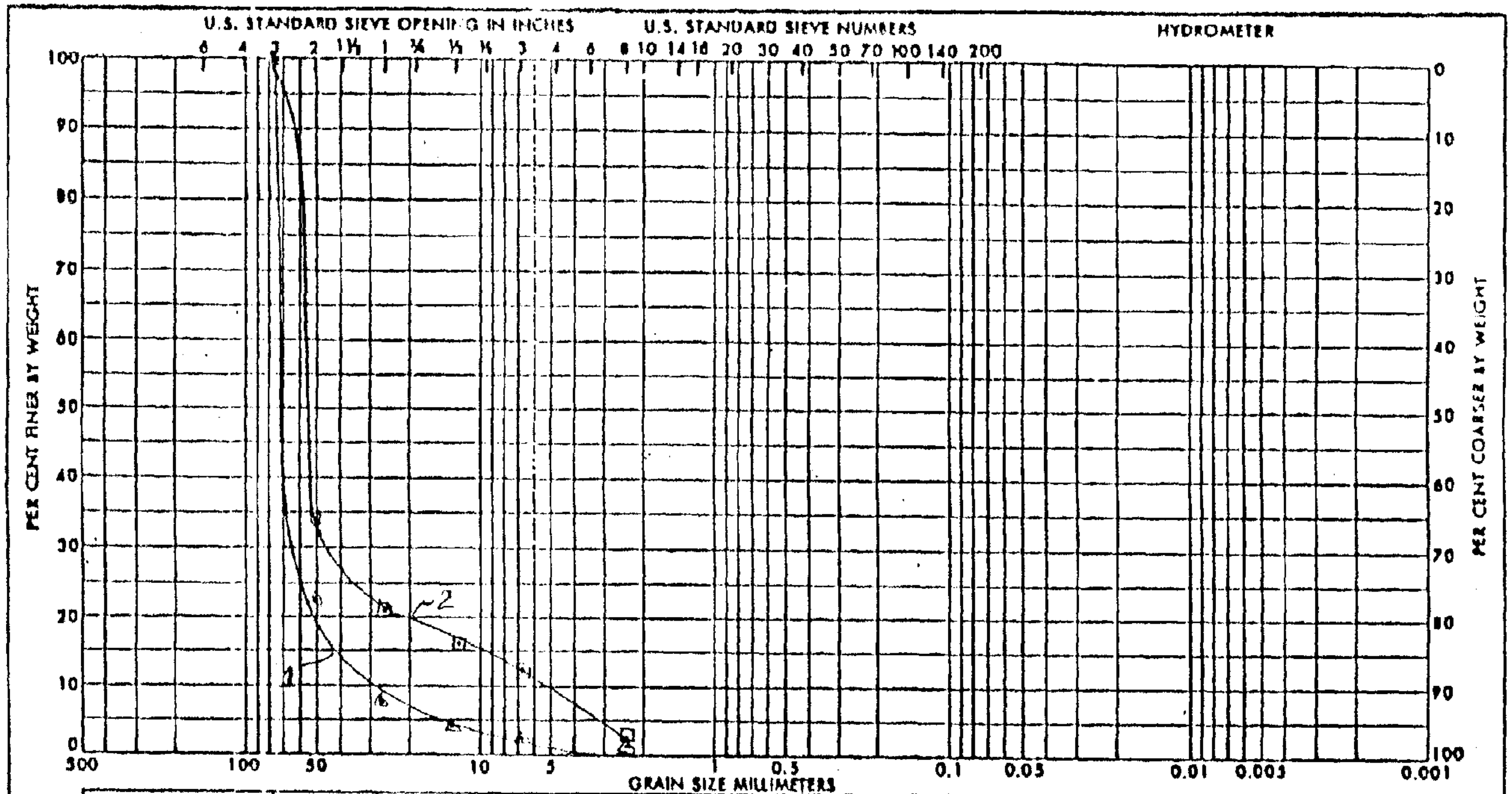
SAMPLE #1 TOTAL WT: 62.19 lb.

e	Wt. Retain	Corrected Wt. Retain	% Retain	% Passing	COMMENTS
	0	0	0	100.0	Start of Degradation Test - "New" 3-inch Anthracite
	47.30	47.48	76.38[2]	23.7	
	10.05	10.09	16.22	7.4	
	1.89	1.90	3.06	4.4	
	0.88	0.88	1.42	3.0	
	0.88	0.88	1.42	1.5	
	0.95	0.95	1.53	0	
	61.95#	62.19	99.98		
	$\frac{62.19}{61.95}=1.0039$	$\frac{100}{62.19}=1.61$	100		

'LE #2 TOTAL WT = 58.73 lbs.

'e	Wt. Retain	Corrected Wt. Retain	% Retain	% Passing	COMMENTS
	0	0	0.00	100.0	13.7 minutes, 45 tons after start of test
	34.55 lb	38.40	65.38	34.6	
	7.52 lb	8.36	14.23	20.4	
	2.42 lb	2.69	4.58	15.8	
	1.61 lb	1.79	3.05	12.8	
	5.47 lb	6.08	10.35	2.4	
	1.27 lb	1.41	2.40	0	
	52.84 lb	58.73	100.0		
	$\frac{58.73}{52.84}=1.1$	$\frac{100}{58.73}=1.70$			

.E 1 Sieve Analyses For Degradation
Samples #1 and #2



SAMPLE NO.	ELEV OR DEPTH	CLASSIFICATION	NAT W%	LL	PL	PI
1,2		Anthracite				

PROJECT Jet Injector Degradation Test
AREA
BORING NO. 5-11-82
DATE

GRADATION CURVES

Figure One - Sieve Analyses for Degradation Samples 1 and 2

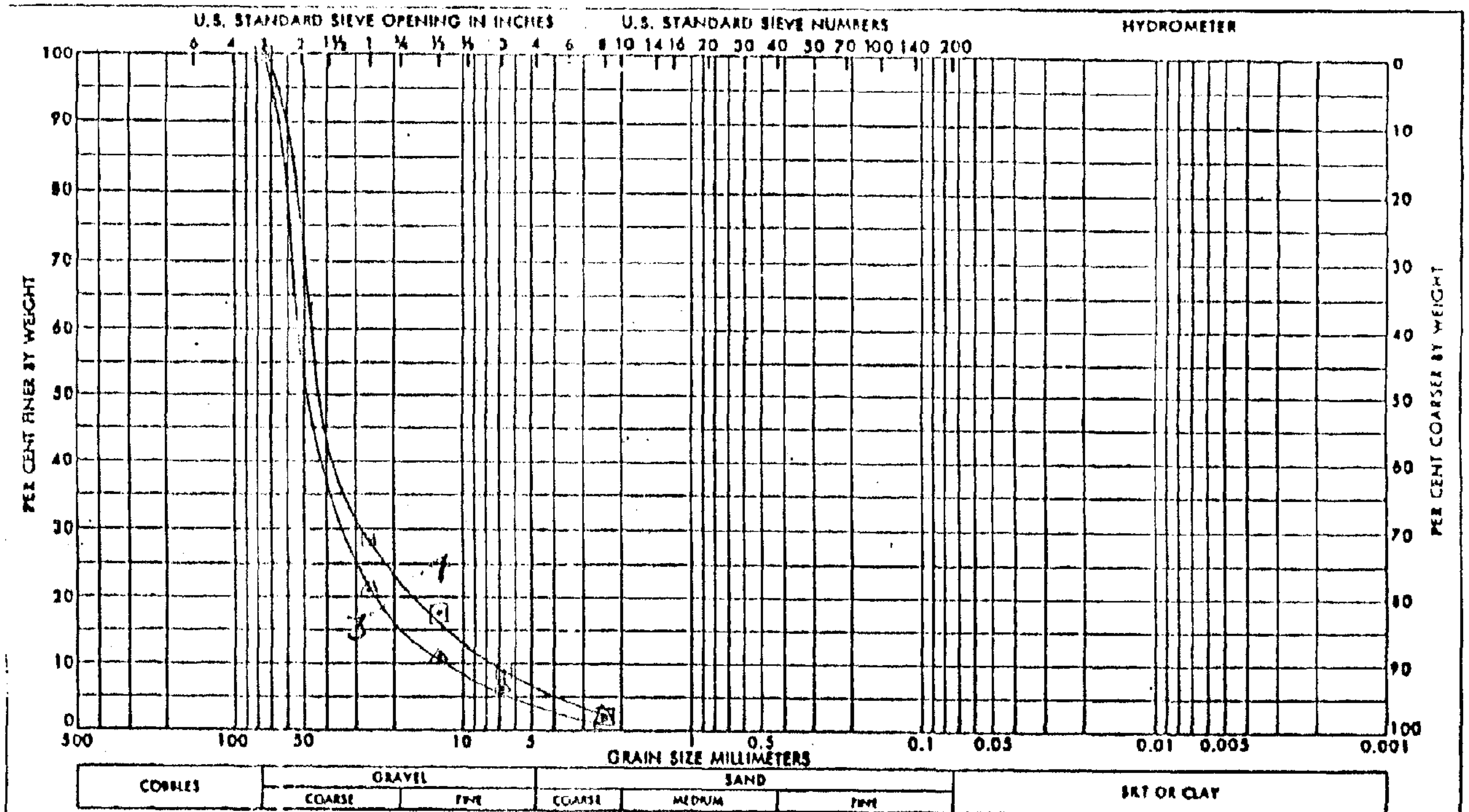
SAMPLE #3 TOTAL WT. = 54.84 lb.

Sieve Size	Wt. Retain	Correct Wt. Retain	% Retain	% Passing	Comments
3	0	0	0	100	15 Minutes
2	25.75	25.73	46.98	53.1	49 tons after start
1	17.84	17.83	32.51	20.6	of test
1/2	4.91	4.91	8.95	11.6	
#3	2.81	2.81	5.12	6.5	
#8	2.41	2.41	4.39	2.1	
Pan	1.16	1.16	2.12	0.0	
	54.88	54.84			
CF	$\frac{54.88}{54.88} = 1.00$	$\frac{100}{54.84} = 1.82$			

SAMPLE #4 TOTAL WT. = 40.91 lb.

Sieve Size	Wt. Retain	Corrected Wt. Retain	% Retain	% Passing	Comments
3	0	0.00	0.00	100.0	20 Minutes
2	14.75	14.82	36.223	63.8	64 tons after start
1	14.63	14.69	35.90	27.9	of test
1/2	4.48	4.50	11.00	16.9	
#3	3.43	3.50	8.55	8.3	
#8	2.73	2.74	6.70	1.6	
Pan	0.66	0.66	1.61	0.0	
	40.73 #	40.91	99.99		
CF	$\frac{40.91}{40.73} = 1.00$	$\frac{100}{40.91} = 2.44$	100.0		

TABLE 2 - Sieve Analyses for
Samples #3 and #4



SAMPLE NO.	ELEV OR DEPTH	CLASSIFICATION	NAT W%	LL	PL	PI
3,4		Anthracite				

PROJECT Jet Injector Degradation Test

AREA

BORING NO.

DATE 5-11-82

GRADATION CURVES

FIGURE 2 - Sieve Analyses for Degradation Samples 3 and 4

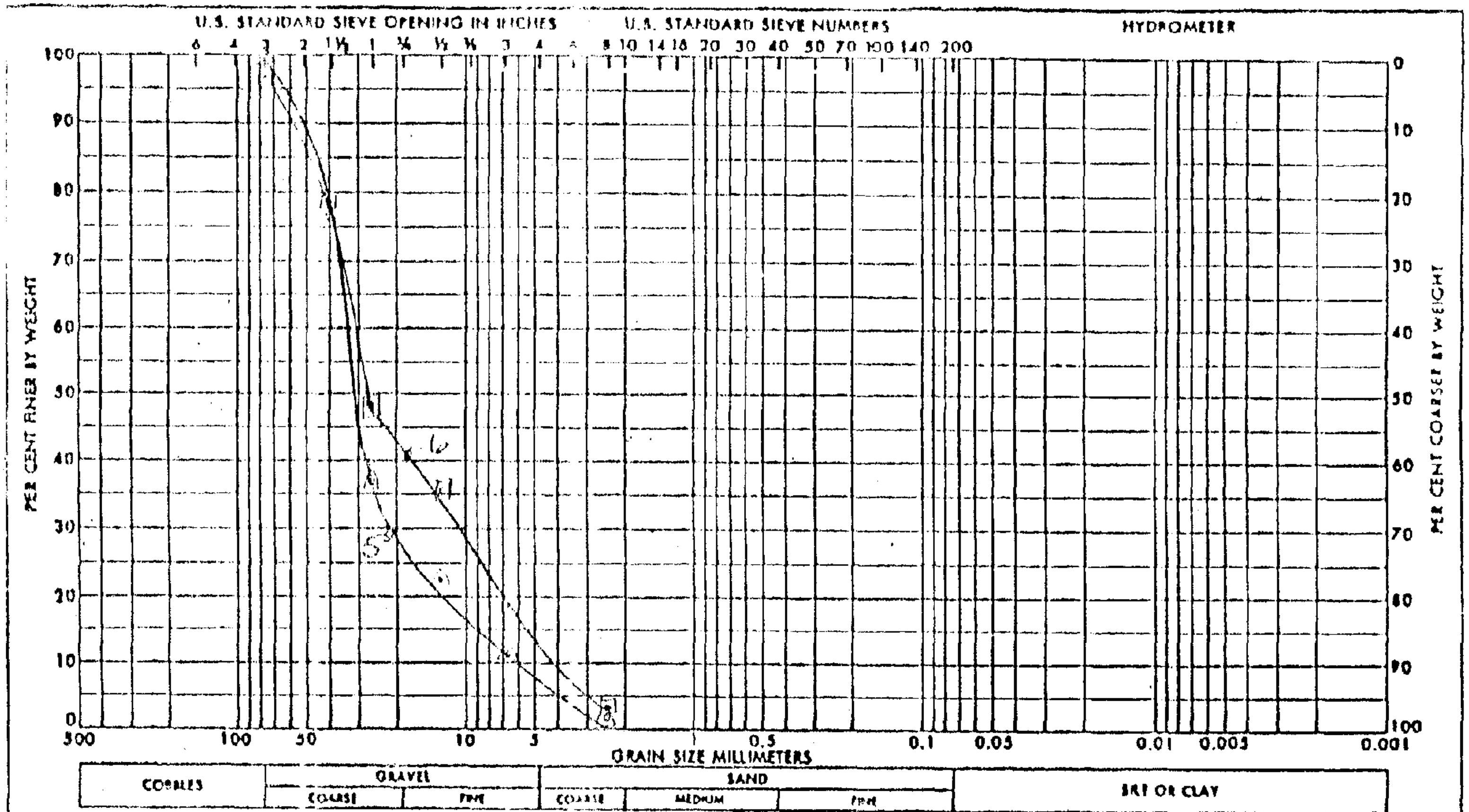
SAMPLE #5 TOTAL WT. 58.92 lb.

Sieve Size	Wt. Retain	Corrected Wt. Retain	% Retain	% Passing	Comments
3	0	0.00	0.00	100.0	27 minutes,
2	14.25	14.40	24.45	75.6	86 tons after
1	22.91	23.15	39.30	36.2	start of test
1/2	7.83	7.19	13.42	22.8	
#3	6.78	6.85	11.63	11.2	
#8	5.75	5.81	9.86	1.3	
Pan	0.78	0.79	1.34	0	
	58.30	58.92	99.98		
CF	$\frac{58.92}{58.30}$=1.01	$\frac{100}{58.92}$=1.70	100.00		

SAMPLE #6 TOTAL WT. = 66.42#

Sieve Size	Wt. Retain	Corrected Wt. Retain	% Retain	% Passing	Comments
3	1.83	1.87	2.82	97.2	37 minutes,
2	12.64	12.95	19.50	77.7	123 tons after
1	19.14	19.62	29.52	48.2	start of test
1/2	8.30	8.50	12.80	35.4	
#3	11.03	11.30	17.01	18.4	
#8	10.48	10.74	16.17	2.2	
Pan	1.42	1.45	2.18	0.0	
	64.84#	66.42#	100.00		
CF	$\frac{66.42}{64.89}$=1.02	$\frac{100}{66.42}$=1.51			

TABLE 3 - Sieve Analyses for
Samples 5 and 6



SAMPLE NO.	ELEV OR DEPTH	CLASSIFICATION	NAT W%	LL	PL	PI
5,6		Antracite				

GRADATION CURVES

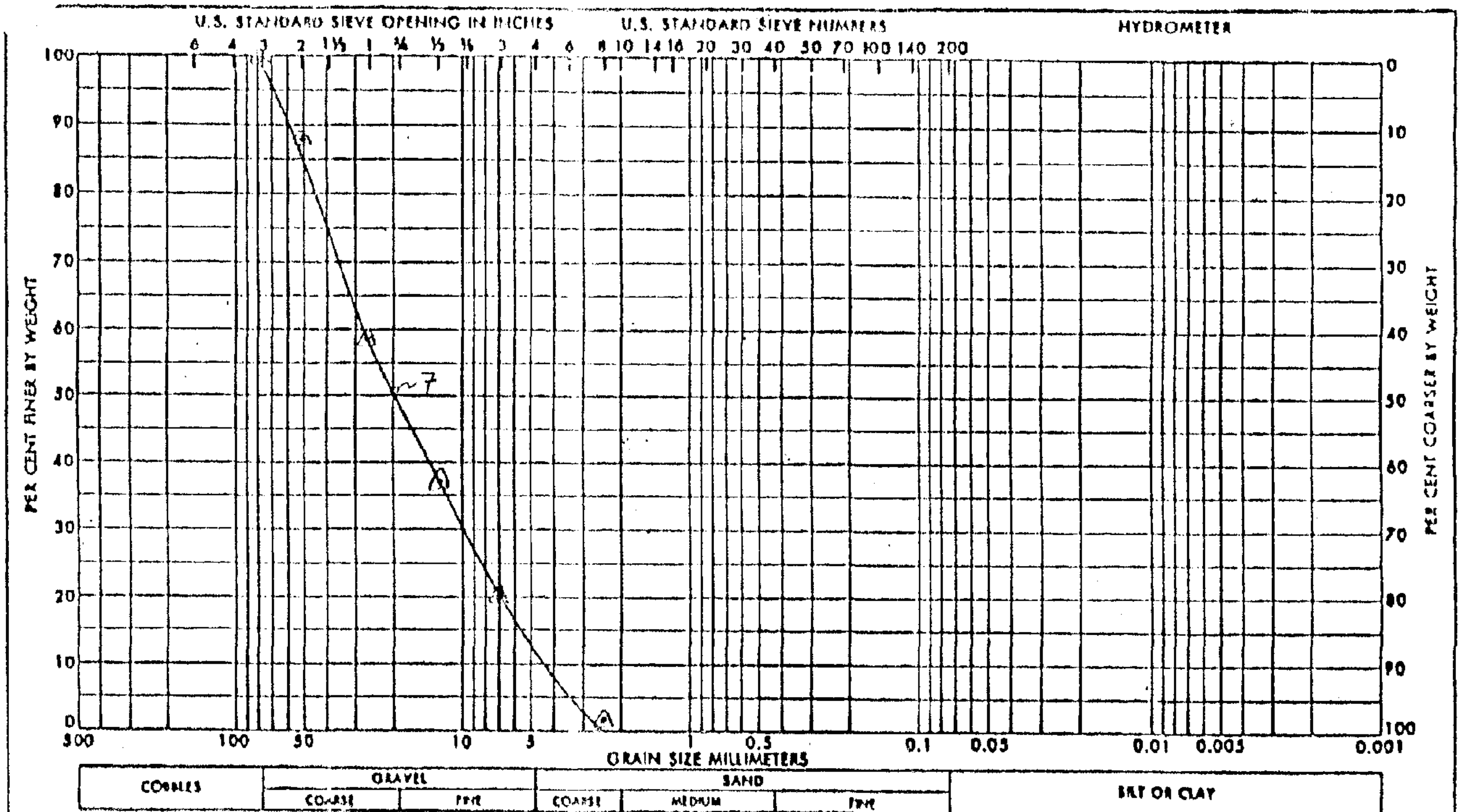
PROJECT Jet Injector
Degradation Test
AREA
BORING NO.
DATE 5-11-82

Figure 3 - Sieve Analyses for Degradation
Samples 5 and 6

SAMPLE #7 TOTAL WT. = 48.25#

Sieve Size	Wt. Retain	Corrected Wt. Retain	% Retain	% Passing	Comments
3	0	0.00	0.00	100.0	41 minutes,
2	6.36	5.79	12.00	88.0	141 tons after
1	15.70	14.29	29.62	58.4	start of test
1/2	11.91	10.84	22.47	35.9	
#3	8.42	7.66	15.88	20.0	
#8	9.70	8.83	18.30	1.7	
Pan	0.92	0.84	1.74	0	
	53.01	48.25#	100.00		
CF	$\frac{48.25}{53.01}=0.91$	$\frac{100.00}{98.28}=2.07$			

TABLE 4 - Sieve Analysis for Sample #7



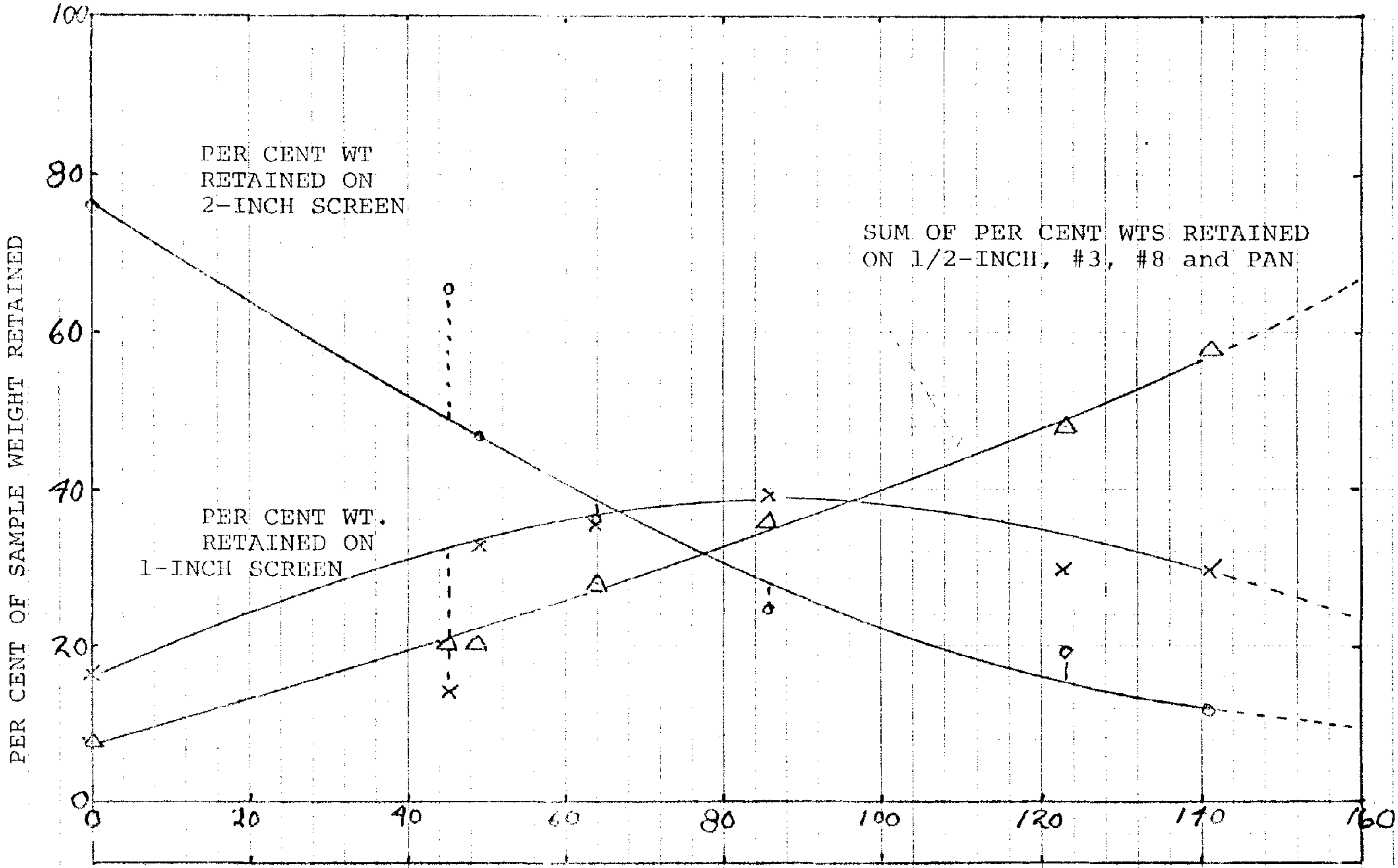
SAMPLE NO.	ELEV OR DEPTH	CLASSIFICATION	NAT W%	LL	PL	PI	PROJECT Jet Injector Degradation Test
7		Anthracite					AREA
							BORING NO.
GRADATION CURVES							DATE 5-11-82

Figure 4 - Sieve Analysis for Degradation

Sample No. 7

TEST	TAPE FOOTAGE	TPM	TIME MIN	TONS	TONS	NOTES
CT-13-1	2250-2278	1.83	2.99	5.5	5.5	Sample #1
CT-13-2	2278-2303	2.57	2.67	14.6	20.1	@ 2250
CT-13-3	2303-2309	4.50	.64	2.9	23.0	LE = 6.1"
CT-13-4	2309-2379	2.89	7.47	21.6	44.6	Sample #2
CT-13-5	2380-2392	3.10	1.28	4.0	48.6	Sample #3
CT-13-6	2403-2453	2.84	5.33	15.1	63.7	Sample #4
CT-13-7	2470-2535	3.29	6.93	22.8	86.5	Sample #5
CT-14-1	2586-2610	3.57	2.56	9.1	9.56	LE = 9.1"
CT-14-2	2610-2636	2.86	2.77	7.9	104	
CT-14-3	2657-2664	5.23	.78	3.9	107	
CT-14-4	2694-2708	4.02	1.49	6.0	113	
CT-14-5	2708-2728	4.36	2.13	9.3	123	Sample #6
CT-15-1	2757-2769	4.04	1.71	6.9	130	LE = 6.1"
CT-15-2	2767-2777	4.33	1.07	4.6	134	
CT-15-3	2777-2789	5.59	1.28	7.2	141	Sample #7

TABLE 5 COAL RATES FOR DEGRADATION TESTS



CUMULATIVE WEIGHT OF ANTHRACITE CIRCULATED - TONS
FIGURE 5 DEGRADATION OF SOLIDS RECIRCULATED
THROUGH THE TEST LOOP 15-20* TIMES

* See Text