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THE DEVELOPMENT OF A LONGWALL WATER JET MINING MACHINE

Final Technical Report
Contractor—University of Missouri—Rolla

July 1981

Contract No. U.S.D.O.E. AC01-75ET12542
(formerly U.S.B.M. H0252037)

U. S. Department of Energy
Assistant Secretary for Fossil Energy
Office of Coal Mining

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THE DEVELOPMENT OF A LONGWALL WATER JET MINING MACHINE

Final Report as of July 1981
on Contract DOE-AC01-75ET12542

Submitted by

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ABSTRACT

The contract initial objective was to construct a prototype longwall water jet mining machine and evaluate its performance by field testing the unit in a strip mine coal panel. The design of the machine, Hydrominer I, was based on USBM Contract H0232064.

The second objective was to design and construct an improved cutting head, Hydrominer II, which would be tested in an artificial coal heading.

The testing of Hydrominer I in coal indicated significant beneficial features resulting from the application of water jet cutting to coal mining. These features were the elimination of dust, the reduction of the danger of gas and dust explosions from machine generated sparks, the ability to cut a wider web with a lower haulage force requirement than for a shearer, and the production of larger coal with fewer fines. However, the production rate was limited by inherent machine design problems which prevented Hydrominer I from achieving its full potential.

The design generated for Hydrominer II was intended to correct the difficulties of the first generation machine and allow the full productive capability of the concept to be achieved while retaining the beneficial features clearly proven in the field tests of Hydrominer I. However, the artificial coal in which Hydrominer II was to be tested did not respond to the jet cutting in a manner similar to that of coal.

Therefore, no conclusions can be drawn in regard to the performance of Hydrominer II based on the artificial coal tests.

EXECUTIVE SUMMARY

On May 7, 1975 Contract H0252037 was entered into between the U.S. Bureau of Mines and the University of Missouri to furnish a prototype longwall water jet cutting machine and to verify its performance by a series of development experiments. The design of the water jet mining machine was proposed and developed under a previous Contract H0232064 which was completed in November 1974.

Originally Contract H0252037 was broken into five phases which are as follows. Phase I would include the fabrication of the cutting head as proposed in the earlier contract, and a study to improve the water jet nozzles to be used in the cutting head would be carried out. The laboratory tests had shown that the existing nozzles could cut a slot approximately 2 inches wide to a depth of 4 inches in coal samples. This study was aimed toward increasing the ability of the nozzles to cut a much deeper slot.

In Phase II, the cutting head would be attached to an equipment sled which would carry the high pressure pump and other equipment necessary to perform the field tests. The experimental unit would be transported to a surface strip coal mine where a series of tests would be carried out to verify the operational parameters of the equipment. Information such as the forces acting on the cutting head during the mining process would be recorded prior to the completion of the prototype unit.

During Phase III an existing production longwall shearer would be modified by replacing the drums, arms, and drive train of the bidirectional unit with two modified versions of the experimental cutting head tested in Phase II. These cutting heads would be designed to include changes indicated by the field trials and would employ two high pressure pumps feeding into the head which was in operation on a given pass of the machine.

In Phase IV, this experimental mining unit would be taken underground and tests would be carried out to prove the practicality of the system and to demonstrate its advantages over existing mining equipment. A final report would be submitted in Phase V describing the work performed.

At the beginning of the contract the scope of the work was reorganized slightly into six phases rather than the original five. There were, however, no significant changes proposed for the complete program. Later, however, several significant modifications were made which included adding a subcontract to Jeffrey Mining Machinery Division of Dresser Industries to assist in adapting the water jet equipment to a BJD shearer.

Later, it was decided to depart from this course of action, and instead build a single second generation cutting head that would be tested in artificial coal at Bruceton, Pa. In this case, a Joy shearer would be used to move the water jet cutting head through the artificial coal, and an equipment sled would be placed between the shearer and the cutting head. The design and construction of this sled became a part of the University effort as a result of these changes. After the difficulties experienced with the artificial coal mixture, the contract scope was again modified to include a study to define a correct artificial coal mixture for testing water jet equipment.

The actual contract work with these modifications followed the pattern described in the following chapters. Hydrominer I was fabricated from standard mild steel structural elements such as plate, angle iron, and channel iron. Extensive use was made in the fabrication process of arc welding to join the basic framework together. Commercially available equipment was purchased to furnish the high pressure rotary swivels needed to permit the jet cutting arms to oscillate. Conventional hydraulic motors were used to drive the linkage which provided the arm motion.

Experiments were conducted to improve the ability of the water jet to remain coherent for a longer distance from the nozzle while the cutting arm was being oscillated. Significant improvements were possible by using good fluid mechanics' guidelines to divide the flow and expel the fluid from the nozzle without gross irregularities in the channel cross-section. The original nozzles would only cut to a depth of 4 inches in laboratory samples of coal. In the field tests, the improved nozzles cut slots to a depth of 24 to 30 inches ahead of the cutting head.

For the field trials a single Kobe 4J high pressure pump was mounted on an equipment sled with a 150 hp electric motor to drive the pump. Also on the sled was a hydraulic system to provide power to oscillate the jet cutting arms. A variable speed winch was designed and constructed to pull the cutting head and sled across the coal panel at fixed speeds of 1, 2, 5, 10, 20, and 30 ft/min.

When the equipment was moved to the field test site and assembled, experiments were begun using the 20^{o} diverging nozzles which had been successful in the laboratory. However, these nozzles were manufactured according to the improved design developed earlier in the contract and they were able to cut much further ahead of the machine than the laboratory tests had indicated. Consequently, the geometry of the nozzle had to be modified to reduce the included angle between the jets.

In addition to the nozzle problem, a major problem arose when particles of coal entered into the cutting head and jammed the motion of the jet cutting arms. This difficulty was partially overcome by designing and installing slot guards to reduce the volume of the debris which entered the interior of the cutting head.

Another problem occurred when lenses of pyrite were present in the area directly in front of the cutting head. Since this material was much more difficult to cut than the coal, a slot wide enough for the leading edge of the cutting head to enter was not always generated. In these cases, the head would jam, stopping its forward motion.

When the head was operating well enough to advance significant distances down the face, it became apparent that the absence of a conveyor to move the coal away from the cutting head was a serious handicap. The volume of coal mined would be loaded onto the equipment sled (where a conveyor would normally be located), but the coal would accumulate in this area and effectively bury the head in coal. The only solution for this problem was to limit the distance the head could advance down the face, and remove the coal between test runs.

Despite the number of problems which surfaced during the field tests, sufficient useful experience was gained in testing the machine to indicate significant beneficial features of a water jet coal mining machine. For example, the coal was mined without generating dust. The size of the coal produced was for the most part larger than 1 inch. No sparks were generated in mining the coal because the portions of the machine which contact the coal are moving with a low velocity. The width of the web taken did not sifnificantly effect the haulage force required to move the head through the coal.

After a careful analysis of the design problems inherent in Hydrominer I, a design was developed for Hydrominer II which was intended to solve these problems. The second generation cutting head was constructed along with an equipment sled which was located between Hydrominer II and the Joy shearer at the Bruceton test facility. The material which was to be used for this series of tests was an artificial coal. The artificial coal did not cut well and its response to the water jet cutting was totally different from the response of coal. Therefore, no conclusions could be drawn from the testing of Hydrominer II.

As a result of the problem encountered with the artificial coal, a search for a suitable material was conducted. The use of a matrix of sand and masonry cement to bind lumps of coal into a solid block worked well in the laboratory tests. This material responds in the same fashion as coal to water jet cutting, and it would be possible to cast large blocks to evaluate the performance of Hydrominer II in a controlled surface test.

At this time, the conceptual idea of the Hydrominer has been successfully proven within the initial parameters established at the start of the contract. However, the full productive capability of the machine has not been proven, and it is unlikely that this will occur until tests are conducted on an underground face. Because there is a great deal of testing which should be done before an underground trial would be possible, it is recommended that Hydrominer II be tested on the surface, initially in an artificial coal, and then in a surface strip panel.

At the conclusion of this test program, it should be possible to fully assess the potential benefits likely from any further development of the Hydrominer. At that stage, a decision should be made as to the future commercial possibilities of the machine.

ACKNOWLEDGMENT

The authors are grateful for the cooperation of the U.S.B.M. and the D.O.E. during the course of the research program and the preparation of this report. In particular we wish to thank Charles Rhoades and John Corwine of the U.S.B.M. for their assistance in the early phase of the program. We also wish to thank Richard Markeley, Anthony Sharkey, and Jasinder S. Jaspal of D.O.E. for their assistance. A special note of thanks should be given to Dr. Lyle Rhea and Dr. Marian Mazurkiewicz for their work on the project. Many others too numerous to mention, provided invaluable help and advice which was appreciated.

Chapter One

CONSTRUCTION OF AN EXPERIMENTAL CUTTING HEAD

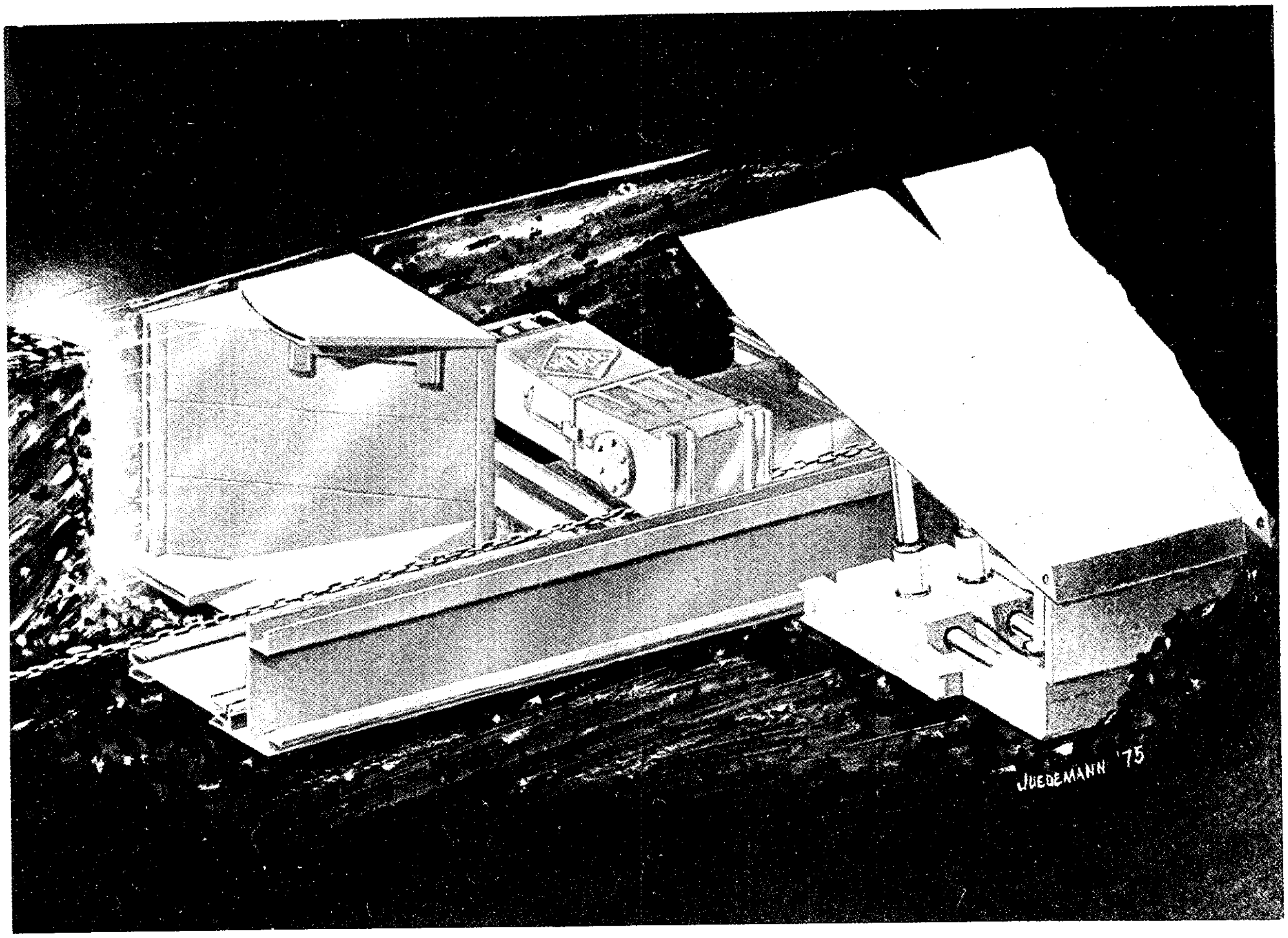
Introduction

The concept of using high pressure water jets as a means of excavating coal from a longwall face was originally developed at the University of Missouri-Rolla for a paper presented (Ref. 1) in October of 1971. This led to a contract between the U.S. Bureau of Mines and the University (Ref. 2) to evaluate the feasibility of the concept through experimentation and a prototype design. The basic performance parameters required for jet cutting to be viable were developed in the experiments and a preliminary machine design completed and submitted to the Bureau in November of 1974. The prototype design conceived for Hydrominer I is illustrated in Fig. 1.1. The Hydrominer water jet cutting head is shown in this illustration working with an existing conveyor and roof support system to mine coal in a longwall face. The artist's concept shows moving water jets cutting at the floor, back, and roof of the coal seam to isolate a pillar of coal which is then mechanically fractured as the body of the cutting head advances into the slots cut by the water jets.

In May of 1975, a contract was entered into between the University and the Bureau of Mines to fabricate a preliminary cutting head based on the design previously submitted, to field test it in a surface mine, and, based on the results of these tests, to modify the head and to develop a working prototype mining machine for use on an actual longwall face.

Figure 1.1

Artist's Concept of Hydrominer I



The water jets shown at the left side of the photograph would oscillate to cut slots at the bottom, back, and top of the coal seam. The wedge shaped body of Hydrominer I would continuously advance into these slots breaking the coal and plowing it over onto the adjacent conveyor. The roof supports and conveyor would be chosen from standard mining equipment.

During the first phase of this program, it was decided to modify the program and instead of the ultimate objective becoming the complete construction of a mining machine, it was intended that the program would be completed with the development and delivery of a second generation cutting head to the Bureau.

Much of the earlier work on this contract has previously been described in detail in the Phase Reports and Monthly Progress Reports submitted to the Bureau of Mines and to the Department of Energy. However, for the purposes of establishing a single complete record, much of that information will be repeated in this document. The major details of the experiments and the designs have been compiled within the phase reports and particularly for the earlier work, only the highlights of the program are described in the main body of the test.

Phase I of the program, which is described in this chapter, essentially comprised two separate tasks. Firstly, the development and construction of the first generation cutting head, and secondly, experimentation leading toward an improvement in the design of the nozzles which could be used on this cutting head. These two sections are herewith described.

Construction of the First Cutting Head

The initial design of the high pressure water jet cutting head was developed under Contract H0232064 and formed at the same time a thesis topic for a graduate student in Mechanical Engineering (Ref. 3). Consequent to the awarding of this contract, a structural analysis of this preliminary design was undertaken and it was concluded that the initial design would have insufficient strength for the purposes required. The head was therefore modified by adding stiffening plates to the assembly. At the same time, an analysis of the drive mechanism for the oscillating arms indicated that the initial methods chosen for this drive would be highly sensitive to the hostile environment likely to be encountered in an operating longwall face, and for this reason, the drive mechanism was also re-designed.

The major changes in the design developed over that initially submitted can be seen by comparing Fig. 1.2 which shows the initial design, with Fig. 1.3 which shows the modified design, as it was ultimately constructed. Major benefits of this modification were not only that a positive drive throughout the stroke of the cutting arms was achieved, but also that all electrical power was removed from the cutting head and replaced by hydraulic power. This would improve the safety of the machine since the cutting head operates in the area of greatest methane emission on the longwall face. This head was constructed (Fig. 1.4) and a film record made of the operation of the head in the laboratory.

The final head configuration achieved for Hydrominer I was, in relation to the design of the second head (Chapter 5), a complicated mechanism. However, it should be understood that the objectives underlying the design of these two heads differ considerably. The first head developed was an experimental device, and it was anticipated at the time that the design was undertaken that a considerable effort would be made in the test program to optimize the operating parameters of the unit. Later,

Figure 1.2
Mainframe of Hydrominer I
with Coverplates Removed - Original Concept

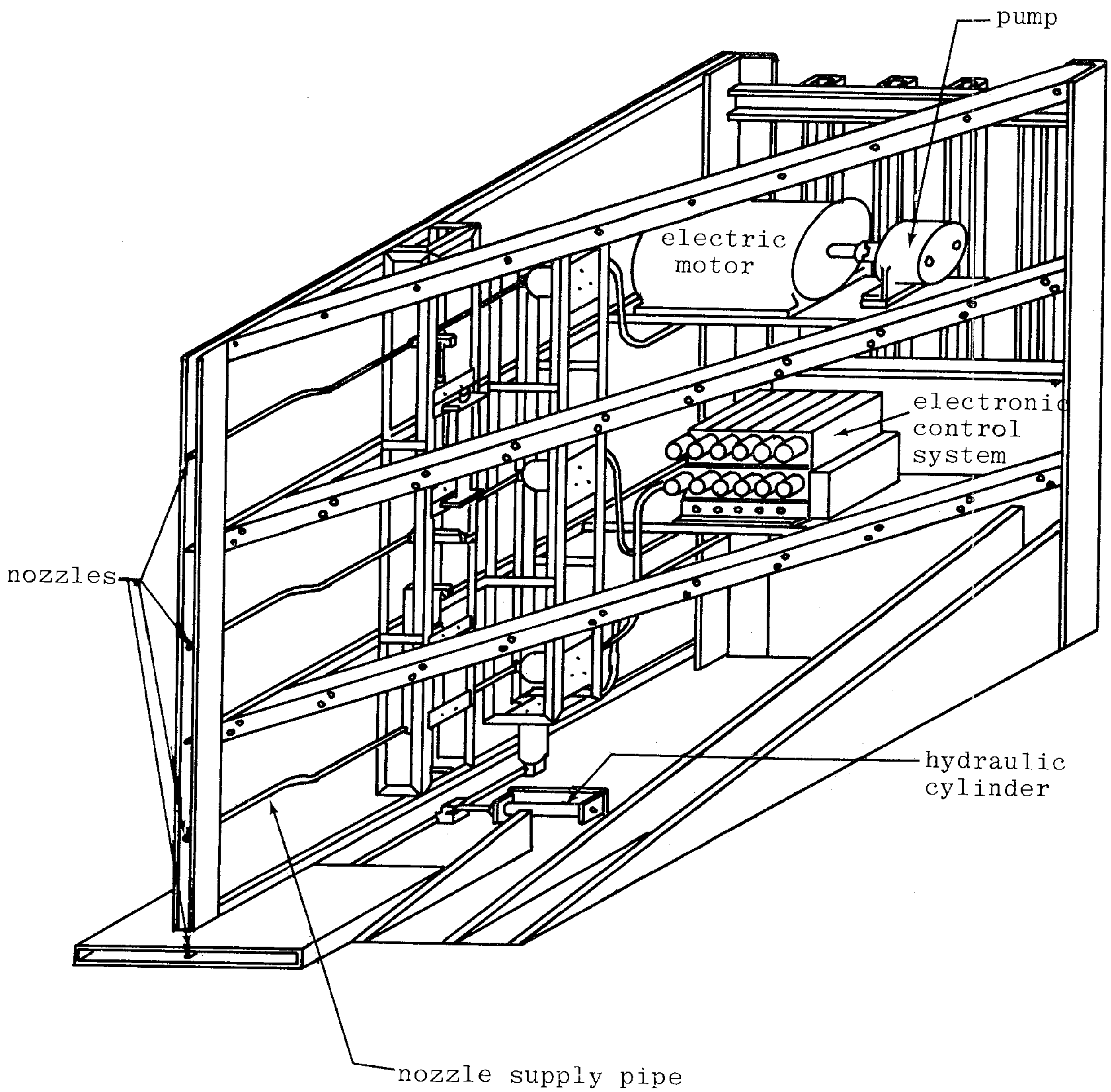


Figure 1.3
Mainframe of Hydrominer I
as Actually Fabricated

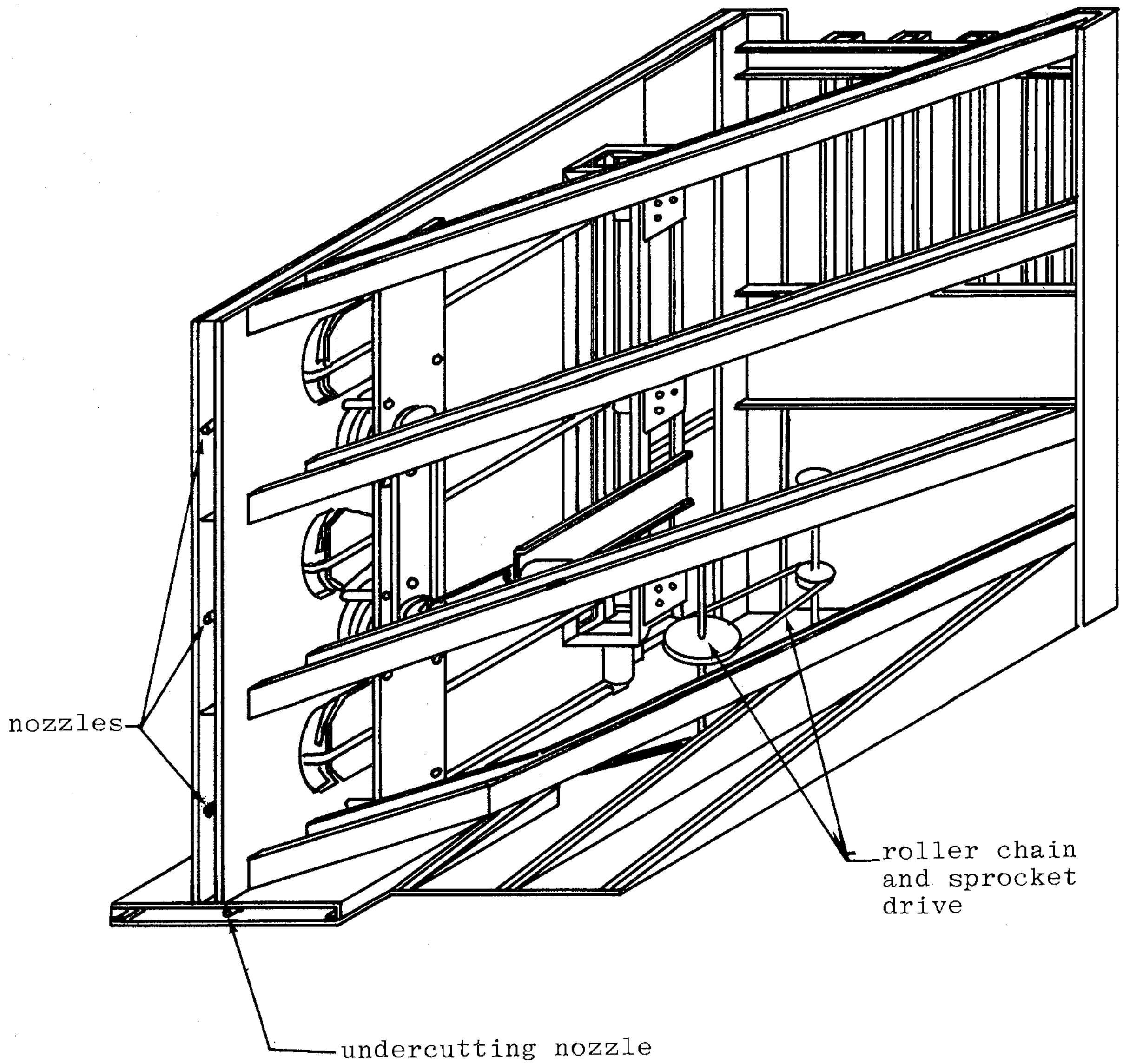
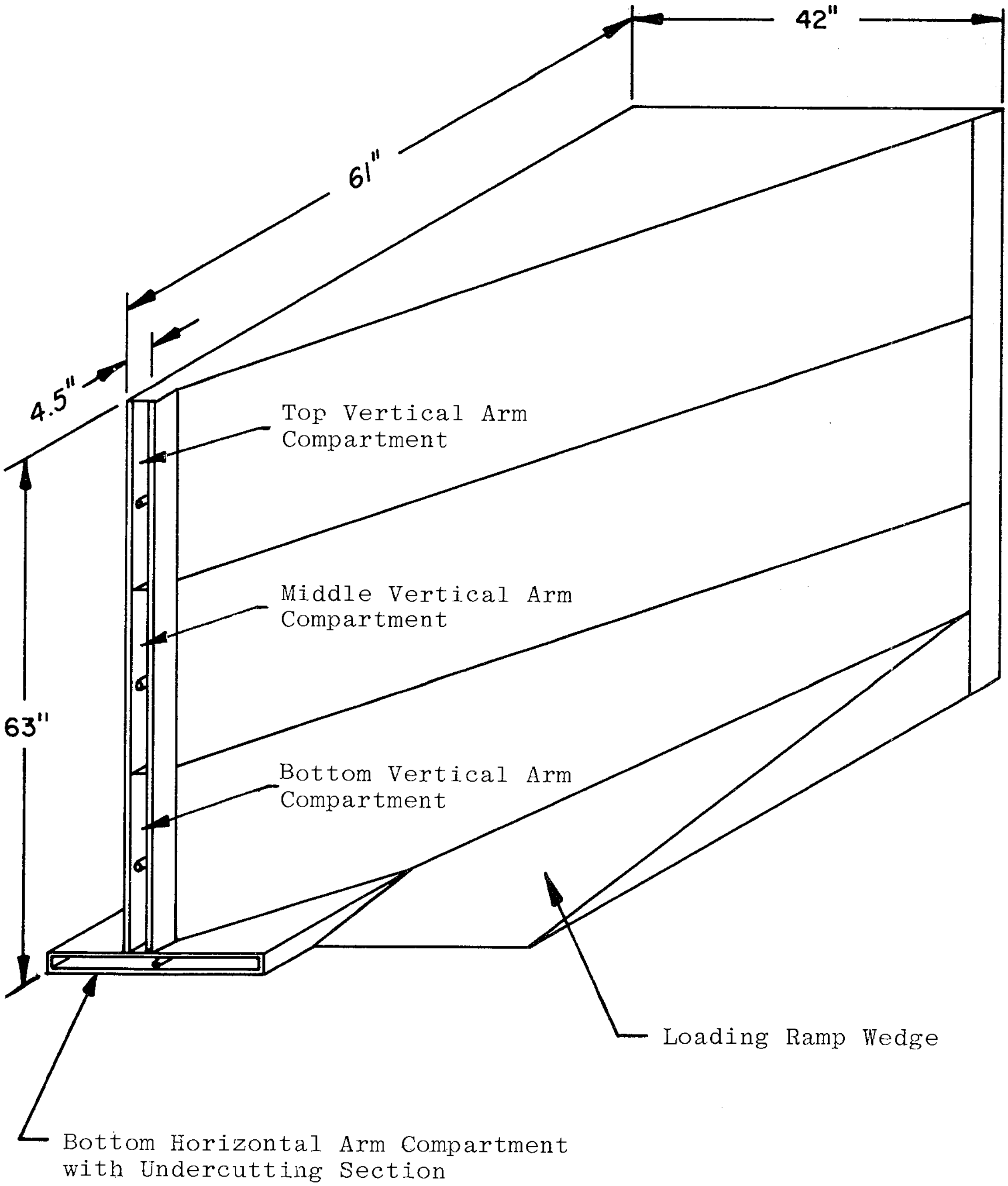


Figure 1.4
Hydrominer I Cutting Head as Used
in the Field Trial Cutting Tests



the contract was modified so that this anticipated research was not included in the program.

Since the second head was designed to correct the deficiences in the mechanical design of the first head and to provide a unit to indicate machine performance under production conditions, the designs are quite different.

Nozzle Configuration Tests

The initial conceptual design of the Hydrominer was based on the method of mining of the Meco Moore mining machine. This mining machine, which was the most productive machine on longwall faces until the advent of the Anderton shearer-loader, mines coal by use of three mechanical cutter jibs. The first jib is located at the bottom of the seam and undercuts to a depth of 6 ft. At the back of this cut, a second vertical cutter jib cuts out the back of the face, while the third cutter jib is located within the seam section to cut the cantilever of coal thus outlined in approximate halves and this controls the size of the fragments removed. The unit is completed with a cross-drive conveyor which travels behind the cutting jibs carrying the coal which falls from the solid under gravity loading over to the face conveyor.

The design was also based on the results of the testing of the Huwood Slicer. The Huwood Slicer was essentially a vibrating coal plough with a series of sharp vibrating knives on the leading edge of the plough cutting out ahead of the machine. Discussions with personnel at the National Coal Board associated with the testing of this machine (Ref. 4) indicated that a 3 in. cut ahead of the machine would give the necessary mechanical advantage with a 35 degree rake of the plough body in order to mine off a 2 ft thick panel of coal.

Laboratory experiments carried out on the earlier contract had shown that a 4 in. depth of cut in coal was achievable with the water jet pressures and flow rates selected for this program. However, increased performance would results if a better nozzle design was developed.

It was also necessary to redesign the nozzles since the jets were initially located too close together and interference between them occurred. The development of the nozzle is shown from the initial configuration (Fig. 1.5) to that developed (Fig. 1.6). Tests were also carried out with this nozzle to ensure that the jet would remain coherent out to the required distance at the nozzle traverse velocities required for the cutting stroke. This was achieved using high speed photography and the results indicated that over the required distance from the nozzles the jet remained coherent up to oscillation speeds of 200 rpm. This speed was above the level anticipated for field operating performances. The details of these experiments are described in the Appendix 2.

The nozzle arrangement shown in Fig. 1.5 is one in which the high pressure water stream from the cutting arm supply pipe is divided into two jets directed at an angle to "diverge" from one another. Thus this type of nozzle is called a dual orifice diverging nozzle. Other nozzles were machined so that the two jet streams met at a common point and these are referred to as dual orifice converging nozzles. In some

Figure 1.5
Initial Nozzle Design

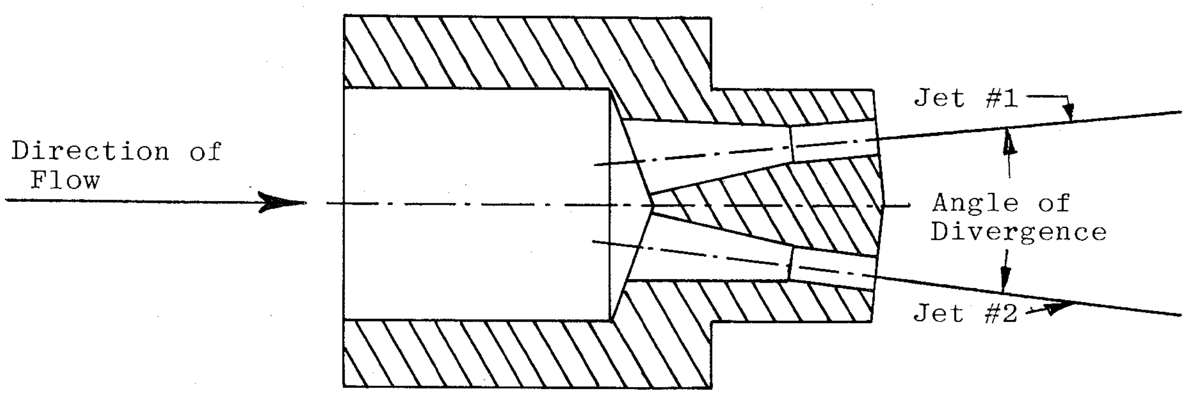
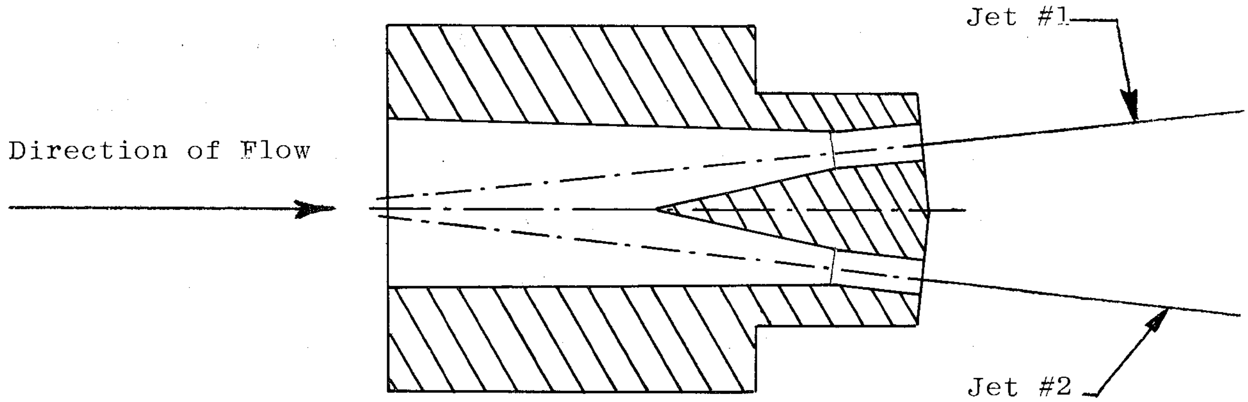


Figure 1.6
Improved Nozzle Design



cases the two jets were neither divergent nor convergent and the resulting nozzle was a parallel configuration.

Support Sled and Field Test Plan

During this phase of the program, the design of the support sled capable of carrying the high pressure water pump, motor, and switch gear was also undertaken and the program of experimentation for the field testing was developed. It was agreed that by field test the machine should prove capable of running at 5 ft/min on a 2 ft depth of cut in a 32 in. high seam of coal. This would entail using only one pump for machine operation rather than the two pumps which would be required if a full 5 ft seam sections were taken.

Chapter Two

SUPPORT FRAME CONSTRUCTION AND FIELD TEST EQUIPMENT DEVELOPMENT

Introduction

The first experimental water jet cutting head was completed during Phase I of the program. Upon approval from the U.S. Bureau of Mines, Phase II of the program was carried out. In this phase the cutting head was attached to a steel platform carrying the support equipment required to generate the high pressure water jets at the nozzles.

Nozzle Selection

Experiments in the laboratory at the beginning of the first phase of the program (Ref. 2) had shown that a water jet pressure of 10,000 psi issuing through five dual orifice nozzles, with each orifice being .04 in. in diameter, would be required as the cutting mechanism on the mining machine. Dual orifice nozzles appeared to be required in order to cut a slot wide enough to introduce the leading edge of the main body of the machine into the cut. For full machine operation, the five nozzles would require a flow rate of approximately 50 gpm at 10,000 psi necessitating 300 hp of pump delivery. It is worthy of note that this flow rate is less than that used on many shearers for dust suppression (Ref. 5).

The operating pressure, flow rate of the water, and input power are related by the equation

$$\text{bhp} = \frac{\text{gpm x psi}}{\text{1714 x eff}}$$

where bhp = brake horsepower
gpm = water flow rate in gallons per minute
psi = pump pressure in pounds per square inch
eff = mechanical efficiency

The flow rate of water can then be divided to power one or more nozzles which must be sized to provide the proper restriction of the flow to maintain the operating pressure. The exit diameter of the nozzles can be calculated for a given operating point, but may have to be adjusted slightly to compensate for pressure losses in transporting the water from the pump to the nozzle.

In general, the smallest number of nozzle exit orifices which will generate the required slot will prove to be the most satisfactory system. The reason for this is because the effectiveness of the cutting is reduced as the nozzle exit diameter becomes smaller. If the flow were to be divided into a large number of small diameter nozzles, the distance that the water jets could penetrate into the coal seam would be severely limited. The rationale behind the oscillating motion of the cutting nozzles is to maintain as large a diameter as possible with the available flow.

Design of the Support Sled

For the first experimental testing of the cutting head it had been decided that it would not be necessary to utilize this full flow and that only one pump would be fitted to the unit requiring 150 hp. This meant that only the bottom half of the cutting head would be operative, and although a top cutting member and an upper vertical cutting arm were included within the design, these arms were not to be used in the test program. Only the middle and lower vertical cutting arms and the bottom horizontal cutting arm of the Hydrominer were expected to be used in the field trials. For this reason, it was assumed that the Hydrominer head would not be tested in coal seams over 32 in. thick in the first test program.

The test program was to take place in a strip mine, and therefore most of the equipment which would be available on a normal operating longwall face would not be available. No conveyor could realistically be used to carry the coal away from the mining machine, and the haulage of the machine down the face would similarly have to be by some other means than that normally available with a shearer.

In order to remove the coal from the cutting area, it was decided that the head would be shaped to deliver coal up onto the back of the sled which would carry the support pump, motor, and switch gear. From the sled it could be dumped behind the sled after the machine had passed. The advance of the machine would be controlled by a winch located at the end of the face.

Details of the Support Sled

The support sled, which had been designed in Phase I of the program, was constructed. In essence it comprised two I-beams over which a steel platform was welded. On this platform the pump, motor, and electrical switch gear were attached (Fig. 2.1), and from it a cross member was run to the cutting head location (Fig. 2.2). The connections to the cutting head were in two parts. There were two pinned joints located toward the bottom of the head which were strain gaged to measure

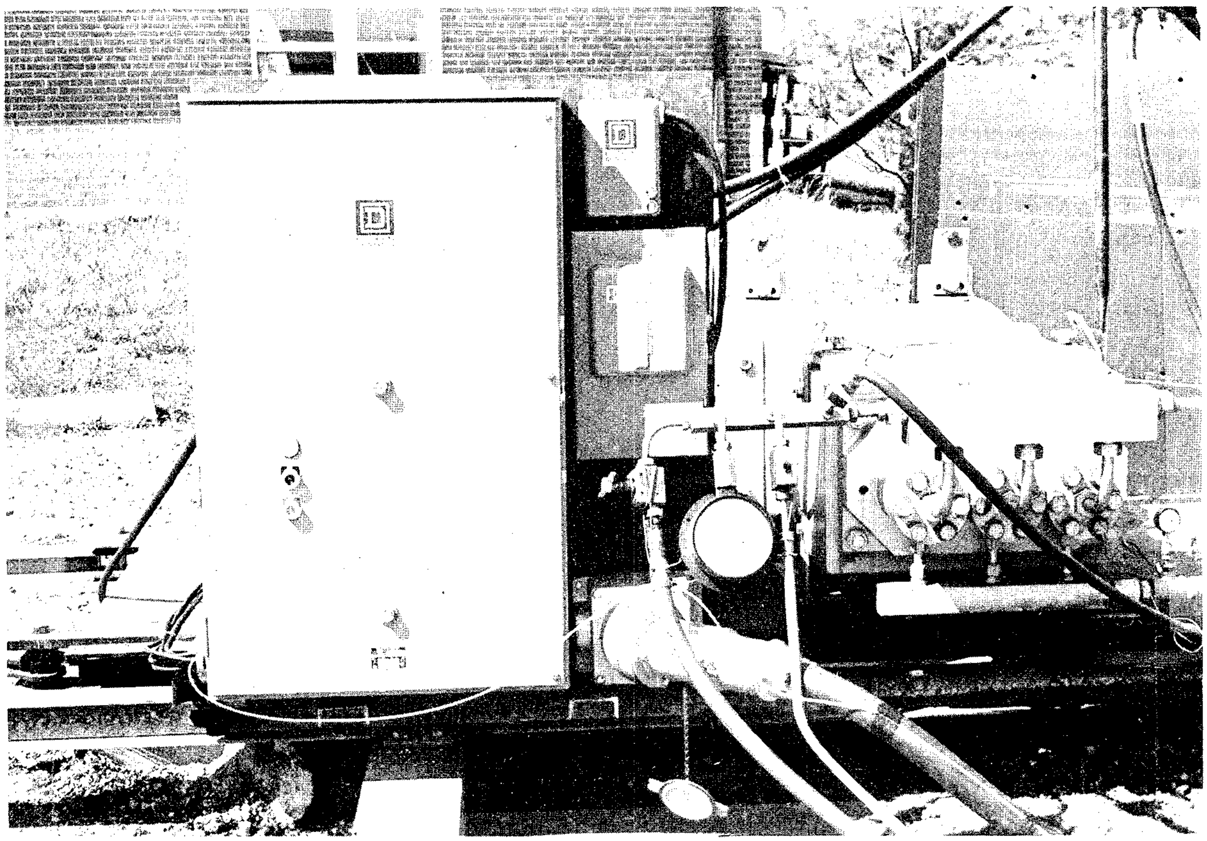


Figure 2.1. Support platform detail showing switchbox assembly and pump head connections.

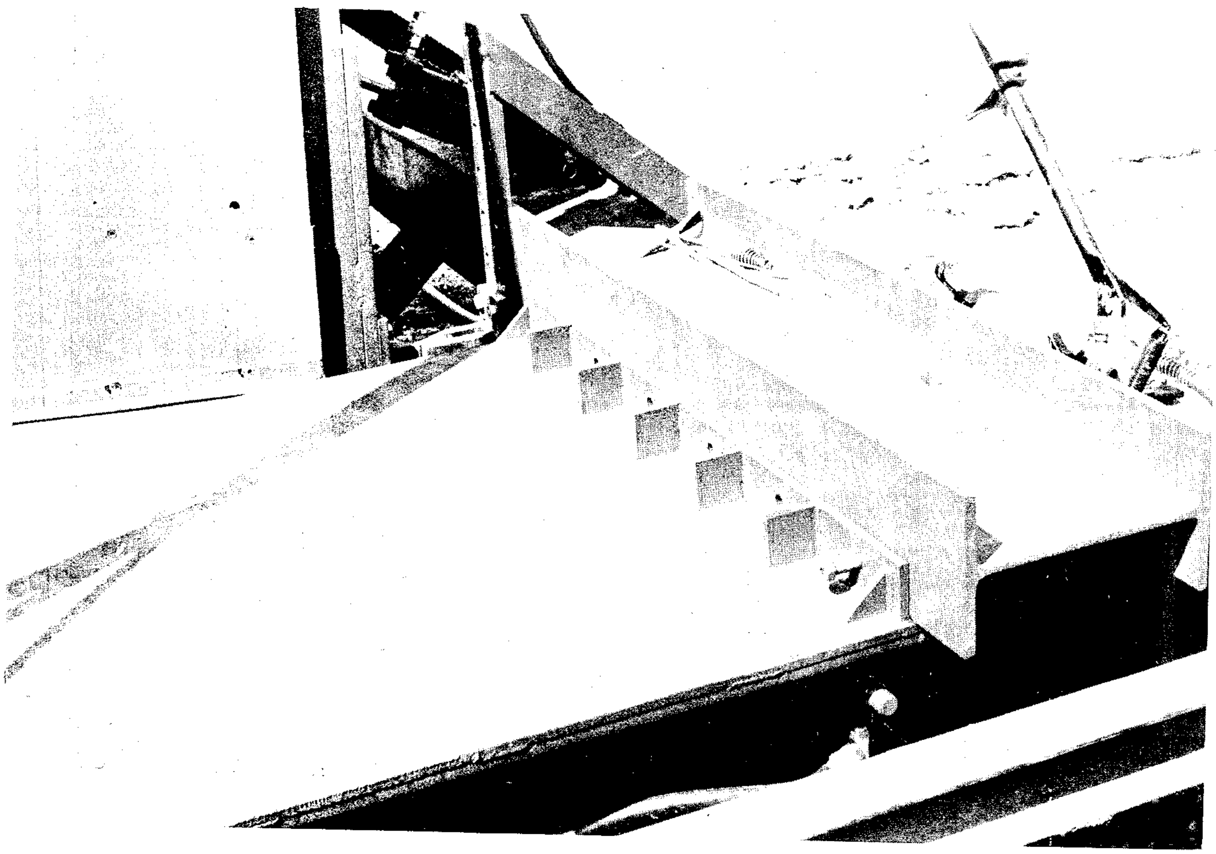


Figure 2.2. Support platform detail showing the support brace to the cutting head connector.

the forces thrusting on the head. A hydraulic cylinder was also attached approximately to the mid-point of the back of the head to raise it and for lowering it for steering (Fig. 2.3).

As originally designed, it was intended that the sled would be guided down the face being connected to a guide bar which in turn would be attached to the mine floor through support ties. This connection is shown (Fig. 2.4). However, as detailed in the following chapter, this particular method of guiding the sled did not prove to be satisfactory in the field.

Variable Speed Winch

Initially it was intended to purchase a variable speed winch capable of providing the required pull to advance the machine down the face at speeds of up to 30 ft/min. However, a survey of available manufacturers through an open solicitation for bids revealed that such a system would cost approximately $20,000 and take 6 months for construction. Accordingly, a simple but effective system built around an automotive clutch, roller chains, sprockets, and two electric motors was designed at a cost of approximately $800 and was constructed within a period of approximately one month (Ref. 6) (Fig. 2.5). This device proved eminently successful during the test program. Its design is detailed in the paper listed as Reference 6.

At the same time as the sled and winch were constructed, a 200 kw generator was obtained and mounted in a van trailer. The trailer was developed as a mobile workshop for the unit. Concurrently, two tanks were mounted on a second truck to act as a water reservoir during the tests.

Once the system was constructed, the assembly was demonstrated to Bureau personnel and approval obtained to go to the field.

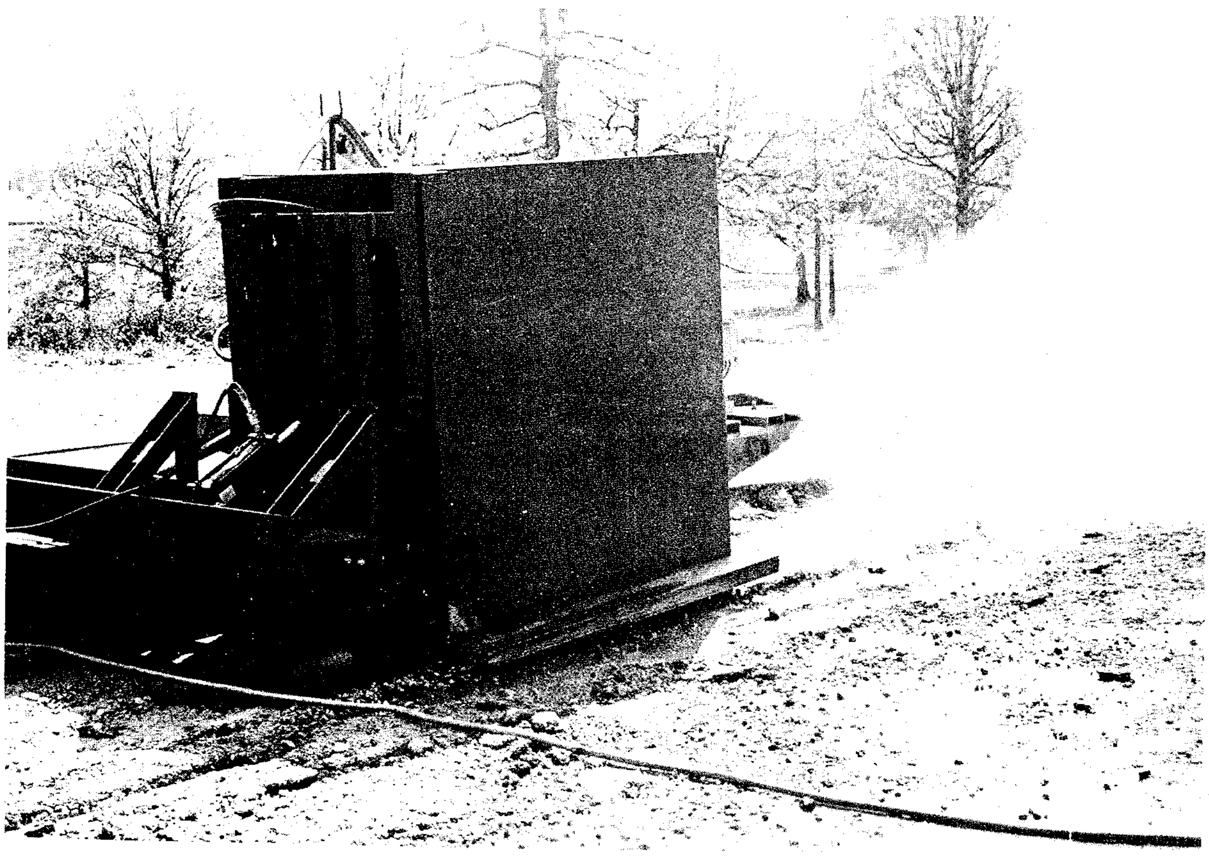


Figure 2.3. Cutting head in operation showing the action of the hydraulic cylinder and pivot points in raising the front lip of the head.

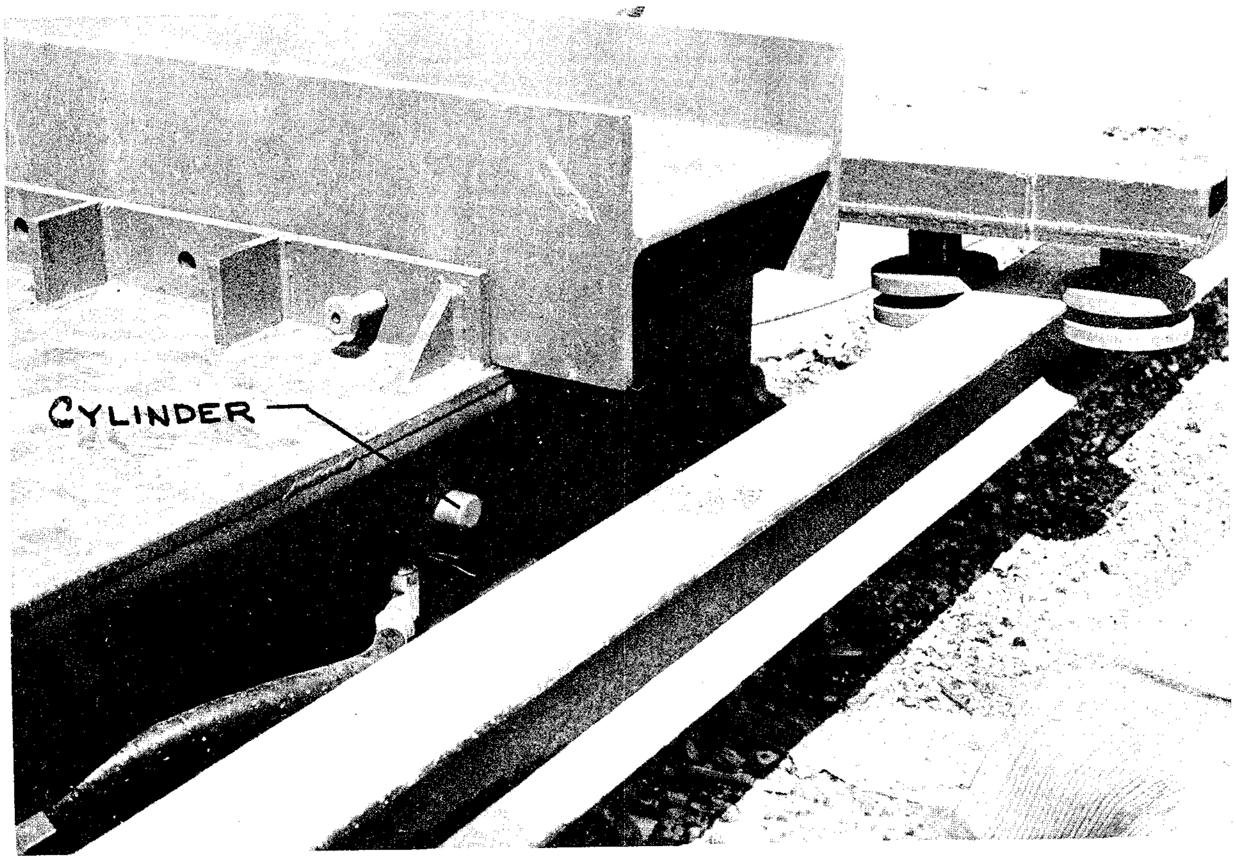


Figure 2.4. Support platform detail showing the location of the lateral positioning cylinder and the roller-retainer mechanism of attachment to the guide-bar.

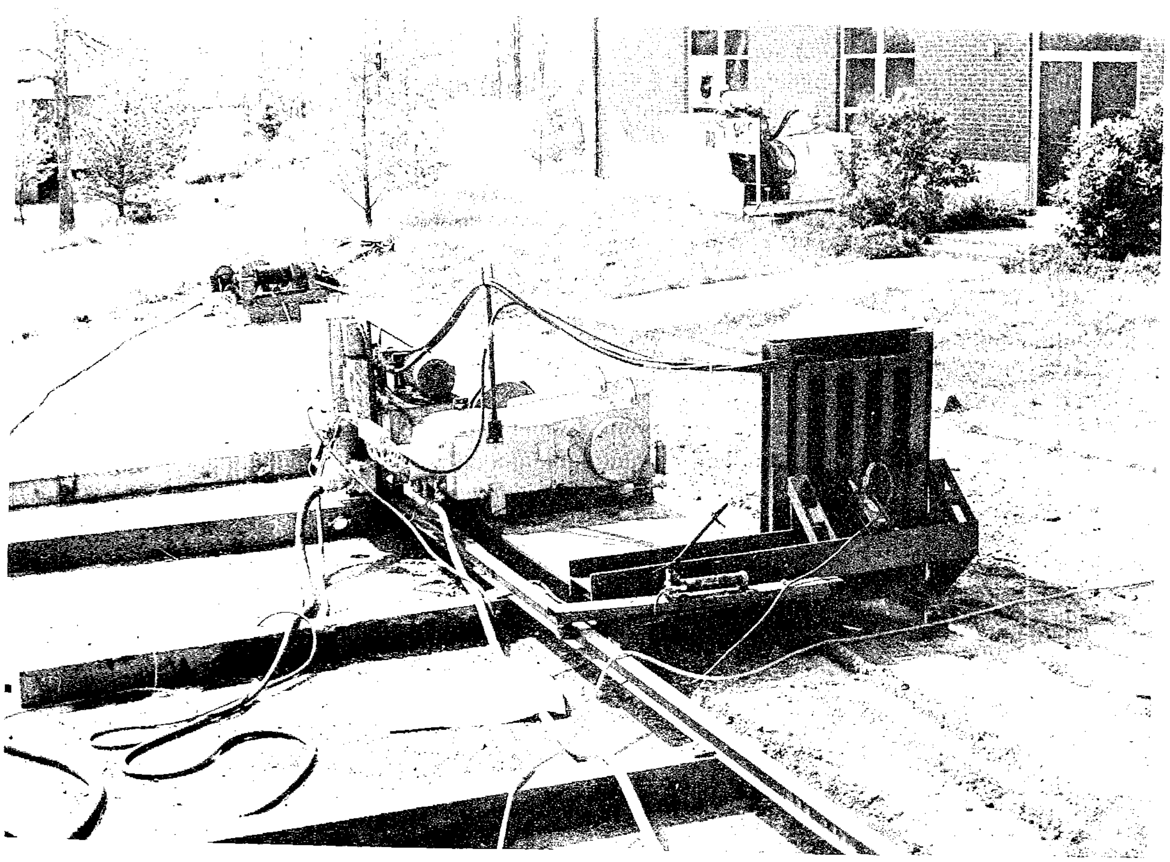


Figure 2.5. The assembled system showing the location of the components on the support platform and the haulage winch, together with the lateral constraining rail and ties.

Chapter Three

INITIAL TRIALS OF THE FIRST EXPERIMENTAL CUTTING HEAD

Introduction

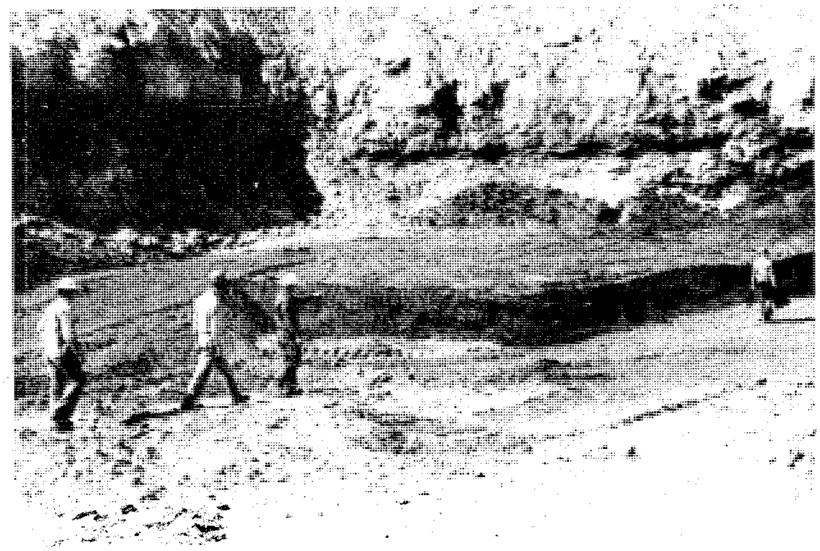
Upon the completion of the construction of the first longwall water jet cutting head and the necessary support sled (hereinafter referred to as Hydrominer I), the unit was demonstrated to Bureau of Mines personnel, and permission was obtained to initiate the field trials of this system.

An agreement was reached between the University and a coal company to utilize a nearby surface mine. Prior to leaving Rolla, it was agreed that the criterion for success of the test program would be the ability of the Hydrominer to achieve a mining advance rate of 5 ft/min on a 2 ft depth of cut in a 32 in. high coal seam (Ref. 7).

Test Site

The unit was taken to an operating strip mine located some 115 miles north of Rolla where a panel of coal 60 ft long by approximately 45 ft wide had been left on the final strip of this section of the mine (Figs. 3.1-3.3). The equipment was located ready to initiate testing. Because of the narrow width initially cleared between the coal panel and the spoil bank, the generator and the water supply truck were located on top of the coal panel and the winch was located some 45 ft beyond the end of the panel and held in place by four resin bolts sunk 4 ft into the underlying limestone. A snatch anchorage point was installed parallel with the face of the coal panel, and it consisted of two resin bolts anchored 4 ft within the limestone. To this anchorage a snatch block was attached by a length of

Figure 3.1
Surface mine test location during preparation



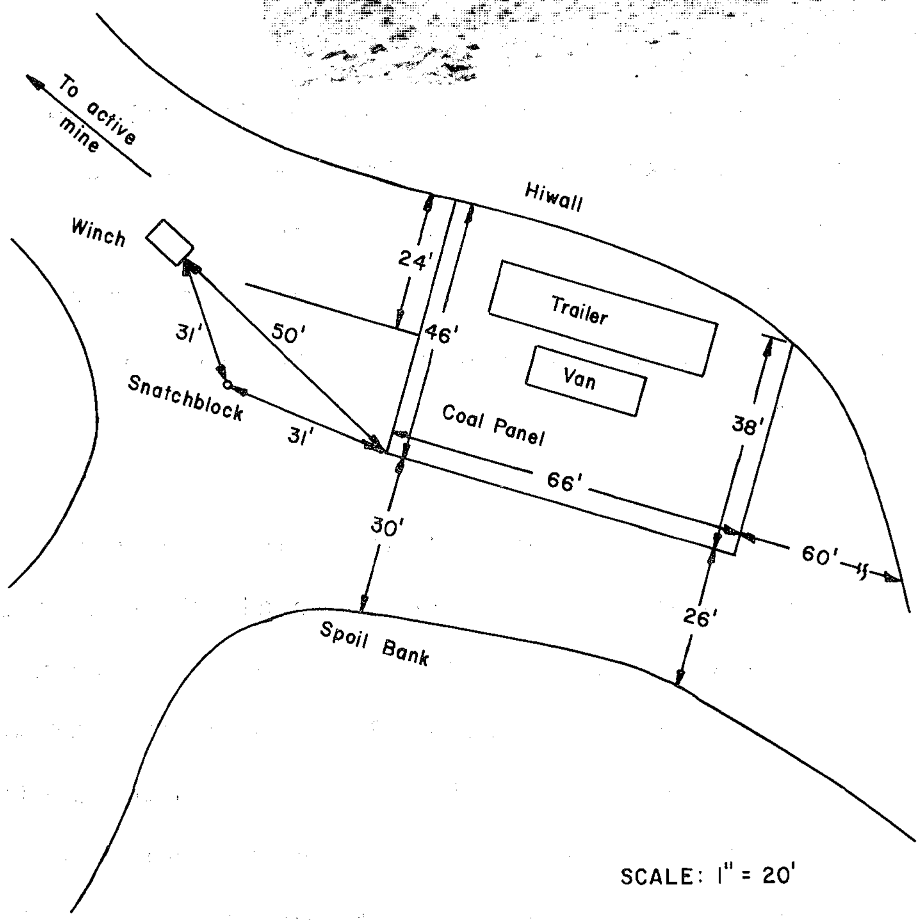
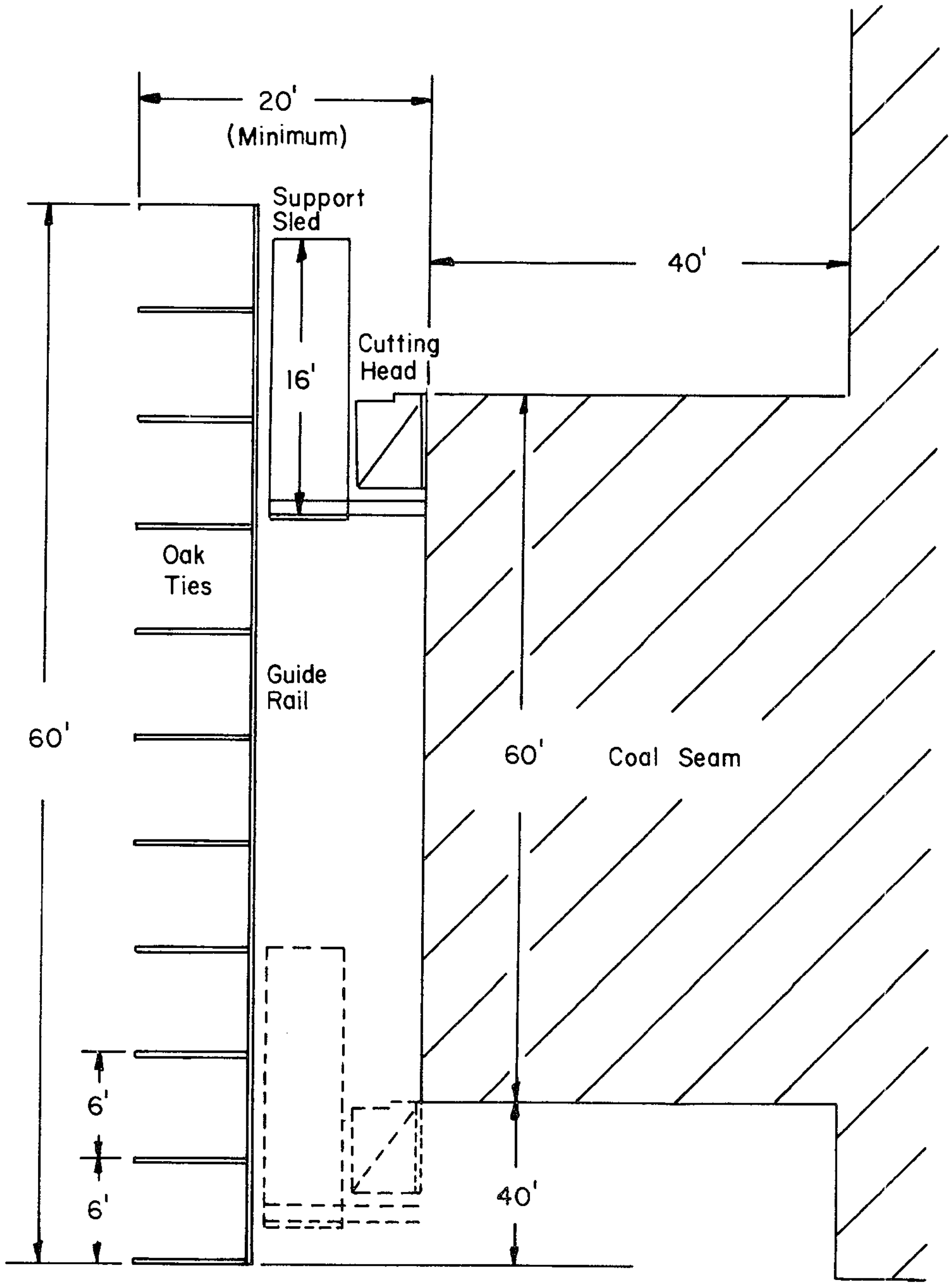


Figure 3.2
Panel layout

Figure 3.3
Schematic of Experimental Layout



chain so that as subsequent strips were mined from the coal panel the snatch block could be relocated to keep the haulage rope parallel to the face. The rock floor adjacent to the coal had initially been scraped down to limestone in the site preparation and prior to set-up this floor was re-established using broken pieces of shale from the spoil bank which were then compacted by a D-9 Caterpillar tractor.

Preliminary Tests and Face Preparation

The unit was assembled and water obtained from a pond in an abandoned area of the strip mine. This water was filtered after it had left the storage tank on the supply truck and a small amount (approximately 5 gal per 600 gal of water) of soluble oil was added to inhibit valve sticking within the pump assembly. After the unit was assembled a preliminary cut was made with 20 degree diverging nozzles with the unit moving forward at an advance rate of 5 ft/min.

Two things became evident during this first run of approximately 40 ft. Firstly, without any guide rail to maintain the horizon of the mining machine, the head climbed up the face, a total height of approximately 1 ft 6 in. over the 40 ft advance, and the head tended to veer out from the face as the coal became harder. Secondly, the 20 degree diverging jets that were used threw some of the coal that was mined from the face ahead of the Hydrominer such that little coal was actually coming back across the plow surface (Fig. 3.4). This indicated that the nozzle angle being used was too large for this particular type of coal. The head was withdrawn and a second cut made to level the face down to the bottom of the seam of coal, again without lateral restraint. However, since the head was again pushed over into the waste as fresh coal was being mined, it was decided to locate and anchor the guide rail before further tests were made.

Guide Rail Installation and First Tests

Oak ties, 8" x 8" x 84" were located at 6 ft intervals along the face section and attached to the mine floor in the manner proposed earlier in the program (Ref. 8) with the rail attached to the top of the oak ties (Fig. 3.5). Only one pair of anchors was used to hold each tie and this was located on the face end of the oak ties and proved sufficient. During the course of locating the ties it was necessary to auger through the fireclay overlying the anchoring horizon of limestone because the fireclay tended to jam the action of the jackhammer. Three tests were then made using a narrower diverging nozzle, 10 degrees included angle, and advances of respectively 7'6", 7'0", and 12' were obtained. The first run was made at 5 ft/min and the second and third at 1 ft/min.

Required Modifications

The tests to date had revealed that particles of coal were floating back into the cutting head and jamming the cutting arms as the head advanced. The head was therefore disassembled from the rig and taken back to Rolla to adjust the drive mechanism and to build slot guards to prevent some debris from entering the cutting head.

During the time that the head was down in Rolla for modification the underlying fireclay became extremely soft and the sled girders started to sink into the floor. Large steel plates were therefore welded between the two girders to provide a large bearing surface and stop this from occurring. This did not,

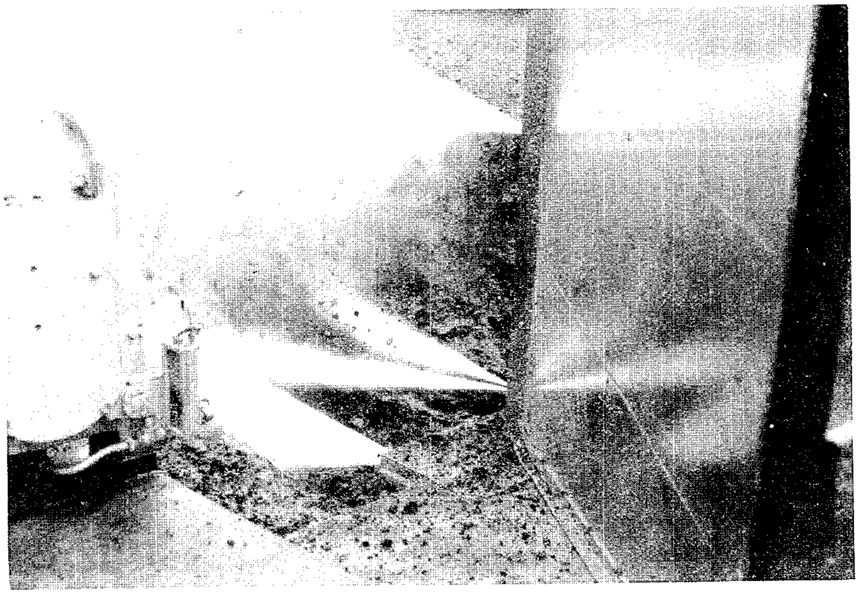


Figure 3.4. First cut - showing jet diversion and the coal being thrown forward from the cutting head.

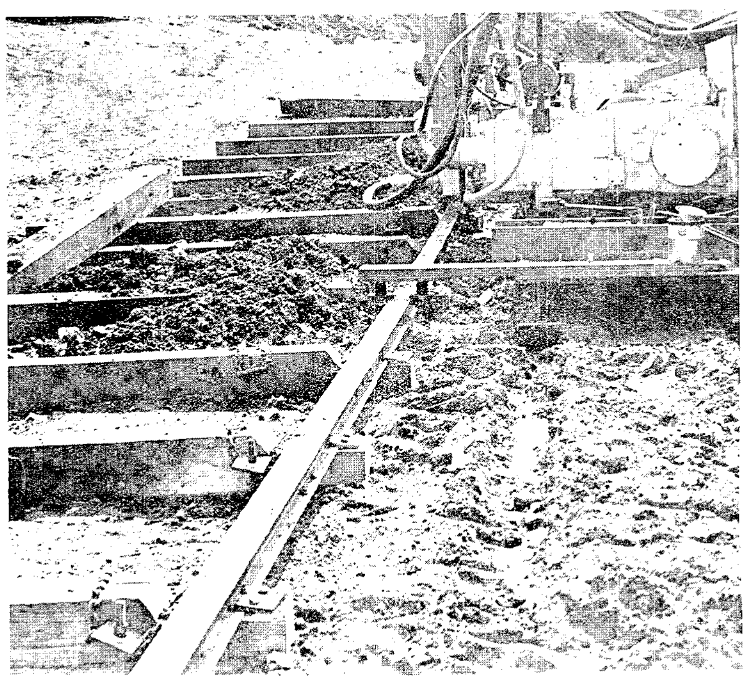


Figure 3.5. Detail showing the method of rail tie-down and sled attachment (initial system).

unfortunately, completely solve the problem because of the very broken nature of the fireclay and while one pass was made down the face at 1 ft/min for 5 ft and then at 5 ft/min, the head had to be stopped several times. This was caused by a very heavy rainfall which flooded the work area and further exacerbated the shale decomposition. Therefore, the floor area was filled with gravel to stop the sled from sinking into the fireclay. In subsequent tests where the fireclay material was fresh the sinking problem did not occur with the same degree of severity, and it was unnecessary to repeat the floor replacement after the first material had been removed. It was also found necessary to change the method of connection from the sled to the guide rail since the seam floor fluctuated quite markedly over the length of the face going through a dip approximately 25 ft from the head end. This dip was too severe for the very rigid connecting system originally used on the head and a more flexible system was therefore designed using a pivoted connection and bracing wire rope members (Fig. 3.6-8).

The head was returned to the test site following the week in which repairs were carried out in Rolla and during the same week a paper was presented at the Rapid Excavation and Tunneling Conference in Las Vegas (Ref. 9). Slot guards (Fig. 3.9) had been installed and a spray unit had been located to direct water into the head to flush out any coal particles which entered the head. A run was then made down the face at an advance speed of 5 ft/min to re-establish the face profile and to clean the face. The depth of this cut varied due to the new manner of connection to the guide rail and two problems were identified during this

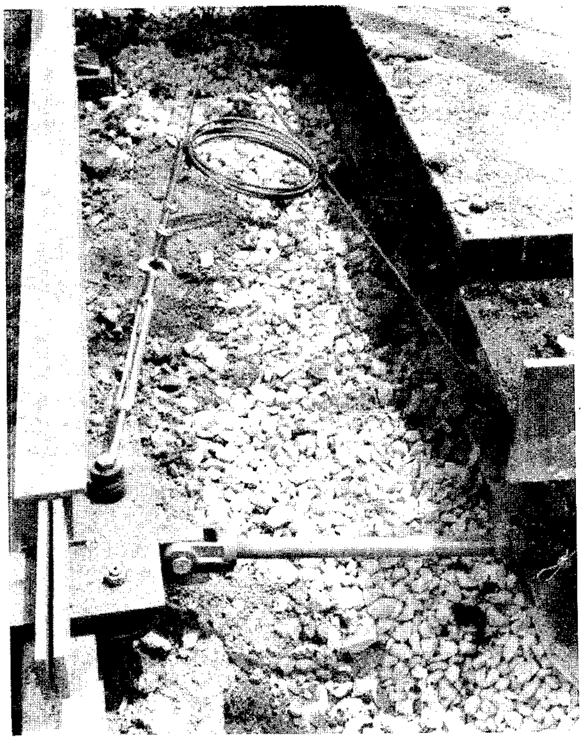


Figure 3.6. Sled attachment

Overall view

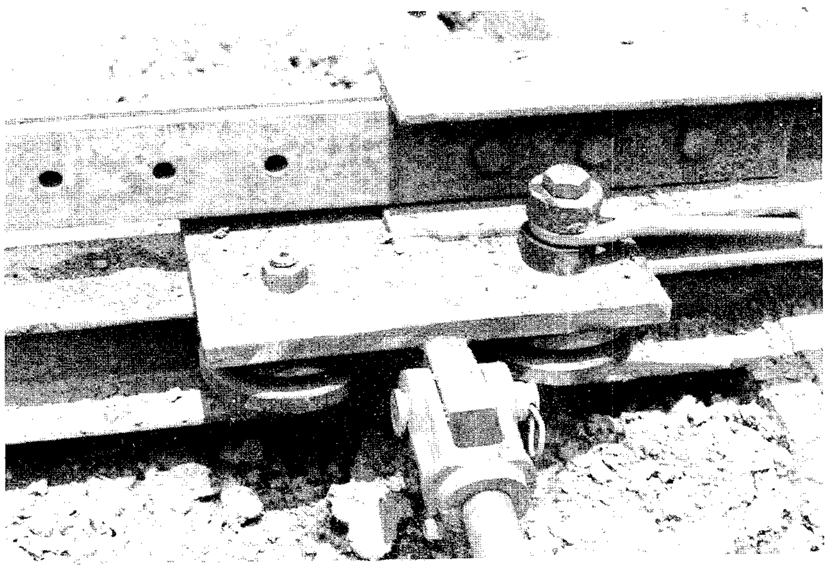


Figure 3.7
Detail-rail connection

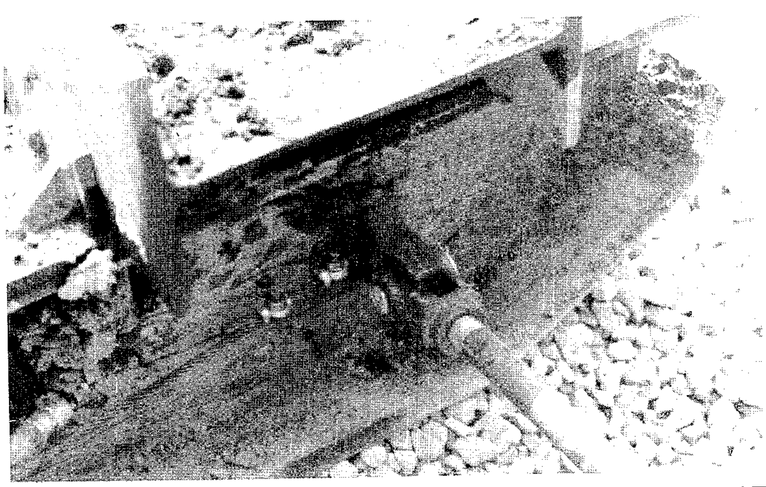


Figure 3.8
Detail - sled connection

Figure 3.9. Initial design of slot guard as installed.

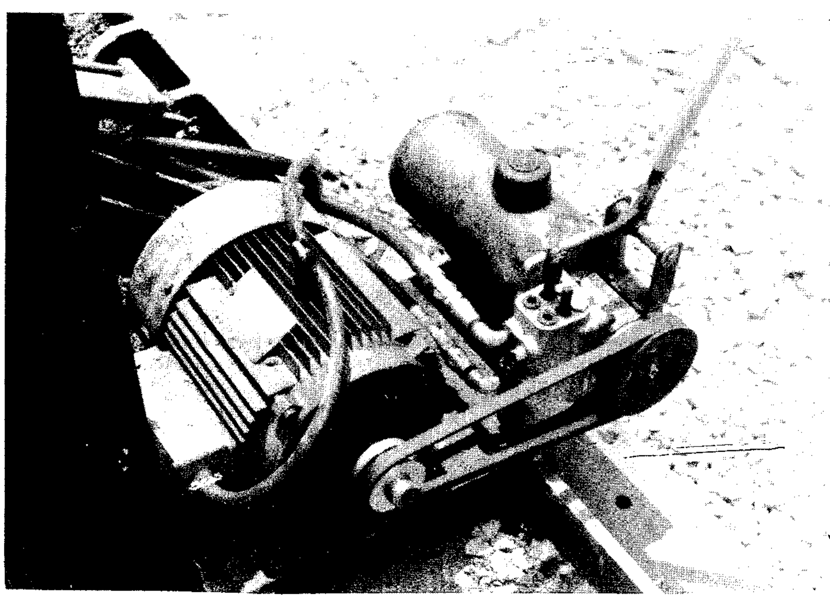


Figure 3.10. Hydraulic drive installed to improve the vertical steering of the head.

cut. The first of these was that the manual steering system was not responding quickly enough to the variation in seam direction. For example, on run 13 the head cut down a depth of 1 ft 6 in. below the sled and was actually starting to cut into the fireclay to the depth of approximately 6 in. This condition could not easily be corrected with the manual steering as the unit advanced. A hydraulic powered steering system was therefore designed and constructed (Fig. 3.10) and connected to the hydraulic cylinder at the back of the head. A second and more serious problem was identified in this run in that thick layers of pyrite were discovered in the seam section (Fig. 3.11-12). This pyrite proved to be extremely strong and because the jets would not consistently cut a wide enough slot in this material, it was found difficult to operate the jet cutting arm successfully.

Changes Made Because of Pyrite Problem

Initially the head alignment was parallel to the face so that as the head advanced the face side of the machine would move parallel with the face of the slot cut. However, where pyrite was present it was not initially cut very effectively by the jets so that some of the pyrite layer would extend into the path of the support structure on the cutting head. This pyrite would then cause deformation of the cover plate on the head so that the cover plate would block the passage of the oscillating arm and the arm would jam. This problem persisted and four solutions were attempted to solve it.

Firstly, the angle of attack of the head to the seam was changed so that the face sides of the cutting head made an angle of approximately 3 degrees to the line of advance of the

Figure 3.11 Seam section showing the presence of a pyrite lens

prior to improving cutting

Figure 3.12

pyrite cut by water jet action

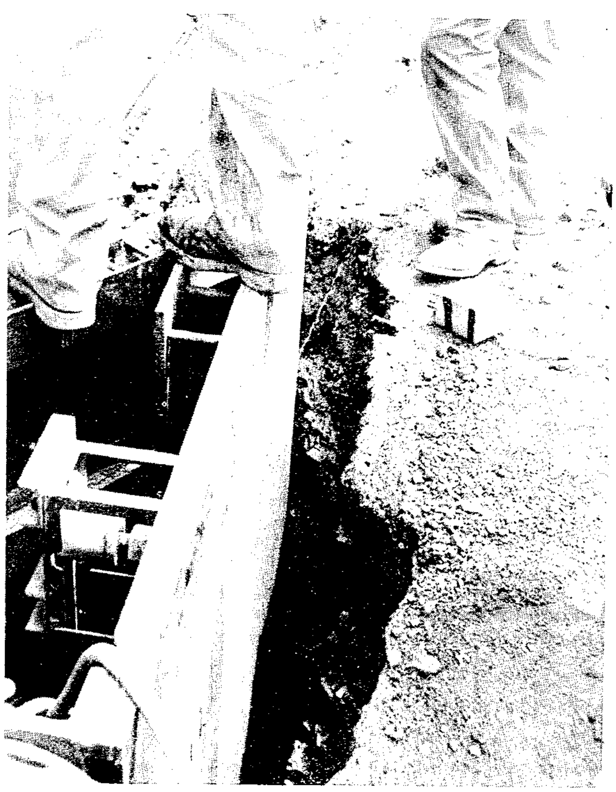


Figure 3.13
Detail showing head rake to overcome pyrite rubbing on the face edge of the hydrominer.

machine (Fig. 3.13). This was achieved by a 6 in. difference in the length of the two connecting arms between the guide rail and the sled. With the back end of the cutting head moved away from the coal face, there was less contact between the pyrite protrusions and the cutting head after the leading edge of the head had passed the protrusion. Secondly, a steel blade was welded to the face side of the cutting head. The intention here was to combine the jet action with the crushing that would be affected were the blade to come into contact with pyrite. The addition of the blade increased the slot width required for the cutting head to advance unobstructed. This created additional problems when the advance rates greater than 5 ft/min were attempted.

A third solution was to change the orientation of the jet stream from the nozzles so the jets intersected at a distance ahead of the cutting head. This arrangement is called a dual orifice converging nozzle system. Where the jets intersect it has been found that there is an augmentation of jet velocity. This has been previously found by others in shaped charge work and with droplet impact. Jet augmentation has been closely investigated by Dr. Mazurkiewicz (Ref. 10), and was found to be sufficient to allow the jet to cut pyrite. In making these determinations cuts were made down the face at 1 ft/min advance rate in order to determine the effectiveness of different approaches to solving the pyrite problem and one complete cut of the face was made in this fashion. However, as a fourth approach to solving the pyrite problem, it was decided to change the nozzle guard system to allow the head to advance into a narrower slot.

By reducing the slot width, the jet energy could be focused into a more narrow band. This was incorporated in the third nozzle guard design and using this system the cutting head was capable of advancing through pyrite. The third nozzle guard design is shown in Fig. 4.11.

Problem of Removing Coal from Area of the Cutting Head

The head was now cutting with 8^{o} diverging nozzles in coal ranging from 40 in. high at one end of the face to over 48 in. at the other, and a problem was now identified with removing the coal which accumulated next to the cutting head. As the pyrite problem was solved, the machine was capable of moving continually over longer distances. However, it was found that as the head was initially designed it was not capable of loading the coal from the head over the sled onto the ground and instead pieces of coal were being jammed between the girder of the sled and the coal face. There was not a sufficiently smooth path for the coal to flow smoothly and this bridging action became sufficient in one instance to break the haulage rope pulling the machine down the face. A modification was therefore made to the cutting head under advice from Dr. Spies and Mr. Fitzgerald of Heintzmann Corporation, and a steel plate was modified and welded onto the structure to increase the loading angle across the plow surface. This modification was successful and in subsequent trials the head was found to load satisfactorily.

Demonstration Run

A demonstration of the operation of the cutting head took place before Bureau of Mines personnel on September 16 when a run was made at 2 ft/min. However, in this demonstration the converging jet nozzles were used in order to ensure that the jets cut the pyrite effectively during the advance and a different procedure was followed than what had been successful in earlier trials. In particular, the system was operated at full pressure without advancing the cutting head and then shut down to allow the observers to inspect the slot generated. The jets had isolated a very large, approximately 36 in. long, cantilever of coal and when the head was advanced this cantilever came out as one large block and jammed between the sled and head. Perhaps the cantilever was not sufficiently infused with water to promote a disintegration when the block was mined. However, this same system had operated successfully in earlier tests and the size of coal produced was reasonable.

Size of Coal Produced

Following the demonstration the diverging nozzles were used instead of the converging nozzles and the remainder of the face was cut out at 5 ft/min. The head was then moved over an increment of 2 ft 6 in. and a second cut was made, again moving at 5 ft/min. The performance of the head during these runs was recorded on film. Typical sizes of coal fragments produced by Hydrominer I can be seen in Figure 3-14. A comparison of coal size with that produced by a shearer can be seen in Figure 3-15.

Figure 3.14. Size of coal produced by Hydrominer.

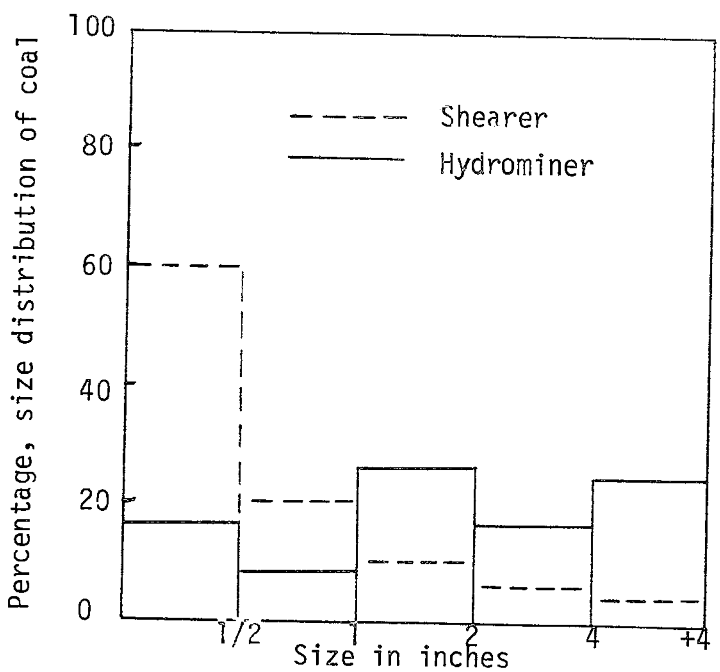


Figure 3.15. Size distribution of coal from a shearer and Hydrominer.

Chapter Four

ANALYSIS OF FIRST EXPERIMENTAL DESIGN

Introduction

In the design of any prototype equipment, it is common to encounter numerous machine design problems. No amount of theoretical analysis or laboratory experimentation will reveal all details of the machine operating characteristics. Further it was not possible to completely solve all the problems discovered in the initial prototype test phase, because these would require, in some cases, extensive equipment modification. Some problems were therefore only partially solved during the field trials, and a complete solution was postponed to be incorporated in the design and construction of a second generation machine. The information gained by analysis of the design problems encountered in the initial testing is therefore an excellent guide to the modification of the machine undertaken in the redesign phase.

Problems Encountered

Three classes of machine design problems were encountered during the field trials:

Type 1. Problems for which a complete and satisfactory solution was achieved in the field.

Type 2. Problems for which only a partial or not completely satisfactory solution was achieved.

Type 3. Problems for which no solution could be achieved in the field

The following discussion will consider each type of problem in turn. The magnitude and importance of the various problems is not considered in the order of presentation. Attention is drawn to Type 2 and Type 3 problems since their complete solution was only achieved during the redesign phase of the development program. The difficulty of modifying the equipment in the field is indicated by the fact that only two problems were completely solved, five were partially solved, and six were not solvable during the field trials.

Type 1. Completely Solved

1a. <u>Insufficient travel in the vertical direction on the guide rail system.</u>

Initially the sled was attached to the guide rail through three pairs of rollers (Fig. 4.1). Channels at the front, rear, and middle of the sled carried pins which allowed the rollers to both slide and rotate relative to the pins. The vertical travel of the rollers was limited to the approximately three inches of available space. When the sled was pulled along the face it was discovered that a much larger vertical travel was required. This requirement was due in part to the fact that the floor was not completely level. In addition, the floor had been ripped to below the desired grade and then back filled. The result was a surface that was neither flat nor solid. The sled sank in the soft material and the problem grew worse as rain softened the material further.

As a result, the guidance system had to be modified. The channels and rollers (Fig. 4.1) were replaced by the system shown in Figure 4.2. The hydraulic cylinders were used as

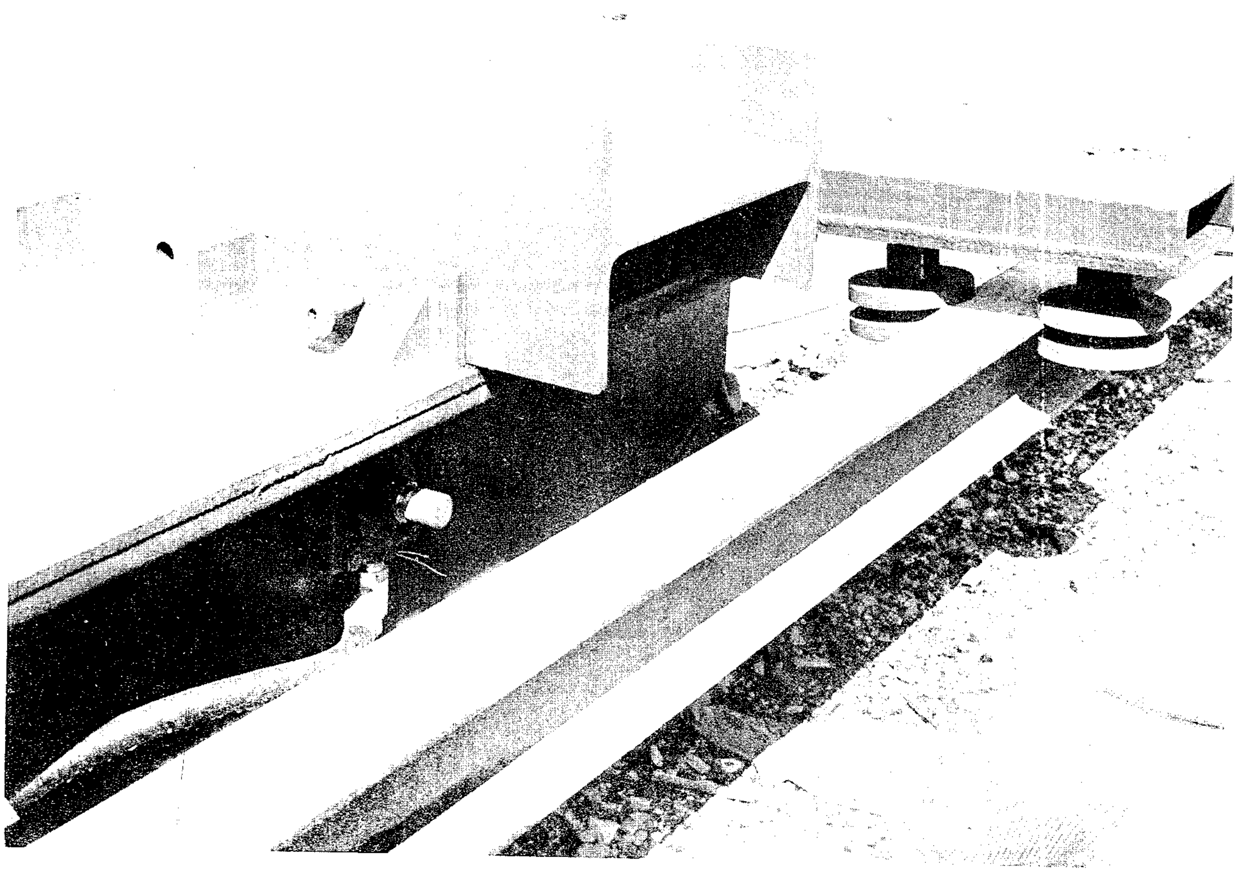


Figure 4.1. Sled connection to guide rail.

connecting links between the sled and guide rail attachment plate. These links were pinned at both ends to give the required vertical travel. Also shown in Figure 4.2 is the strain gaged load cell used to measure the side thrust forces exerted on the guide rail by the sled. An added benefit of this guidance system was that the guide rail did not have to be moved after every cut. Instead the piston of each cylinder was extended to move the sled and head over for the next cut.

1b. No rake provided on the cutting head.

The orientation of the head was initially parallel to the face and the direction of advancement. With this orientation the back vertical side of the head could contact the coal face over its entire length, and coal particles which were trapped between the machine and the face would rub along the length. In order to correct this, the front end of the sled was positioned slightly closer to the coal than the rear end of the sled. This was accomplished by setting the hydraulic cylinders to slightly different lengths. The result was a rake of approximately 3 degrees which prevented the head from rubbing against the coal.

Type 2. Partially Solved

2a. Short nozzle life.

The life of the brass nozzles was relatively short and after approximately 30 minutes the quality of the jets and their ability to cut deteriorated rapidly. This problem was not addressed directly since the cost and speed of

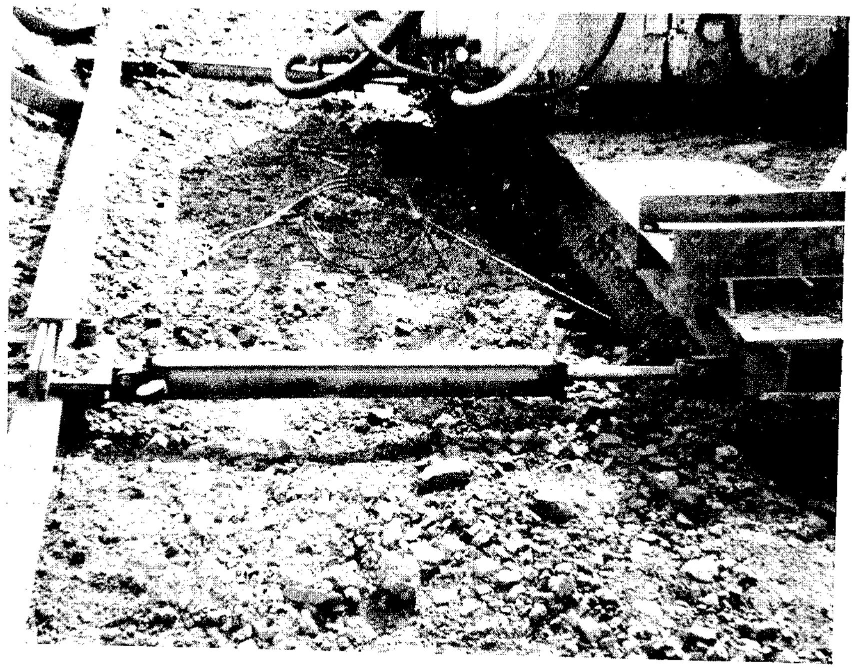


Figure 4.2. Hydraulic cylinders used in new guidance system.

Figure 4.3. Spray arm compartment openings.

manufacturing gave overriding advantage to the use of brass while an optimum design had not been established. Late in the field trials, however, some brass nozzles were plated with an electrolysis nickel plating and then hardened to Rockwell #60. None of these nozzles were worn out during the test program. In the following stage of the program this problem will be readdressed.

2b. <u>Coal enters the interior of head through the cutting arm compartment openings</u>.

The original head design had a rectangular opening on the front of each cutting arm compartment (Fig. 4.3). Each of these openings were approximately 1-1/4 in. wide by 20 in. long. In the early test runs a major problem was that large pieces of coal, up to 1-1/4 in. in size, would float, in a slurry of water, through the openings into the head and jam the cutting arms. The worst problem occurred in the horizontal arm compartment and was most severe when the entire front and bottom area of the head was covered with coal. Under this condition there was no place for the water to go but back through the head. Hence, this problem was aggravated by the fact that no conveyor was present to carry coal away from the head as it was cut out of the face.

The solution to this problem was challenging since it was essential to stop the coal from entering the head without interfering with the ability of the jets to exit from the head. The first attempt at solving the problem was to fabricate a nozzle guard from 3/32 in. thick sheet metal (Fig. 4.4).

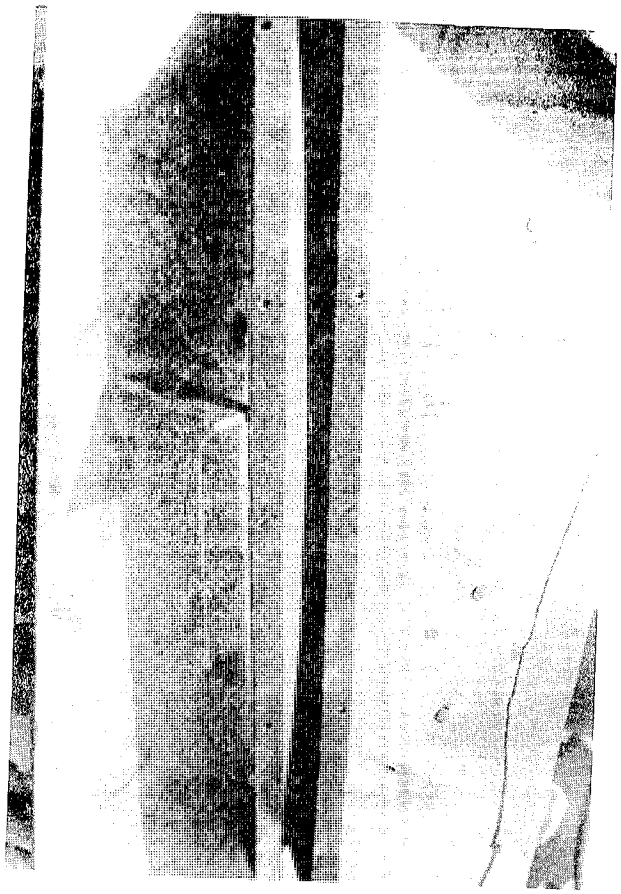


Figure 4.4. First guard design.

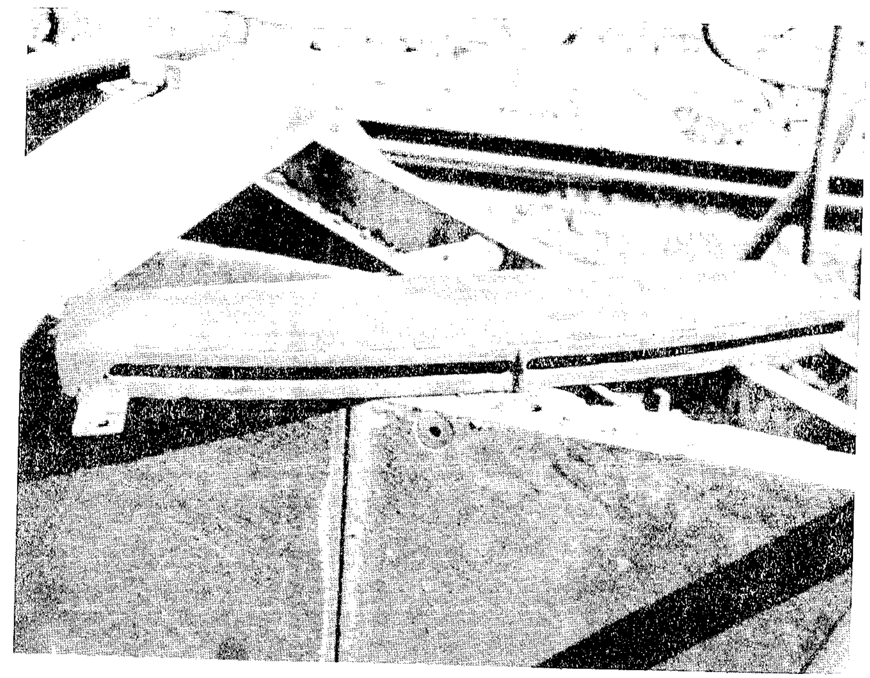


Figure 4.5. Second guard design.

This guard was attached to the outside front of the cutting head with sheet metal screws and the water jets had to pass through an opening 0.2" wide to exit from the head (Fig. 4.6). This guard was reasonably successful since it greatly reduced the size and volume of the coal that entered into the cutting head. The difficulty with it was that it interfered with the water jets, since there was no provision to keep the nozzle aligned with the slot in the guard. When the nozzle was not accurately aligned the jets would touch the guard and be prematurely disrupted. The slots were therefore milled wider in successive increments until all jet interference was eliminated. The result was a final slot width of 0.375" which greatly reduced the ability of the guard to keep coal out of the head.

A second guard design (Fig. 4.5) was constructed with two plates which extended behind the nozzle. The purpose of these plates was to align the nozzle with the exit slot. A small cylinder of plastic was pressed on the outside of the nozzle holder to prevent metal to metal contact as the nozzle holder moved relative to the guard plates. With this arrangement, a slot width of 0.25" successfully reduced the volume of material entering the head to approximately 10% of the initial amount. The only problem encountered with this guard was that many of the nozzles could not be used since the spacing between the orifices was too great, the nozzles having been manufactured before the need for the guards was discovered.

Both of the above designs had a relatively short entrance path to the interior of the head (Fig. 4.6). An improvement of the length of this entrance path was obtained with a third design. The discussion of the third guard is incorporated in the next section, because it was designed to accomplish more than preventing coal from entering the head.

In summary, all of the guards used were effective in reducing the volume of coal that entered the head. However the use of such guards cannot be considered a complete solution since it only postponed the time at which the arms would jam. Once the coal had entered the head, there was no place for it to exit from the head. In time, the volume would build up until the arms jammed. A low pressure water flushing system, added to flush the coal from inside the head, helped considerably, but did not provide a complete solution.

2c. <u>The slot cut in the coal is not always wide enough</u>.

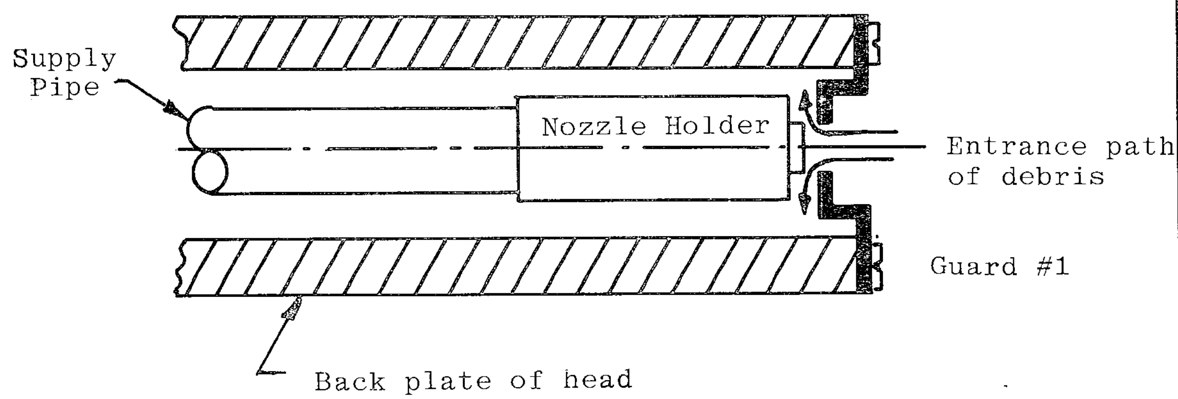
2d. <u>The slot cut in the coal is not always continuous</u>.

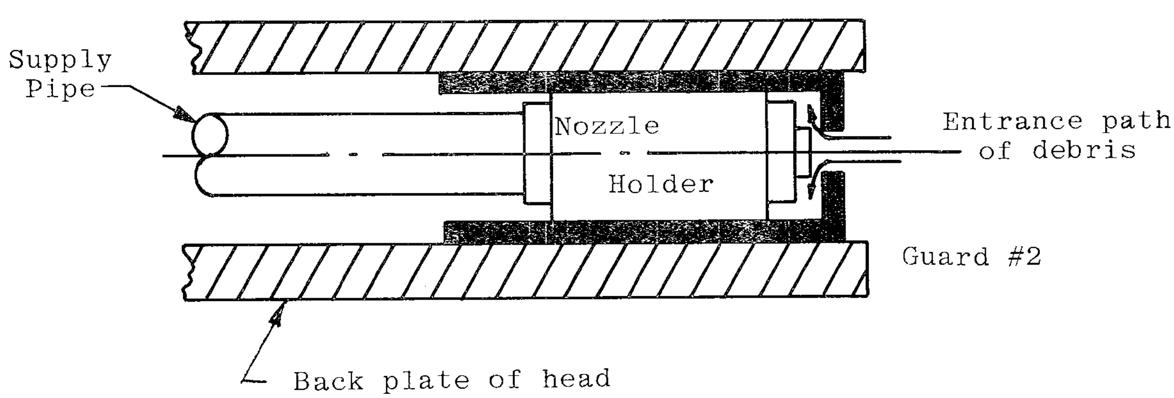
2e. <u>The best nozzle design is not defined</u>.

The above three problem areas related to the effectiveness of the nozzle, the manner of cutting and the material being cut. As these are interrelated the problems will be discussed together.

With 150 hp available at the pump, this is translated into a flow of 23 gpm at 10,000 psi at the manifold. Where the coal was thick, dual orifice nozzles, 0.04 in. diameter were located in each of the three cutting arms, giving a

Figure 4.6. Cross-sections of guard designs #1 and #2.



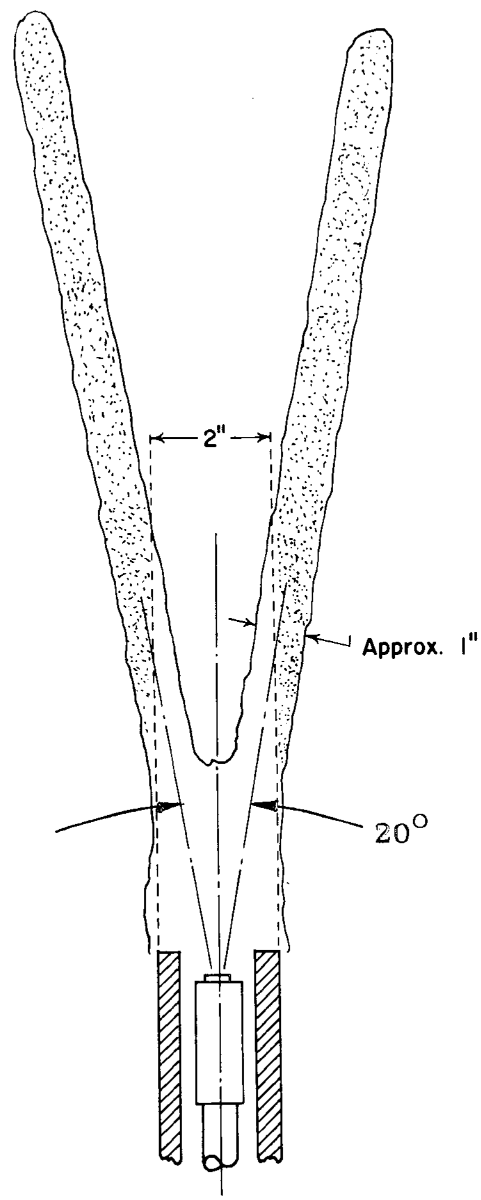


flow of just under 8 gpm per nozzle or 4 gpm per orifice. In order for the unit to advance continually the jets issuing from each nozzle must cut a slot 2 in. wide and at least 20 in. long at the same speed as the machine advances.

For the machine to advance unimpeded the slot cut must be unobstructed and wide enough (2 in.) to allow the head of the wedge to enter the slot. In the field trials neither condition always occurred. Where the slot was not continuous the force required to move the machine through the obstruction exceeded the strength of the shear pin in the winch which, in failing, stopped the advance. Where the slot was not wide enough for the head to enter the slot, the constriction in turn bent the walls of the machine inward. This, in turn, stopped the movement of the nozzle, the required slot was not cut and the shear pin overloaded. The problem was exaggerated by the increasing density of pyrite lenses in the coal as successive passes were made.

Initially a solution to the problem was sought by a change in the design of the nozzles being used. The initial design incorporated two jets diverging at 20 degrees to cut the type of slot shown in section in Figure 4.7 and seen in Figure 3.4. With this design each jet is cutting an independent path, leaving a central core which, in harder material, can reach into the nozzle guard before being broken. If the path cut by the jets is compared with the slot width required it can be seen that the jets are too divergent. Since the original design was predicated on the jets only cutting a 4 in. deep slot and the slot achieved was up to 24 in. deep, this was an obvious result.

Figure 4.7. Slot cut by 20° diverging nozzles.



Successive designs of nozzle at 15^{o}, 10^{o}, and 8^{o} included angle were constructed and tested. At 10^{o} the central core was not present, nevertheless, in comparison with the required slot width (Fig. 4.8) the jets still cut beyond the required distance. The 8^{o} divergent nozzle was, however used from this point forward for the lower horizontal nozzle and where the guard and nozzle were correctly aligned it cut effectively at distances up to 22" ahead of the hydrominer, although the slot cut at these distances exceeded that necessary to introduce the head.

However two other considerations must be taken into account in the nozzle design, the hardness of the material being cut and the use of the jets to infuse the coal in the web being mined. These problems were addressed in further design modifications tested in the vertical arms.

Were the seam to have been a clean coal then these modifications would not have been necessary. Because of the potential problem with pyrite, however, it was felt necessary to increase the angle of the jet on the face side of the unit to 12^{o} in order to ensure a free path for the machine edge. On the goaf side a 5^{o} angle was found sufficient to satisfactorily infuse the coal pillar and give adequate coal fragmentation. A greater angle was not required and would have proved counterproductive since the 5^{o} - 12^{o} combination (Fig. 4.9), with an included angle of 17^{o} was not always able to remove the core between the jets much in advance of the head (Fig. 4.18).

One solution to the pyrite problem had been to strengthen

Figure 4.8. Slot cut by 10^{o} diverging nozzles.

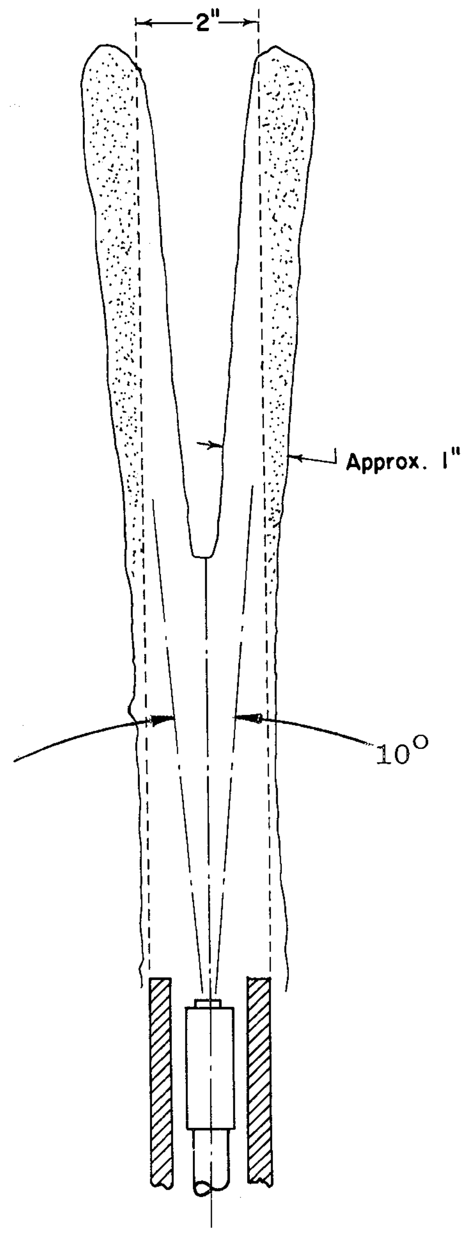
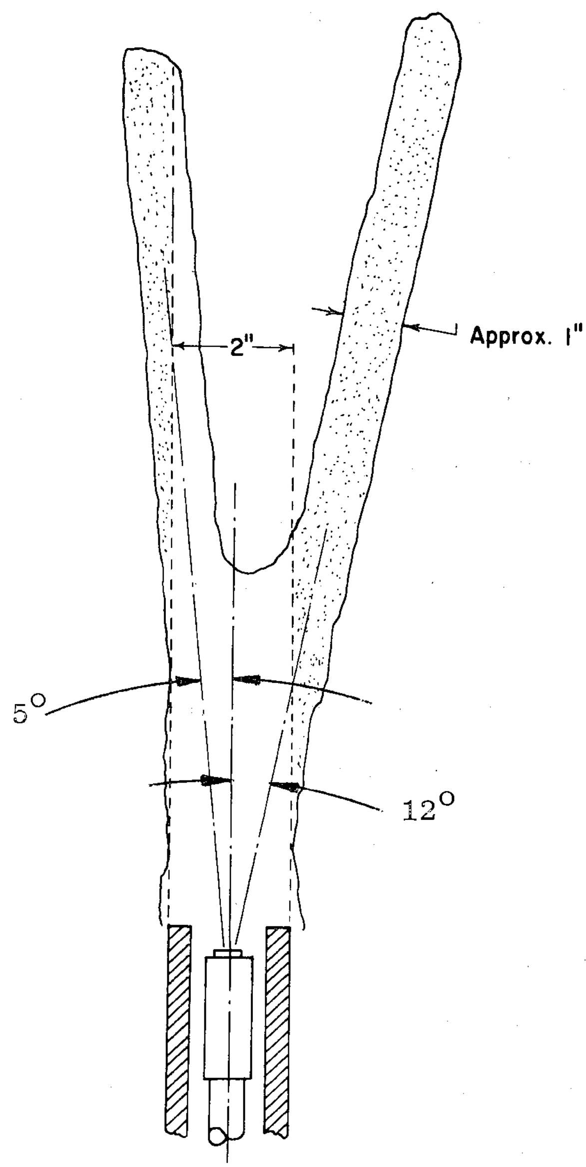


Figure 4.9. Slot cut by 5°-12° diverging nozzles.



the face side of the machine with a scraper blade but, while this was used with some success, the additional width of slot which was required with this attached was not always attainable at higher advance rates, although one run of approximately 10 ft at 10 ft/min was made with this attached.

An alternative approach to the cutting of pyrite was therefore investigated. Where two high velocity streams impact Taylor (Ref. 11) has shown that the ensuing flow will be divided into a faster flow and a slower flow. This principle, used in shaped charges and investigated by Bowles (Ref. 12) had previously only been considered advantageous where symmetrical droplets or perfectly flat face jets were used as the impacting members. Mazurkiewicz (Ref. 10). had however shown that an augmentation could occur with two high speed jets continuously impacting and this design was therefore tested.

The results of tests with jets converging at an angle from 0^{o} to 30^{o} indicated that the most effective nozzles of this type were with parallel or 1^{o} to 2^{o} convergent jets. Whether or not there was a great deal of augmentation is unproven, however when an equivalent flow was directed through a single orifice no cutting of the pyrite occurred. Some pressure increase on the pyrite must, in consequence, be assumed but whether this was from augmentation or from the pressure pulses from droplets caused by disruption of the jets is not clear. However, since the converging jets were capable of cutting up to 36 inches ahead of the nozzle a too-rapid jet disruption would seem unlikely.

While this greater depth of cut could be advantageous, this is not always the case if the result is that the large coal blocks are produced. The exact role of the infusion in controlling coal size is currently imperfectly understood and no experiments could be undertaken to define the relationship. At the present time therefore it is not possible to determine whether infusion is a necessary prerequisite for satisfactory coal sizing or whether differential thrust locations across the seam section, with wedging occurring sequentially rather than simultaneously, would give an equivalent or better solution.

Because the slot cut by a convergent nozzle is approximately 1 in. wide (Fig. 4.10), a guard modification, removing the scrape blade and inserting a stronger guard section with a wedge face (Fig. 4.11) was developed. This guard was located within the opening, thus reducing the required slot width from 2 in. to 1.125 in. At the same time, the access slot to the head was lengthened and narrowed, effectively stopping coal from entering the nozzle compartment.

A final modification was made to the guard (Fig. 4.12), milling out one side to allow a divergent jet to be used with this design and this proved successful in maintaining protection for the nozzle chamber while the coal mined was adequately sized.

Figure 4.10. Slot cut by converging nozzles.

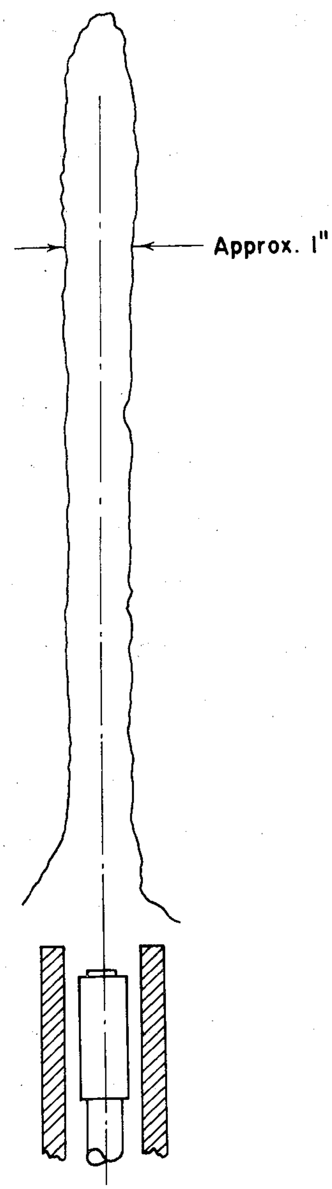


Figure 4.11. Third design for slot guard.

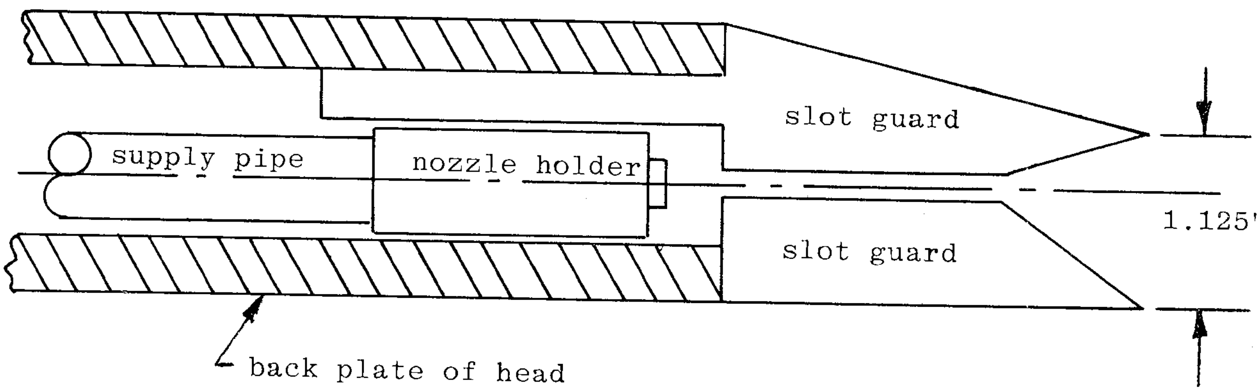
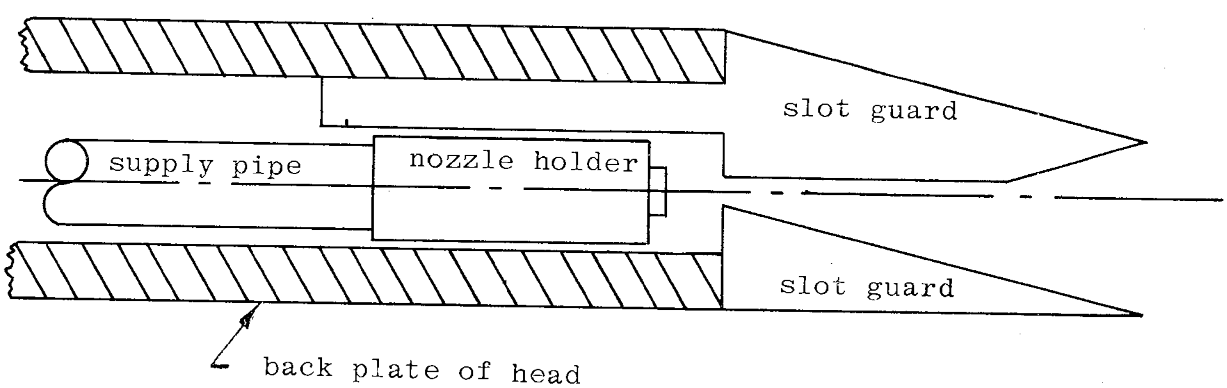


Figure 4.12. Modified version of third slot guard.



Type 3. Problems not Solved Satisfactorily in the Field

3a. Inadequate horizon control.

The connection between the sled and cutting head was through two pins located at the base of the rear of the head and an intermediate hydraulic cylinder attached to the support rail and the head (Fig. 4.13). The pins allowed the head to rotate, relative to the sled, when the hydraulic cylinder was retracted or extended. With this arrangement the head has only one degree of freedom relative to the sled and that is rotation about a horizontal axis perpendicular to the face. This configuration proved to be an unsatisfactory method of guiding the head.

The problem had several components. Firstly, the jets cut an access slot for the head, in which the bottom nozzle oscillated protected by a guard. This guard extended into the precut slot being made for the next pass (Fig. 4.14). Because the cut in the face was only slightly wider than this guard the head could not be raised without lifting the superincumbent coal.

Secondly, since the head pivoted at the back the lifting or lowering of the head moved the entire bottom surface, and since the space under the head was filled with debris as the unit advanced attempts to lower the head instead caused the sled to elevate.

Thirdly, the jets cut the required slot width some distance ahead of the nozzle and thus when the head was either raised or lowered the slot was often initially

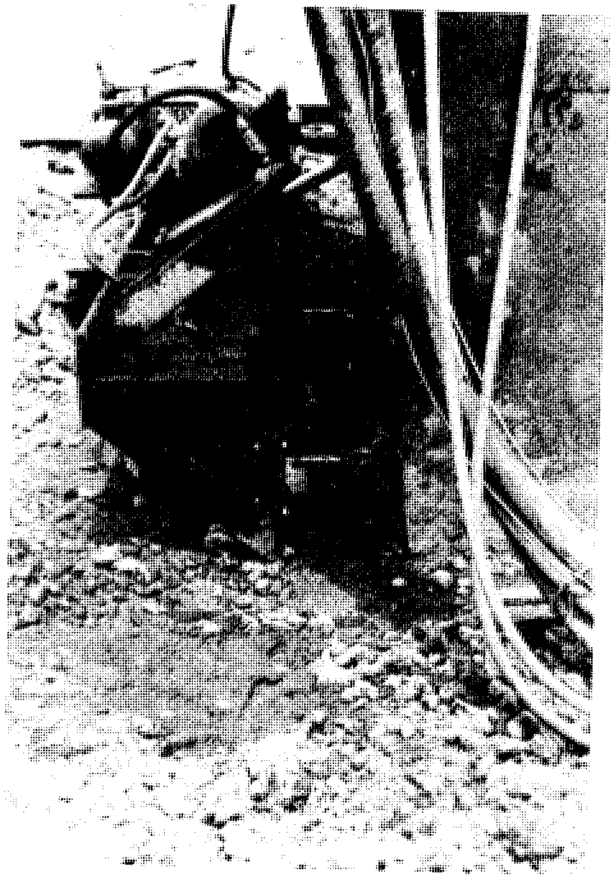


Figure 4.13. Connection between head and support sled.

Figure 4.14. Overhanging portion of cutting head.

not wide enough for the head to advance without catching the edge of the cut.

Another difficulty with the head guidance system was that the response time on the hydraulic cylinder was too slow, despite the change from manual to mechanical drive. While a measure of control could be achieved at 1 ft/min, with advance rates of 5 ft/min and above the unit had advanced too far during the reaction time of the system for the steering to be effective.

These guidance problems could not be solved in the field. To have done so would have required removing completely the overhanging portion of the bottom horizontal compartment, a major structural modification. In addition, it would have required redesigning the linkage drive system for the bottom horizontal arm.

3b. <u>Too much coal trapped between the head and the sled</u>.

Referring to Figure 4.15, the path of the coal over the head and sled is clearly illustrated. The head successfully gathered the coal and could also elevate it to the proper height for conveyor loading. However, since the conveyor was not present to carry the coal away, the coal tended to build up between the head and sled. A modification to the shape of the bottom wedge, increasing the slope to widen the passage onto the sled proved sufficient to move coal onto the sled.

No complete solution for this problem could, however, be achieved without a major revision of the location of

Figure 4.15. Coal trapped between sled and face.

the head relative to the sled. A complete solution would require using a conveyor to transport the coal away from the area of the head.

3c. The head is too open at the back and on top.

The interior of the head was open to the outside at all times along the back as shown in Figure 4.16. In addition, the top was sometimes open when the cover plate was not installed or improperly positioned. These openings caused problems when the unit was hauled back or run without the cover, allowing debris to enter the interior of the head.

3d. The head is structurally weak in some areas.

Figure 4.17 shows a section of the leading edge of the vertical wedge. The two steel plates protecting the nozzle are 0.375 in. thick and were intially unsupported across a span of 20 in. When this section of the head was jammed into a slot in the coal that was less than 2 in. wide, a squeezing action resulted which caused a large structural deflection of these plates. In some cases, the deflection was large enough to pinch the rubbing block on the nozzle holder and stop the cutting arm from moving. In the original design considerations, the wedging action of the coal was not anticipated this far forward on the cutting head, since with a 2 in. slot this area would not be required to withstand any appreciable load.

The 0.375 in. plate on the goaf side of the head was stiffened by welding the plate shown in Figure 4.18 to it.

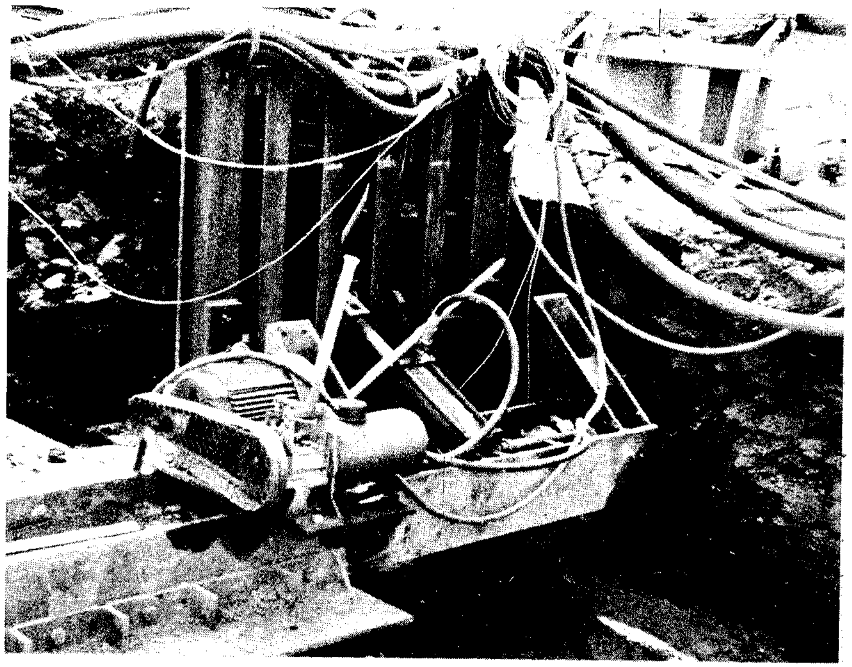


Figure 4.16. Open areas on back of cutting head.

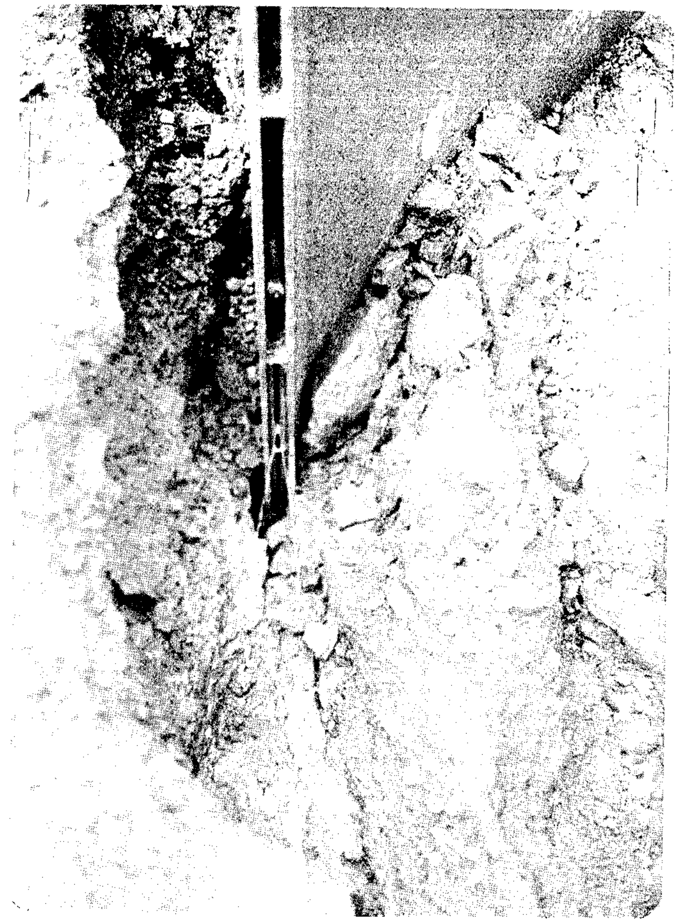


Figure 4.17. Weak construction on nose area of cutting head.

Figure 4.18. Stiffening plate added to nose section of head.

Figure 4.19. Weak construction with vertical wedge sitting on top of 3/8" plate.

It was not possible to stiffen the face side plate in the field, although this was attempted when the scraper blade was welded to the face side of the head.

A second area of the cutting head that was found to be too weak is shown in Figure 4.19. This is the connection between the base plate and the vertical wedge through a 0.375 in. plate spanning a 24 in. distance. When the hydraulic cylinder at the back of the head forced down on the base plate, the 0.375 in. plate shown in Figure 4.19 deflected, in some cases, sufficiently to pinch the rubbing block on the bottom horizontal arm and stop its motion.

3e. <u>The linkage drive to the cutting arms is not strong enough</u>.

The cutting arms and linkage drive were manufactured from relatively light weight structural components (Fig. 4.20). For this reason, a pressure release valve was installed in the hydraulic system to protect the mechanism from damage. The components were simply not strong enough to plow through the volume of debris that surrounded the arms. When the setting on the hydraulic release was increased, the linkage failed on one occasion when it attempted to operate surrounded by debris.

It would be better if in future designs the cutting arms each had their own independent drive mechanism and hydraulic motor. With this arrangement, the speed of each arm could be controlled separately and in the event of a malfunction it would be obvious which arm drive was at fault.

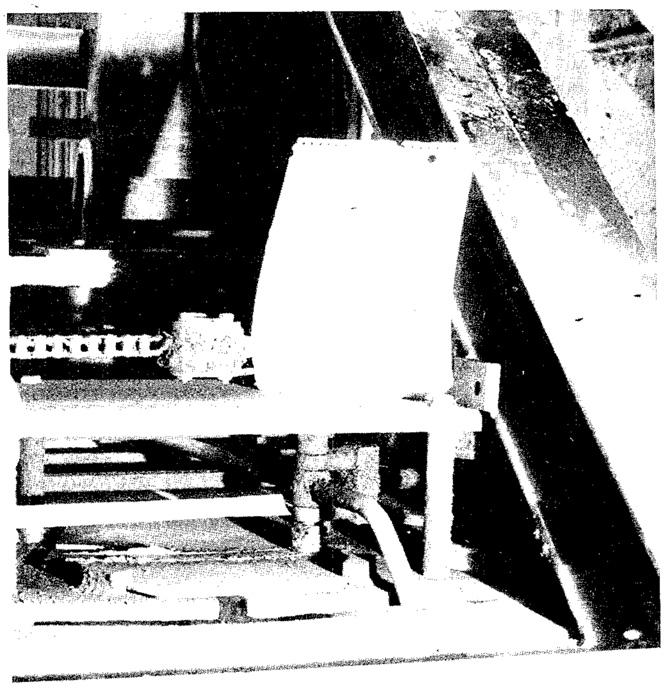


Figure 4.20. Light weight linkage components.

3f. <u>The nozzles were hard to install in correct orientation</u>.

The brass nozzles were held against the ends of the stainless steel supply tubing by a steel holder. Each nozzle had to be oriented so that the orifice holes would be aligned either horizontally or vertically. Since there was no provision to index the nozzle relative to the tubing, it usually took several tries before the nozzle holder was tight at the same time as the orifices were properly positioned.

Chapter Five

DESIGN FOR THE NEW CUTTING HEAD

Introduction

In the previous chapter the discussion reviewed specific machine design problems encountered in the field tests with the first cutting head. The objective in designing the new head was to ensure that all of these faults were corrected and that no new problems were generated. The following discussion presents the design for the new head and references the solution to the specific problem listed in the previous chapter by enclosing the relevant number in parenthesis.

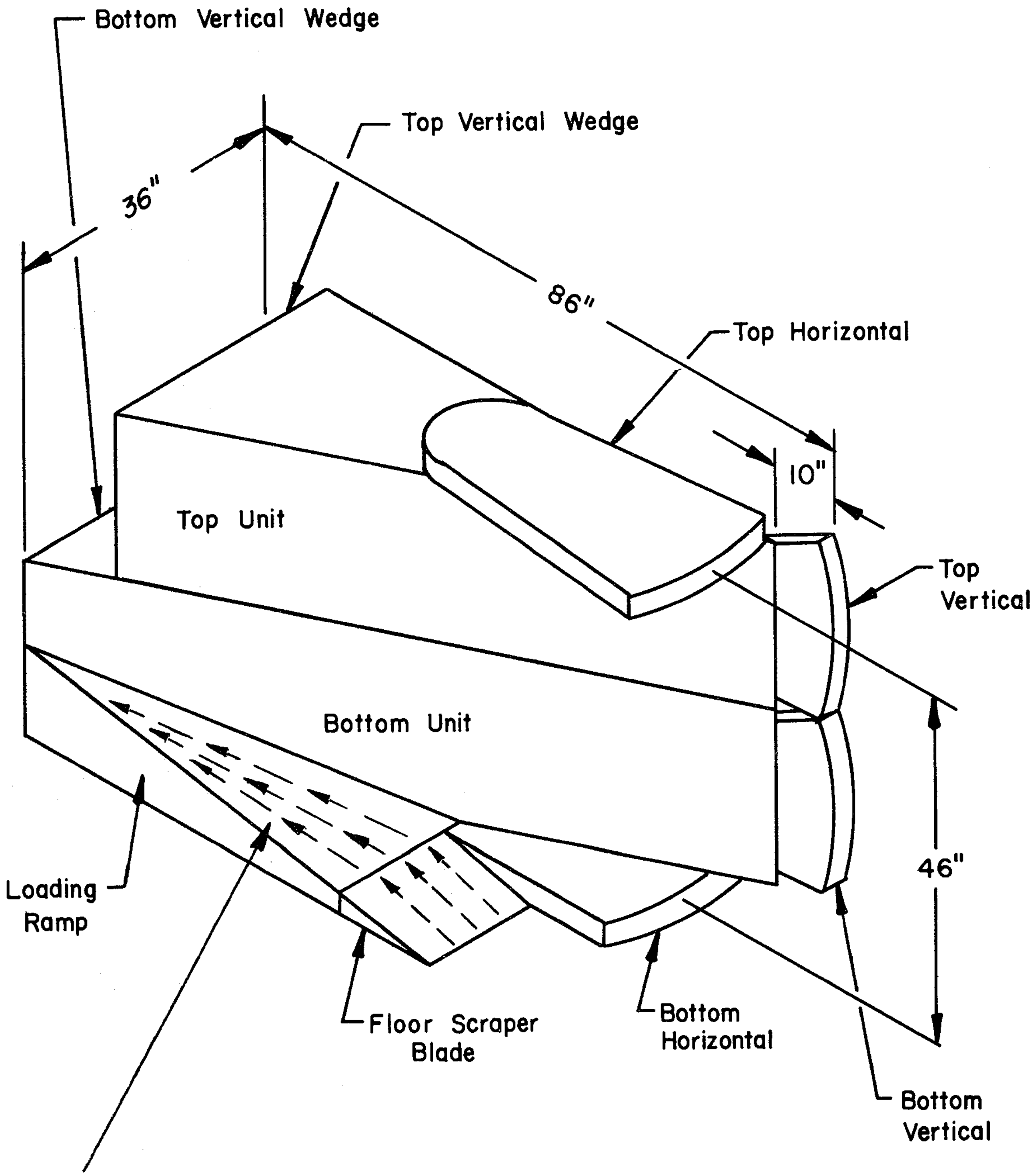
Exterior Features

The design for the new cutting head is shown in Figures 5.1 and 5.2. At first glance the new head appears similar to the old head. In many respects they are quite similar, but in the details of the design the new head differs markedly from the first version. For example, the new head is slightly smaller than the old but weighs roughly twice as much. The increased weight reflects the advice of numerous people associated with mining equipment that the design needs to be "heavy and rugged". Also, additional weight is required to correct the specific structural weaknesses (Prob. 3d) of the old head. The maximum dimensions of the new head are shown in Figure 5.1.

The design shown in Figure 5.1 consists of two separate units: the top and bottom. The top unit has a vertical wedge shape to which a top horizontal and top vertical cutting compartment are attached. The cutting compartments contain the

Figure 5.1

Proposed design for Hydrominer II



Coal flow up loading ramp to conveyor height and over away from bottom unit.

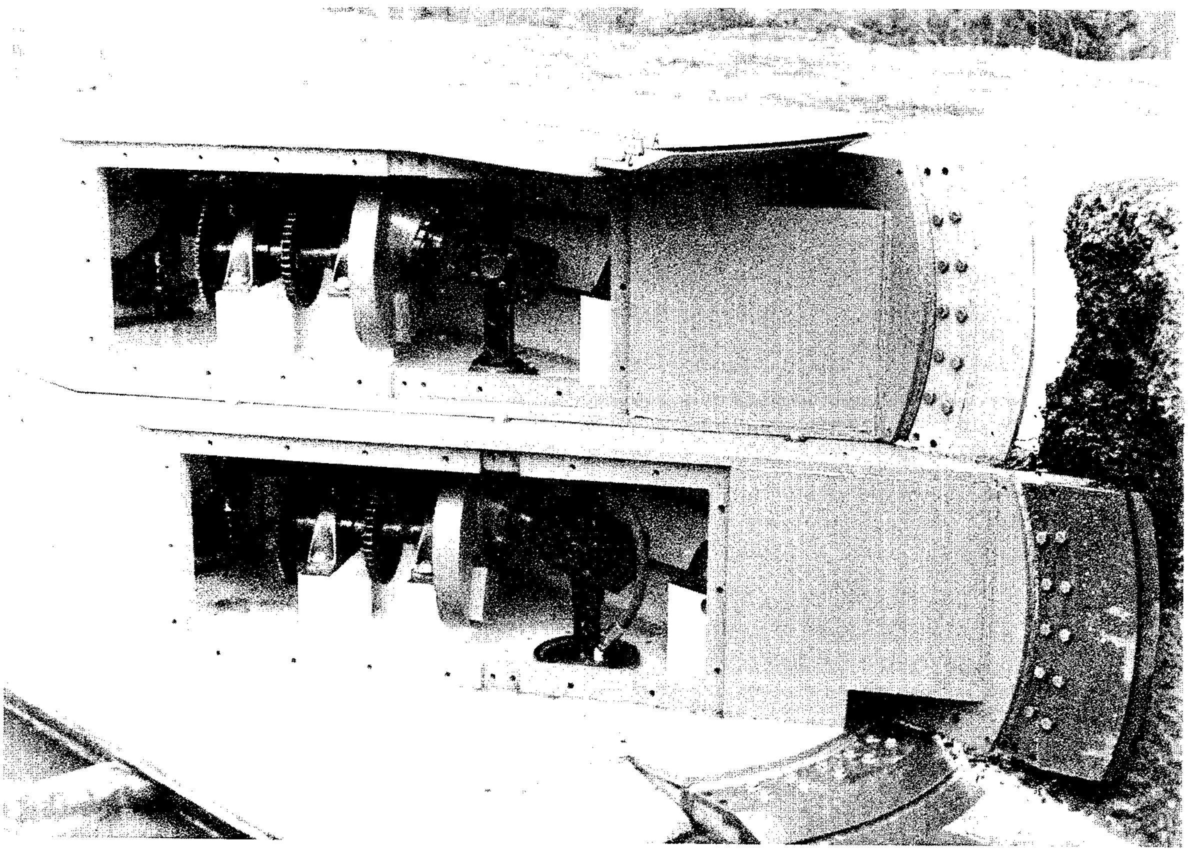


Figure 5.2. Hydrominer II shown with cover plates removed to expose the linkage to move the jet cutting arms.

cutting arms and nozzles, which are located outside of the main body of the machine to eliminate the drag (Prob. 1b) of the entire face area of the head on the coal surface.

Notice that the leading edge of each cutting compartment is curved. The purpose of this shape is to maintain the nozzle as close to the coal as possible in all the positions of its motion. This shape also wedges the coal at point locations rather than simultaneously wedging across a broad area, as occurred with the experimental head. The bottom unit has similar horizontal and vertical cutting compartments and in addition it has a loading ramp and floor scraping blade. Note that the bottom horizontal compartment and floor scraping blade are located below the level of the bottom unit base plate. This reduces the drag on the bottom surface (Prob. 1b), but more significantly makes it possible for the head to move relative to the support sled (Prob. 3a) by:

1. Rotating about a horizontal axis perpendicular to the face.
2. Translating along a vertical axis.

<u>Interior Features</u>

Both the top and bottom units are completely sealed (Probs. 2b and 3c) to prevent any debris from entering into the head interior. The drive mechanism for the individual arms transmits motion to the cutting arms through a permanently sealed ball bearing. This excludes debris from the interior of the top and bottom units, but allows passage of the high pressure water and the proper motion to be transmitted to the cutting arm. The individual cutting compartments open through guards on the leading edge to allow the high pressure jets to exit.

The jets exit through a slot made as narrow as possible consistent with not disrupting the jet flow. These slots are formed by a removable guard. The purpose of this is to match the leading edge profile of each guard with the type of nozzle being used (Probs. 2c, 2d, and 2e). For example, if a 10 degree diverging nozzle is used, the cross-section of the required guard is shown in Figure 5.3. If a converging nozzle is used, then a different guard design (Fig. 5.4) is bolted in place. Alternate guard designs are shown in Figures 5.5 and 5.6.

Since all of the guard designs have a long entrance path and a slot minimum width of approximately 0.2 in., only a small amount of coal enters (Prob. 2b) the cutting compartments. The consideration then was what to do with the coal that does enter. The solution for the vertical cutting compartments was relatively simple since gravity, along with a flushing system, causes the coal to accumulate in the lowest area of the compartment. Figure 5.7 illustrates that the coal and water can exit from the vertical compartments at the locations designated through an opening provided in the compartment wall.

The two horizontal compartments required a more elaborate system to remove coal from their interior. Some of the debris is pushed out of the interior of the horizontal compartments by the motion of the cutting arms through openings provided at each end of the arm's stroke. It was initially proposed that the remainder would be removed by the combined action of a flushing system and wet or dry vacuum system. This system

Figure 5.3

Guard design for diverging nozzles

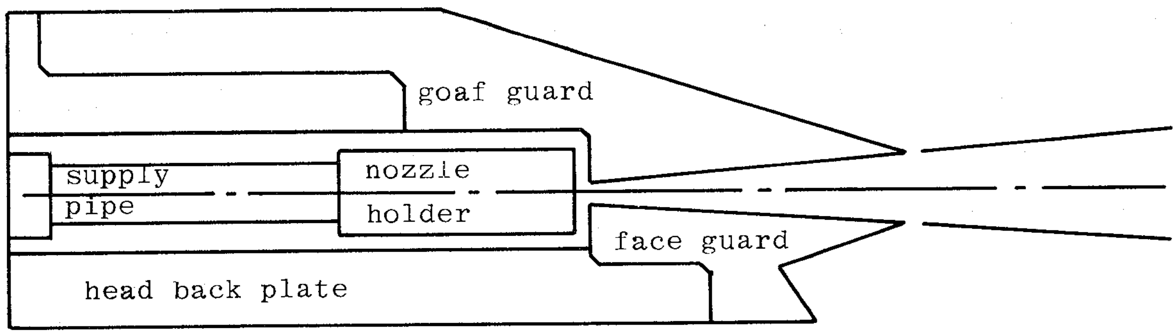


Figure 5.4

Guard design for converging nozzles

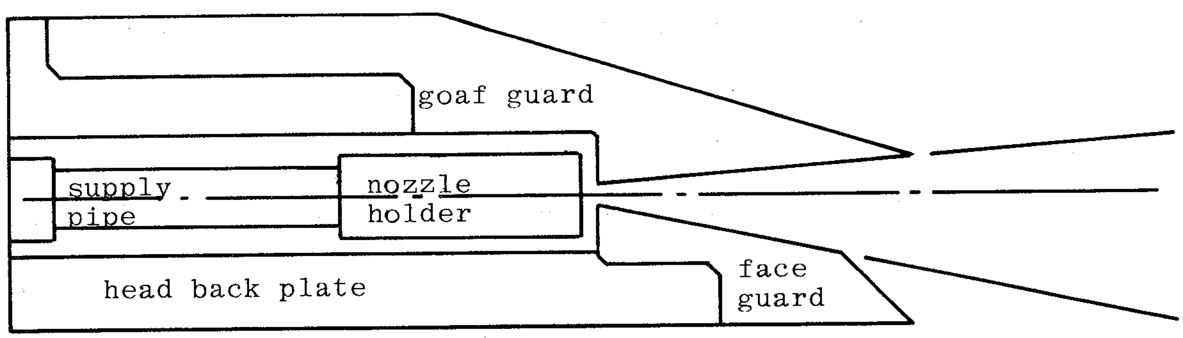


Figure 5.5

Alternate guard design for
diverging nozzles

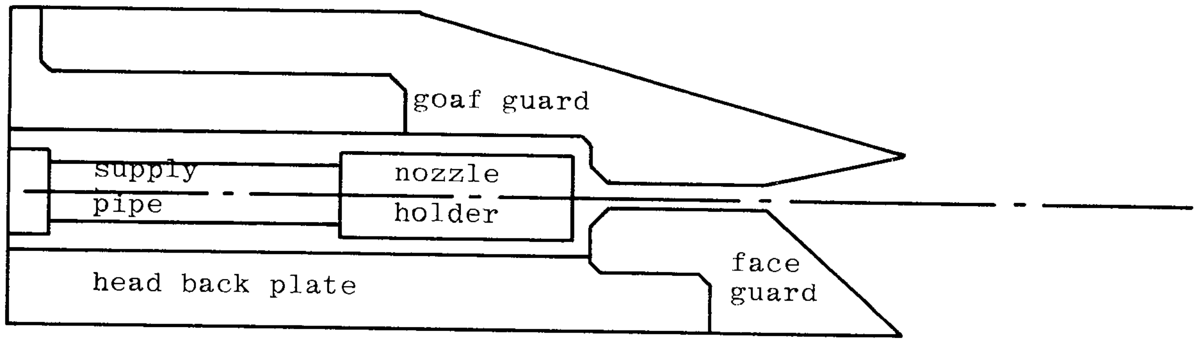


Figure 5.6

Alternate guard design for
converging nozzles

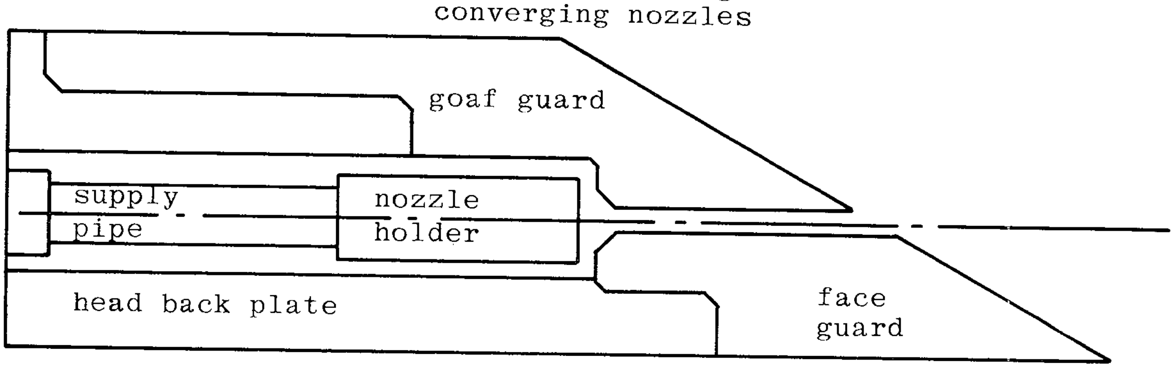
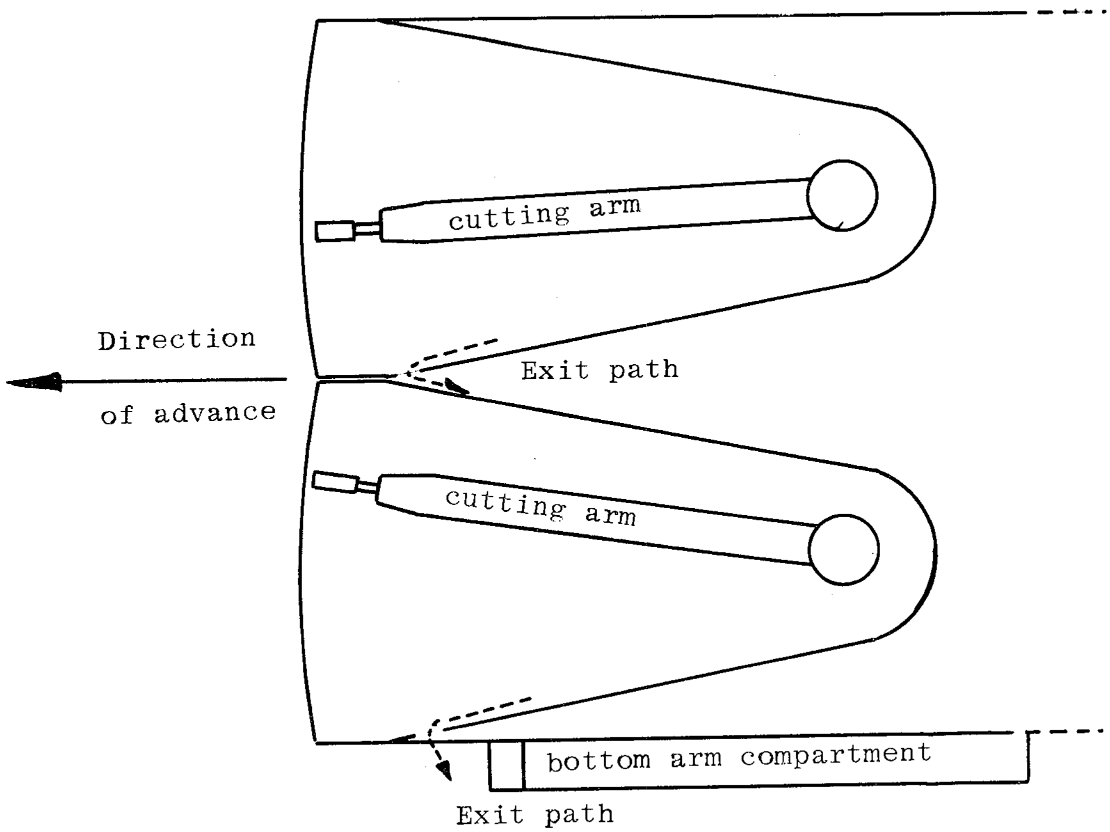


Figure 5.7
Exit paths for coal from
vertical compartments



was not, however, installed, and, in the trials at Bruceton the compartments remained satisfactorily clean, although the tests were not long enough to ensure the absolute validity of this conclusion. It will take a certain amount of experimentation with the new cutting head before the correct combination of flushing and forcing debris out of the cutting compartments is defined.

A photograph of the drive mechanism used to oscillate the individual cutting arms is shown in Figure 5.8. This preliminary prototype working unit was constructed and tested to verify that it had the desired kinematic characteristics and ruggedness required (Prob. 3e). Operation of the drive is achieved by power from the hydraulic motor which is transmitted, through a roller chain, to the 2 in. diameter input shaft. A flywheel attached to the drive shaft carries a spherical flange cartridge bearing assembly. A 1-1/2 in. diameter shaft extends through the spherical bearing and transmits motion to the vertical output shaft. The cutting arm is attached to the output shaft and it executed a sinusoidal oscillation when the input shaft rotates at a constant rate. All of the bearings in this mechanism are permanently sealed ball bearings.

The drive mechanism was tested at operating speeds to 300 rpm on the input shaft corresponding to 300 oscillations per minute of the arm. The unit operated smoothly and quietly at this speed. A high pressure water line was connected to the arm and the resulting jet observed as it exited through the nozzle while the linkage was in motion. The linkage performed satisfactorily in all tests and was therefore adopted for use in the new head.

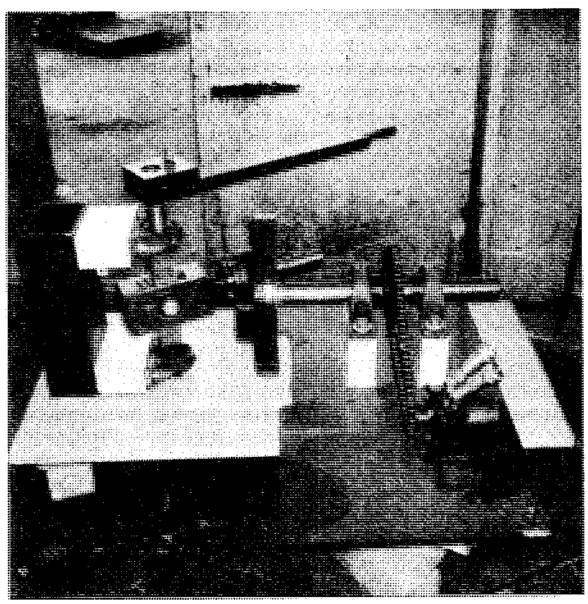


Figure 5.8. New linkage drive system.

There are two drive mechanisms located in both the top and bottom head units. These provide an independent drive for each arm so that their operating speeds can be independently adjusted in the range from 0 to 300 oscillations per minute. The actual arrangement for transmitting power and water to the arms is shown in Figure 5.9. Note the division between the interior of the cutting head and the exterior of the cutting compartment.

Figure 5.10 shows the nozzle modification adopted to correctly orient the nozzle relative to the cutting arm (Prob. 3f). Two small pins were located in holes drilled in the end of each arm. When the nozzles were manufactured, matching holes were located in the nozzle body at the proper locations. When the nozzle holder rotates to press the nozzle against the end of the arm, the pins prevent the nozzle from rotating away from its correct position. In addition every nozzle was manufactured from electroformed nickel to reduce wear (Prob. 2a).

The nozzle design was also changed. In the earlier program the dual orifices separated from the main flow in the horizontal plane. By redesigning this separation to occur in the vertical plane (Fig. 5.10) the gap required to allow the jets to exit from the guards could be further reduced (Prob. 2).

Additional Features of the New Head Design

Many components of the new design are interchangeable. For example, each of the removable guard components can be installed in any desired location. All of the cutting compartments are identical except in minor details. All of the

Figure 5.9

Detail of separation between inside of cutting head and arm compartment

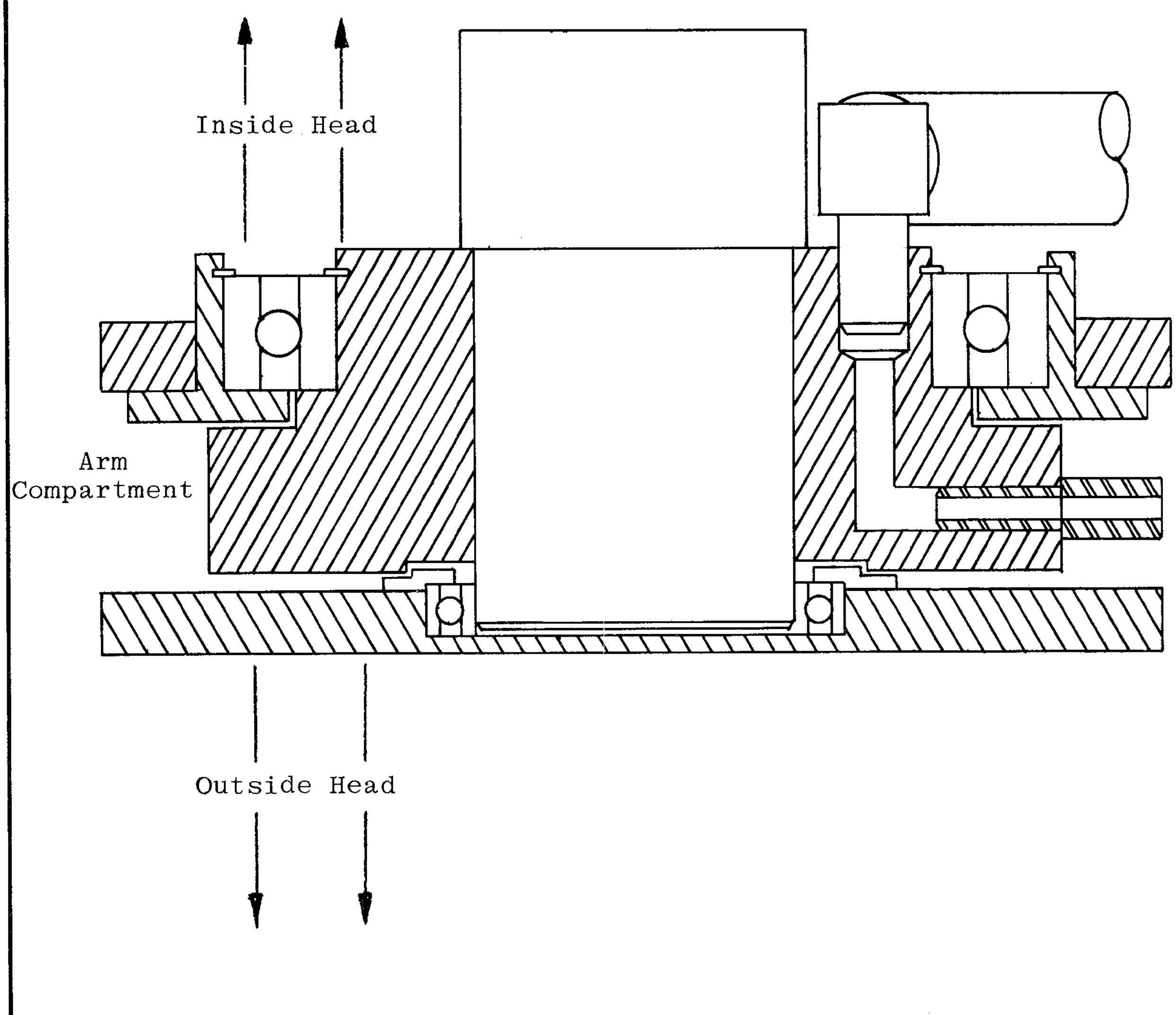
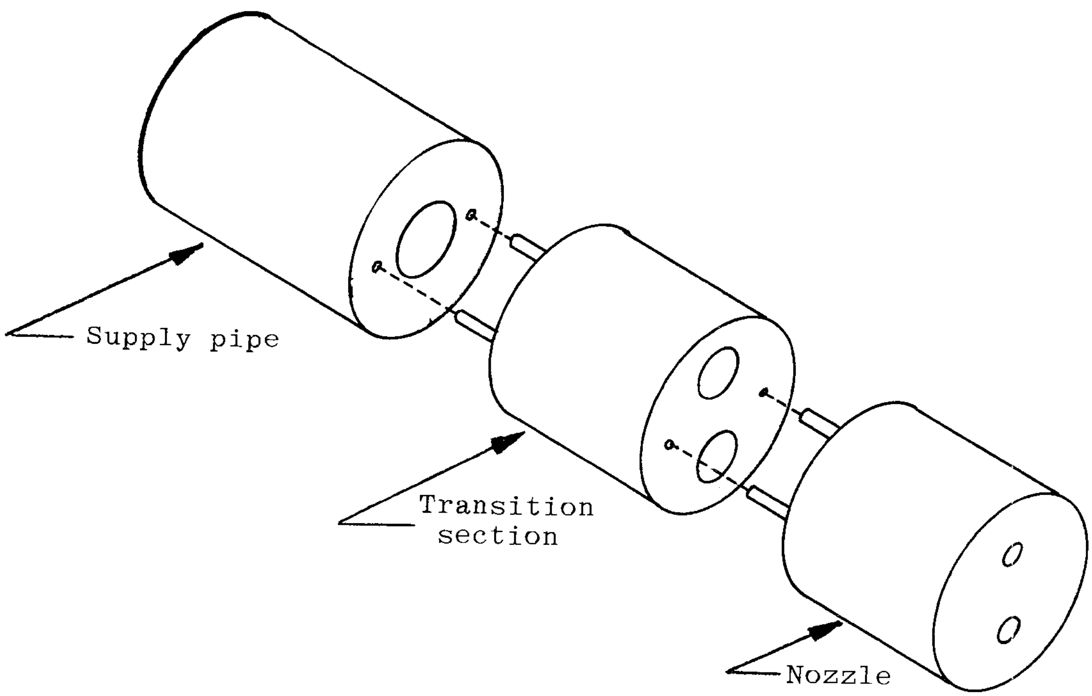


Figure 5.10

Nozzle locating system



drive mechanisms are identical. This feature reduced the requirements for spare parts and should make servicing the machine much easier.

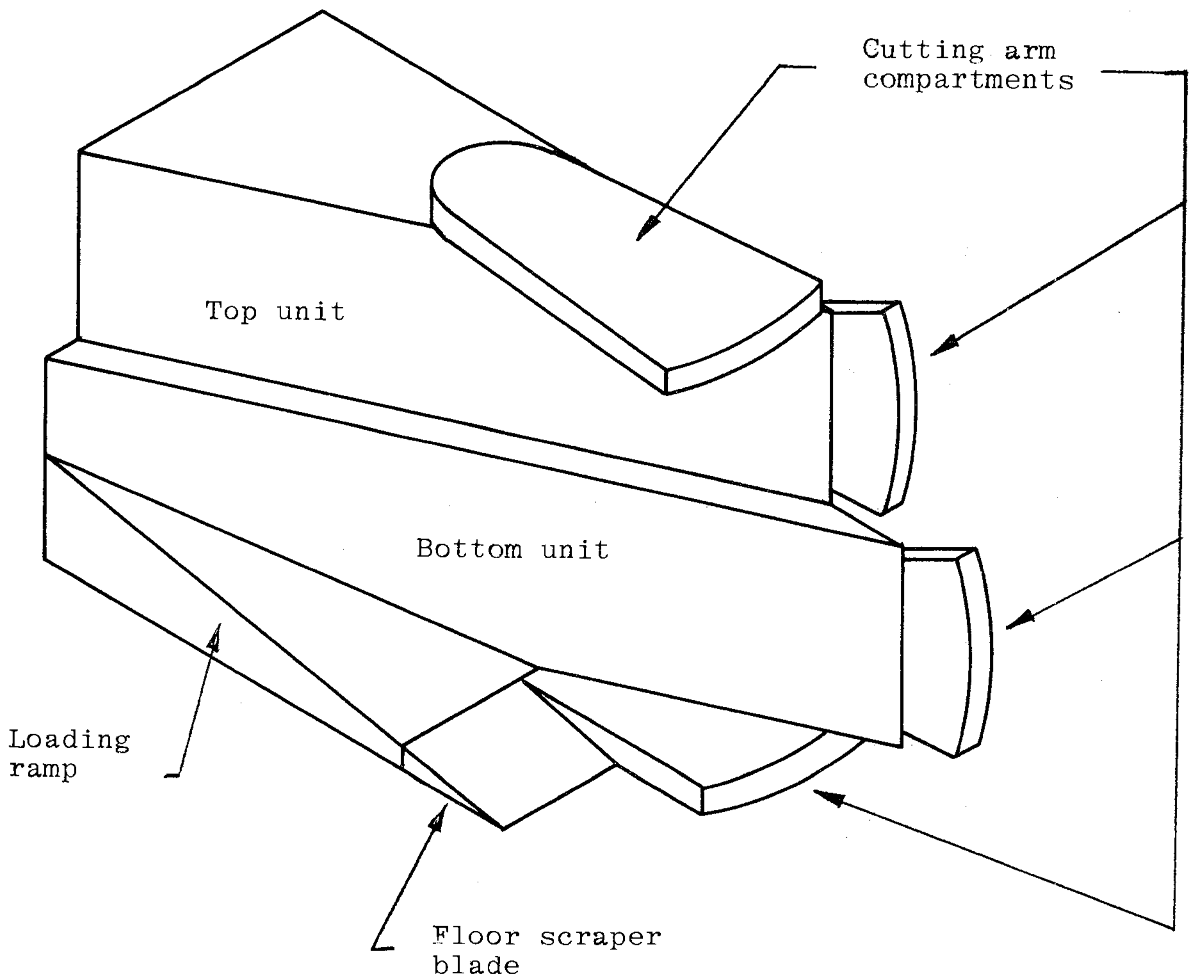
Another feature which is present in the second generation head is shown in Figure 5.11. The top unit has been positioned relative to the bottom unit approximately 10 in. farther away from the coal seam in the direction of the advance. This should allow better mechanical fracturing since the mining sequence will be to break the coal first by wedging it horizontally with the bottom vertical compartment. Next the bottom horizontal and top vertical wedging will occur, and finally the top horizontal unit will wedge any remaining coal away from the roof. This feature may prove to be a significant advancement over the old head design. For example, the available flow of high pressure water can be redistributed to place a larger portion of it into the bottom unit nozzles. When the coal is fractured by the bottom unit, much of the top may collapse and fall away as happens with conventional plows. The top unit can also be positioned 10 in. ahead of the bottom unit to determine the effect this has on the force required to fracture the coal.

Support Sled Design

In the initial design of the Hydominer it was intended that two pumps would be affixed, one to each end of a power pack which would not only power the pumps but also provide the haulage unit to pull the machine down the face (Ref. 13).

Figure 5.11

Alternate location for top cutting unit



The shearer available at Bruceton, however, was not configured so that this was practical, since the shearer drum motors were, for this unit, separate and mounted on the ranging arms. For this reason, a separate "support sled" was required, to travel with the cutting head along the face, to carry the equipment necessary to provide high pressure water to the head. This sled would carry two size 4J Kobe pumps and their motors, the hydraulic drive unit required for the cutting arms, and the necessary electrical switch gear for these units.

Discussions were held with the shearer manufacturer and agency staff both in Washington and Bruceton, as a result of which the University was asked to build the support sled.

The support sled was designed and its construction supervised by Dr. Rhea, an Associate Professor of Mechanical Engineering employed on the contract for this specific purpose.

Design Considerations

Discussions between the shearer manufacturers, the agency staff and ourselves indicated that the best course of action would be for the sled to be manufactured, to as great a degree as possible, at Rolla. To this end, based on the drawings supplied by Joy and measurements made on site, the unit was manufactured to fit on the Joy conveyor and to be pushed by the shearer.

Two initial points were considered in the design; firstly, should the sled be placed between the shearer body and the cutting head (Fig. 5.12), or trail the shearer (Fig. 5.13). The advantage of the former design would be that the length of high pressure tubing required would be reduced, improving unit safety and reducing friction losses to the head. The advantage of the latter configuration was that the sled would not have to carry the thrust (50,000 lb) between the cutting head and the shearer.

The second point related to the stability of the cutting head and the sled as it traveled along the conveyor. Because of the weight of the cutting head, some concern was expressed as to the effect that this would have on the total system equilibrium. Detailed examination of the problem indicated that, since the cutting head was approximately half the weight of a single shearer drum and ranging arm, the design was inherently stable (Fig. 5.14). This analysis, in itself, went beyond normal operating conditions for the unit since, in such situations, the cutting heads normally rest on the floor of the cut.

An overall engineering analysis revealed that the arrangement with the sled between the head and the shearer had the following advantages:

1. Only one precision hookup is required.
2. There is no possibility of the machine tipping about the front shoe.
3. The maximum side force required to support the conveyor is less than half of that force required by the alternate arrangement.
4. The cutting head can be supported more rigidly by elimination of the cantilever effect on the support beam.
5. The pumps are located closer to the nozzles which minimizes flow losses and increases cutting efficiency.

6. Supply hoses can be secured and protected.
7. The complete system can be assembled and tested in Rolla, Missouri.

It was therefore concluded that the equipment should be arranged so that the shearer pushes the sled and cutting head.

Because of the limited time available, and in order to facilitate transportation, the sled was constructed in three parts (Fig. 5.15), and, wherever possible, standard structural shapes were used. These were sized and oriented to ensure proper integration with the force combinations which were anticipated between the shearer, the cutting head, and the conveyor and drive rack.

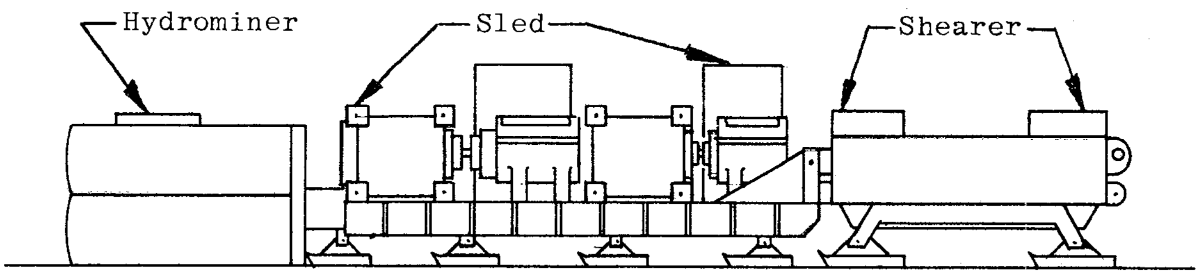
The sled length was minimized, based on the arrangement of the components which would locate all the controls on the operator side of the machine. A factor of safety of 6 was used in the design.

Because of modifications which were made to the shearer and conveyor units at Bruceton, post manufacture, final completion of this sled could not be made until the unit reached Bruceton.

In essence the sled consisted of a two component platform mounted on twin I-beams, filleted for extra strength. The I-beams in turn rode on rotatable shoes, designed to fit on the Joy conveyor. At one end of the sled a pin mounting was established to hold the connecting pin which would attach the unit to the shearer. At the other end a cross beam was attached and braced to connect to the cutting head. The details of the arrangement can be seen in Figures 5.15 and 16.

Figure 5.12

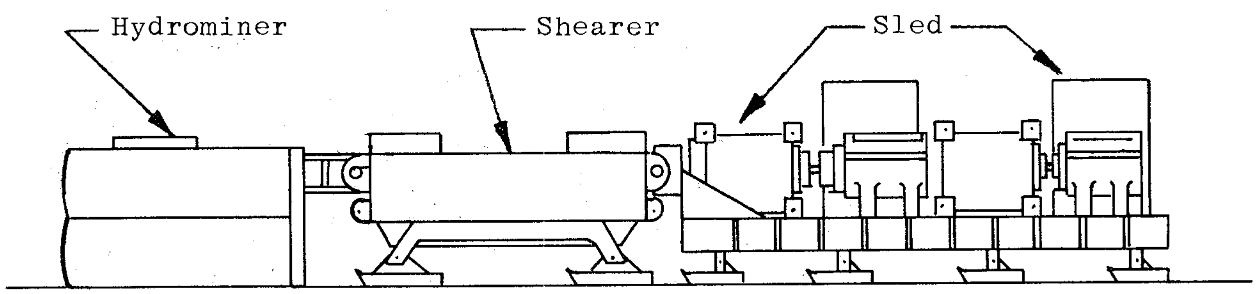
Equipment arrangement for Case 1



Case 1. The shearer pushes the sled and cutting head

Figure 5.13

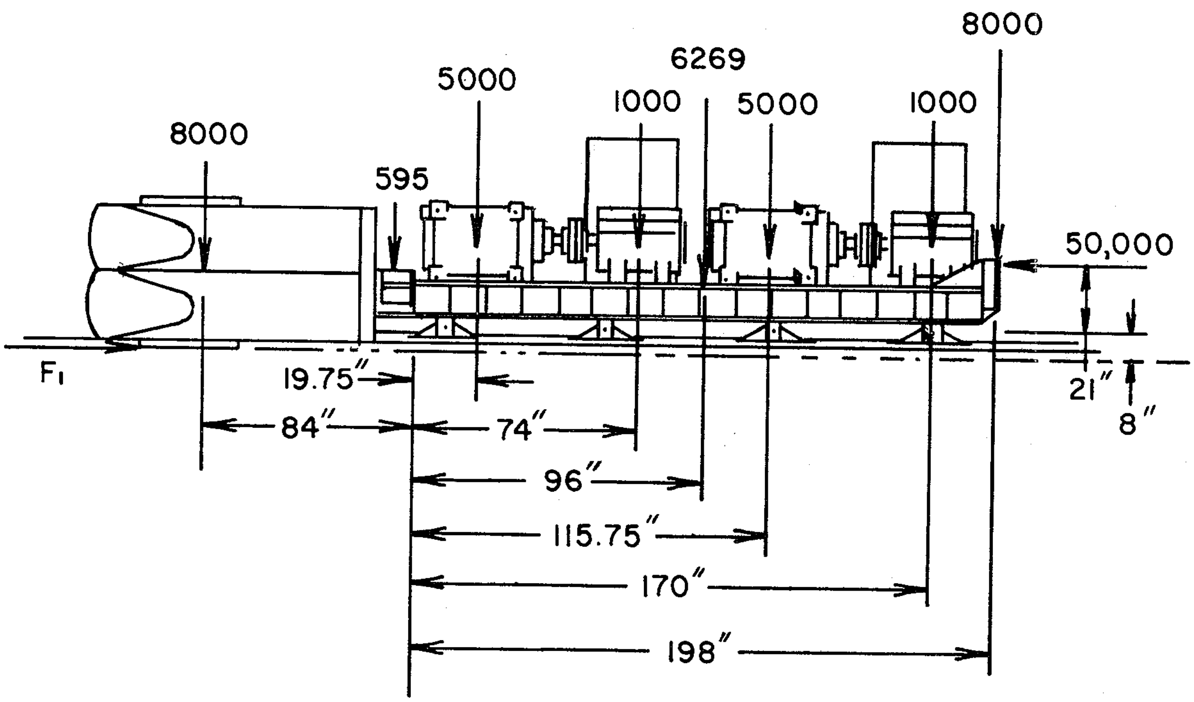
Equipment arrangement for Case 2



Case 2. The shearer pulls the sled and pushes the cutting head

Figure 5.14

Force and location identification
for balance analysis - side view



(all forces are in pounds)

Figure 5.15

Three major components of the sled

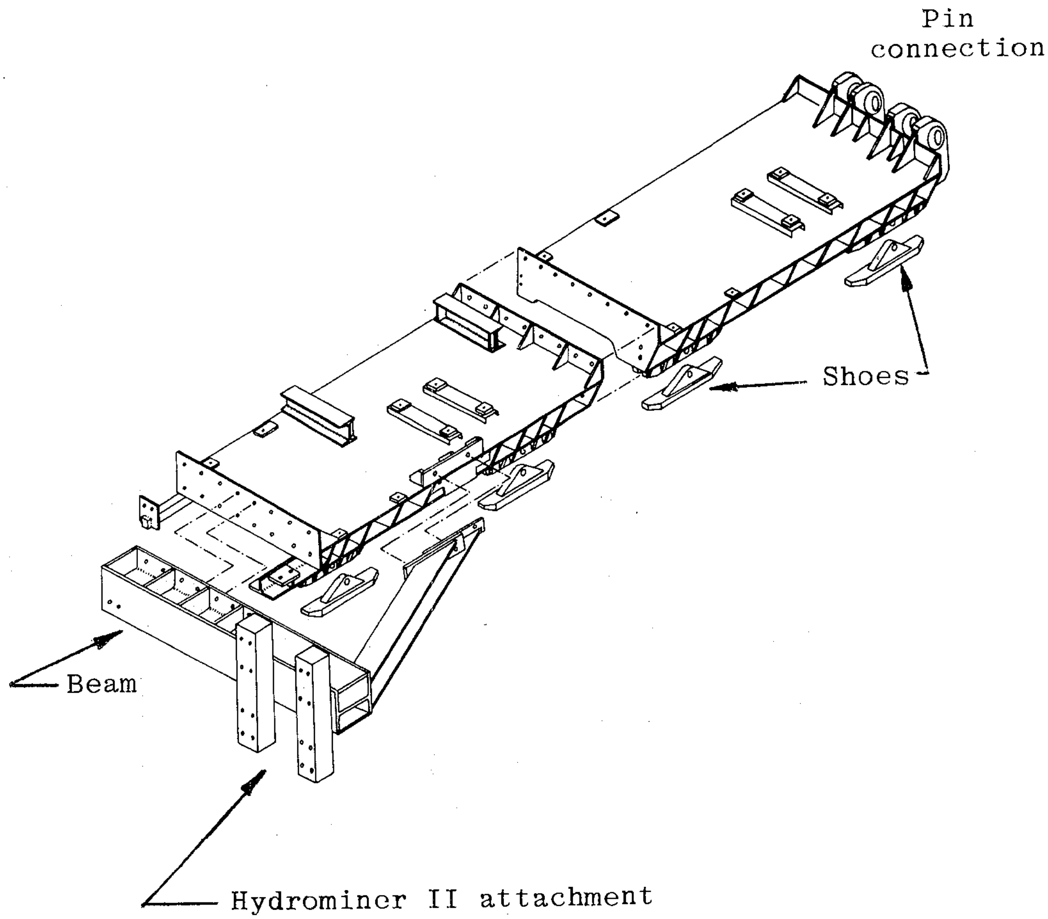
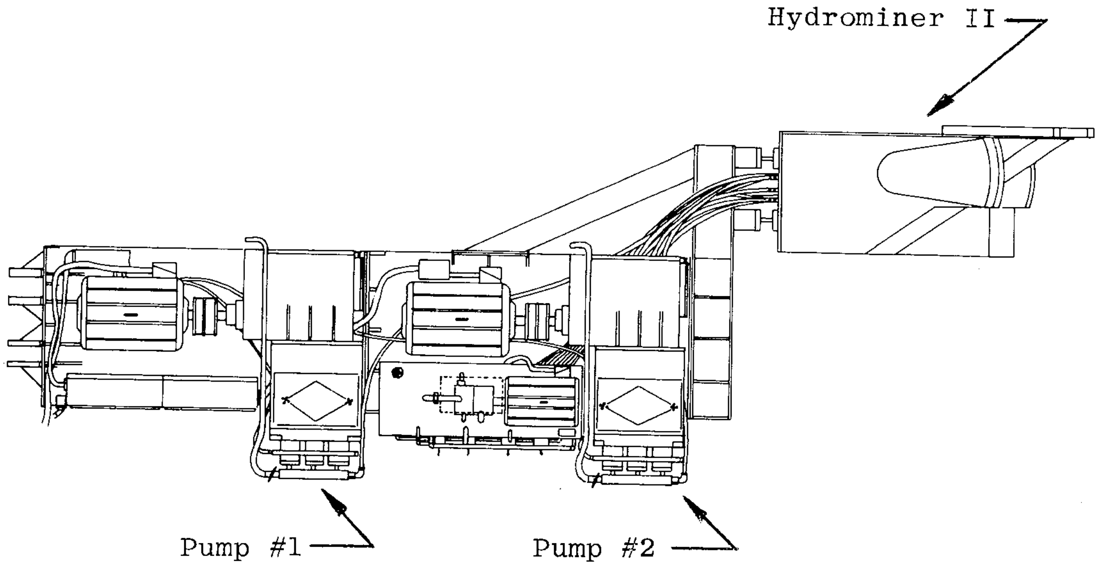


Figure 5.16

Top view of sled and cutting head



Chapter Six

TESTING OF HYDROMINER II

Introduction

The experimentation carried out with the first prototype model of the Hydrominer had shown that it had the potential for attaining equivalent or greater productivity than existing shearer units, the most popular mining machine used on longwall faces. However, there were many problems encountered in the use of the machine in a surface coal mine that increased the difficulty of performing the trials. One particular problem was due to the lack of a conveyor in the test system to remove the coal mined from the longwall face. For example, it was frequently difficult for the machine to advance more than 10 feet without the head becoming buried in coal (Fig. 6.1). The natural variation of material properties from point to point within the coal also made it difficult to compare test performance data from trial to trial.

The need for a consistent material in which to conduct trials of a prototype machine is clearly of paramount importance when the number of variables is as large as it was for a machine such as Hydrominer I. For example, over 30 different nozzle configurations alone were tested. In addition, different cutting arm oscillation rates, advance rates, angles of head rake, etc. were tested. It was anticipated that a similar scale of investigation would be required for the second generation head. Accordingly, it was decided to initially test the second generation of the Hydrominer in the artificial coal seam at the longwall surface test facility in Bruceton, Pennsylvania. The

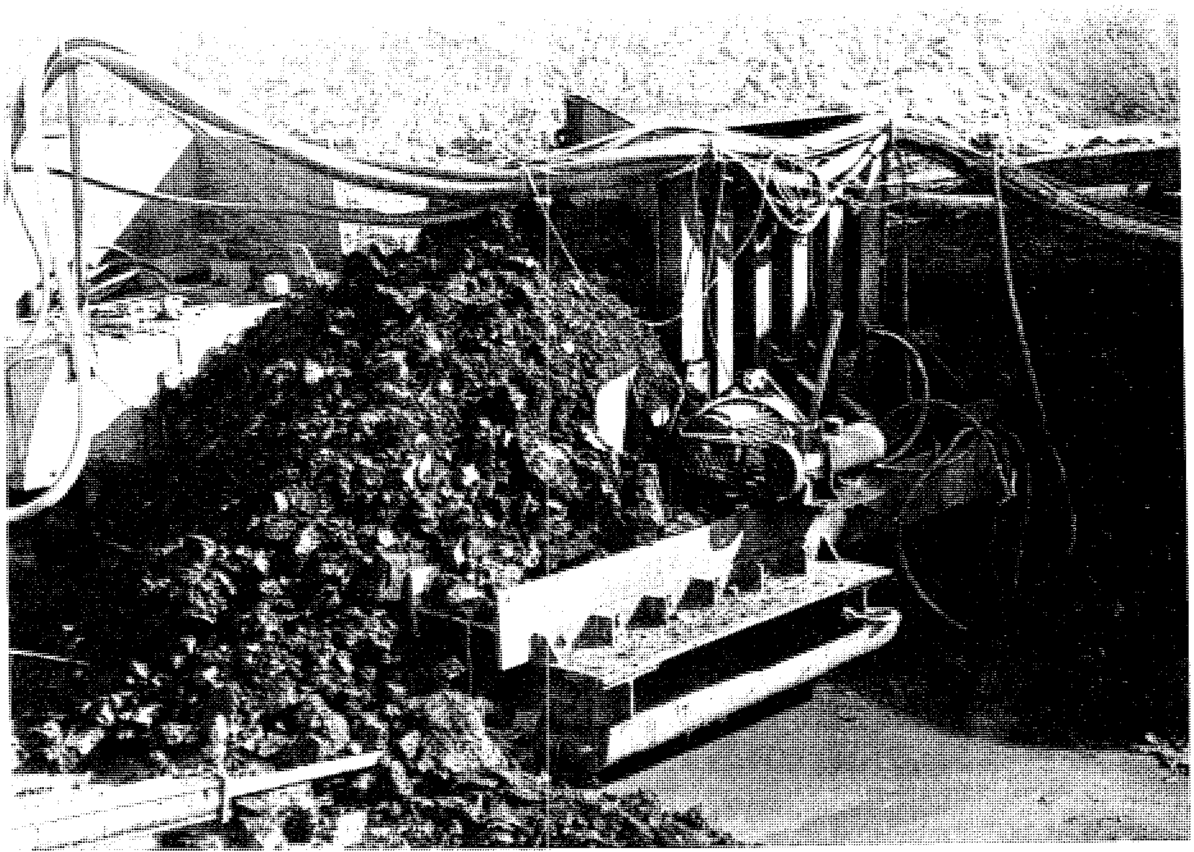


Figure 6.1. Condition of the head after advance through about 15 ft of face at 2 ft web.

intent was for the artificial coal to provide a material whose mechanical properties were constant from point to point, and a sound experimental base would thus exist for comparative purposes.

Laboratory Tests in Rolla

The artificial coal used at Bruceton was based on mixtures developed by the National Coal Board for use in the test headings at the Swadlingcote test site in England (Ref. 14). Prior to taking the machine to Bruceton, however, the University requested test blocks of the mix to ensure that the water jet would be capable of cutting this material during the test program. Concurrently, an evaluation was made of the use of artificial coal as part of a Master's thesis by one of the students working on this project (Ref. 15).

While some adequate cutting depths were achieved in relation to the depths predicated in the initial design of the Hydrominer (Ref. 2), at no time was the artificial coal cut to the depths which were achieved in the testing of Hydrominer I in the surface coal seam. The artificial coal did not respond to water jet attack in a manner which resembled coal. The artificial coal was much harder to cut and the use of greatly improved nozzles, compared to those used in Hydrominer I, failed to produce a slot of consistent width and depth. In one block particularly, the slot cut fell well below the dimensions required if the head were to be advanced effectively. The field trials had established that the machine would not function if it were advanced into a region where a slot did not exist.

There was a considerable concern in evaluating the cuttability of these test blocks, particularly the blocks received from Bruceton, insofar as there were no weakness plane present in the samples, relative to the cleat and bedding planes of a typical coal seam. This concern was because the Hydrominer operates in part through the water jet's exploiting the cleat and bedding planes of a typical coal seam (Fig. 6.2). The coal fragments themselves used in making the block were sufficiently small that they also did not contain the large weakness planes prevalent in the coal massif.

During the course of this phase of the program, some small degree of experimentation was carried out on different mixes of coal that might be used to test water jet cutting effectiveness. However, the program was stopped when a decision was made to use the artificial coal block already cast at Bruceton. As mentioned above, three blocks of this material were sent to Rolla, and were tested under water jet attack. The block which looked most promising as a conceivably representative sample, was retained and used as the demonstration test block during the demonstration of the cutting head performance, before representatives of the Bureau of Mines in September of 1977 (Fig. 6.3). The block was shown to cut adequately, although again the depth of cut was in no way comparable to that achieved in the field demonstration. Tests of the other blocks indicated that even smaller effective cutting distances were achieved (Table 6.1). In coal it has been found, consistently to be practical to cut to depths of up to 22 inches under field conditions. In the laboratory, depths of only 1 or 2 inches could be cut in artificial coal despite a repeated number of passes.

Figure 6.2. Slot cut into coal showing the damage done to the sample by water penetration of the bedding planes and cleat.

Figure 6.3. Slot cut in artificial coal sample at the time of the demonstration to Bureau of Mines personnel, September 1977.

Table 6.1

DATA SUMMARY FROM JET CUTTING EXPERIMENTS IN COAL

	Depth of cut (in.)					
Speed of passes ft/min	10	20	50	100	160	AVE
At 1 pass	2.82	0.70	1.44	1.04	1.78	1.56
At 2 passes	4.72	4.17	2.53	1.64	0.84	2.78
At 4 passes	6.38	5.19	3.59	2.72	3.09	4.19
At 8 passes	7.61	6.99	4.86	4.34	1.91	5.14
At 16 passes	14.34	11.78	8.53	6.34	4.61	9.12

Total number of passes = 31

$$\text{Depth of cut per pass} = \frac{\Sigma\ \text{Depth of Cut}}{31}$$

	1.15	0.93	0.67	0.51	0.40	0.73

A matter of more concern, reflected in correspondence (Ref. 7), was that the material had considerably greater tensile strength than is normally prevalent in coal. A simple wedge indentation test was performed on a block of the artificial coal which had been previously slotted with a saw cut (Fig.6.4). This indicated that where the machine was taking a 2 ft wide slice on a 4 ft section with the coal breaking at 45 degrees forward of the machine, as was found to be the case in field trials, that this would require the entire haulage force of the machine to break off the cantilever of the coal. This situation is totally unrepresentative of that which was found in the trials of the machine in coal.

The depths of cut which were achieved in the tests of the artificial coal at Rolla were, however, similar to the initial 3 inch depths of cut upon which the initial design of the machine was predicated (Ref. 2). This sufficed as justification of the decision to move the equipment to the artificial coal test facility in Bruceton, Pennsylvania. This move was made under accelerated conditions, due to the onset of winter in Bruceton, with the problems which this would entail in the operation of the high pressure equipment. The unit was taken to Bruceton and added to the front end of a Joy shearer, which had been undergoing tests at the Bruceton facility. The support sled carrying the pumping units was mounted ahead of the shearer, which had had the cutting drums removed. The cutting head was mounted on the front end of the support sled (Fig. 6.5).

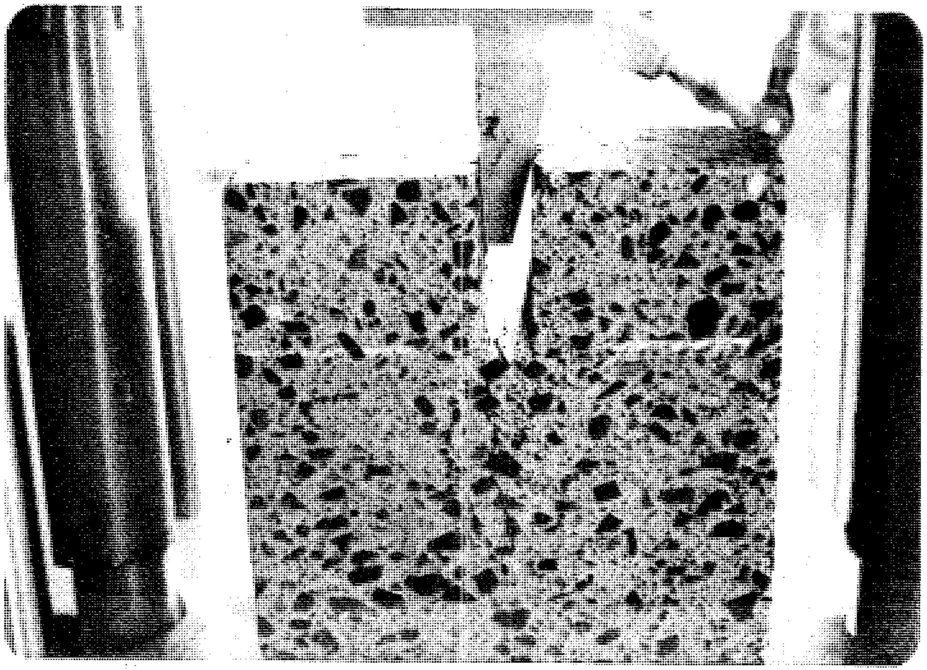


Figure 6.4. Configuration of wedge test to determine "ploughability" of the artificial coal from Bruceton.

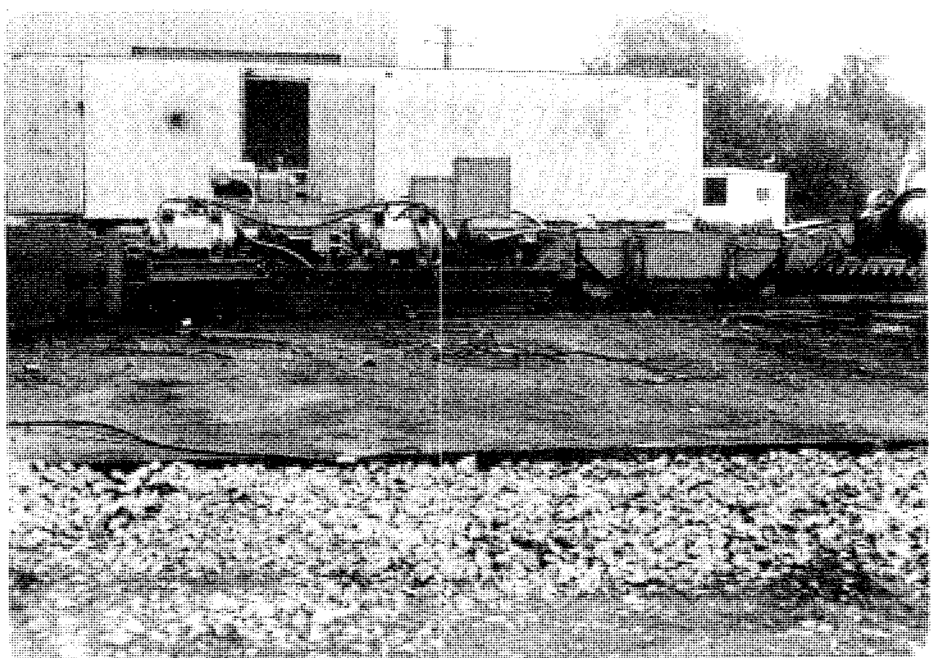


Figure 6.5. Configuration of test equipment used at Bruceton.

Testing at Bruceton

The investigators on this contract were very concerned with the quality of the pour of this first test block. It had, for example, been noted that there was more water added than called for in the specification. This had led to a segregation of concrete in the lower sections of the pour, with different degrees of void ratios established in the upper layers of each pour and a segregation of the coal from the cement (Fig. 6.6). Nevertheless, despite these conditions which would make accurate correlative testing difficult, it was decided to continue.

The Principal Investigator was present at Bruceton during the week of October 10-14, 1977 to supervise the attachment of Hydrominer II to the equipment sled designed by Dr. Rhea. At the end of the cutting head installation, Mr. Ronald Robison, Senior Laboratory Mechanic at UMR, and the Principal Investigator performed a series of equipment checkout procedures to ensure that everything was operating satisfactorily. As part of this procedure, the cutting head was advanced to the end of the artificial coal face and several different nozzles were tested to see how the material would respond to jet attack at 10,000 psi operating pressure. Single orifice nozzles, dual orifice diverging, and dual orifice converging nozzles were all tried without advancing the head into the coal seam.

Because the head was not being advanced, the same area of the material was exposed to attack by the water jets from each of the various nozzles tested. The results of these tests were very disappointing since the material was even harder to

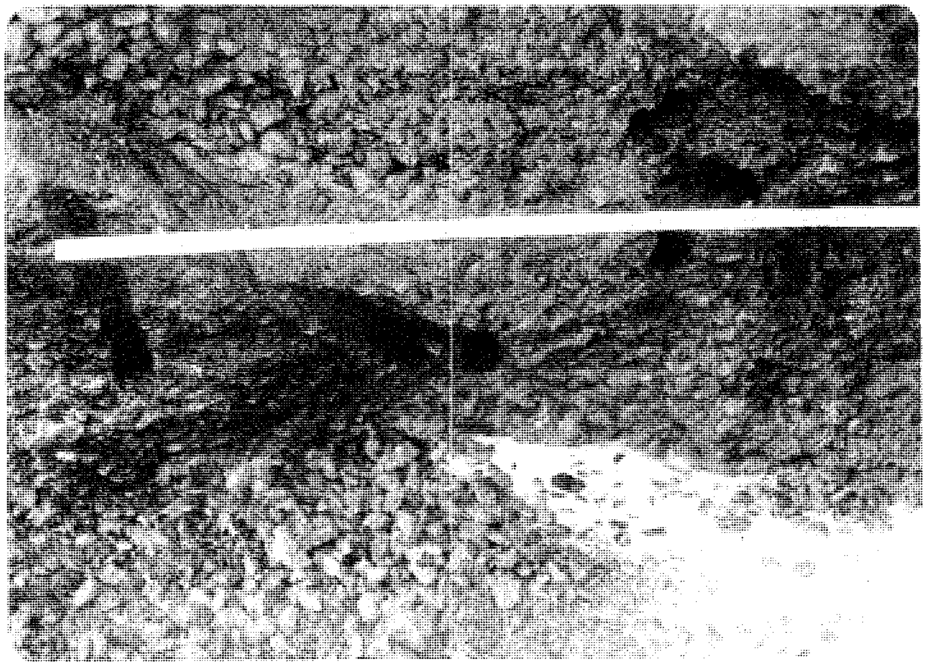


Figure 6.6. Section of the face at Bruceton broken out by a splitter, showing the wide variation in coal and concrete mixes.

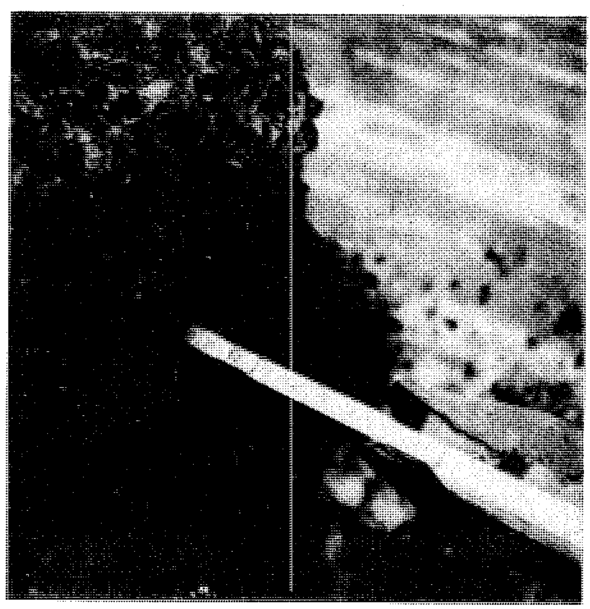


Figure 6.7. Depth of slot cut into the face prior to the initial advance of the machine.

cut than the test blocks shipped to Rolla. Even after repeated passes with different nozzles, the slot created (Fig. 6.7) was not equivalent to that which would have been generated in coal. During the process of these preliminary trials, a plunger seized in one of the high pressure pumps and a return trip was required to Rolla to secure a replacement plunger.

After the pump was repaired, the cutting head was started and advanced forward. However, the width of the slot generated by the water jets was insufficient to allow the head to feed into the coal. This experience was directly contrary to our findings where the water jet was initially tested in coal at the surface coal mine.

The head was backed off from the coal and a pair of diverging jets inserted into the vertical cutting arms. A single orifice, large diameter jet was maintained on the bottom, since this was cutting close to the plastic interface with the underlying concrete, and a break did appear to be developing. The machine was then advanced forward again, and advanced a distance of approximately 5 ft before it was stopped due to an overload condition on the machine.

In this advance, the material that came out from the face was very large and there was no fragmentation of the artificial coal as had been found where real coal had been mined (Fig. 6.8). The water jets were again not cutting very effectively. Nevertheless, a sufficient amount of cutting was taking place so that the machine could be advanced. It was therefore decided to test the effectiveness of different nozzle combinations in cutting through the coal mixture.

Figure 6.8. Size of the original fragment cut from the artificial coal at Bruceton.

The procedure was to insert the nozzle set, start the equipment, bring it up to pressure, and then advance over a distance of two feet with the machine initially one foot away from the artificial coal face. (This standoff distance was necessary to change the nozzle). The arms were then stopped from oscillating prior to the jet shutdown, and, at the same time as the arms were stopped from oscillating, the machine was also stopped. This allowed measurement of the distance that the jets were cutting ahead of the machine, as well as an analysis of the cutting effectiveness of the water jet nozzles.

Three nozzle sets were used in this program, generating diverging jets at included angles of 12 degrees, 10 degrees, and 8 degrees. In the test program at the surface coal mine, it had been found that an 8 degree diverging included angle was adequate to allow the equipment to advance, and much of the testing was done under those conditions. With the artificial coal at Bruceton, the 8 degree diverging angle was insufficient to cut clearance for the cutting head on the face side of the machine (Fig. 6.9). This caused considerable rubbing between the machine and the artificial coal, and the machine itself had to do some cutting of this material. A 10 degree angle (Fig. 6.10) indicated that the water jet was cutting approximately 6 inches from the nozzle to the back of the cut and that material was being moved between the two jets out to a distance of four inches from the nozzle path, and that there was barely adequate clearance for the cutting head to advance. When a 12 degree nozzle was inserted, clearance on the face side of the machine was achieved (Fig. 6.11), the jets cut to a distance of 8 inches ahead of the machine, with approximately 5 inches of material being removed between the jets. While these trials did allow some evaluation of the best nozzle design for this material, the wide variation in the seam properties precludes holding even this conclusion as absolute, and in either case the results do not relate to those which would be obtained in coal.

A series of tests was also performed at an elevated pressure. An operating pressure of 14,500 psi was achieved by inserting an obstruction in one orifice of a dual orifice nozzle located in the bottom cutting arm. Hence the 150 hp pump powering the bottom head unit was supplying only three 0.045 in. diameter orifices or one 0.045 in. and one 0.064in. diameter at any given time. The top cutting unit had the horizontal cutting arm completely disconnected so that it was possible to achieve 14,500 psi at the pump while still bypassing water from the nozzle supply line. This did not seem to markedly affect the performance, although this pressure is sufficient for use in quarrying granite.

It is also significant to record that the head and equipment sled were repeatedly forced away from the face as the equipment was advanced. The reason for this was because the nozzles were not cutting a slot wide enough for the leading edge of the guard to advance into. Consequently, no clearance was generated and an excessive amount of rubbing occurred on the face side of the arm compartment cover plates. The advance would continue down the face until the forward thrust

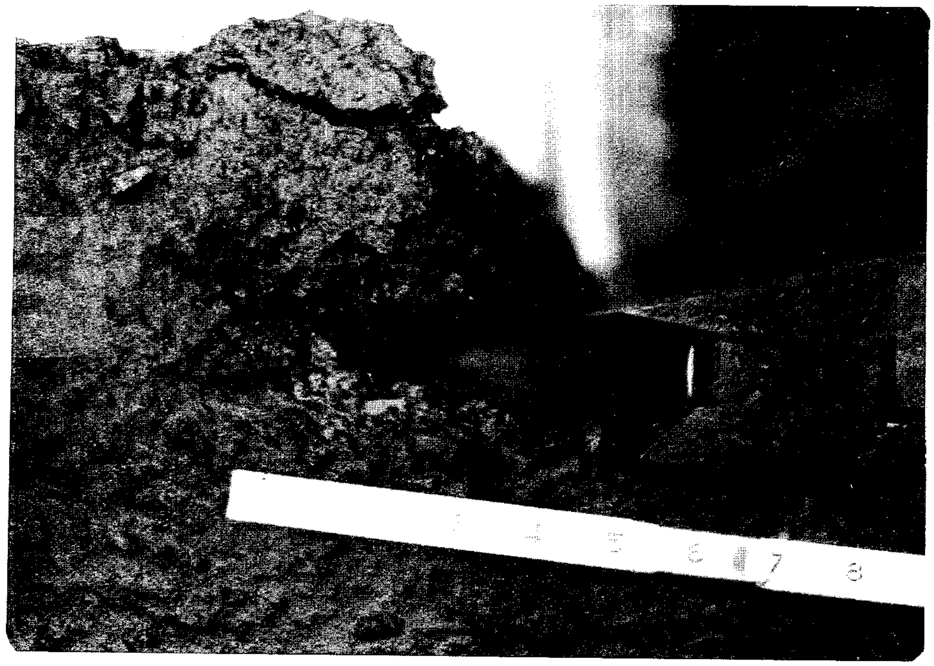


Figure 6.9. Slot configuration cut by an 8 degree nozzle in the test block.

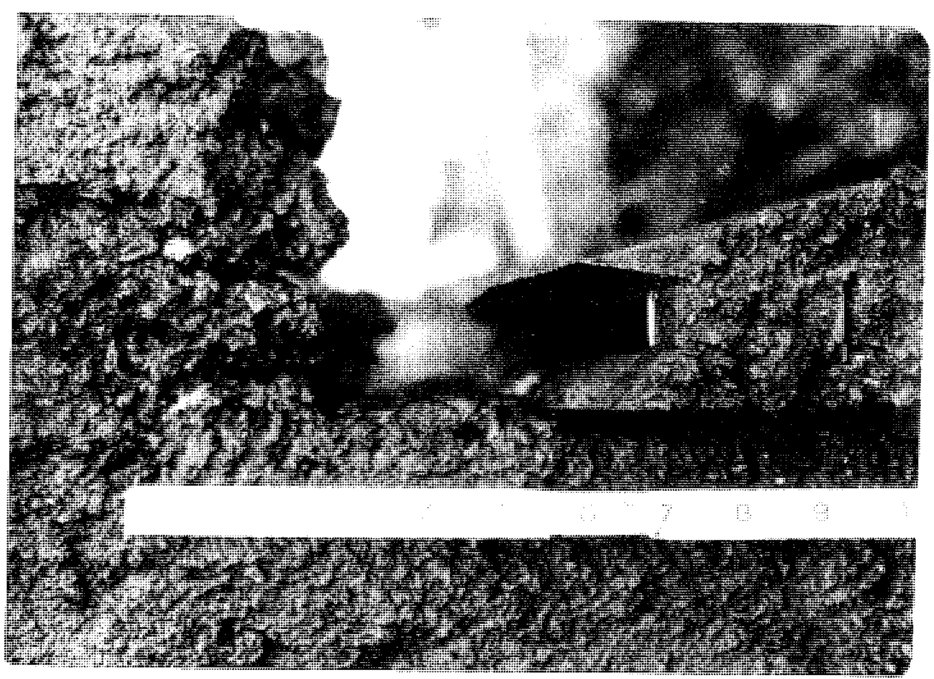


Figure 6.10. Slot configuration cut by a 10 degree nozzle in the test block.

Figure 6.11. Slot configuration cut by a 12 degree nozzle in the test block.

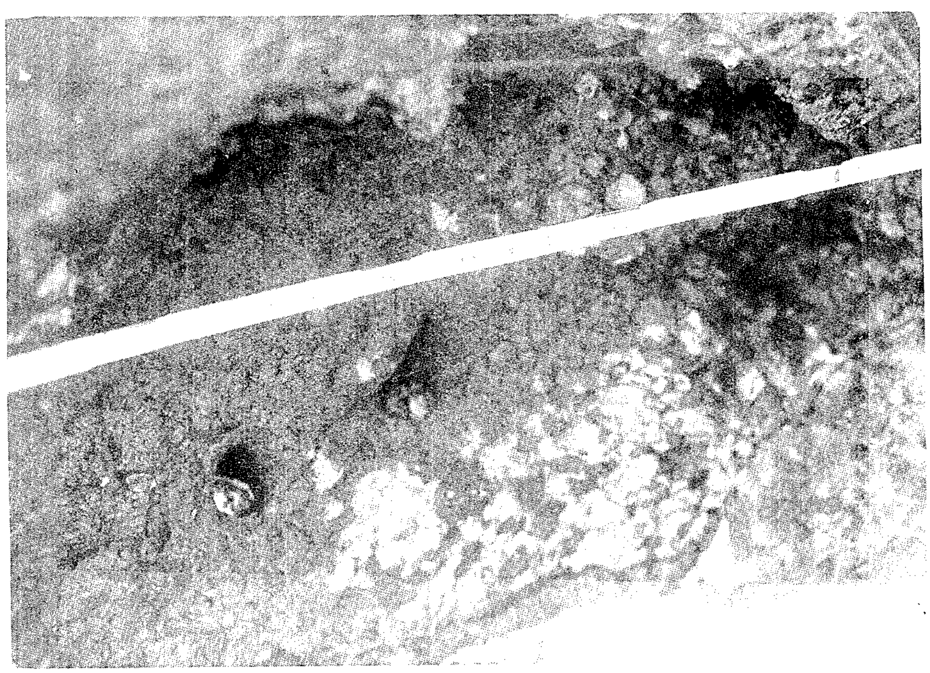


Figure 6.12. Showing the artificial coal at Bruceton, with the distance required between adjacent holes to effectively break out the material (6-12 in.).

capacity of the shearer was exceeded. This could, potentially, have been due to one of three conditions:

1) A condition in which the movement away from the face completely removed the shoe clearance and a metal-to-metal friction drag stopped the advance,
2) a direct bearing force from attempting to advance the head into an area where a slot did not exist reached the stall magnitude, or
3) the force required to break out a portion of artificial coal was greater than the thrust capacity of the Joy machine.

It is likely that the halt was due to either of the first two conditions or a combination.

During the course of one of the advances, the head struck a piece of angle iron which was imbedded in the artificial coal matrix. The lower cutting arm compartment was slightly damaged and a nozzle was broken during the incident. The equipment was withdrawn, the nozzle replaced, and the angle iron was cut out from the face.

The machine then again advanced; however, due to the wide variation in the quality of the artificial coal the jet streams were not cutting consistently along the jet path. Since there was a particularly dense concrete layer ahead of the machine at this time, the Bureau of Mines personnel were requested to remove this, using jackhammer drills and a Darda rock splitter which had been used to break out the angle iron from the face. The holes were drilled at an advance rate of less than 1 ft per minute by the jackhammer through this material, and it was found that the holes had to be spaced within one foot of each other to remove the material, comprising the artificial coal section at this point. This indicated that considerable difficulty would be encountered in breaking out a 9 inch web to the full seam section, as the Hydrominer was designed to do, and that a 2 ft web would not be practical with the force which was then available from the Joy shearer, due to the exceedingly high tensile strength of the material.

This was a result which could not be changed during the course of tests in the particular block available, and since the quality of the artificial coal along the seam section currently being taken varied so widely that no consistently meaningful results could be achieved from the jet cutting system, it was decided to terminate the trials in Bruceton at that time.

It is our opinion that no attempt should be made to draw conclusions about or formulate projections of the performance of Hydrominer II based on the Bruceton testing. The major problem was that the material used for the tests was completely different from coal in its mechanical properties and its response to water jet attack. The extent of the testing was so limited and the level of performance so low, that not a single trial was repeated to ensure that the same result would occur.

Therefore, it would be incorrect to conclude that the trials demonstrated that the machine failed or would have failed if the material had been coal.

Prior to the Bruceton tests it had been agreed, in discussions with both our monitor and the project Staff Engineer from Washington, that the machine would be designed to mine coal, rather than the artificial material developed at Bruceton. While this was a realistic decision, the differences in the material response of this artificial coal over the mine coal response were such that the trials were inconclusive. However, with the advantages of a surface trial of the unit being evident, it was necessary to develop a more realistic artificial material, relative to water jet cutting of coal. This was the subject of the next phase of the investigation.

Chapter Seven

STUDIES ON THE COMPOSITION OF AN ARTIFICIAL COAL FOR TESTING OF A WATER JET SYSTEM

Introduction

Within the past few years there has been a growing awareness of the need for surface testing of mining equipment before it is introduced to the mining industry. While historically major efforts in this area have been carried out in the United Kingdom and Germany, there has been a major construction program undertaken in Bruceton, leading in 1979 to the commissioning of the first U.S. surface test facility. Prior to establishment of the permanent facility, a preliminary artificial coal heading was developed in 1977 and formed a base for initial testing of a Joy shearer and also for the preliminary experiments with Hydrominer II. However, although the artificial material proved sufficient for the testing of the shearer, the properties which it displayed made it an impractical medium for evaluating the performance of the Hydrominer. For this reason a section of this contract was set aside to develop such an artificial mixture.

Considerations in the Composition of an Artificial Coal

In order to evaluate a proper mixture for the simulation of coal response to water jet attack, it is necessary that the means by which the water cut through coal be first evaluated. If, for example, one compares the slot cut across a coal section by high pressure water jet (Fig. 7.1) with the type of cut achieved across the artificial coal at Bruceton (Fig. 7.2) a considerable difference in the response of the

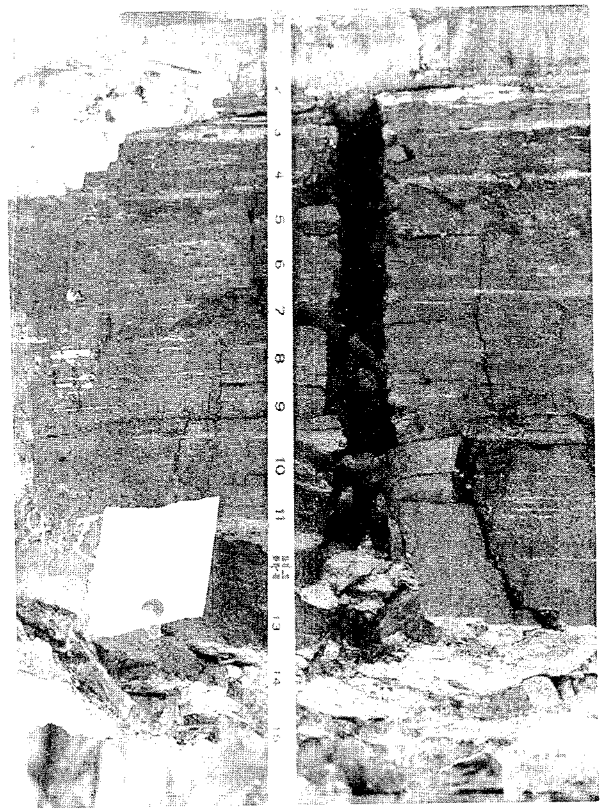


Fig. 7.1. Slot cut in coal by a water jet.

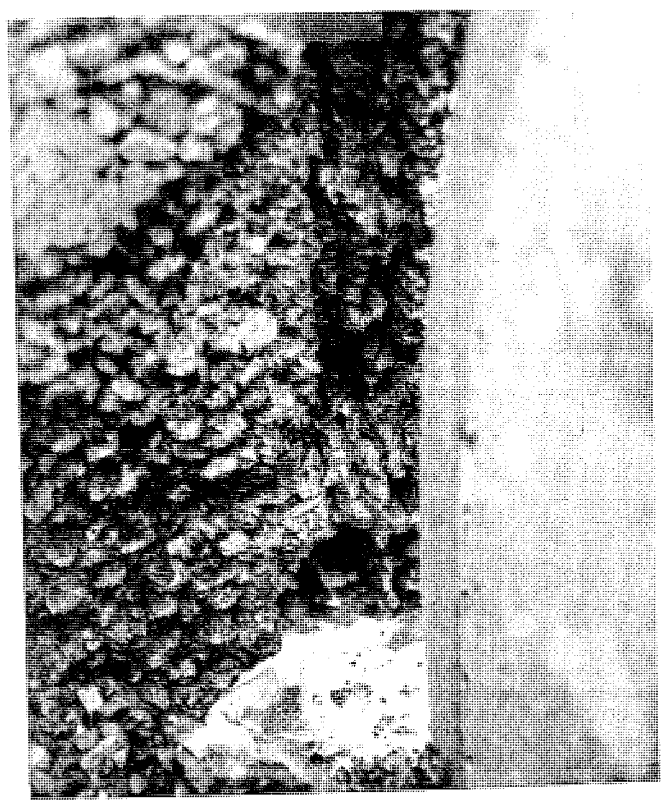


Fig. 7.2. Slot cut in artificial coal block in Bruceton.

material is evident. Coal is a solid mass in the ground interlayered with cleat planes in the vertical and bedding planes in the horizontal which divide the solid into blocks which can often be separated from the mass by hand, and in relatively large pieces. The artificial coal at Bruceton contains a large number of voids and it was not possible in the majority of the material to remove any of the pieces embedded in the matrix by hand.

Early evaluations of the response of material to water jet attack have shown (Ref. 16) that the grain size, the presence of weakness planes, and the tensile strength are critical parameters in the response of that material. These properties are not normally modeled when an artificial coal is developed for a test program at the National Surface Test Facility.

Historically, the methods of mining have involved the use of large rotating drums with picks mounted in them or the application of a single wedge to plane off a thin section of the material. In these cases, the structure of the material used for surface testing is not as critical, and the property most often modeled is the compressive strength of the coal rather than the tensile strength. Unfortunately, it has been shown that the compressive strength of a material is, in and of itself, no criterion for evaluating the response of the material to water jet attack (Ref. 17). Also, because of the cost factors, it is not normally possible to model the weakness planes present within the coal mass.

Two separate constituents exist within the material normally developed as an artificial coal. Historically, the seam has been constructed of pieces of coal mixed together with a matrix made up of fly ash and Portland cement to produce the required seams. This mix is poured in place in layers approximately 18 in. thick to prevent segregation of the coal from the matrix. The water content is of particular concern in such mixes and a considerable degree of skill is required in the practice to develop an accurately reproducible seam section. The development of an artificial coal at Rolla was built around examination of these two components, i.e., the matrix and the coal particles used.

The Matrix

If the water jet is to cut through the artificial material in the same way as it cuts through solid coal seams, it is essential that the matrix itself must respond to jet attack in a manner similar to coal. In addition, the matrix must have an extremely low tensile strength and be of such composition that it can produce a void-free final mass. Based on past experience with jet cutting research, a mixture of sand, water and masonry cement was chosen as a possible matrix composition. The mix is commonly used by masons to bond bricks and blocks together in construction projects. The masonry cement itself is a blended cement made by combining natural or Portland cements with fattening materials such as hydrated lime.

The University of Missouri-Rolla, Rock Mechanics and Explosives Research Center completed in September 1978, work on a grant from the National Aeronautics and Space Administration. The program objective was to develop a means whereby water jets under high pressure could be used to differentiate between coal and the over-or-underlying rock within the stratified sections of a longwall face. The concept considered was that a water jet cuts through coal in a manner much different to the way in which it cuts through rock. The basic differentiation is due to the fact that coal contains many weakness planes and sensibly has no strength in tension across the joint planes. Therefore, when a high pressure jet is rotated eccentrically around an axis so as to describe a cylindrical slot in the test material where that material is coal, then the central core of the cylinder will be removed by the jet action (Fig. 7.3). In rock, on the other hand, which has strength in tension, the central core will not be removed (Fig. 7.4); in fact, in sandstone, it is possible to run adjacent traverses of a high pressure water jet within 1/4 in. of one another without having the material between removed (Fig. 7.5). Thus, this program was successfully concluded when it was shown, inter alia, that an eccentrically rotating water jet would cut up through 8 inches of coal (in under 2 seconds) to a rock interface, remove the core of the coal outlined, but only slot the rock surface, thus identifying the interface (Fig. 7.6). It was necessary to develop a material very weak in tension which could be cut through extremely quickly by a high pressure water jet. In the testing carried out for the NASA contract, for example, the water jet proved capable of cutting through 9 in. of coal in less than 5 sec.

The criteria established for the matrix of the artificial coal was that the water jet be capable of a rapid penetration through the material and that up to a 2 inch diameter

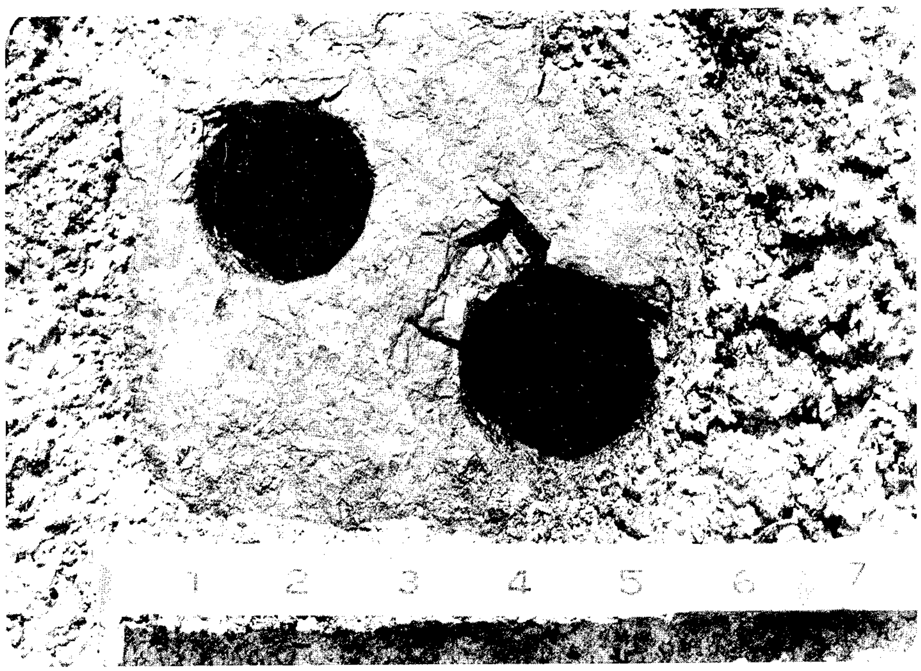


Fig. 7.3. Slot cut in coal showing removal of the central core.

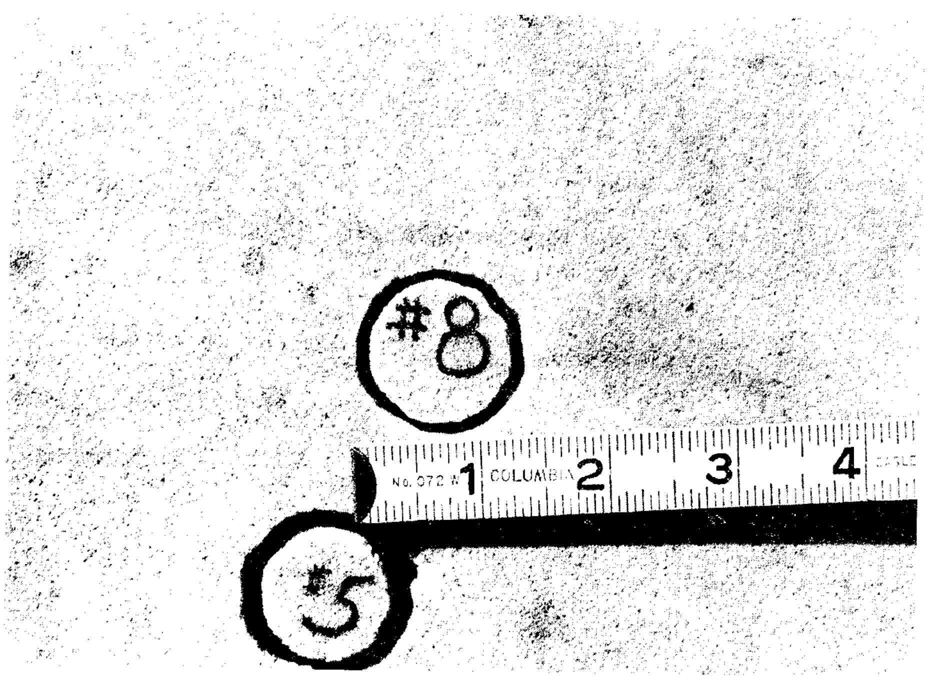


Fig. 7.4. Slots cut into sandstone showing the residual central core.

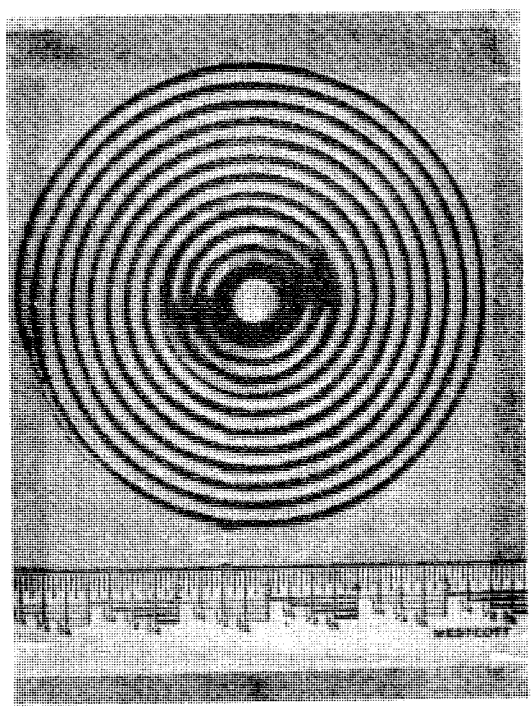


Fig. 7.5. Circular cuts in Berea sandstone showing that ribs less than 0.1 inch wide are not removed.

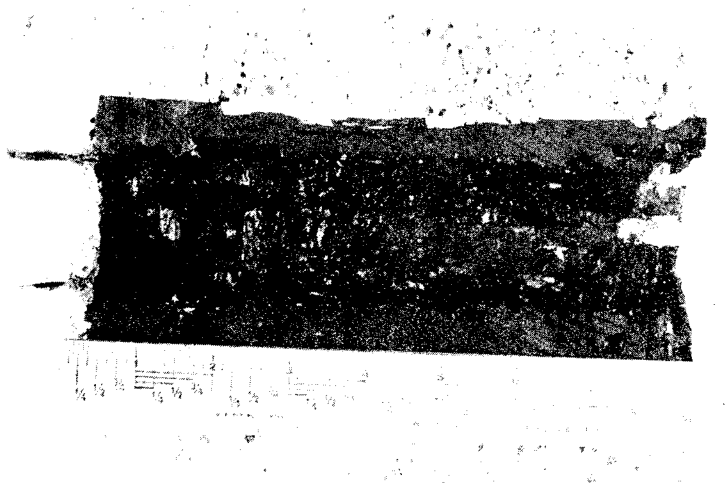


Fig. 7.6. Successful cut through 8 inches of coal to sandstone showing the differentiation in cutting.

the central core would be removed by the water jet action when a slot was cut around the cylinder of the material.

Initially, samples of the artificial coal matrix 6-1/2 in. deep were cast in concentrations by volume of sand and masonry cement in ratios of 5:1, 7:1, 9:1, 11:1, and 13:1, in each case, with 2 parts water to determine the correct proportions of such a mix. These samples were mixed and dried for two weeks prior to test under water jet attack. Upon testing it was found that these samples were, if anything, too small but that the approach was in the right direction insofar as the water jet penetrated through the average 6-1/2 in. of the sample in approximately half a second or less (Table 7.1).

At the same time, various different sizes of sand were tested in a 5 to 1 mixture, looking at screening sizes in the range from #6 mesh (.131 inches) to #28 (.0232 inches). However again, the speed with which the water jet cut through the samples indicated that there was very little effect that could be ascertained due to this variation in size. It was therefore decided to prepare mixtures in larger blocks containing relatively large pieces of coal which had been obtained from the test site and to determine how easily the material cut relative to coal fragments located in the matrix.

<u>Testing of the Coal:Matrix Combination</u>

The results of the jet cutting tests of the matrix alone indicated that a mixture in the proportion by volume of 5:1 (sand to masonry cement) was reasonable. Therefore, the samples cast using both lump coal and matrix used matrix

Table 7.1. Test Results for Artificial Matrix

Composition (by volume) sand:cement:water	Jet Pressure (psi)	Rotation Speed (rpm)	Depth of Cut (in.)	Time
5:1:2	8,000	810	5.75	< 2 sec(1)
7:1:2	10,000	810	6.75	< 1 sec
9:1:2	10,000	810	6.00	< .5 sec
11:1:2	10,000	810	6.75	< .3 sec
13:1:2	10,000	810	6.875	< .3 sec

Stand-off distance 2", offset eccentricity 1".

(1)Test result disregarded due to early opening of shutter.

proportions of 5:1, 4:1, and 3:1. Two procedures were followed in casting the blocks (16 in. cubes). In the first case, the lumps of coal were placed in the form by hand to preserve the normal orientation of horizontal and vertical axes. Then a relatively wet mix of matrix was poured around the lumps making sure that all of the voids were filled with the matrix. After a layer approximately 6 in. thick was poured, a sheet of 6 mm thick polyethylene was inserted before the next layer of coal was placed. This procedure was continued until a form was completely filled. The objective in inserting the polyethylene was to prevent the layers from bonding together in an attempt to simulate a weakness plane. In the second case, the matrix and lumps of coal were mixed together with the proper amount of water in a small cement mixer. This material was then poured directly into the form making no effort to properly orient the coal, but being careful to eliminate voids. A typical result of the mixing process is shown in Figure 7.7. The size of the sample in the photograph is approximately 5 in. by 6 in. and the curved black line is a layer of polyethylene.

Two sets of cutting tests were performed on both types of samples. Slot cutting tests were performed using the same mechanism to move the nozzle as was used in Hydrominer II. These cutting tests were conducted at the same oscillation rate as that of the traverse tests in the coal seam experiment. In addition, the hole drilling experiments were repeated to ensure that these mixes were responding correctly to the jet attack.

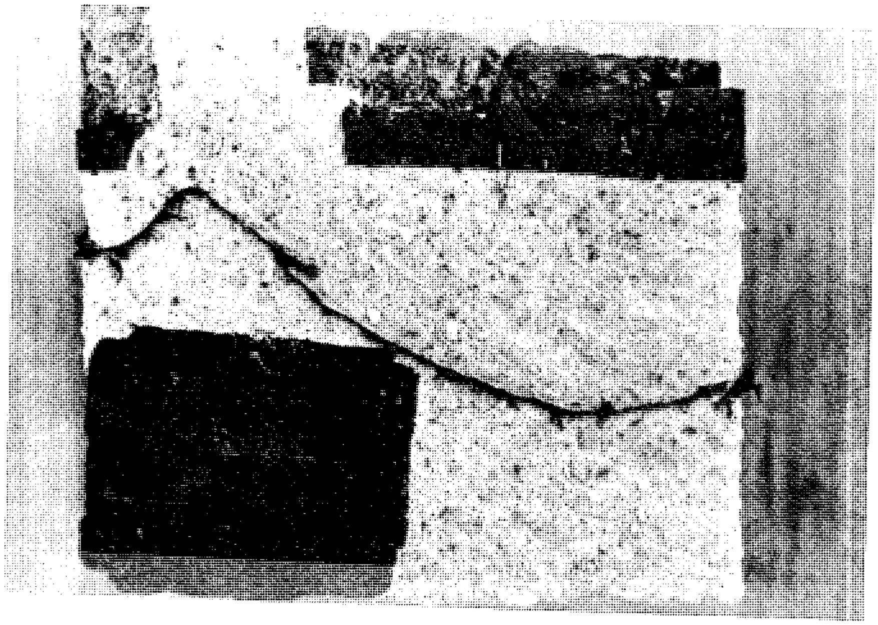


Fig. 7.7. Lump coal held in place by a sand-masonary cement matrix.

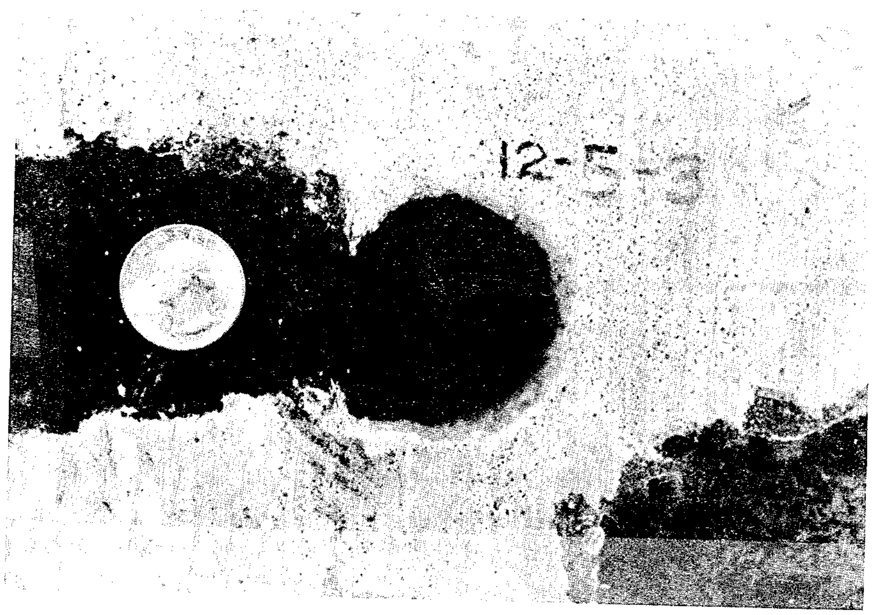


Fig. 7.8. Hole drilled by a water jet in lump coal and matrix showing equal cutting response.

The results of the cutting tests were extremely encouraging. In most instances the matrix and the coal were cut equally whether the jet action was to produce a slot or drill a hole into the material. For example, a hole drilled in a sample#12-5-3 with the 5:1 matrix and coal in random orientation was positioned such that part of the hole would be created in coal and the remainder in the 5:1 matrix (Fig. 7.8). The diameter of the hole created remained essentially constant and the depth of the hole was nearly uniform whether the final material being cut was coal or matrix. A sample with a 5:1 matrix was cut by the traversing test using a 4 degree diverging nozzle and 5 passes. The same sample was drilled with an eccentric jet. This sample was then cut in half to expose the interior details (Fig. 7.9). The slot cut is on the left in Figure 7.9 and clearly shows a constant slot width when the jet has cut from matrix through coal into matrix. The hole drilled is shown in section in the right side of Figure 7.9. The jet began cutting in matrix, cut through coal into matrix, sectioned a small lump of coal, again cut through matrix, and finally ended in coal. The diameter of the hole was nearly uniform and the matrix was completely removed from the hole as was the coal. Typical slots cut by an 8 degree diverging nozzle are shown in Figure 7.10. The right slot was the result of one pass of the nozzle while the left slot was from 5 passes. This material is a 3:1 matrix.

Another favorable result of the traverse cutting tests on these mixtures was that the artificial weakness planes

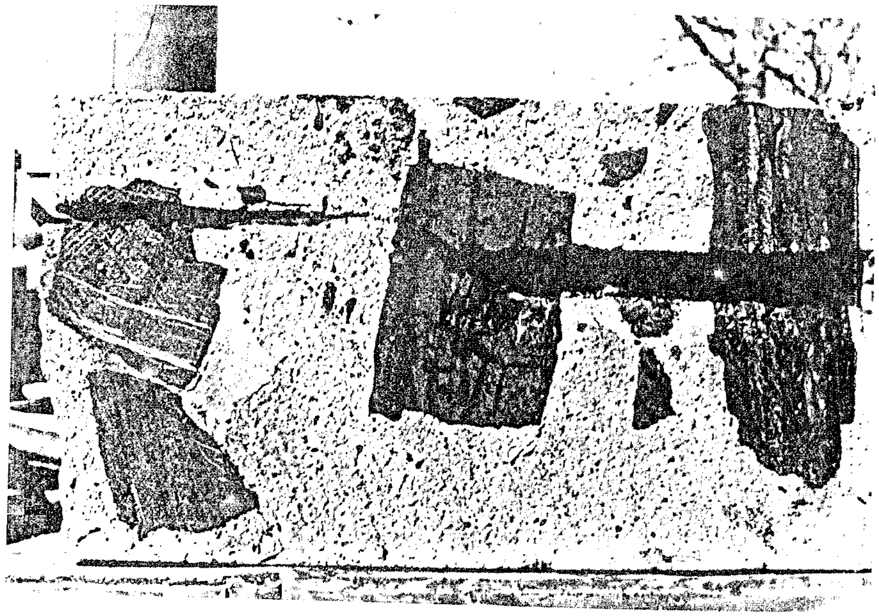


Fig. 7.9. Section through a slot on the left and a hole on the right, both cut by a water jet in the artificial mix.

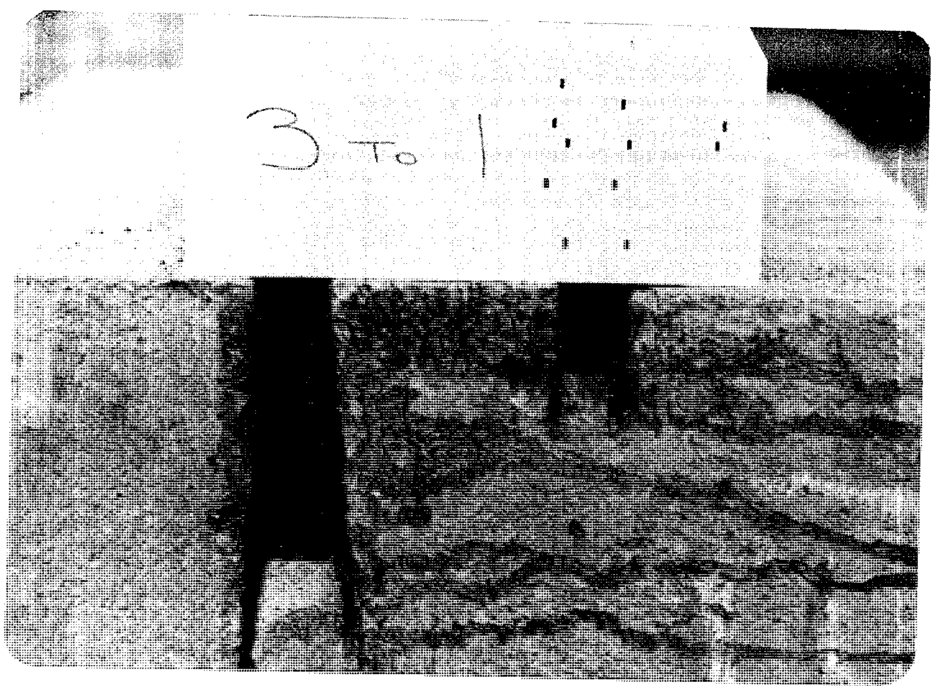


Fig. 7.10. Slot cut by 1 and 5 passes of a dual orifice 8° diverging nozzle.

generated by the polyethylene functioned as planned. The jets were able to cut through the weakness planes without a distortional effect and the jets were able to exploit the weakness planes in fracturing the sample (Fig. 7.11).

To conclude the study, samples were poured using the matrix of 5:1 and inserting increasing proportions of 1 in. stoker coal into the mix to determine the difficulty of filling the voids as the proportion of coal was increased. These samples were mixed in the cement mixer and combined as 5:1:1, 5:1:2, 5:1:3, through 5:1:6 with the components being sand:masonry cement:stoker coal (Fig. 7.12). These mixes would be more nearly like the mix at Bruceton than the samples tested earlier.

The response of these mixes to jet attack was substantially different from the mixes which contain the lump coal and matrix. Using small coal greatly increased the resistance of the material to water jet cutting. It was not possible to attain the proper depth of cut in the traverse tests and frequently the material between the cutting zone of the two jets could not be removed (Fig. 7.13 through 18). Since the results of the traverse tests were radically different from what would be acceptable as an artificial coal mix, no drilling tests were performed.

Fig. 7.11. Jets cutting in artificial coal have fractured sample along a weakness plane.

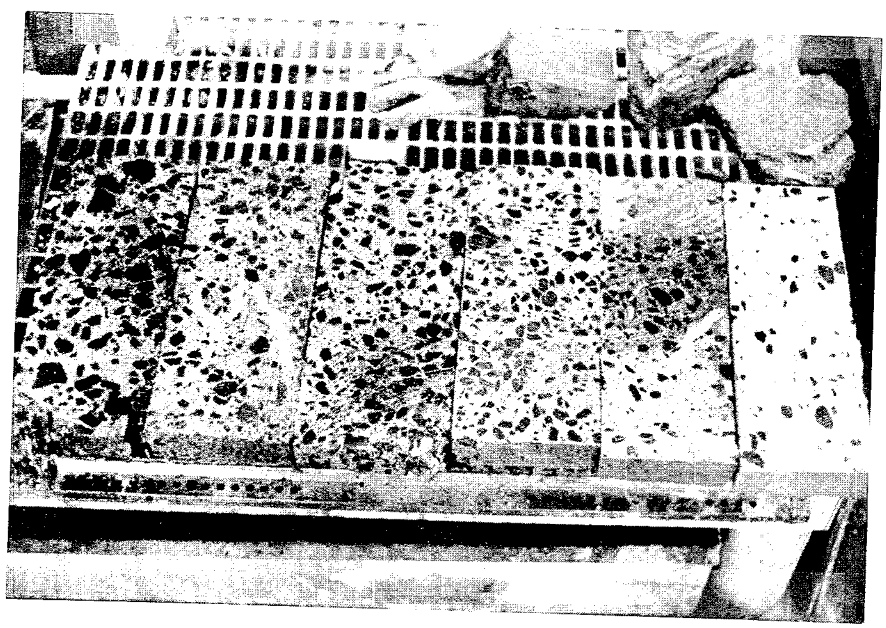


Fig. 7.12. Typical samples showing increasing proportions of stoker coal from right to left.

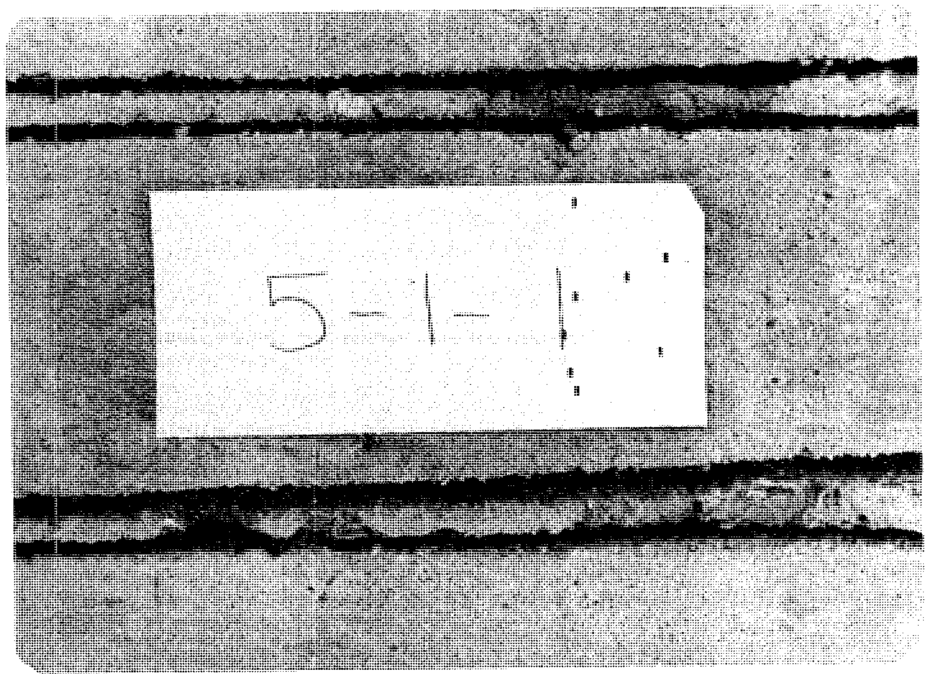


Fig. 7.13. Typical slots cut in stoker coal-matrix mixture showing failure of the water jets to remove the material between the jets.

Fig. 7.14. Typical slots cut in stoker coal-matrix mixture.

Fig. 7.15. In this sample the presence of excessive voids is beginning to reduce the effectiveness of the jet cutting.

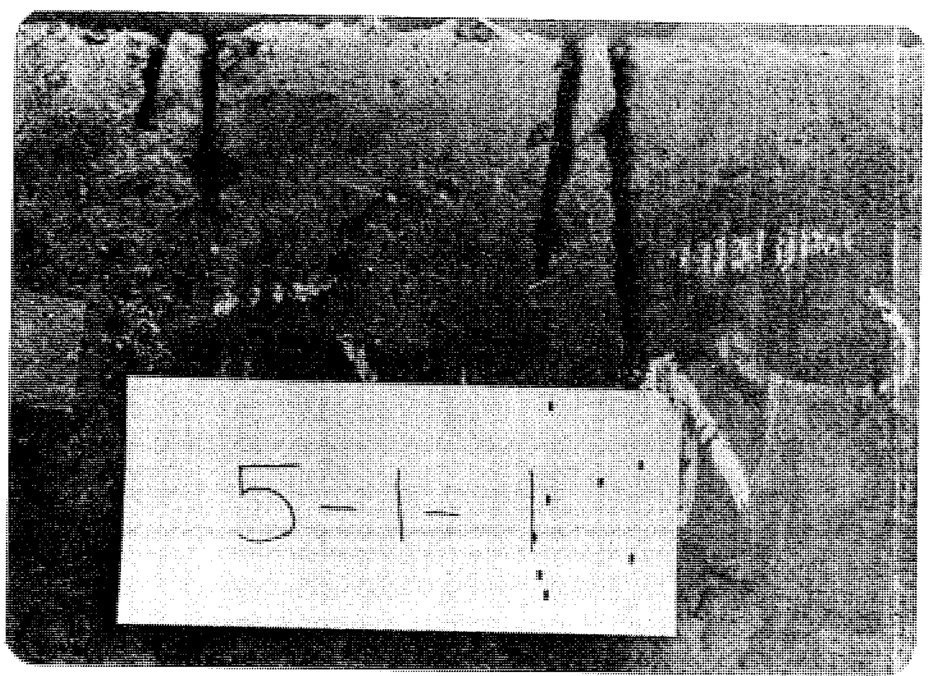


Fig. 7.16. End view of slots cut by 1 and 5 passes of a dual orifice nozzle across the stoker coal-matrix mixture.

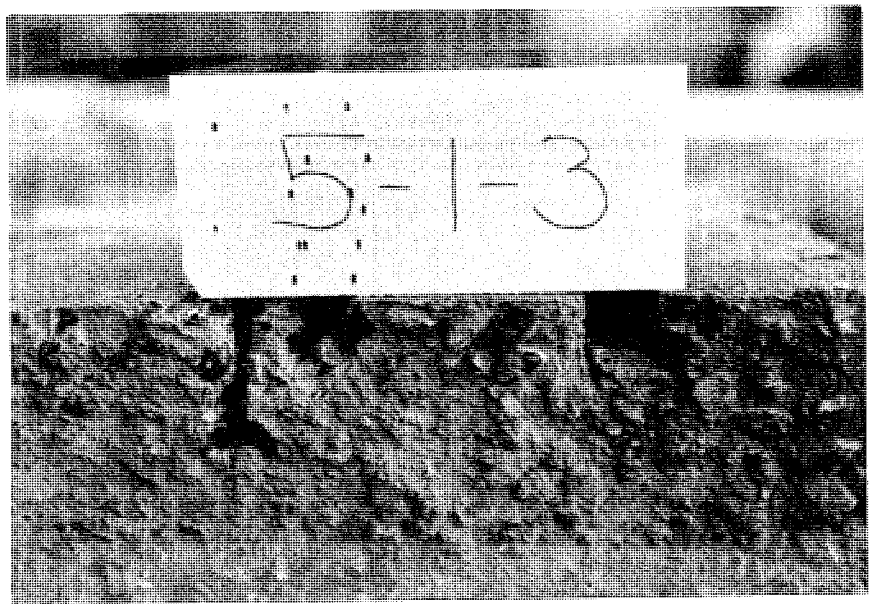


Fig. 7.17. End view of slot showing poor depth of penetration.

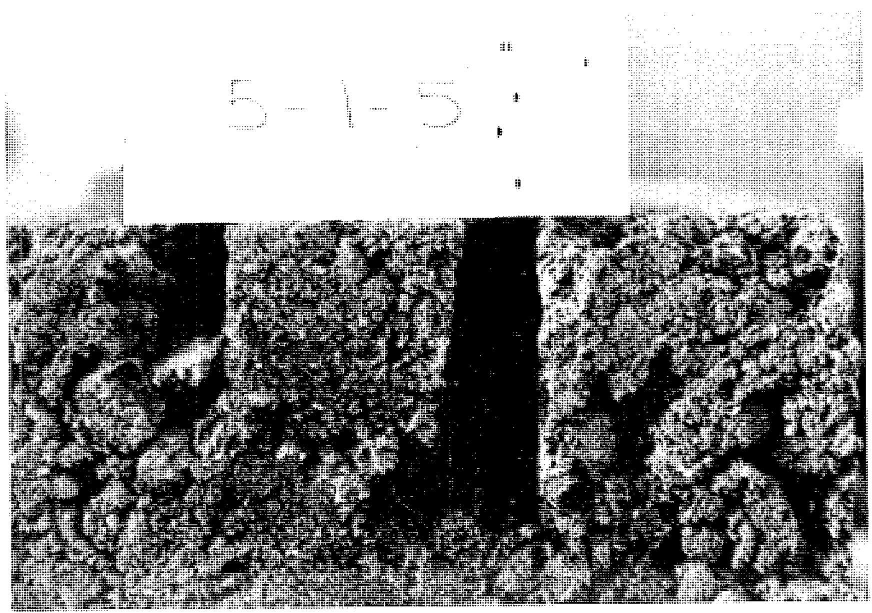


Fig. 7.18. End view of slot.

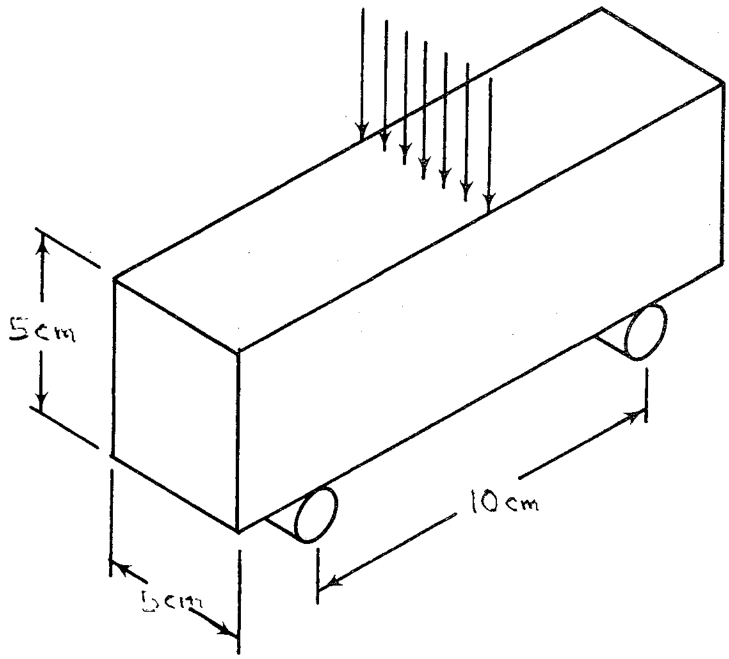


Fig. 7.19. Three point bending test.

Strength Testing

In addition to the artificial mixture responding correctly to jet attack, it is essential that the tensile strength be very low since coal has almost no strength in tension. Consequent to the cutting tests, samples of the various mixtures along with some coal samples were selected for determination of indirect tensile strength in three point bending (Fig. 7.19). The results are summarized graphically as shown in the figures (Fig. 7.20, 7.21).

The average value of tensile stress at failure for the sand and masonary cement ratios of 3:1, 5:1, 7:1, and 9:1 were 360, 119, 122, and 100 psi respectively. The coal samples failed at an average value of 91 psi. However, a great deal

Fig. 7.20. Strength in Tension of Matrix Material of Sand and Masonary Cement.

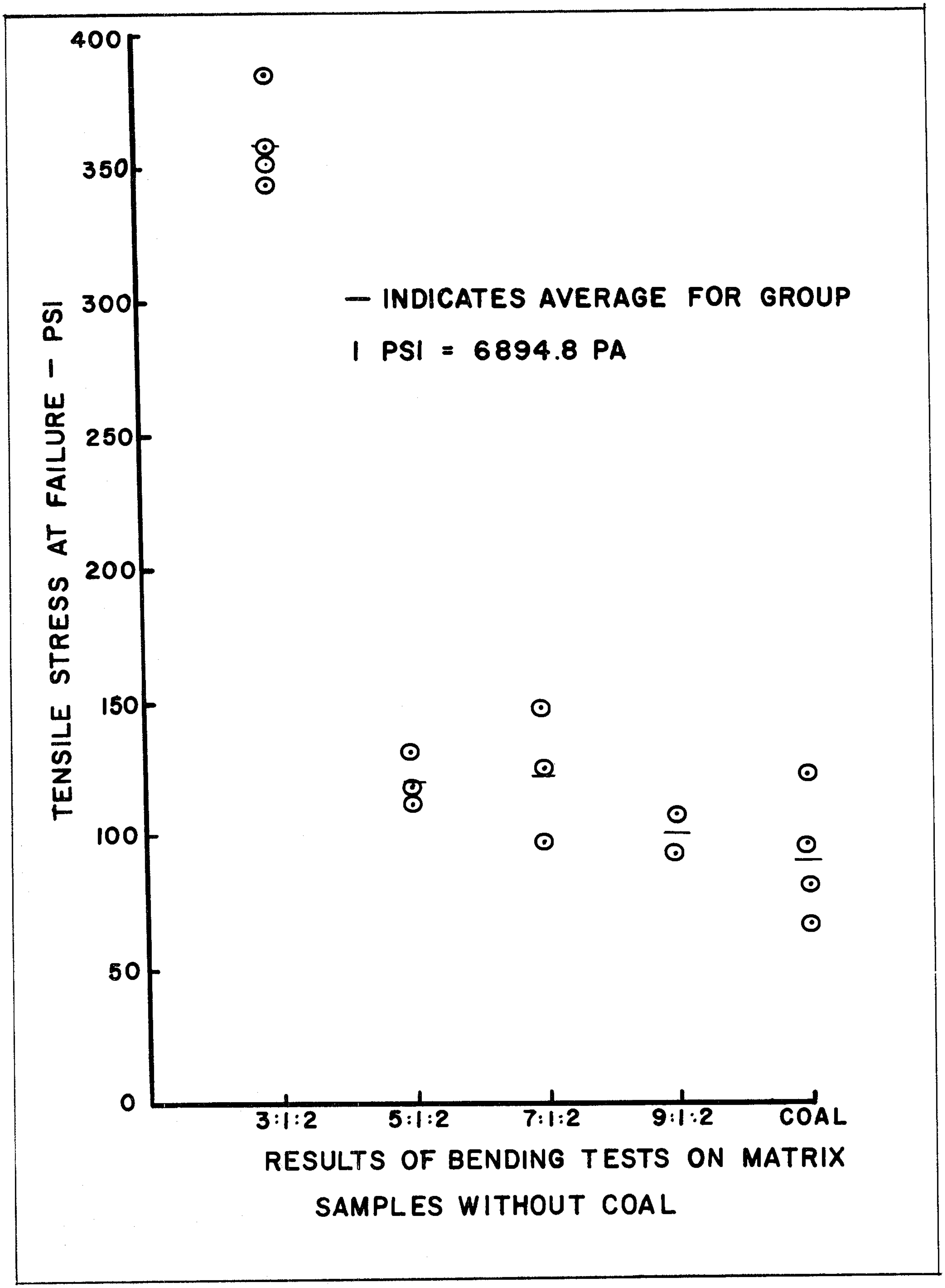
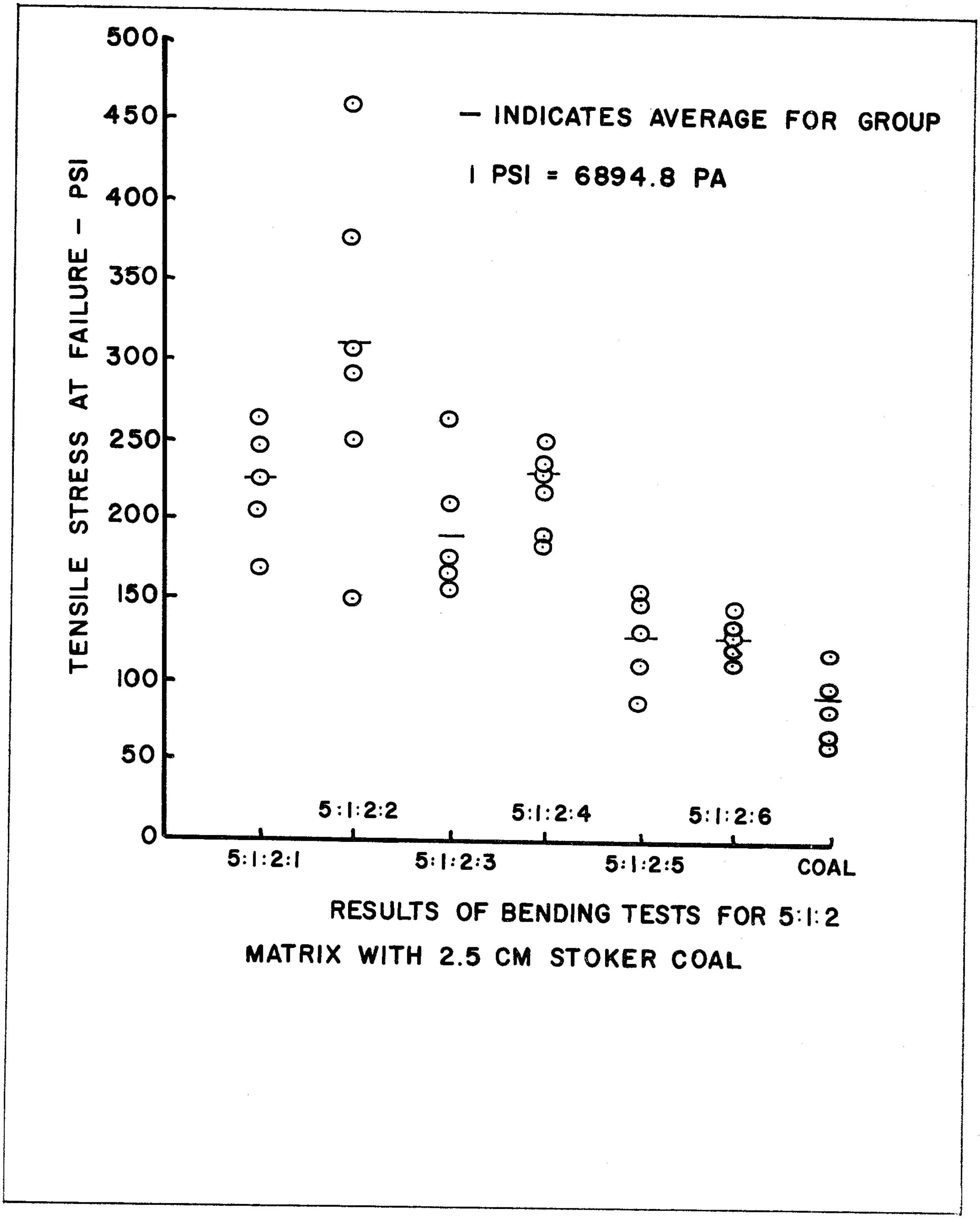


Fig. 7.21. Strength in Tension of Samples Using Sand, Masonary Cement, and Stoker Coal.



of care had to be exercised in selecting and preparing the coal samples to provide a specimen which would not fail at a much lower value of stress. In other words, the coal samples tested must be regarded as the upper limit strength values since the majority of the coal would fail at much lower tensile stress values.

Conclusions Regarding Artificial Coal

The results of these studies indicate a very promising approach to the simulation of coal for studies involving the penetration of the artificial mixture by high pressure jet cutting. By using lump coal and a matrix of sand and masonary cement to bond the coal together and fill all the voids between the lumps, it is possible to provide an artificial material which responds the same as coal does to jet attack. The weakness planes of coal can also be simulated by layering the mix with a plastic sheet between layers to inhibit bonding. However this procedure would require more labor to construct a large block of coal and would require a longer period of time to allow the layers to solidify before additional material was poured.

Additional testing should be conducted on larger blocks to ensure that the strength in tension is low enough to simulate coal. The results of the bending tests indicate that the artificial mixture tends to be too strong in tension in comparison to coal. Another factor which should be studied is the change in the property of the mixture with prolonged curing. Most of the samples tested in this study were cured for two to three weeks before being cut or broken. If this material follows the pattern of concrete, the maximum strength would not have been reached in this period.

Chapter Eight

CONCLUSIONS AND RECOMMENDATIONS

Introduction

Insofar as the program carried out had three separate sections, it is perhaps germane to divide our conclusions as they were drawn from each part. The three cases are the field trials of the experimental cutting head, the experiments which were carried out using the second generation cutting head, and the trials which are being carried out using the artificial coal samples and mixtures developed at the University of Missouri-Rolla.

1. Testing of the Experimental Cutting Head

The initial objective of the tests carried out at the surface mine site was to verify the concept upon which the Hydrominer had been designed, namely, that high pressure water jets could isolate a cantilever of coal which would require very little force to break out this cantilever and load it over onto the conveyor. The target for verification of this concept was that the machine should be capable of moving at 5 ft/min, taking a 2 ft web in 32 in. high coal.

In the field, the machine was able to exceed all three specifications, moving at speeds up to 10 ft/min, being capable of taking a 39 in. web at 5 ft/min, and operating in coal sections up to 54 in. The experimental rig was unable, however, to complete the full pattern of the test program designated. At a web depth of 1.2 meters, a 200 kw shearer is only capable of achieving an advance rate of 3 meters per minute (Ref. 18). It can thus be seen in a 1.2 meter section that the Hydrominer achieved approximately 50 percent of this rate at a lower horsepower, and while restricted, since no conveyor was available to remove the coal from the face.

The results indicated that the water jets, when cutting on an advancing machine, were capable of outlining a cantilever at least 2 ft long ahead of the machine in contrast to the designed distance of some 3 in., thus making it easier for the cantilever to be fractured. Results showed that the effect of the width of this cantilever on the force required to fracture it was relatively insignificant over the range from 18 in. to 39 in. wide, indicating the potential of this machine for taking a wider web than existing machines at a greater advance rate.

Examination of the films taken during the testing of the Hydrominer indicated that in the range where the water jet was cutting out to its full coherent length, that the jet is capable of cutting to a depth of at least 1 in. per stroke. Since the arm was cutting at 120 strokes per minute this gave a potential advance rate of 20 ft/min. The potential of the machine, therefore, to equal the performance of existing mining machines, was demonstrated in this field test program.

Certain other advantages to the Hydrominer became apparent after the tests progressed. As had been surmised, no dust had been generated by the mining operation, and by suitable choice of nozzle design the size of the product can be controlled. In all cases the coal fragment size was larger than that conventionally achieved with a shearer, where typically up to 80 percent of the coal particles are less than 1 in. in dimension, whereas with the Hydrominer, the converse is the case, 80 percent approximately lying at above 1 in. in dimension. This makes the coal easier to clean and information from elsewhere (Ref. 19) indicates that preparation plants could be operated at a higher efficiency and lower costs.

The tests showed also that if the Hydrominer is suitably designed, it is capable of cutting through thin pyrite lenses (up to 2 in. thick) with no diminution in the performance of the Hydrominer and, since the cutting takes place away from the cutting head itself, with no appreciable wear on the machine, this indicates a considerable potential saving over the operation of a shearer where pick costs can rise to an appreciable level due to the presence of abrasives within the coal seam. For example, in 1975, one mine in Germany (Ref. 20) spent approximately a half million dollars on pick costs, alone.

At the end of this contract, the principal investigators had the opportunity to discuss the performance of a longwall hydraulic mining machine developed in Poland which operated at less than 1,000 psi. It required much higher water flow rates to achieve the same result as the Hydrominer, and was unable to cut through the lenses of pyrite and abrasive material present in the coal face (a problem which the Hydrominer has demonstrably solved). Nevertheless, this machine was found capable of achieving advance rates of up to 24 ft/min (8 meters per minute) in the hard Silesian coals, indicating that there should be no appreciable problem in achieving the advance rates projected for this current generation of machine.

During the course of the trials it was also noted that the machine operates, once the head becomes buried within the coal, at noise levels at or below 85 db, the particular noise level being a function of the amount of shielding surrounding the pump and motor, and not coming from the cutting head itself. This noise level is well within the regulations for noise generated by the Mining Safety and Health Administration for exposure of mine operatives. Because the cutting area is completely flooded with water, the machine also will be safer to use than conventional mining equipment, insofar as there is no risk of a face ignition of methane in the vicinity of this machine.

It was also noted that the haulage forces required to pull the unit down the face were reduced from those normally reported for advancing a shearer through coal on a longwall face.

2. Conclusions from the Testing of the Second Generation Cutting Head

The development of the second generation head resulted in a machine of relatively few moving parts and in comparison with the components of a shearer, much simpler in operation and much more easily capable of access and repair in case of any equipment failure within the machine.

The trials at Bruceton, unfortunately, did not achieve their objective in any manner, although there is an indication that as material gets harder it becomes necessary to widen the angle of divergence of the water jets in order to achieve clearance for the leading edge of the machine as it advances into the face. The tendency of the artificial coal to break out in large lumps because infusion was impractical made it difficult to develop a suitable analysis of the behavior of a water jet on a true coal face, and did not allow the experiments to be properly evaluated as to the relative benefits of having the lower or upper head of the machine leading. Nevertheless, the machine did demonstrate its robustness and it was able to withstand the full haulage force of the shearer. However, the material of which the artificial coal face at Bruceton is composed is not suitable for testing high pressure water jet equipment due to its relatively high tensile strength and grain structure.

3. Testing the Artificial Coal at Rolla

It would appear that the most advantageous mixture for testing of artificial coal under water jet attack comprises 5 parts sand, 1 part masonry cement, 2 parts water, 2 parts coal, with the coal to be in as large a fragment size as possible, since below a certain size no cleat or sensibly weak bedding planes are likely to occur. This mixture is an abrasive one, and therefore would not be most suitable for testing of shearers where the abrasion of the sand on the pick would cause a very rapid deterioration in the performance of the machine, but this material, nevertheless, has the characteristics of coal in regard to strength and does much better model the cutting characteristics of coal than does the material at Bruceton. It was found important that the solid have no voids in it, since the presence of voids negates the ability of the water jets to pressurize the target material and using larger scale failure, a necessary component to a suitable performance of the mining machine.

Recommendations

At the present time, although the conceptual idea of the Hydrominer has been successfully proven within the initial parameters set down at the start of the contract, there still remains a considerable question as to the true performance of the machine. Unfortunately, it is likely that the machine can only be adequately tested where a conveyor is available to remove the cuttings as they are mined by the machine. The use of an artificial coal face is therefore very likely to be the site for any first stage of any subsequent work, since the time required to establish a face conveyor and suitably anchor it, and the amount of money that would thus be incurred would make it impractical for a site to be set up in any active surface mining operation.

Experiments at Rolla with small blocks have shown that to a first approximation an artificial material can be constructed to simulate the response of coal under high pressure water jet attack. It is therefore recommended that as a first stage in any subsequent contract that a small block of this material be set up in Rolla and that the Hydrominer cutting head be pulled through this material to ensure that the conclusions of the initial laboratory study hold true. Should this be the case, then it is proposed that a section of the artificial longwall panel developed at Bruceton be poured to the mixture developed in Rolla, and that this pour be to a height of 5 ft. This will then allow the series of tests to be carried out, projected in the earlier phases of the current program to verify the capabilities of the Hydrominer. Among the tests which should be carried out are evaluations of the optimum head configuration (lower or upper head leading) and the possible need to rake the head.

It must, however, be borne in mind that the projected tests at Bruceton, in themselves, do not adequately model the likely productivity of the hydraulic mining machine, insofar as the coal on an underground face has already been preweakened by front abutment pressure and is under some degree of stress, it can therefore be expected to fail much more easily in many instances than the artificial coal. This, for example, can be illustrated by examining the coal at the Mid-Continent Coal Company operation in Colorado, where the coal is extremely easy to cut and would be very easily loaded by the Hydrominer without the great problems of dust and methane generation currently encountered at that face (Ref. 21).

There is, however, a need for a surface trial of the second generation of the machine, to satisfy such criteria as the loading capability of the machine and the ability to differentially size the coal particles that fall on the conveyor. This is important to establishing the logistical operation of the machine on a longwall face.

It is not practical to simulate this behavior on the artificial coal seam insofar as the artificial coal, albeit it can model the cutting capability of real coal, cannot economically model the weakness planes within the coal structure, such as bedding and cleat, because of the problems of generating such faces accurately. Thus, therefore, while the artificial coal trials will indicate how fast the machine can advance, it will not, in and of itself, allow an estimate to be made of the true characteristics of the mining machine. It is therefore proposed that in a third phase of any subsequent contract that the machine can be taken to a surface coal mine and that the machine be tested in this operation to facilitate this program. It is, however, recommended that instead of operating the Hydrominer from a support sled as was the practice in the first series of tests, that the cutting head would be mounted on the front end of a Caterpillar tractor front end loader, and that this will power the cutting head down the face very rapidly, giving a true assessment of the failure characteristics of the coal under water jet attack and the capability of the machine to load the coal from the face. This will allow a true estimate of the cutting speeds which the machine is capable of undertaking.

At the conclusion of this test program, it should be possible to fully assess the potential benefits likely from any further development of the hydraulic mining machine Hydrominer. Therefore, at that stage a decision should be made as to the future commercial possibilities of the machine.

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APPENDIX 1

CUTTING HEAD FABRICATION AND CONSTRUCTION MODIFICATIONS

The fabrication of the cutting head is described in the following discussion of the assembly procedure. The fabrication progressed smoothly and major structural modifications were limited to two changes. The first was stiffening the base plate (see Step 4 below) and the second was redesigning the drive train for the spray arms. This discussion is presented later. All of the minor structural changes are presented in the assembly procedure in the order in which they were performed.

Assembly Procedure

Step 1. Holes were drilled in cover plates B47, B48, B49, B50, and B51* and matching holes were drilled and tapped in bars B9b, B10b, B11b and beams B16, B18, B20, and B20a. Instead of using 1/2 - 13 UNC cap screws, it was necessary to change to 7/16 - 11 UNC Allen head flat headed machine screws. This change eliminated the bolt heads protruding from the faces of the cutting head. The cover plates were spot faced at each hole location with an 82 deg countersinking tool. Note that beams B16 and B18 were changed to 4' x 4' H-beams since the original steel bars were not available. This change resulted in a 50 percent reduction in weight of the members with the same structural strength.

Step 2. In order to maintain hole alignment and straightness of the members, Bars B9b, B10b, and B11b were bolted to plate B51 and then plates B9a, B10a, and B11a were tacked with an electric welder to the bars. These assemblies were then removed from plate B51 to finish welding a and b parts together.

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* Numbers refer to drawings in "Excavation of Coal Using High Pressure Water Jet System," USBM Report H0232064.

Step 3. Assemblies B9a, B9b, B10a, B10b, B11a, and B11b were then bolted back onto plate B51 and positioned on base plate B5. The bottom edges of plates B9a, B10a, and B11a were then tack welded to the base plate B5. It was again necessary to remove plate B51 to complete the welding.

Step 4. At this point in the fabrication it was obvious that the base plate B5 was too thin (1/4 in. thick). It did not provide a solid flat surface to work from and would warp when other members were joined to it. Rather than delay the fabrication until a heavier plate could be obtained, temporary stiffening steps were taken at this time. These measures involved welding 3-in. channel irons onto the top surface of the base plate to stiffen it. Later a 11/16-in. thick plate was added to the original base plate to provide a permanent solution.

Step 5. Next 1-1/4" x 1-1/4" angle irons were substituted for plate B6 and B7 and welded to the base plate B5. This substitution was also made to increase the stiffness of the base plate. Plate B8 was positioned on top of the angle irons and welded in place. It was necessary to add a plate 3" x 8-3/8" x 3/8" to plate B8 in order for it to extend to the back of plate B5 as shown in the assembly drawings.

Step 6. Large angle irons B12 were placed on the shop floor in a parallel location and squared by making the diagonal distances equal. The channels B14 and B15 were welded to them with the spacing between B14 and B15 changed to match the supplied length of channels B21. The assembly was then turned over so that B21 channels could be welded in place. Note that spaces were added inside B12 at the ends of B14 and B15 in order for the assembly to match the print.

Step 7. After the welding in Step 6 was complete, the assembly consisting of B12, B14, B15, and B21 was located on the base plate B5. It was necessary to notch plate B11a and bar B11b in order to correctly position this assembly. Rear plate B47 with four bars, B20 and B20a (two of each), bolted to it was then positioned relative to the first assembly. After proper alignment was achieved, clamps were used to hold the position until the welding was completed. These welds joined angles B12 to the base plate, bars B20 and B20a to angle B12 and plate B8, and joined angle B12 to plate B11a. As many welds as possible were completed with plate B47 bolted to bars B20 and B20a.

Step 8. Plate B19 was then welded to four bars, B20 and B20a, and also welded to plate B8.

Step 9. The next step was to fit the H-beams to the cutting head structure. These beams were the replacements for bars B16 and B18. The procedure used was to make the H-beams and cut them to fit between angle B12 and plate B8. Then before welding in place, the cover plates B48, B49, and B50 were bolted in place to ensure proper alignment of the holes. An extra plate 1/4" x 5" x 60" was added directly behind plate B8 to increase the surface to which the H-beams could be welded at the front end of the cutting head. Step 9 was by far the most difficult and required a great deal of care to execute correctly.

Step 10. The last step was to reduce the height of plate B48 by 1-1/8 in. to keep it from protruding above the cutting head and to add an angle iron to the bottom of plate B50 to cover a crack between plates B50 and B51.

This completed the basic assembly procedure required to construct the cutting head. Due to a complete redesign of the linkage system used to drive the spary arms, the following parts were not required:

B6,* B7,* B13, B16,* B17,* B18,* B25, B26, B27, B28, B29, B30, B31, B32, B33, B34, B35, B36, B37, B38, B39, B43, B44, B45, B46, B52, B53.

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* Parts B6, B7, B16, B17, and B18 were replaced by other material, and the others were not used at all.

DESIGN MODIFICATIONS OF THE JET ARM DRIVE TRAIN

Linkage Design

The original design proposed using five hydraulic cylinders to move the five jet arms through prescribed arcs. Early in the stages of fabrication this concept was reconsidered because of the complex electrical system required to modulate the cylinder stroke. The expense of the components along with the doubtful reliability prompted a search for a linkage drive mechanism that could be powered by hydraulic motors. The design criteria selected stated that the phase relationship between the top horizontal and bottom horizontal arms was not important. However, the middle vertical arm must be 180 deg out of phase with the top and bottom vertical arms. This condition will cause the top vertical arm to meet the middle vertical arm at one point in their travel. Also, the middle vertical arm will meet the bottom vertical arm at a different point in their travel.

When two vertical arms meet, their water jets both impinge on a given spot and a reinforced cutting action results. In an attempt to meet this criteria, the mechanism shown in Figure A.1.1 was designed and a working model constructed on the laboratory test stand. As shown in the figure, the position of the top and bottom arms is determined through the connecting links by the position of the middle arm. This device worked well at slow operating speeds, but as the speed was increased the link flexibility became significant. The link deflections were so large that the spray arms would bang together. Attempts were made to stiffen the links, but the added mass aggrevated the problem and hence this approach was abandoned.

Testing of the device shown in Figure A.1.1 demonstrated that the drive shown in Figure A.1.2 would work in the required speed range for the middle arm if the other two arms were disconnected. The input link 2 and connecting

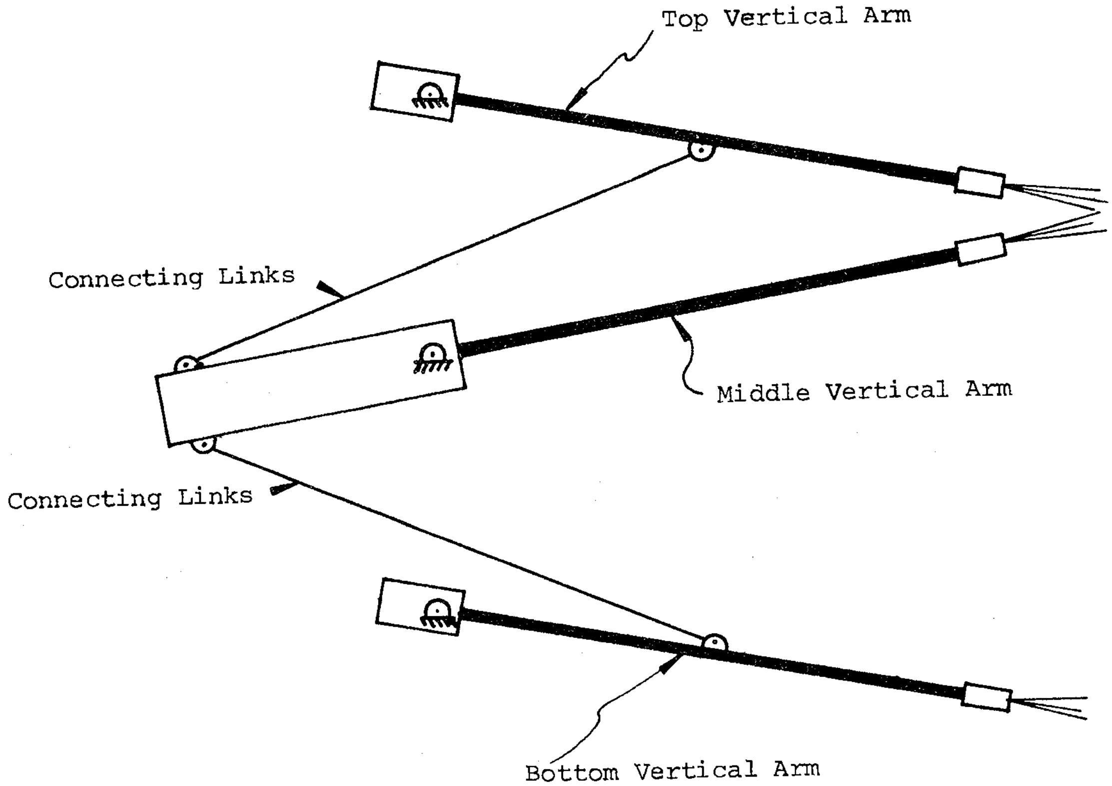


Figure A.1.1. The initial proposed linkage system.

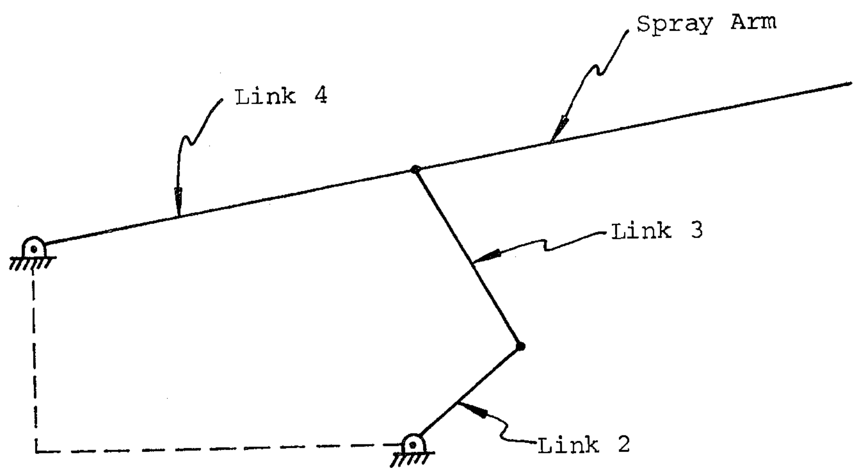


Figure A.1.2. The modified linkage system.

link 3 are proportional so that the arm oscillates through an angle of $\pm$ 14.5 deg. Since this drive worked well on one arm alone, it should work for the other two arms if a means of timing the motion of the links 2 could be provided. This was accomplished by using a roller chain and sprockets so that the driving link 2 of the top vertical arm mechanism is exactly 180 deg out of phase with the driving link of the middle vertical arm mechanism.

Space did not permit driving the bottom vertical arm in this fashion. Hence, a parallelogram linkage was formed between the top and bottom vertical arms by a connecting link. The final configuration is shown in Figure A.1.3. The sprocket on the middle arm drive mechanism is driven by a hydraulic motor also using a roller chain and sprockets. The hydraulic motor drive train has a 2 to 1 reduction so that for an input link speed range of 0 to 200 rpm the motor speed range is 0 to 400 rpm.

Similar four-bar linkages were used to drive the top and bottom horizontal arms. These linkages were also powered using hydraulic motors and roller chains with 2 to 1 speed reduction in the drive train.

Hydraulic Motor System

The power required to drive the arms is very low and nearly constant. Gear-within-gear motors were selected because they deliver smooth output torque and have high starting torques. The oil reservoir for the hydraulic system has a 40 gallon capacity. Oil is drawn from the reservoir by the 166 pm pump and the oil normally returns directly to the reservoir. However, the three-way control valve can divert part of the flow into the motor circuit. Coarse speed control is provided by the position of the three-way valve while fine adjustments are made with the flow control valve. The relief valve serves as a protective device in case the spary arms become locked by an obstruction in their travel. All three hydraulic motors run at

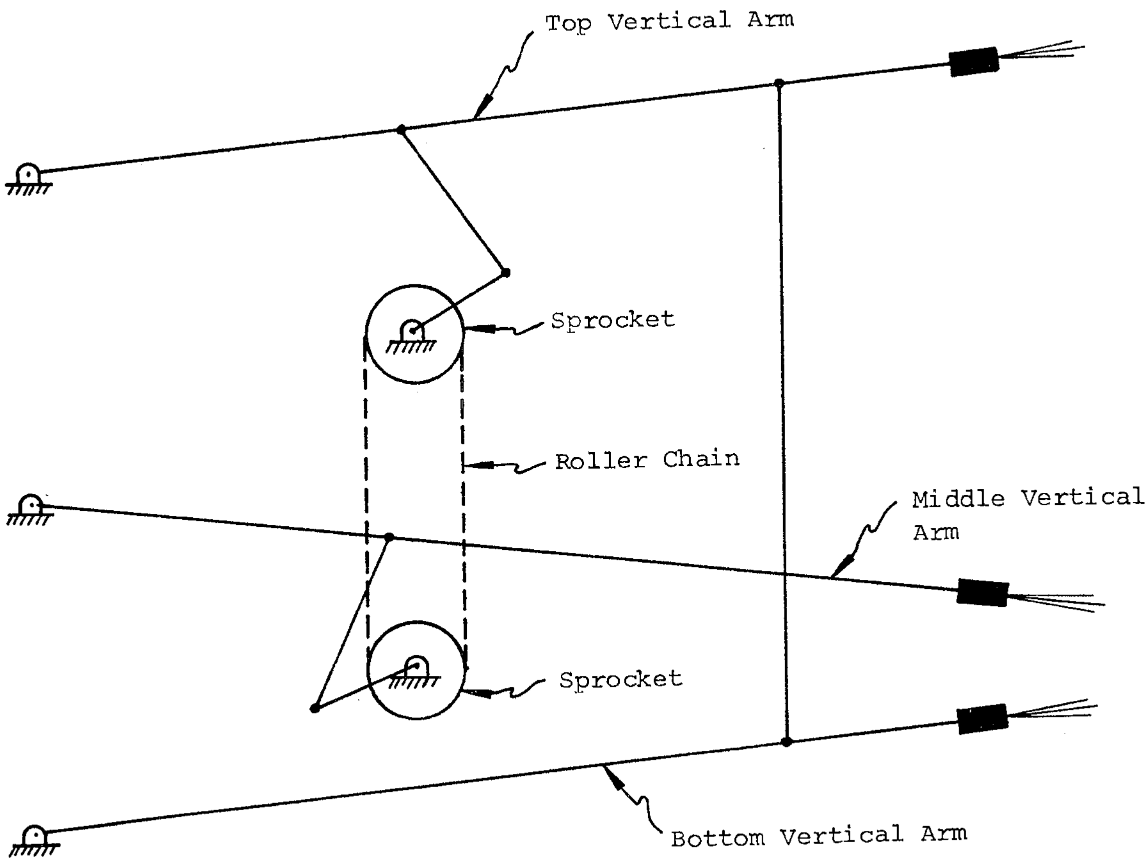


Figure A.1.3. The linkage system used in the cutter head.

the same speed since they are connected in series. Figure A.1.4 shows the complete hydraulic system for the motor drive train.

The modification of the cutting arm drive system removed the electric motor and pump unit, together with the control valve unit, from the cutting head. In the final machine design it is proposed that the hydraulic pump for the drive system be located between the cutting heads; the exact location to be selected in discussions with the engineering services subcontractor.

Three small Char-Lynn No. 107-1006-003 hydraulic motors were used in the final assembly to power the cutting arms. This simplified the drive to the three systems since a separate motor was used to drive the jets in each plane. The feed from the Air Hydraulics S30S17 AJ22R supply pump, driven by a Lincoln 10 hp electric motor, was fed in series through the motors using a high pressure hose with quick-connect couplings.

The horizontal drive mechanisms have been modified to the extent that instead of the drive coming from a linear piston and cylinder, the drive is taken from the hydraulic motor through a toothed drive to a sprocket chain, which in turn drives a flywheel. The eccentric connection from this flywheel to the cutting arm provides the oscillatory motion (Figure A.1.4).

The details of the change in the vertical drive mechanism is shown illustrated in the assembled drawing A.1.5, with the detailed component drawings given in A.1.6 through A.1.10.

The assembly is mounted upon an additional cover plate A.1 in Figure A.1.6 which provides a protection for the drive mechanism from the rebounding water and ejecta from the cutting area. The drive system itself is attached to the carrier plate B.1, attached to the cover plate by 12 spacer bars (Figure A.1.6.a).

Two flywheels are composited to give a single unit made from the two halves E (Figure A.1.7) and F (Figure A.1.8). The drive from the flywheel

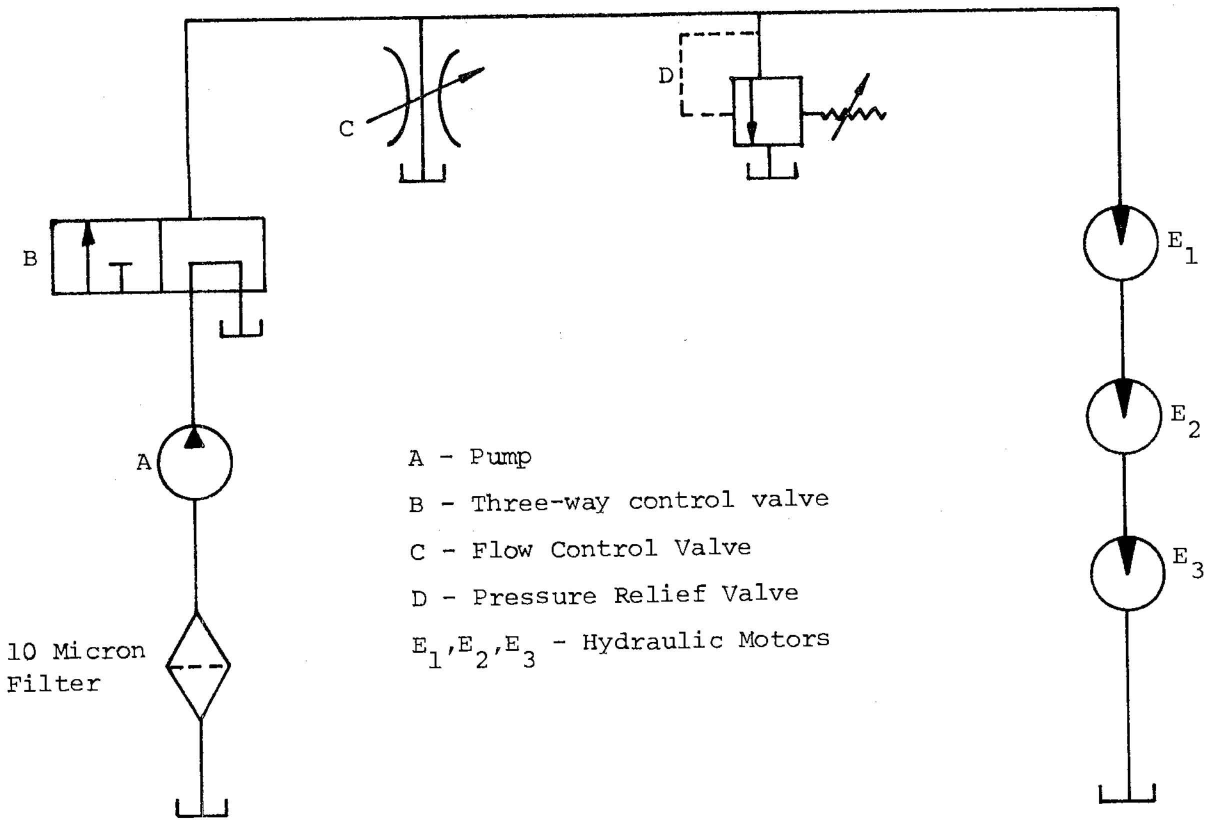


Figure A.1.4. Hydraulic system for the motor drive train.

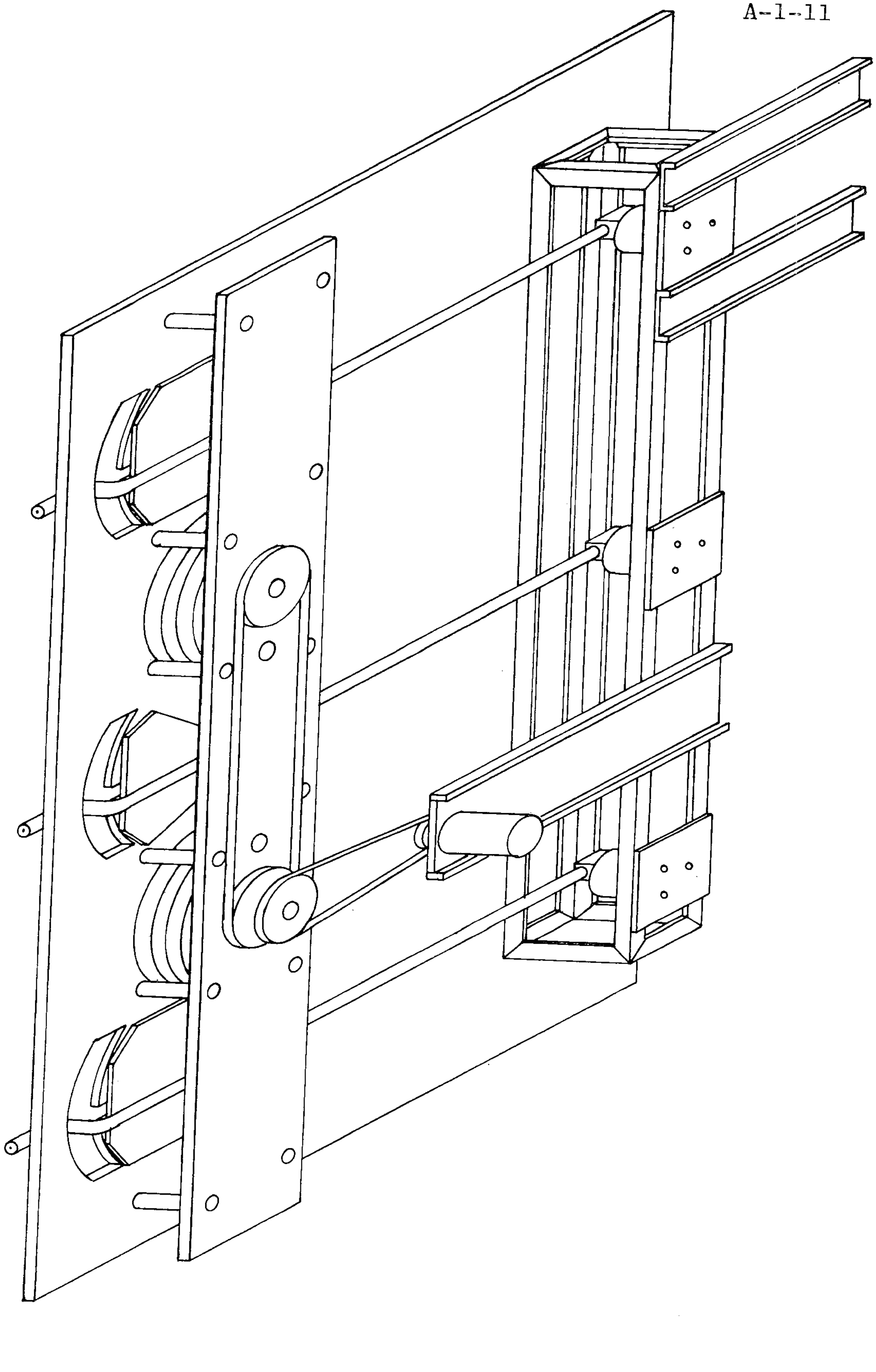


Figure A.1.5. The details of the vertical drive mechanisms.

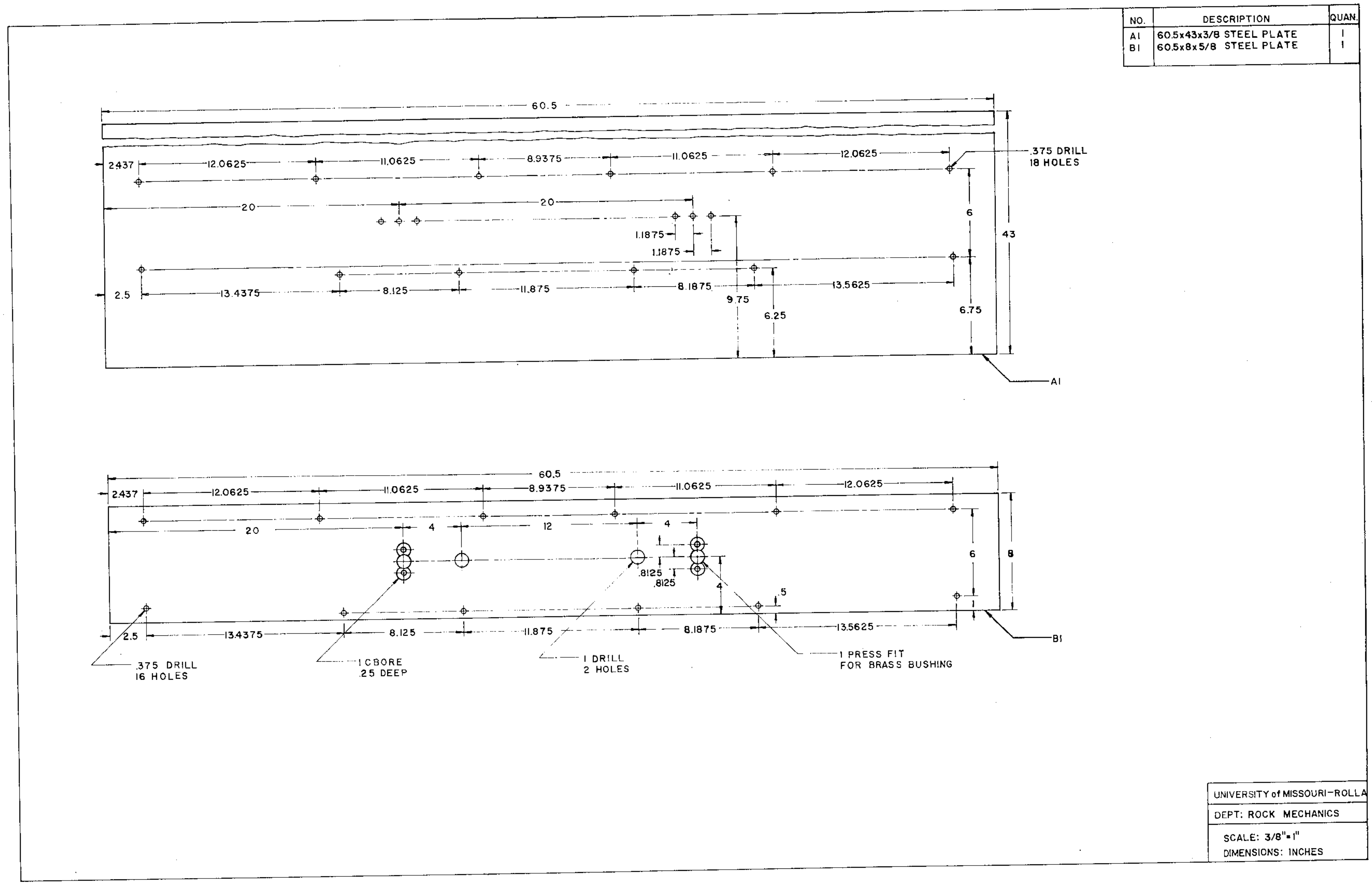
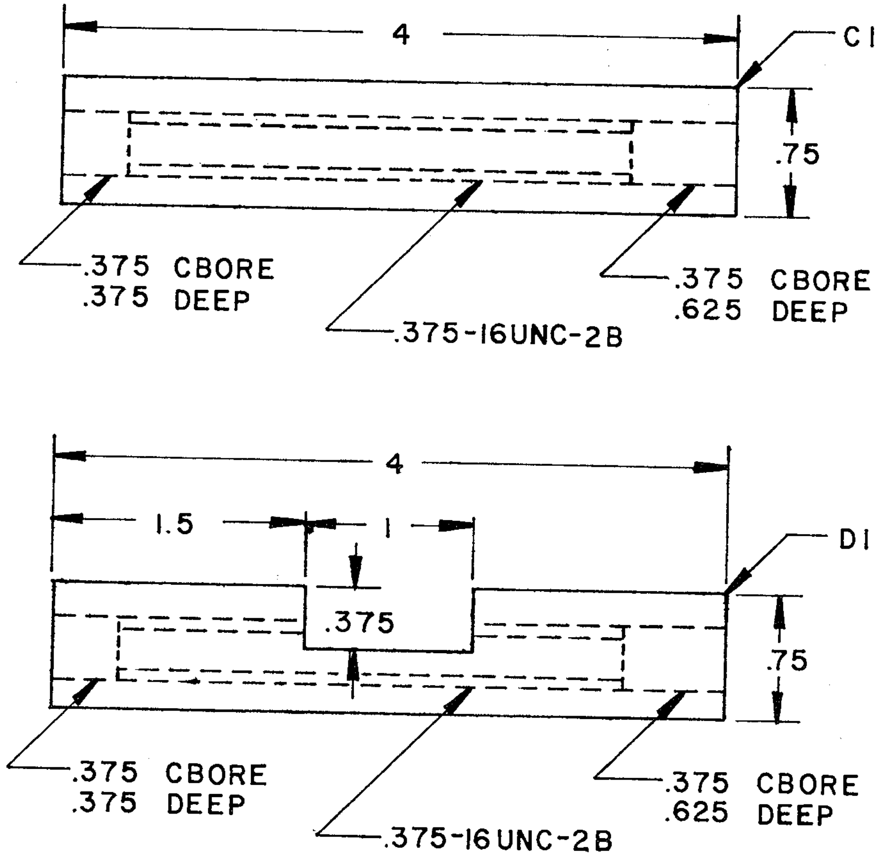


Figure A.1.6. Support plates for the vertical drive.

NO.	DESCRIPTION	QUANT.
C1	4x3/8 DIA. STEEL ROD	10
D1	4x3/8 DIA. STEEL ROD	2



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DEPT: ROCK MECHANICS
Scale: 1"=1"
Dimensions: inches

Figure A.1.6.a. Spacer bars detail.

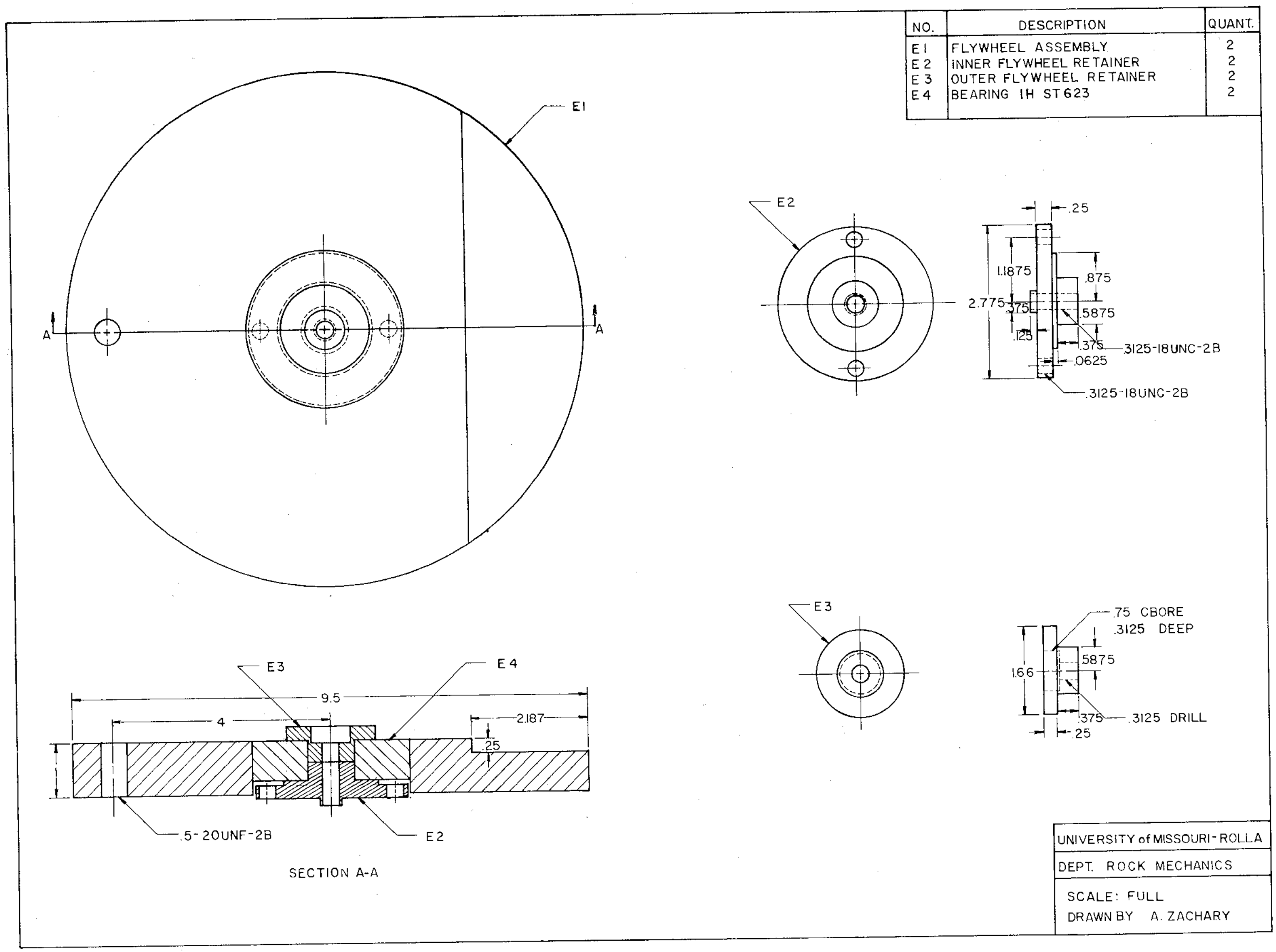


Figure A.1.7. Upper flywheel assembly.

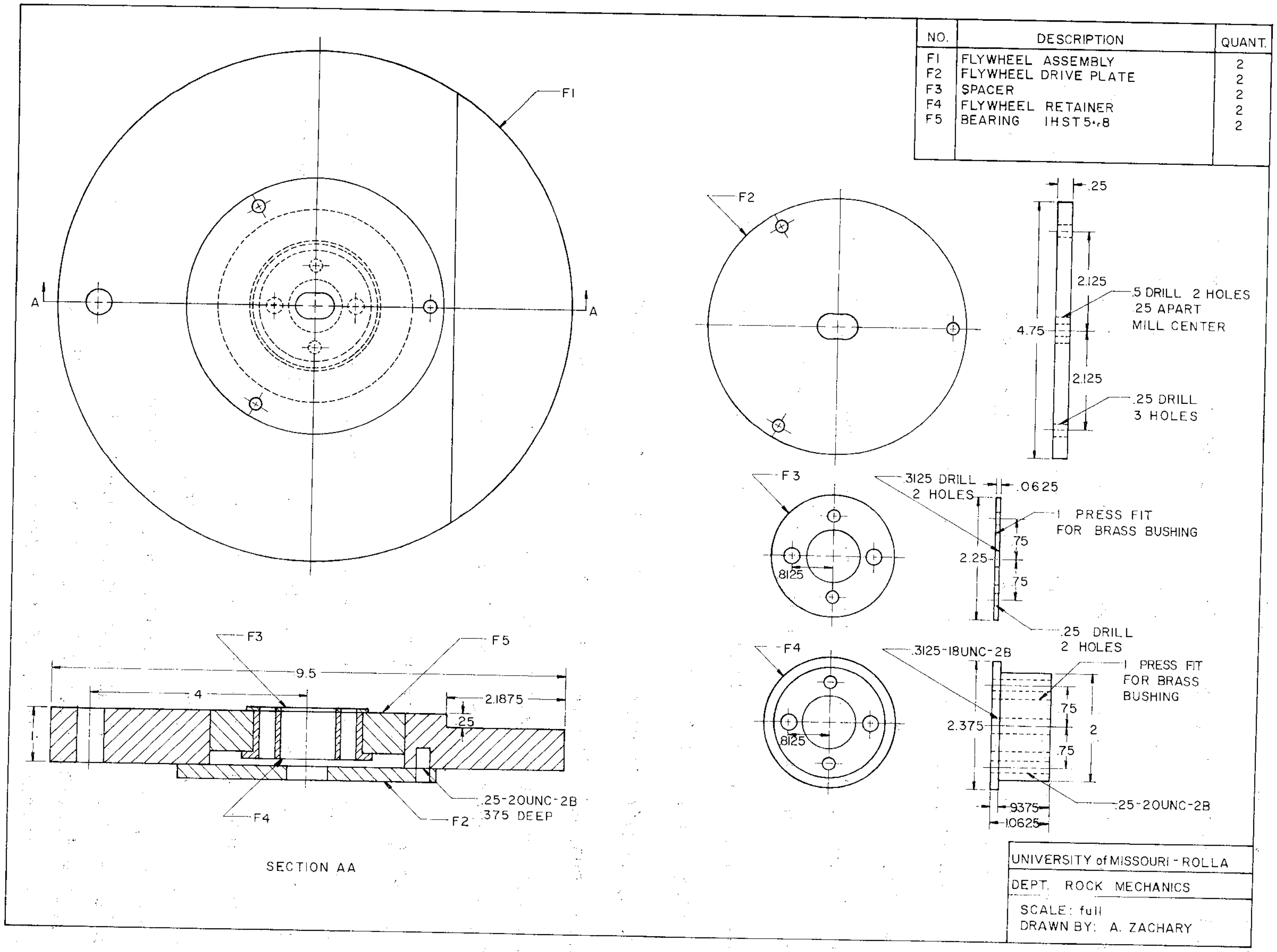


Figure A.1.8. Lower flywheel assembly.

to the cutting arm is through a connecting rod (Figure A.1.9) attached to the flywheels through the eccentric hole drilled in them. A commercial ball joint rod end (K44) is used to allow freedom of rotation at the coupling points. The attachment to the cutting arm is through a sliding sleeve. In the initial trials this was locked to the cutting arm with an Allen screw but this proved unstable at the higher oscillation speeds proposed, and the connection was, therefore, made with silver solder. Rubbing blocks were fitted to the connection to hold the jet arm in the horizontal plane and increase stability of the arms.

The drive to the lower cutting arm was taken directly from the top cutting arm through a rigid linkage made up of two link rods and four K11 rod ends, commercially available, and used to give the required flexibility in the joint (Figure A.1.10).

The hydraulic driving motor was spaced from the coupling support arm on a short channel (Figure A.1.4) and drove the flywheels through a chain and sprocket assembly made up of commercially available units.

The final cutting head configuration is shown in Figure A.1.11.

Load Measuring Instrumentation

The proposed instrumentation utilizes four strain-gage bridges with two active bridge arms each. A diagram of a typical strain-gage bridge circuit is shown in Figure A.1.12, where

R = nominal resistance of strain gage
$Rg(t)$ = instantaneous strain-gage resistance
$e_1(t)$ = instantaneous voltage at node a with respect to ground
$e_2(t)$ = instantaneous voltage at node b with respect to ground
$e_0(t)$ = output voltage of differential amplifier
E = bridge supply voltage.

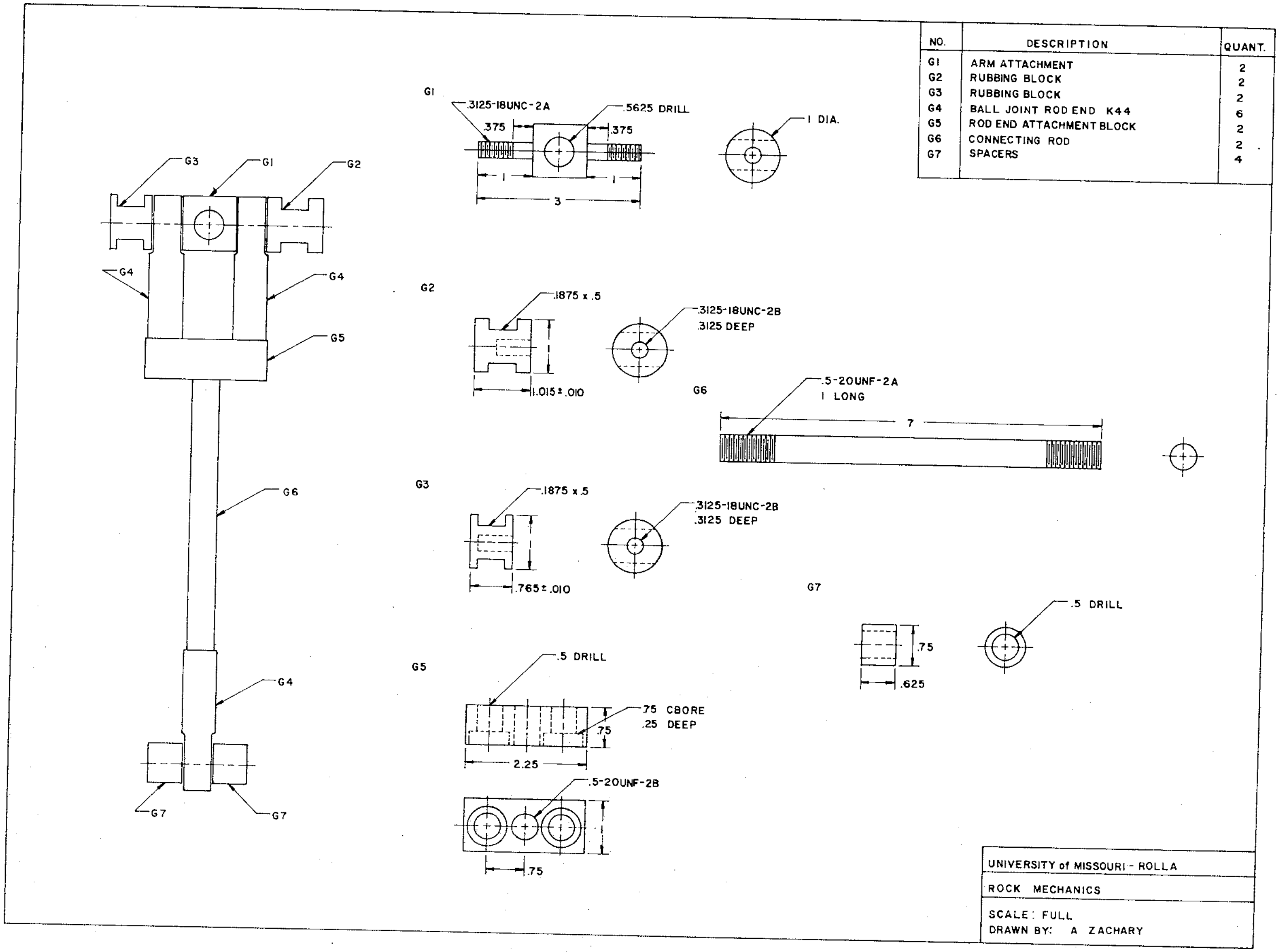


Figure A.1.9. Connecting load assembly.

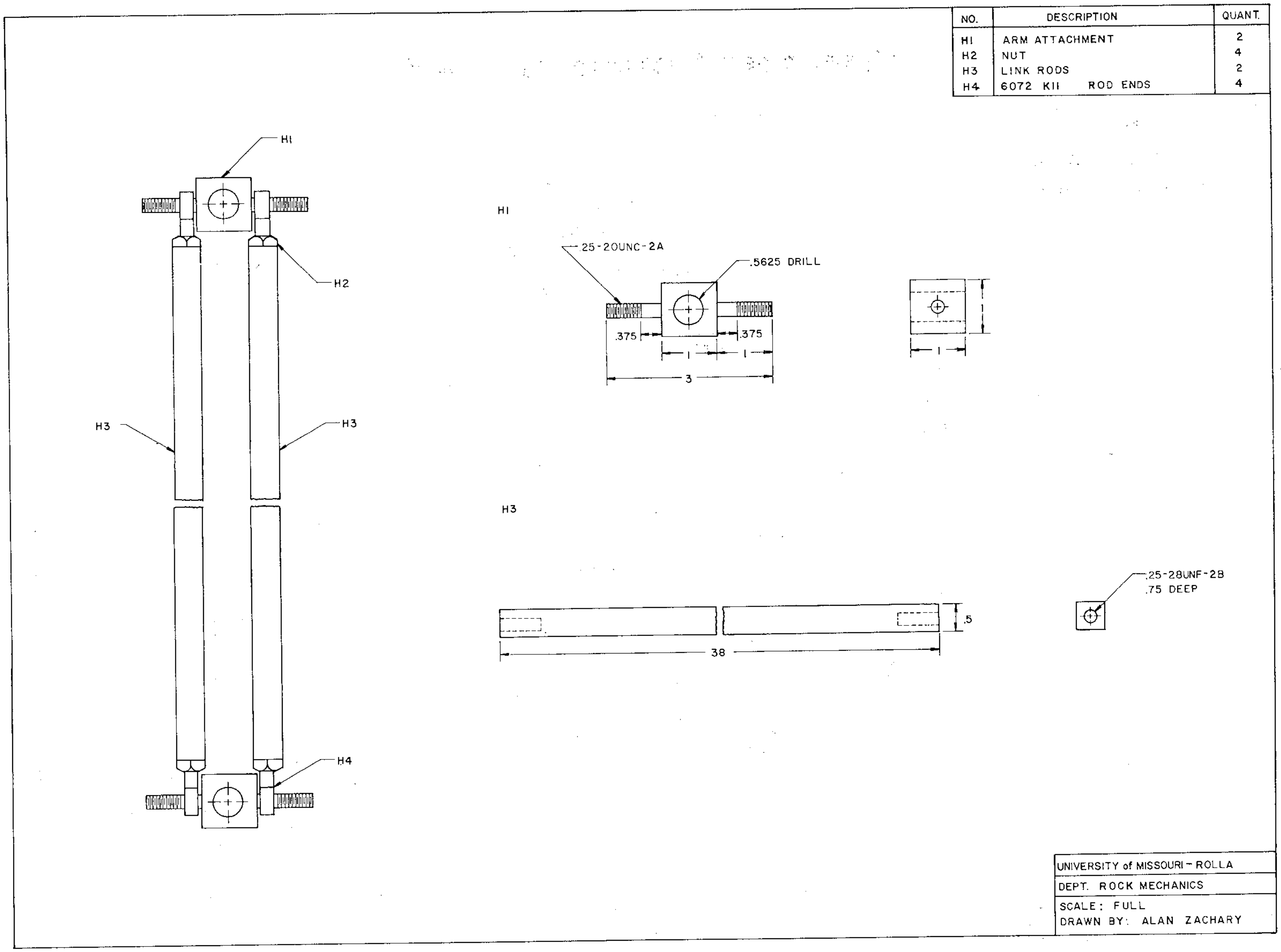
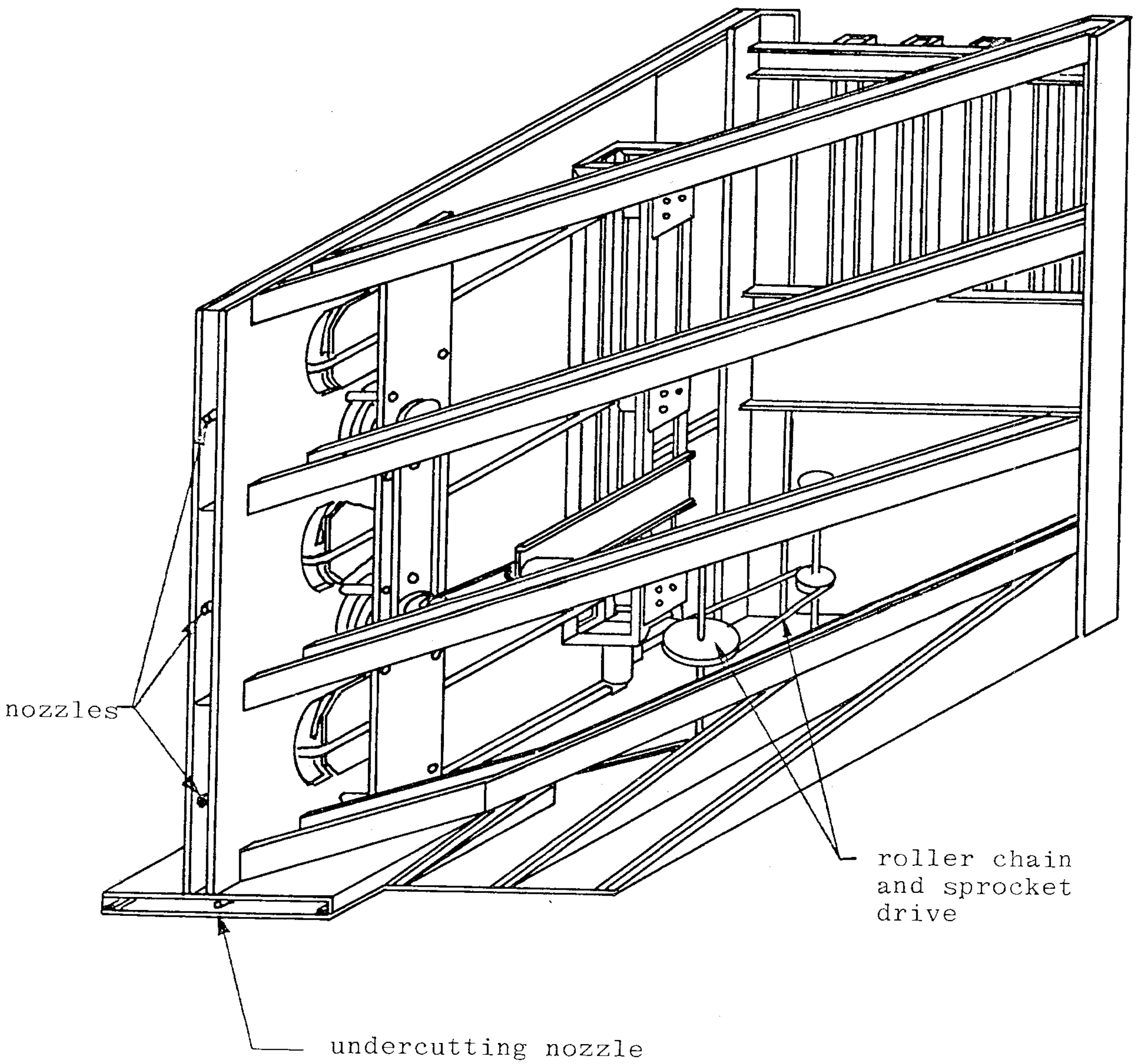


Figure A.1.10. Linkage load assembly.

Figure A.1.11
Mainframe of Hydrominer I
as Actually Fabricated



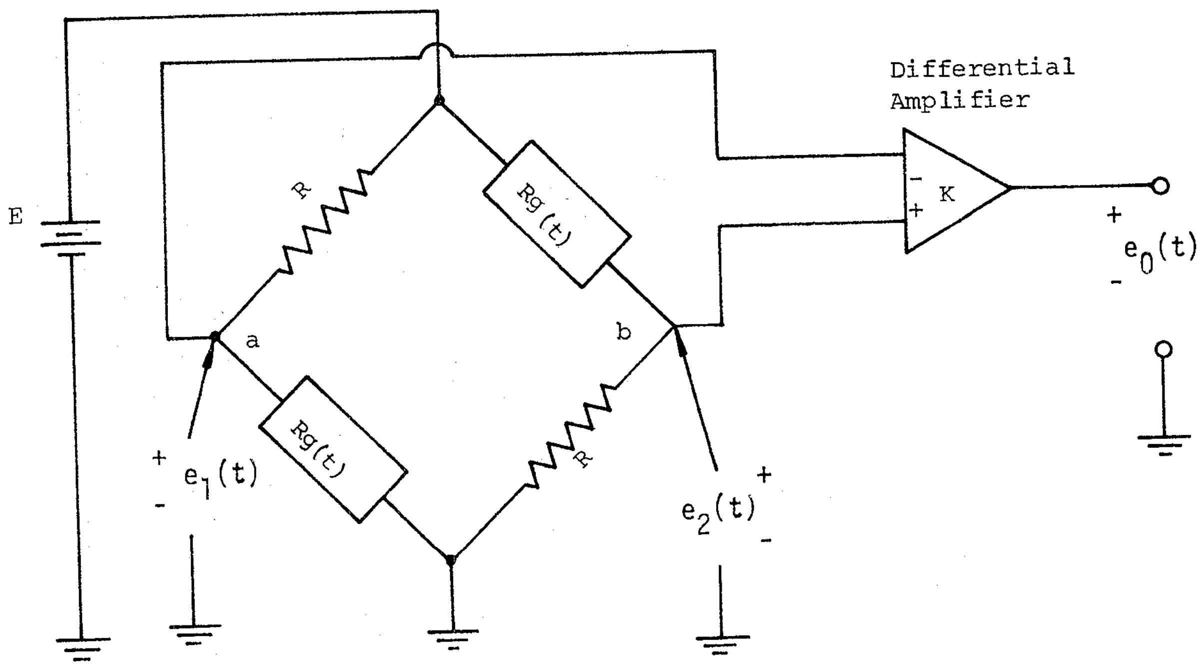


Figure A.1.12. Strain gage bridge circuit.

Expressing Rg(t) as a function of its nominal resistance, R, the following equation results:

$$Rg(t) = R + \Delta R \tag{1}$$

The term ΔR represents the difference between Rg(t) and its nominal resistance, R.

The following equations describe the operation of the circuit shown in Figure A.1.12:

$$e_2(t) = \frac{RE}{R + R + \Delta R} \tag{2}$$

$$e_1(t) = \frac{(R + \Delta R)E}{R + R + \Delta R} \tag{3}$$

$$e_2(t) - e_1(t) = \frac{RE}{2R + \Delta R} - \frac{(R + \Delta R)E}{2R + \Delta R} = \frac{-\Delta RE}{2R + \Delta R} \tag{4}$$

Defining

$$S = \frac{\Delta R}{R},$$

equation (4) can be written as

$$e_2(t) - e_1(t) = -\frac{\frac{\Delta R}{R}E}{2 + \frac{\Delta R}{R}} = -\frac{SE}{2 + S} = -\frac{E}{2} \cdot \frac{S}{1 + \frac{S}{2}} \tag{5}$$

Now then

$$e_0(t) = K \quad e_2(t) - e_1(t) \quad = -\frac{KE}{2} \cdot \frac{S}{1 + \frac{S}{2}} \tag{6}$$

The equation relating strain to resistance is given by

$$\frac{\Delta R}{R} = S = F\,\frac{\Delta L}{L}, \tag{7}$$

where

F = gage factor

$\frac{\Delta L}{L}$ = strain in inches/in.

Combining equations (6) and (7) an expression for output voltage in terms of strain is obtained; namely,

$$e_0(t) = -\frac{KE}{2} \cdot \frac{F\frac{\Delta L}{L}}{1 + \frac{F}{2} \cdot \frac{\Delta L}{L}} \tag{8}$$

Solving for $\frac{\Delta L}{L}$

$$\frac{\Delta L}{L} = -\frac{2e_0(t)}{e_0(t) + KE \quad F} \tag{9}$$

$$\frac{\Delta L}{L} \simeq \frac{2e_0(t)}{KEF}, \text{ if } e_0(t) << KE \tag{10}$$

The acquisition of the data, Figure A.1.13, will be accomplished by taking the output of the differential amplifier shown in Figure A.1.12, correcting for any dc errors, low-pass filtering, amplifing, frequency-modulating, summing with other three signals, and recording on cassette recorder (Figure A.1.14). At an appropriate time the data will be played back, demultiplexed, demodulated, an analog-to-digital conversion made, and stored in the calculator. The calculator will process the data and output the results to the digital plotter and the graphic terminal hard copy unit.

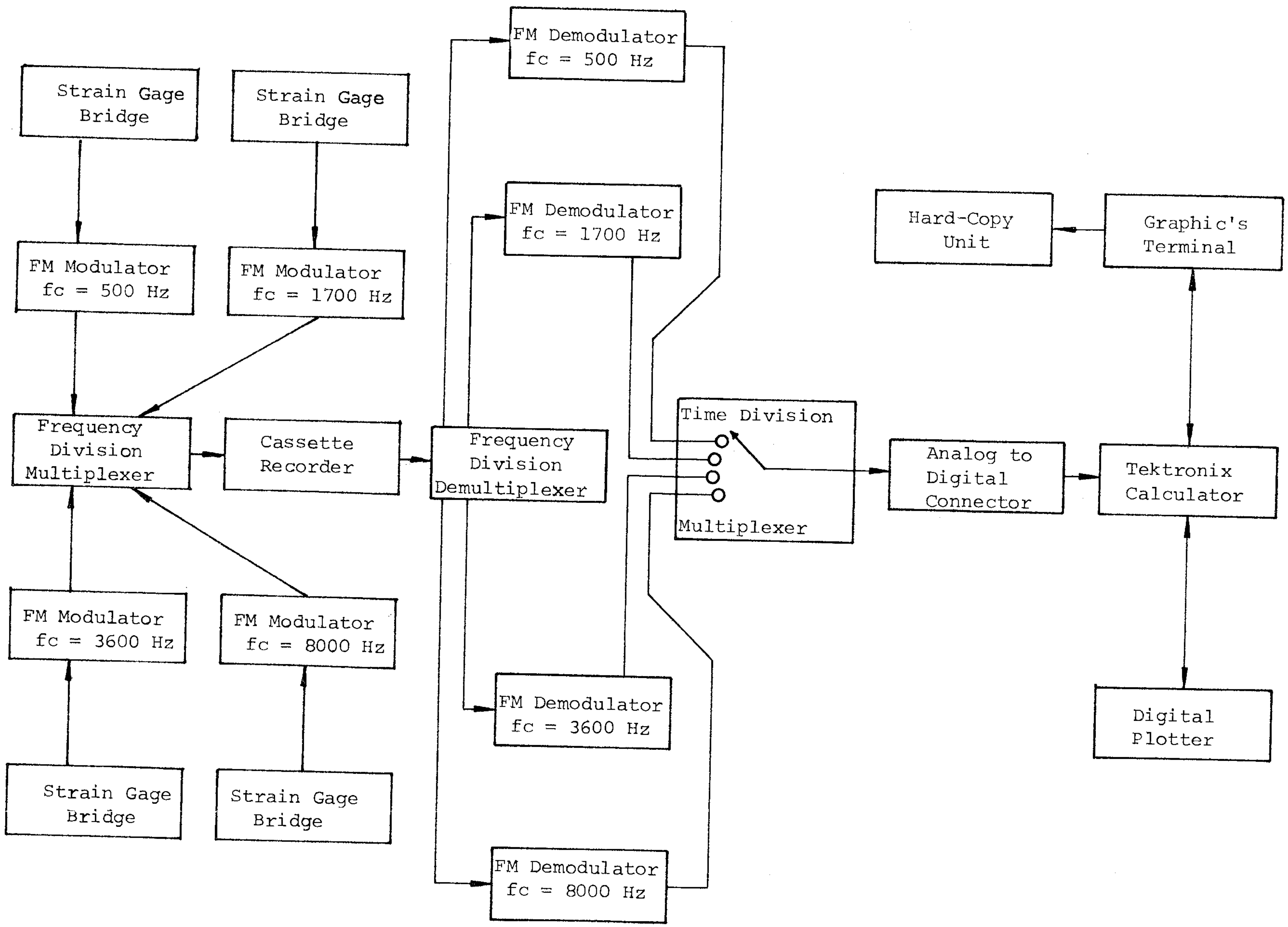
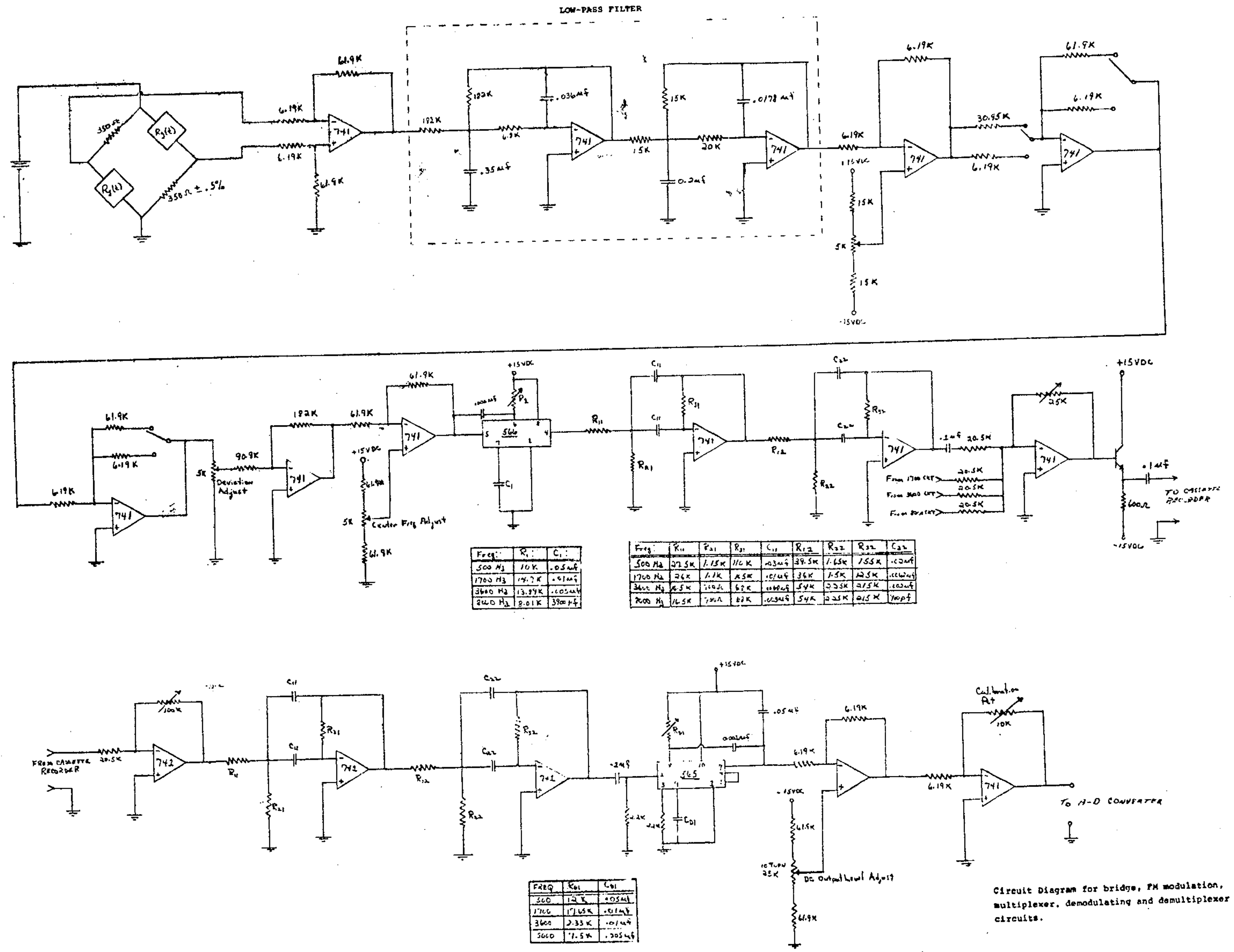


Figure A.1.13. Block diagram of instrumentation circuitry.



Freq:	R_1:	C_1:
500 Hz	10K	.05 µf
1700 Hz	14.7K	.01 µf
3600 Hz	13.84K	.005 µf
8400 Hz	8.01K	3900 pf

Freq:	R_{11}	R_{21}	R_{31}	C_{11}	R_{12}	R_{22}	R_{32}	C_{22}
500 Hz	27.5K	1.15K	110K	.03 µf	39.5K	1.65K	155K	.02 µf
1700 Hz	26K	1.1K	85K	.01 µf	36K	1.5K	125K	[illegible]
3600 Hz	16.5K	[illegible]	62K	[illegible]	54K	2.25K	215K	.002 µf
7000 Hz	16.5K	[illegible]	62K	.0034 µf	54K	2.25K	215K	700 pf

FREQ	R_{01}	C_{01}
500	12K	.05 µf
1700	17.65K	.01 µf
3600	2.35K	.01 µf
5000	7.5K	.005 µf

Figure A.1.14. Circuit diagram for instrumentation circuitry.

APPENDIX 2

EXPERIMENTATION ON THE SHAPE OF THE MOST EFFECTIVE NOZZLE

Research carried out in the earlier contract to design the cutting head had shown, Table A.2.1, that the jet cutting achieved with the dual orifice nozzle was not as deep as that achieved where only a single orifice of equivalent size was used. This indicated that there was some improvement potentially available in the jet cutting if a better nozzle design were obtained.

Three features of the nozzle design could be considered. These were: firstly, the closeness of the two issuing jets, one to the other; secondly, the angle made by the two jets to the axis of the cutting arm; and thirdly, the shape of the converging section of the nozzle leading up to the nozzle throat. In the experiments each of these sections was looked at seperately. While experiments were carried out on the nozzle angle, the experimentation carried out in the earlier program, which had shown that a 25 deg included angle between the jets was optimum, was still considered valid.

The first part of the investigation was to determine how close two orifices could operate without interference. It was possible that, where the two jets were too close, the flow interference between the two was disrupting the jet closer to the nozzle than would otherwise occur. Further, the jets could be pulled towards each other by a variation of the Coanda effect. In order to investigate this, a series of experiments was carried out using dual orifice nozzles with the jets directed parallel to the axis of the cutting arm. A set of nozzles were constructed with varying distances between the two orifices to provide a range of values.

The jets were directed into a metal target box, aligned parallel to the jet axis. This contained, at 1-in. intervals down its length, slots

Table A.2.1. Data summary from previous design experiments.

Depth of cut - single nozzle (in inches).

Speed of passes ft/min	10	20	50	100	160	AVE
At 1 pass	2.82	0.70	1.44	1.04	1.78	1.56
At 2 passes	4.72	4.17	2.53	1.64	0.84	2.78
At 4 passes	6.38	5.19	3.59	2.72	3.09	4.19
At 8 passes	7.61	6.99	4.86	4.34	1.91	5.14
At 16 passes	14.34	11.78	8.53	6.34	4.61	9.12
AVE of passes	7.17	5.77	4.19	3.22	2.45	4.56

Depth of cut - dual nozzles (in inches).

Speed of passes ft/min	10	20	50	100	160	AVE
At 1 pass	2.83	1.23	1.59	3.91	0.67	2.05
At 2 passes	2.81	2.80	2.30	0.95	1.44	2.06
At 4 passes	4.50	5.73	3.24	1.65	1.90	3.40
At 8 passes	6.90	6.76	3.39	3.81	2.25	4.62
At 16 passes	7.25	4.64	3.77	4.13	3.32	4.62
AVE of passes	4.86	4.23	2.86	2.89	1.92	3.35

into which target plates of aluminum could be inserted. Aluminum targets were used since these would be eroded by water jets at 10,000 psi; but because of the material grain size, erosion only would occur. This would, therefore, give the shape of the crater removed, in turn, a reflection of the shape of the impacting jet. The aluminum would be eroded not only by the water jet impact but also by droplet impact from the spray where this was of sufficiently high initial velocity. (Much work has been recently carried out in droplet erosion elsewhere, and this has been a subject of many works written by other investigators).

A series of experiments was undertaken, the results of which are shown in Figures A.2.1 through A.2.7. The relevant nozzle was inserted in the nozzle holder and the pressure in the jet raised to the required level, while a steel shutter was placed between the nozzle and the target box. When the pressure had been stabilized at the required level, the steel shutter was withdrawn and the jets entered the target box. The jets played on the first aluminum target, at a distance of 1 in. from the nozzle, for a fixed time interval, either 30, 60, 90, or 120 seconds. This plate was then withdrawn, and the jet impacted the second target plate in like manner. The sequence of aluminum targets was withdrawn from the box at the chosen time intervals, thus giving a profile of jet performance with standoff distance. The evaluation of the results was in two parts. There was a visual analysis of the data obtained and exemplified in the attached figures, and the target plates were also weighed to observe changes in the mass loss with standoff distance (Table A.2.2). Of the two methods of analysis, the visual observation proved the superior. The reason for this is that as the jets break up, the structure changes and thereby also the method by which they erode the target surface. The effect of this change in jet structure on the mechanism

Figure A.2.1. Erosion pattern in 2-in. aluminum target blocks, single nozzle, .06-in. diameter, 10 ksi, 2 minutes.

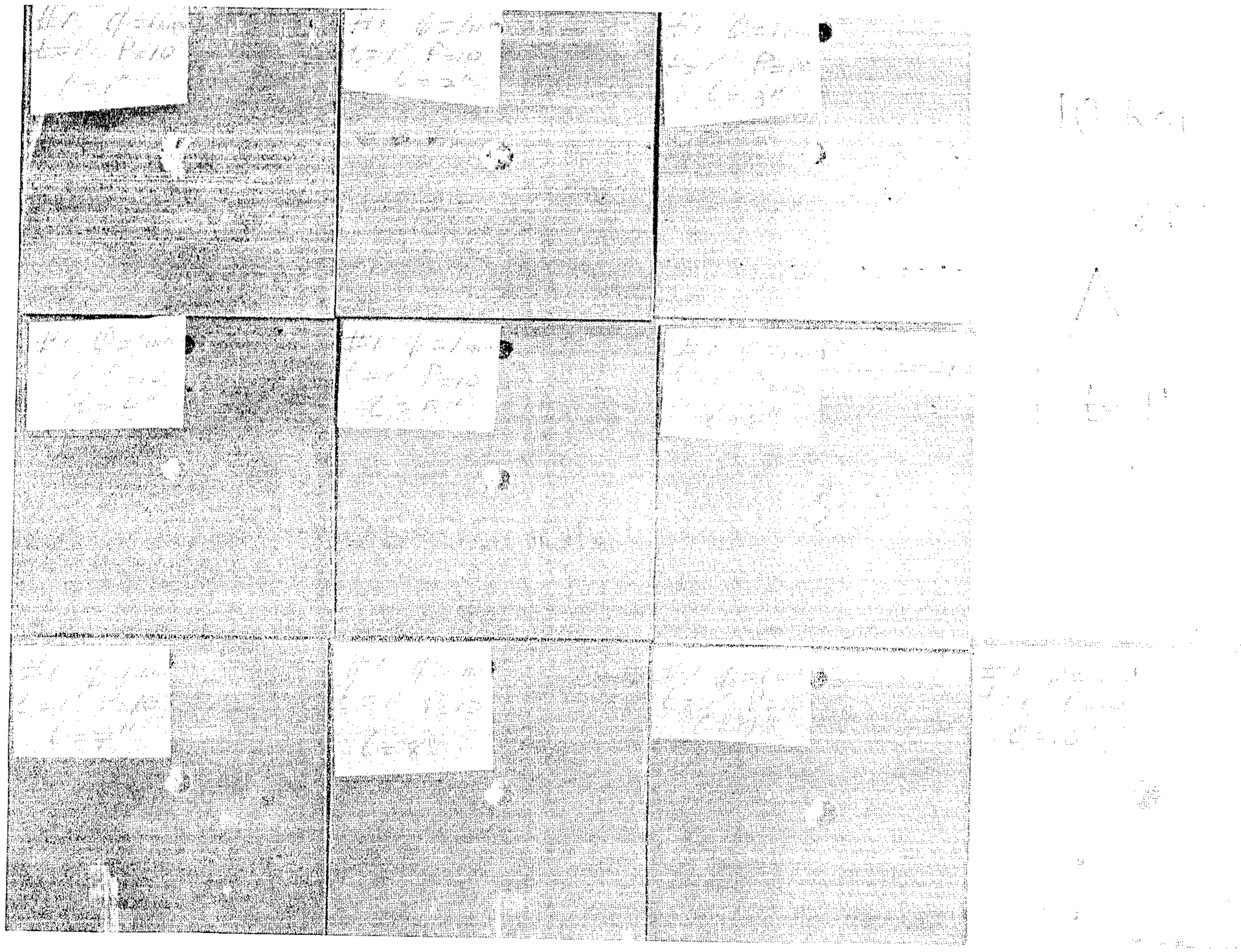


Figure A.2.2. Erosion pattern in 2-in. aluminum target blocks, single nozzle, .04-in. diameter, 10 ksi, 2 minutes.

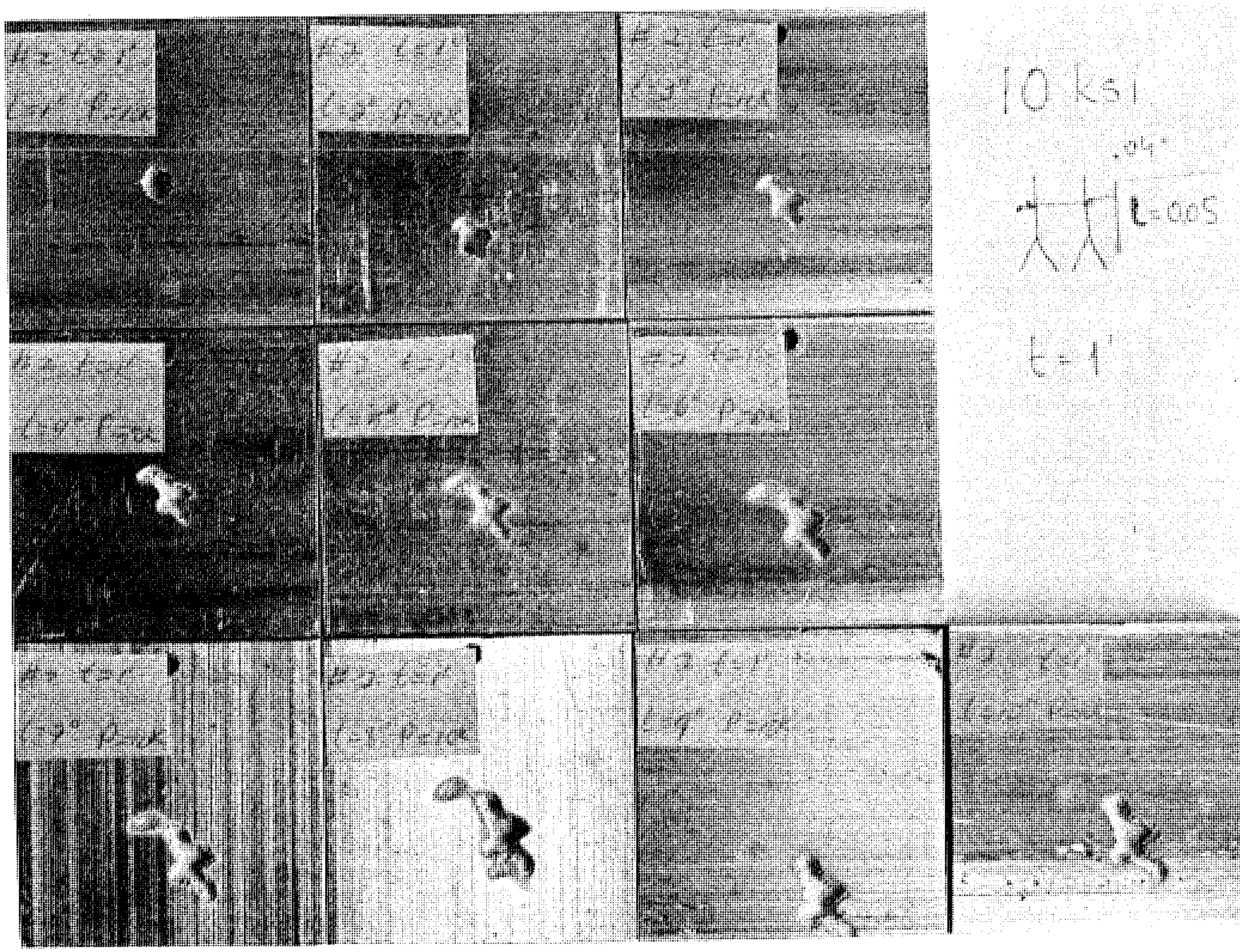


Figure A.2.3. Erosion pattern in 2-in. aluminum target blocks, dual nozzles, 0.04-in. diameter, 0.05 in. apart, 10 ksi, 1 minute.

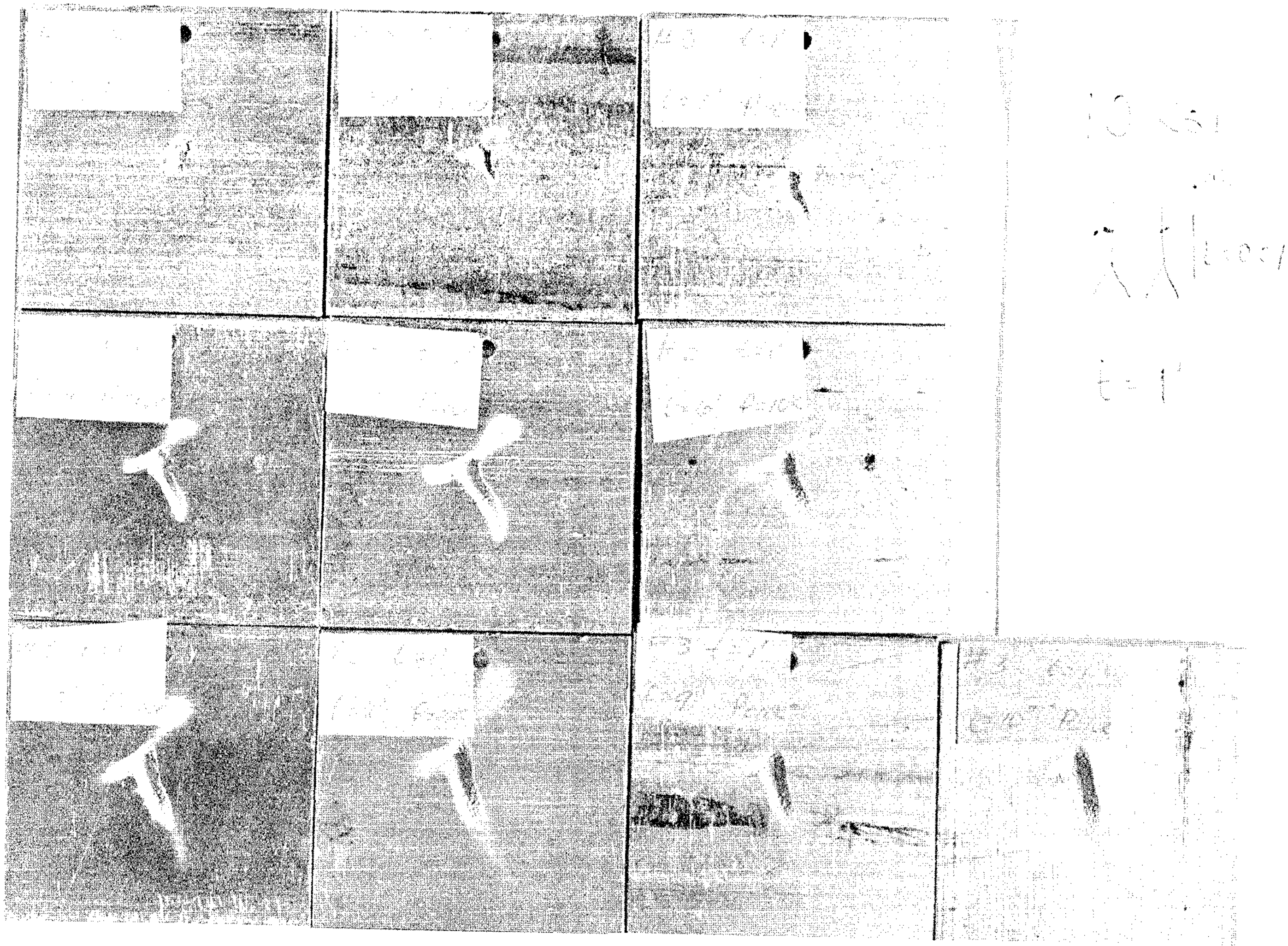


Figure A.2.4. Erosion pattern in 2-in. aluminum target blocks, dual nozzles, 0.04-in. diameter, 0.07 in. apart, 10 ksi, 1 minute.

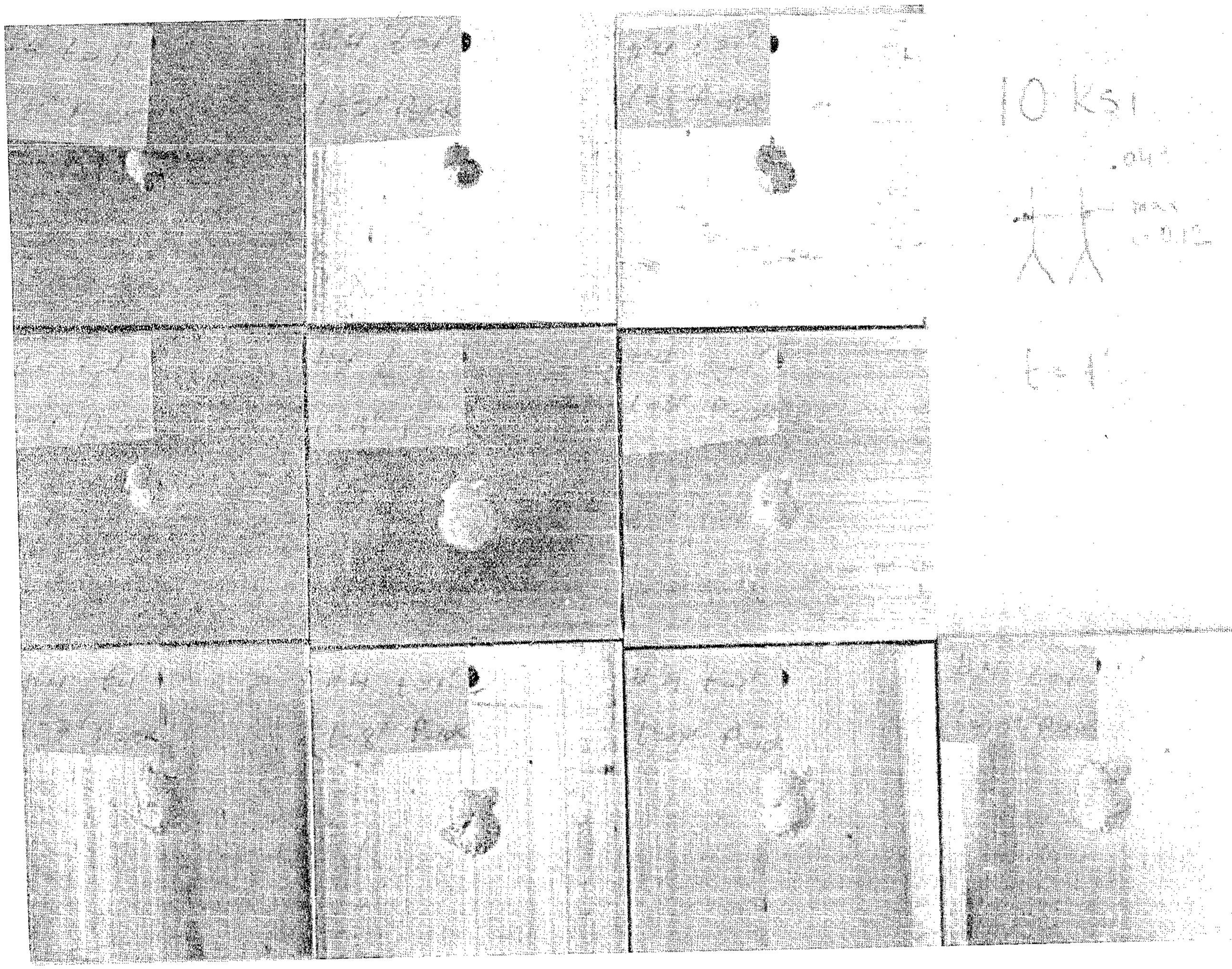


Figure A.2.5. Erosion pattern in 2-in. aluminum target blocks, dual nozzles, 0.04-in. diameter, 0.12 in. apart, 10 ksi, 1 minute.

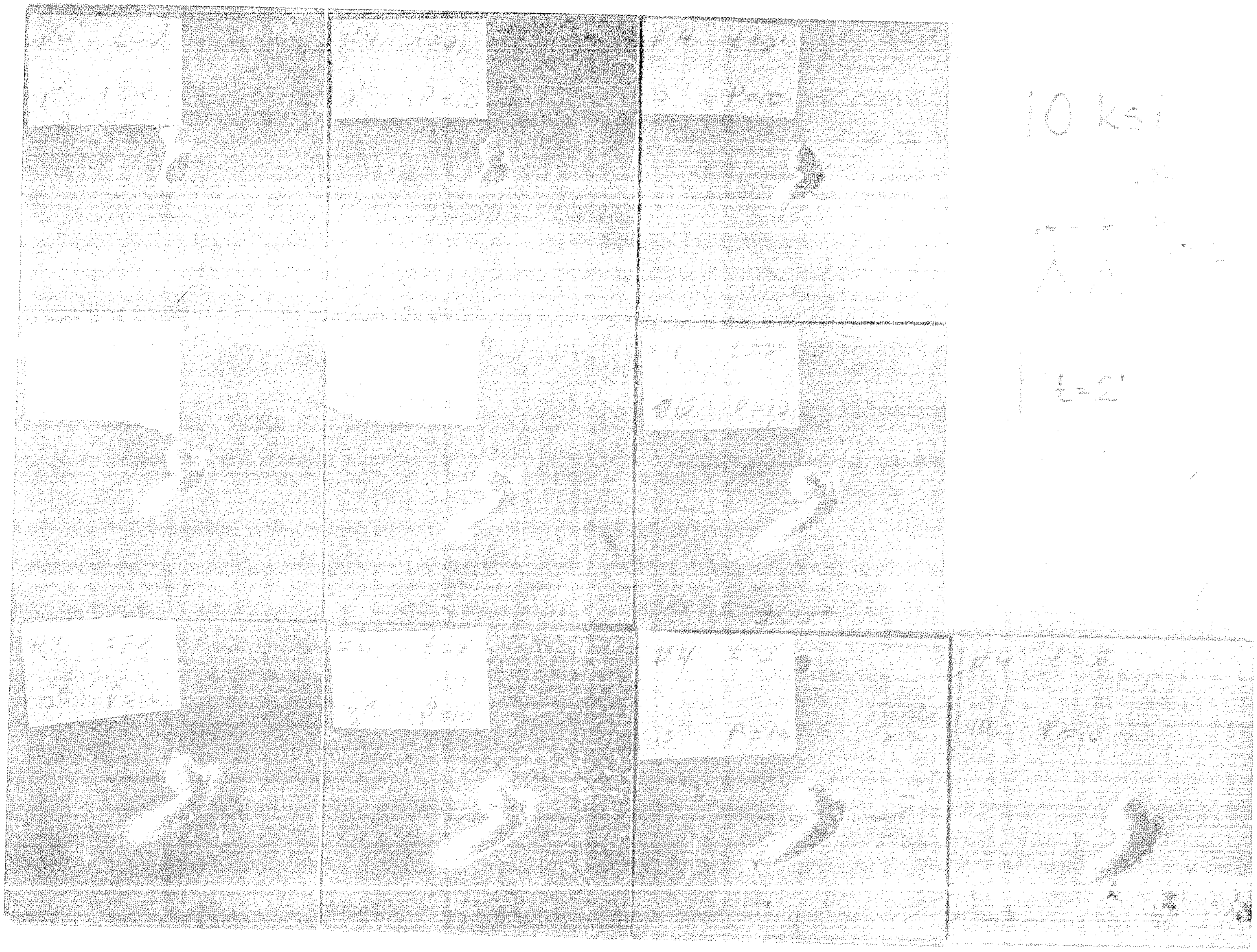


Figure A.2.6. Erosion pattern in 2-in. aluminum target blocks, dual nozzles, 0.04-in. diameter, 0.12 in. apart, 10 ksi, 2 minutes.

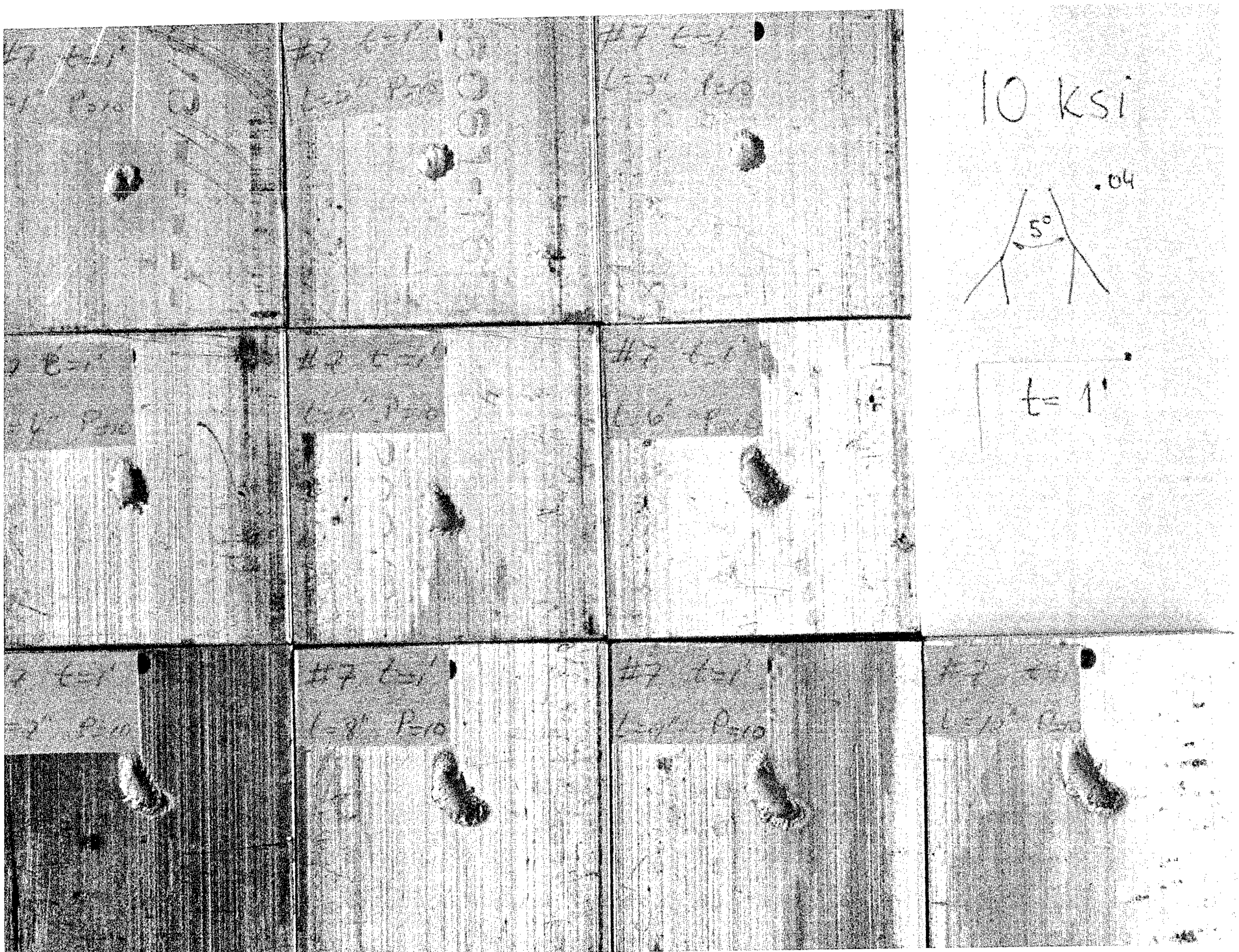


Figure A.2.7. Erosion pattern in 2-in. aluminum target blocks, converging jets, included angle, 0.04-in. diameter, 10 ksi, 5 deg, one minute.

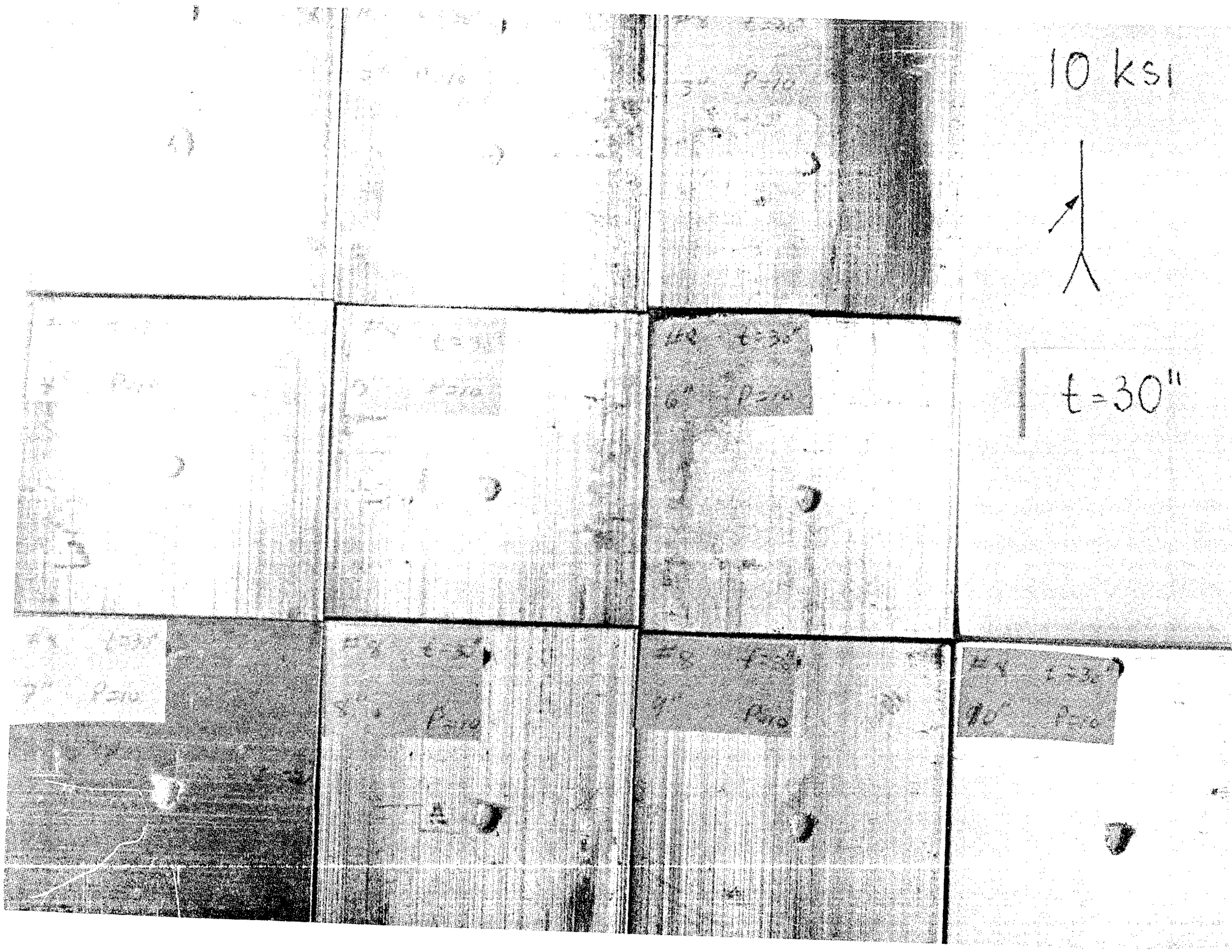


Figure A.2.7.a. Erosion pattern in 2-in. aluminum target blocks, single nozzle with air in-take, 0.04-in. diameter, 10 ksi, 30 seconds.

Table A.2.2. Weight loss on 2-in. target blocks during the test program.

Test #	1"	2"	3"	4"	5"	6"	7"	8"	9"	10"	Noz-zles	Time
1	.064	.191	.170	.288	.204	.177	.185	.262	.311	.265	1	2
2	.005	.006	.027	.018	.027	.208	.085	.147	.116	.167	7	½
3	.232	.296	.255	.260	.236	.241	.245	.258	.276	.340	4	2
4	.022	.007	.017	.015	.013	.017	.024	.032	.024	.025	8	½
5	.002	.001	.004	.005	.005						10	¼
6	.016	.012	.031	.076	.072						11	½
7	.034	.078	.067	.107	.127	.160	.189	.194	.155	.224	3	2
8	.023	.028	.054	.061	.087	.081	.097	.087	.129	.122	3	1
9	.244	.152	.121	.141	.224	.178	.225	.263	.360	.264	2	2
10	.115	.154	.104	.081	.095	.180	.137	.191	.165	.181	2	1
11	.104	.082	.083	.102	.156	.182	.423	.352	.235	.205	7	1½
12	.151	.226	.288	.293	.279	.266	.236	.185	.246	.265	4	1
13	.045	.085	.146	.080	.121	.190	.182	.288	.269	.249	7	1
14	.010	.014	.012	.014	.024	.022	.023	.024	.029	.051	1	½
15	.011	.033	.014	.024	.034	.054	.028	.033	.055	.045	1	1
16	.087	.076	.080	.138	.130	.169	.157	.238	.251	.299	7	1
17	.042	.041	.073	.072	.058	.185	.130	.141	.122	.171	7	½
18	.003	.015	.010	.012	.020	.025	.027	.035	.031	.045	8	½
19	.002	.004	.006	.007	.008	.009	.015	.013	.021	.025	8	¼
20	.025	.035	.055	.062	.077	.135	.096	.109	.174	.138	6	½
21	.002	.020	.119	.045	.035	.100	.116	.095	.103	.168	9	½

Weight loss in grams.

of jet erosion can clearly be seen in Figure A.2.1. The set of target plates are for a single 0.04-in. orifice, 10,000 psi jet directed at each target plate for two minutes, the distance increases from left to right in rows downward.

At the closest distance to the nozzle the jet has a solid core of relatively constant pressure within a sheath of decelerating water as the surface approaches. The failure of the target is shown to be due to the lateral jet velocity following impact, rather than the absolute jet pressure. This is indicated by the solid central core under the central impact point in the 1 in., 2 in., and 3 in. target blocks. In this zone, although the jet pressure is the maximum generated, the gradient across the jet is low and erosion, at 1 in. and 2 in., does not occur. (It is interesting to note that at an exposure time of one minute a slight central core can be seen in the impact crater in targets up to 9 in. from the jet). The effect extends beyond the radius of the impacting jet for the 1 in. target thus indicating that the high lateral flow velocities generated around the impact point must be removing material. This is evidence that water jet erosion of material within 50 nozzle diameters of the nozzle is due to shear and tension failure mechanisms, rather than compressive failure; and that theories and dimensionless analysis of the failure phenomenon which include the material compressive strength are inherently incorrect.

At the 1-in. standoff it can be seen (Figure A.2.1) that even after two minutes exposure there are sectors around the central core where no erosion has occurred. This suggests that at this point the velocity gradient across the jet must be very rapid and that radial flow from the impact is not axisymmetrical. In the second stage of erosion the size of the central core diminishes, and the high lateral jet velocity due to its presence is also removed. Material removal is caused by the velocity gradient across the impacting jet and where the relative shear value falls below the strength of the material, erosion stops and a clean sharp cavity is eroded, of diminishing diameters with increasing distance. This is more clearly perceived at the shorter impact time (Figure A.2.2). The velocity gradient is more gradual than at shorter standoff distances so that the relative pressure drop across a grain is sufficient to generate the shear forces required for failure, and it is postulated that failure in this zone will be a combination of shear and tensile failure.

As standoff distance increases further, a third method of failure becomes predominent. The central core of the jet breaks into discrete droplets, traveling at velocities initially close to that of the solid jet. When these droplets impact the surface, they generate a water hammer pressure (computed as the product ρCV, where ρ is the fluid density, C is the sound velocity, and V is the impact velocity) which is of much greater magnitude than the original jet stagnation pressure. The duration is extremely short, of the order of 1 μsec, and thus a series of such impacts is required to remove material. At short standoff distances, droplet size and number is insufficient for material erosion; but as the distance increases beyond 100 nozzle diameters, the size and number of the droplets is sufficient for material erosion to occur. The jet will also be dispersed radially as distance increases and so, the droplets disperse and the craters increase in size. The droplets lose velocity with distance; but because of the much higher pressures generated by the water hammer, their range of effectiveness is much greater and exceeds the 250 nozzle diameter (10 in.) range of the experimentation.

Droplet erosion craters are wider and do not generally have the sharp edges of the jet cut craters. The surface is also much rougher as is shown by the surface at the crater edge of the 10 in. target in Figure A.2.1.

The change in the crater surface characteristics was used to evaluate the presence of interference between adjacent jets. Where interference occurs, then the droplet erosion will occur nearer the nozzle than previously and the jet structure will be distorted. Three orifice separation distances were evaluated at 0.05, 0.07, and 0.12 in., (Figures A.2.3 to A.2.5) with nozzles numbered 2, 3, and 4 (Table A.2.2) and at different exposure times; since in general the longer the exposure time, the greater the evidence of droplet erosion. The results show that while interference occurs at 0.05 and 0.07 in. separations, at one minute exposure time there is no evidence of interference between the jets at 0.12 in. However, when the exposure time was increased to two minutes some evidence of interference was shown (Figure A.2.6). Nevertheless, since this was at greater standoff distances and longer exposure times and since the design to be used has divergent rather than parallel jets, a 0.125 in. spacing between jets was considered sufficient to preclude jet interference.

This phenomenon is not to be confused with that which occurs when two jets converge, as in the manner of shaped charge jet formation. This method, which has previously been suggested by Bowles,[2] was studied with two angles of convergence, 2.5 deg and 5 deg (nozzles 6 and 7); however, the results were not remarkable in this mode of testing (Figure A.2.7). Ongoing research has, since that time, indicated some promise for this technique and the results of this testing will be reported as results develop.

An attempt to introduce air bubbles into the jet, as suggested by Russian investigators,[3] using nozzles with holes in the side (nozzle 8) has so far shown no improvement in jet performance. A four orifice nozzle (nozzle 10) and a 10 deg divergent jet system (nozzle 11) were also tested but the small jets disrupted too rapidly and the divergent jets exceeded the dimensions of the test sample. A cavitation nozzle (nozzle 9) was also tested, but the data from this single test was nonconclusive, although a greater mass loss was measured from this test (Number 21) than where an equivalent nozzle without cavitation (Number 14) was used (Table A.2.2), (Figure A.2.8).

Experiments on Nozzle Geometry

The initial design of the water jet nozzle, Figure 3, had incorporated a single converging section to the nozzle followed by a separate converging section for each orifice from the central area. In another concept, a single throat convergence was maintained from which the two straight sections were taken. In a third nozzle design, the two orifices were prepared by taking converging cone sections from the start of the nozzle to the orifice, with separate but overlapping cone sections for each of the two nozzles. Experiments were prepared with diverging jets where the angle between the jets, the shape of the orifice, and the distance between the jets was varied. Concomitantly, there was some concern that the jets remain stable at the high traverse velocities since lateral nozzle traverse velocities up to 14 ft/sec are proposed for the field trials. In order to evaluate these various conditions, a factorial experiment was designed, to examine the effect of nozzle shape, jet spacing, and divergence angle and traverse velocity on jet structure.

The test matrix proposed (Table A.2.3) included these variables over a wide range of conditions; however, preliminary experimentation and the practical reality of the required product (a 2 in. wide slot) reduced the number of tests to be carried out.

The nozzles were in turn placed in a nozzle holder on the end of a model cutting arm (Figure A.2.9) which could be operated at oscillation speeds up to 400 rpm. Tests were, however, held to speeds between 40 and

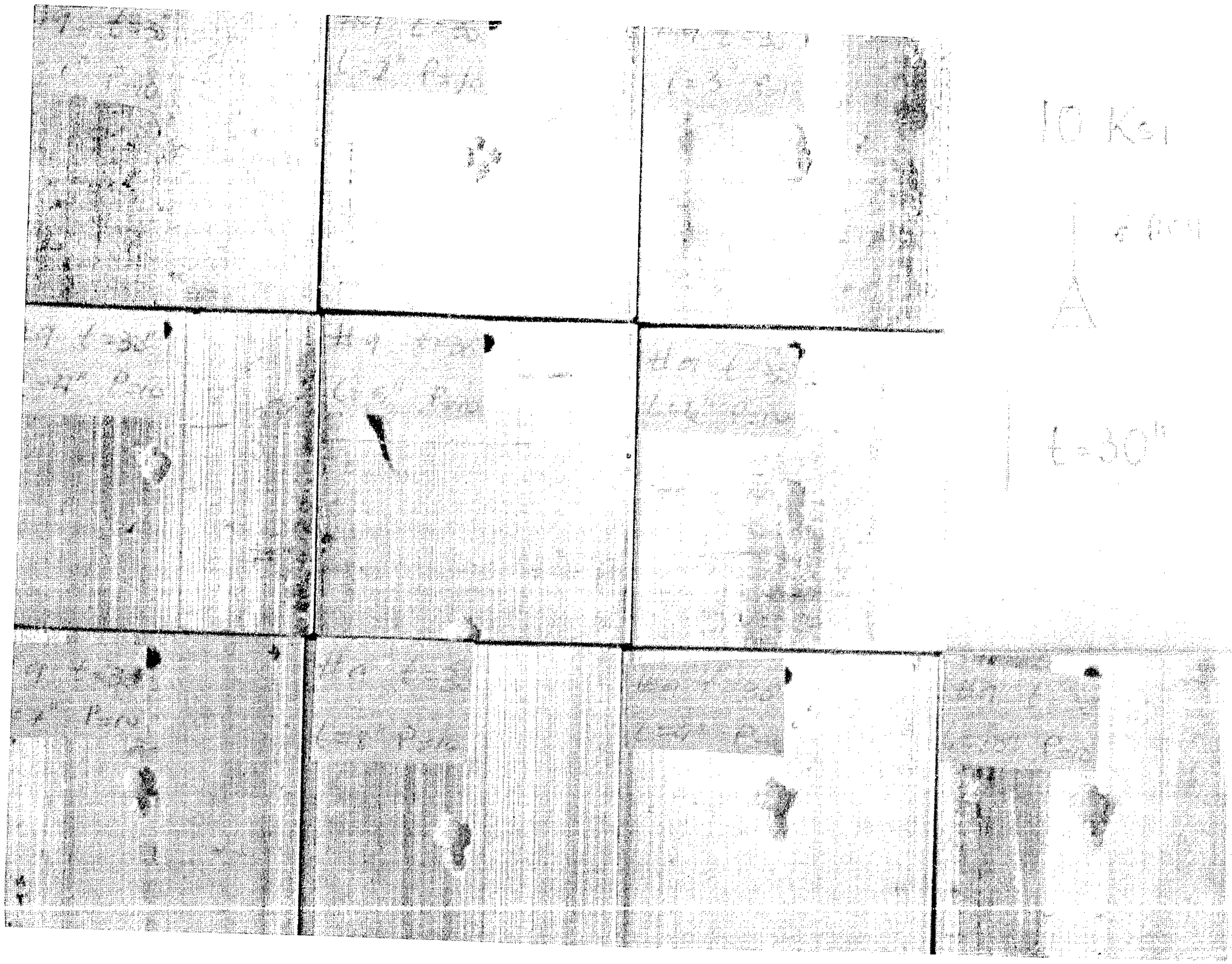


Figure A.2.8. Erosion pattern in 2-in. aluminum target blocks, cavitating nozzle, 0.04-in. diameter, 10 ksi, 30 seconds.

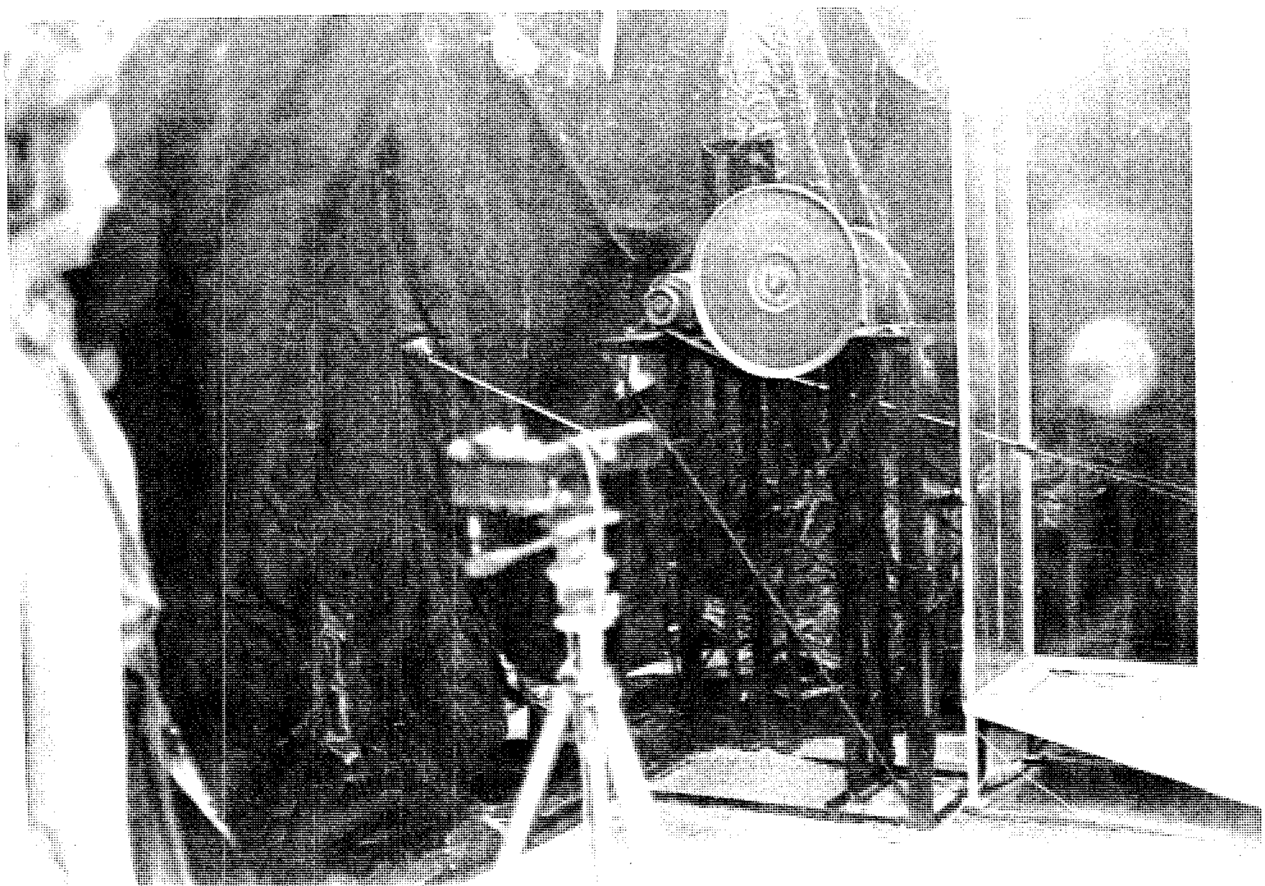


Figure A.2. 9. Equipment layout for moving nozzle experiment.

Table A.2.3. Test variables used in the moving nozzle experiments.

Nozzle Shape	Orifice Spacing	Included Jet Angle	Oscillation speed (rpm) 40	80	120	160	200
1	.125 in.	20	X	X			
2	.125 in.	10	X	X			
3	.08 in.	10	X	X	X	X	X
	.125 in.	10	X	X	X	X	X
	.250 in.	10	X	X	X	X	X
	.08 in.	15	X	X	X	X	X
	.125 in.	15	X	X	X	X	X
	.250 in.	15	X	X	X	X	X
	.08 in.	20	X	X	X	X	X
	.125 in.	20	X	X	X	X	X
	.250 in.	20	X	X	X	X	X
	.08 in.	25	X	X	X	X	X
	.125 in.	25	X	X	X	X	X
	.250 in.	25	X	X	X	X	X

All nozzles were tested at 60, 6,000, and 10,000 psi.

200 rpm. In order to assess the jet structure, it was decided that a photographic method would be most effective while remaining cost effective. Accordingly, the test area was enclosed in a heavy duty black plastic, which cut out incidental light so that an "open shutter" method could be used. The precedure developed was to start the jet oscillating at the required speed and then to open the camera shutter. A single flash at 1.8 μsec from a stroboscopic light source was used to illuminate and "flash freeze" the jet motion. Both front and back lighting positions for the strobe were used in the analysis. In order to get the required detail from the jet, without exposing the camera to the jet spray (to which previous experiments had shown it to be susceptible) a telephoto lens was fitted to the camera.

To ensure that the light triggered at the center of the strobe, the manual trigger circuit to the strobe had a second trigger included which was attached to the test frame and activated by the cutting arm. Because of the energy of the cutting arm, some experimentation was required to get the required flexibility in the trigger. A rubber sleeve extending into the arm path and fitted over a spring steel arm which rested upon a microswitch was eventually found to be satisfactory.

The procedure developed for the test program was to run each nozzle at five traverse speeds, 40, 80, 120, 160, and 200 rpm with each range run successively at 60 psi (the main pump supply pressure), 600 psi, and 10,000 psi. A full range of test conditions was thus recorded on a single 20 exposure roll of Kodak Tri-X film.

Thirty-six nozzle shapes were tested, a single cone with two throat orifices, a complete cone to each throat, and a nozzle design partly between the two. For all nozzles the cone angle was 13 deg and the throat length four times the diameter of 0.04 in. The single cone results were not found to be very satisfactory but, together with the photographic evidence, there was physical evidence to the superiority of the design in which the two orifice cones were started at the edge of the nozzle. The jets were directed into a vapor trap consisting of a tall wooden box built around an angle iron frame. Three slotted plastic pipe sections in the frame were to act as traps for the jet flow. Where the superior nozzle design was used, the jets had sufficient force upon impacting these pipes (some 36 in. from the nozzle) to move the frame backwards. Photographs of the jets showed that the jets remained solid, at speeds of up to 200 rpm, at distances greater than the 4 in. ahead of the nozzle at which they are expected to operate in the field.