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FLEXIBLE ROOF DRILL FOR LOW COAL

Final Technical Report
Contractor—Ingersoll-Rand Research, Inc.

July 1979

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



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Flexible Roof Drill for Low Coal

FINAL TECHNICAL REPORT
AS OF

July, 1979

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US DEPARTMENT OF ENERGY
Assistant Secretary for Energy Technology
Division of Fossil Fuel Extraction
Mining Research and Development

This report represents work on a program that was originated by the Bureau of Mines and was transferred to the Department of Energy on October 1, 1977.

FOREWORD

This report was prepared by Ingersoll-Rand Research, Inc. (IRRI), Princeton, New Jersey under USBM Contract Number HO242063. It was administered under the technical direction of the Spokane Mining Research Center with Mr. R. Burnham and later Mr. John Bevan as the Technical Project Officers. Mr. R. Simonich was the contract administrator for the Bureau of Mines.

This report is a summary of the work completed between November, 1975 and December, 1976. This report was submitted by the authors in June, 1977.

Material in this report was prepared by Robert N. Eakin, Program Manager, and Dean E. Rich and Thomas J. Wall, Project Engineers, under the supervision of Dr. Wallace A. McGahan, Director of Research.

IRRI wishes to express its appreciation to the Bethlehem Steel Corporation for the use of the Nicholas No. 81 mine, Drennen, West Virginia, as the test site.

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1.0 Abstract

The IRRI Flexible Roof Drill for Low Coal, referred to as the Rod Changer Drill, evolved as a result of the Phase I and II efforts on this contract in which several design concepts were evaluated. The Low Coal Drill program was initiated to provide a roof drill which, in conjunction with the Bendix Roof Bolt Inserter, would be capable of operating in coal seams as low as 30 inches. This drill would have the ability to drill and install bolts of up to eight feet in length, thus retaining productivity and increasing operator safety by removing the operator from the immediate area.

The IRRI "Rod Changer Drill" performs the drilling of longer than seam height roof drills by automatically coupling up to 6 short drill steel sections to form the drill string. This concept will permit the operator to perform drilling from a remote location, although bolt insertion requires his presence in the immediate area.

Volume I of the Final Report discusses the multiphase contract which began in July of 1974. The preliminary design efforts, Phase I and II, were completed in December 1974. The design effort, Phase III, began in March 1975 and was completed by mid-July 1975. The fabrication and aboveground test effort began in November 1975 and was completed in September 1976. The underground test phase began on September 27, 1976 and was completed on November 30, 1976 with the drilling of 400 "in-cycle" roof bolt holes. The original Bureau of Mines contract number was HO242063. The contract is now administered by Department of Energy as contract ET-74-C-01-9090. A second volume discusses a modification and test program conducted to implement the changes found desirable in the underground testing.

The underground test evaluation was conducted at Bethlehem Nicholas Mine No. 81 located in Drennen, West Virginia. The evaluation began with a demonstration of the prototype drill performing in a previously bolted section. This out-of-cycle test accumulated a total of 31 holes. The in-cycle test followed in which a total of 471 holes were drilled. The field trial was completed in two and one-half months in which the vehicle performed adequately in all areas. The coal seam height varied from 46" to 56" and the majority of the bolts secured were four foot in length utilizing four drill steels to drill a six foot hole. A ten foot bolt was the maximum length bolt secured to the top.

The average cycle time was approximately four minutes without the Bendix Bolt Inserter and indeterminate with the Bendix Bolt Inserter.

After reviewing the field trial results it is recommended that development of the Flex Drill be continued. The utilization of the Bendix Bolt Inserter is an area that requires immediate review for this concept to be accepted in a mining environment. A major redesign improvement must be the reduction of the overall cycle time which can be accomplished by examining the hydraulic circuit. It is also recommended that a family comprised of various carousel lengths be considered for various seam height applications.

The test demonstrated the ability of the Rod Changer Drill to successfully function in an underground environment without major malfunctions. The test also revealed the necessity of further refinement of the drill to improve productivity and reliability.

2.0 Introduction

The IRRI Flexible Roof Drill for Low coal, referred to as the Rod Changer Drill, evolved as a result of the Phase I and II efforts on this contract in which several design concepts were evaluated. The Low Coal Drill program was initiated to provide a roof drill which, in conjunction with the Bendix Roof Bolt Inserter, would be capable of operating in coal seams as low as 30 inches. This drill would have the ability to drill and install bolts of up to eight feet in length, thus retaining productivity and increasing operator safety by removing the operator from the immediate area.

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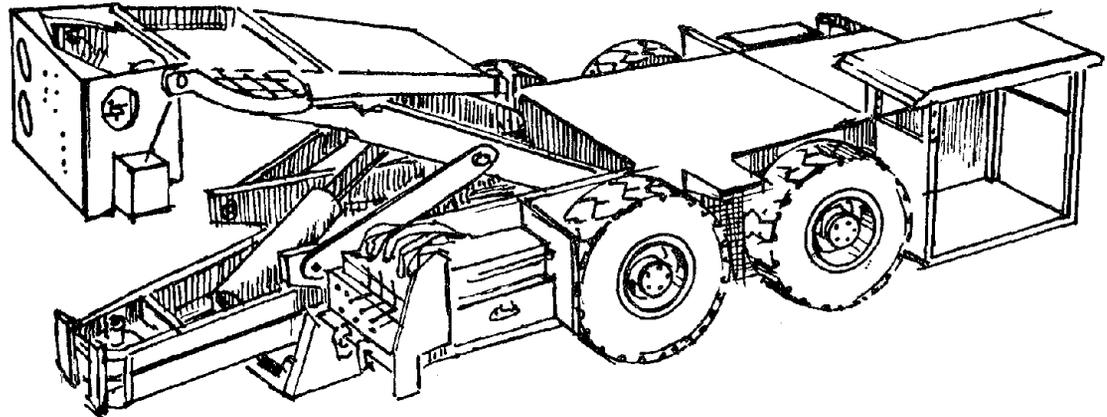
A series of sketches detailing the machine's features are shown in Figures 1 through 4.

The rod changer is a mechanical-hydraulic device that will automatically engage and separate short lengths of drill steel to allow a miner to drill an 8 foot hole in a low coal seam mine roof. The mechanism will drill a 1-3/8 inch diameter hole in approximately four minutes. The bolt bender is swung into position to insert a longer-than-seam-height roof bolt into the hole. Both the carousel and the bolt bender are swung from a stowed position to the axis of the desired hole with no detectable misalignment.

The rod changer shown in Figure 2 is completely powered by hydraulics. The hydraulic system is controlled both manually and by pneumatic pilot pressure from an on-board air compressor. The semi-automatic load and automatic

① - ROD CHANGER MECHANISM

③ - AIR LOGIC CONTROL BOX
AND AIR COMPRESSOR



② - OPERATORS CONTROL PANEL

Figure 1: Flexible Roof Drill For Low Coal

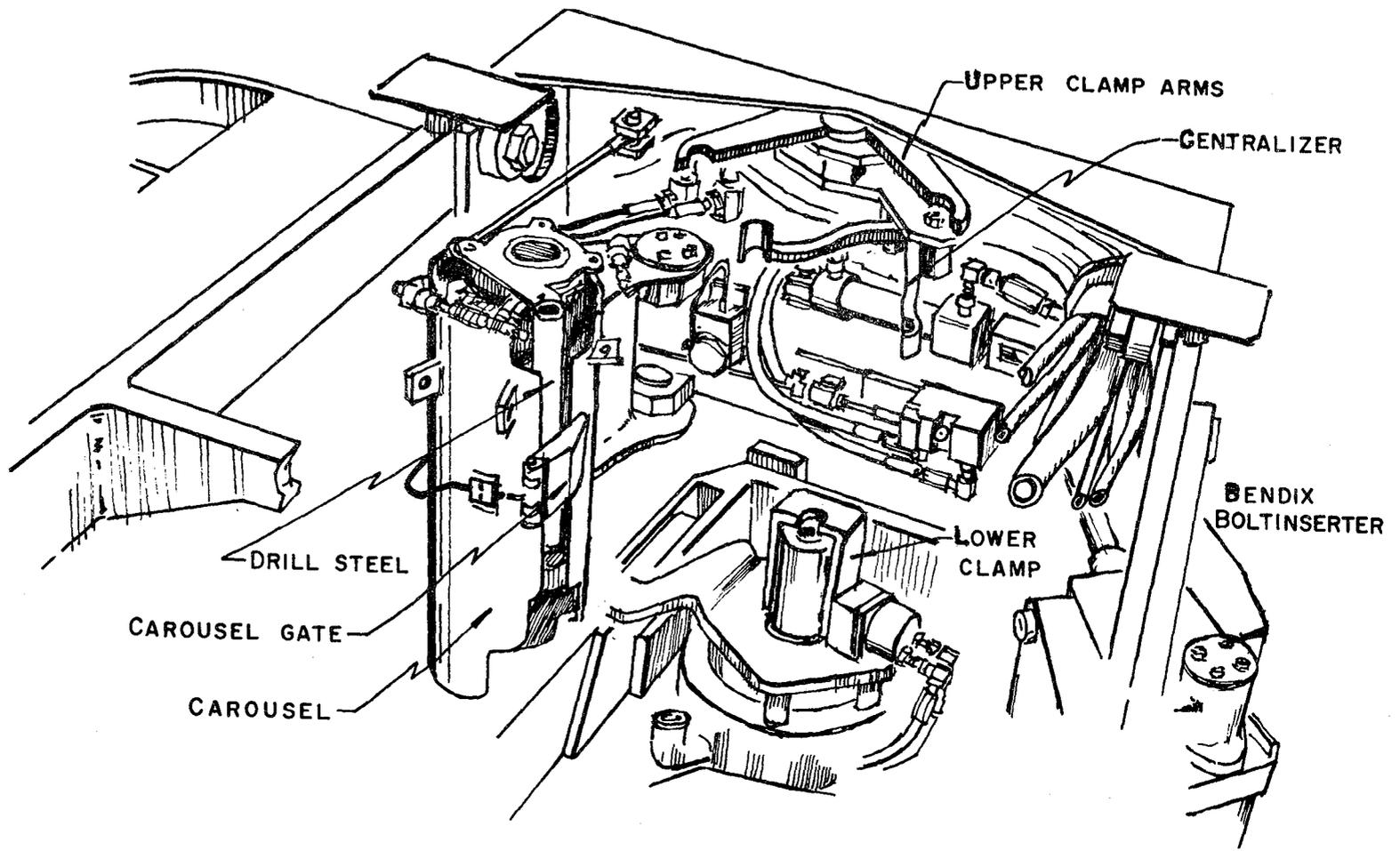


Figure 2: Rod Changer Mechanism - Exploded View

Figure 3: OPERATORS CONTROL STATION
EXPLODED VIEW (2)

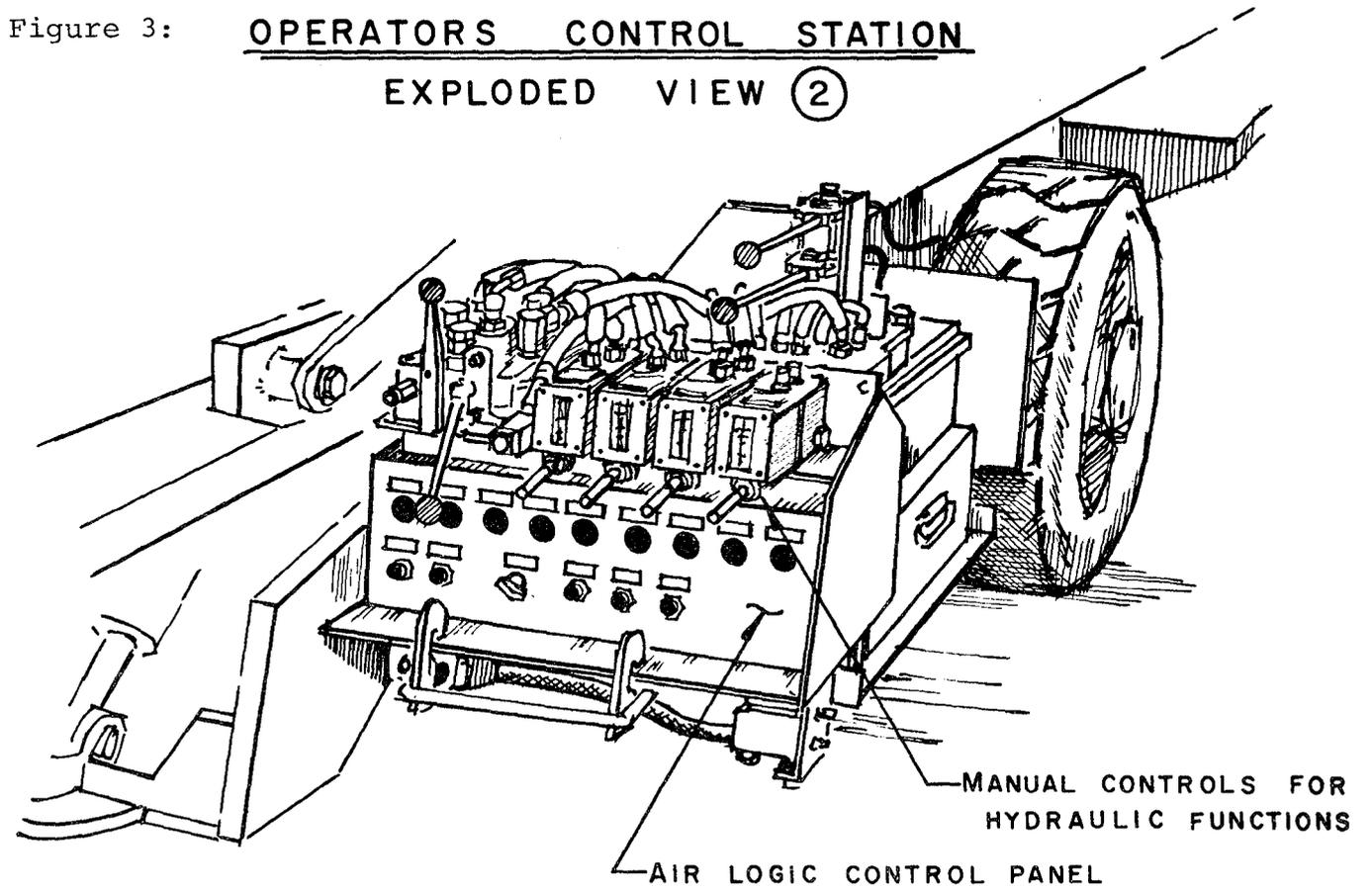
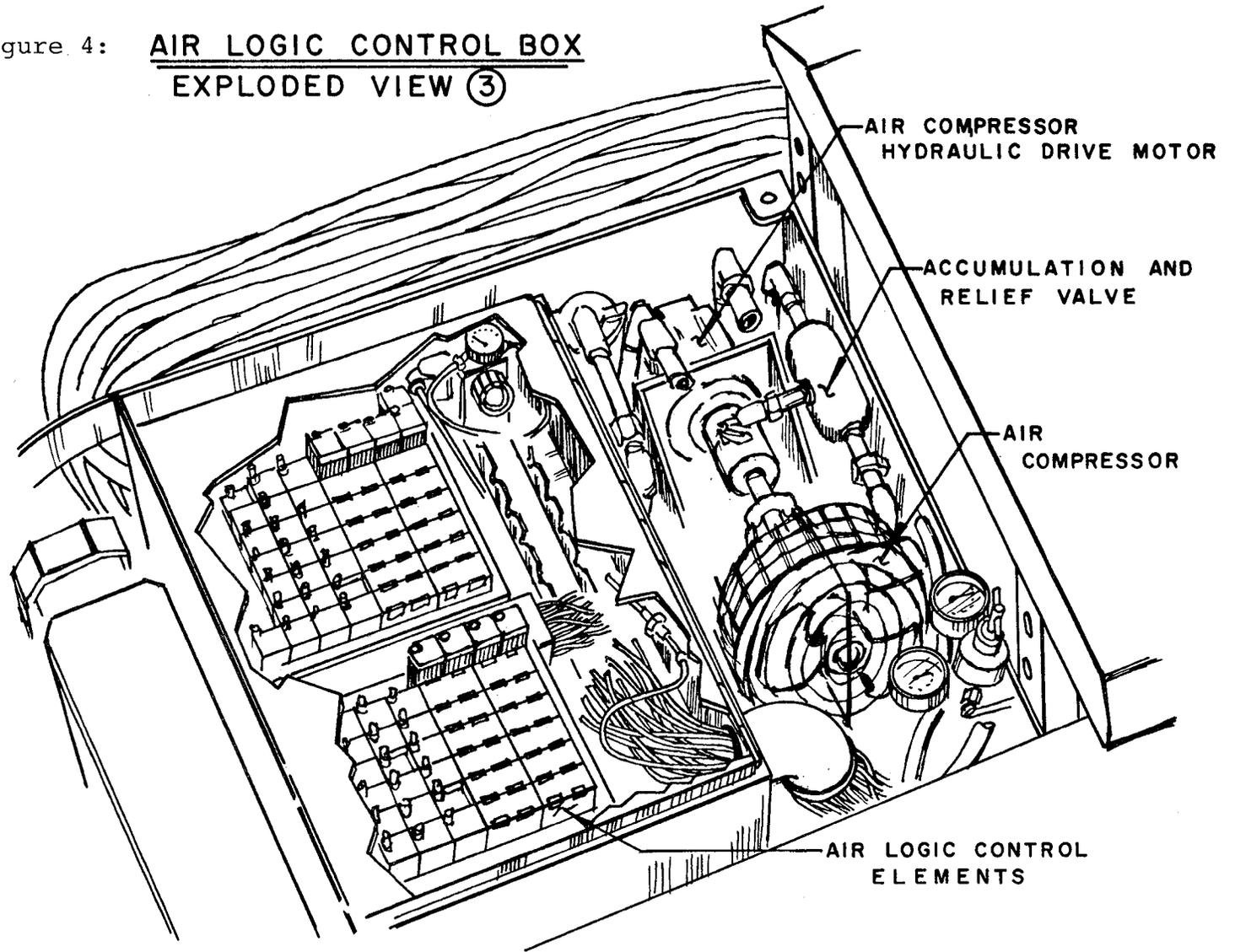


Figure 4: AIR LOGIC CONTROL BOX
EXPLODED VIEW ③



unload cycles use moving part air logic circuits to control mechanism functions and positions. Limit switches, also using air pressure, detect and control the movements and positions of the drill boom and carousel.

The carousel carries a beginning, or lead drill steel (consisting of a drill bit, chuck, drill steel, and coupling) five middle steels (consisting of a drill steel and coupling). To drill an 8-foot hole, all steels must be used. A shorter hole can be drilled by using fewer middle steels. The rod changer is capable of engaging the lead steel plus five or less middle steels.

The rod changer has two prime modes of operation: load and unload. Load is selected to drill the hole. Unload is selected to remove all drill steels from the hole just drilled. The load cycle is semi-automatic; it will position a steel beneath the hole, switch to manual control of drill boom lift and thrust rates, then return to automatic control to couple another steel. The unload cycle is completely automatic. Initiating the unload sequence will remove a steel from the hole. It will store the steel in the carousel in preparation for removing another steel. This is repeated until all steels are removed.

Manual control of the drill boom lift and penetration rates in the load cycle is provided to allow the machine operator to adjust for varying roof conditions.

3.0 Summary

The combined test period, comprised of five (5) months of aboveground test and evaluation (Phase IV) followed by a two and a half (2-1/2) month underground test (Phase V), has demonstrated that a "rod changer" type of low coal drill can successfully function in a mine environment.

The summary of the test will be categorized into three groups:

1. Results
2. Conclusions
3. Recommendations

3.1 Results

1. The Rod Changer - Flexible Drill operated underground for 502 holes without a major malfunction (see Section 17).
2. A total of 32 minor failures occurred during the underground test. The repetitive minor failures occurred in the following areas:
 - a. Carousel Ratchet Key
 - b. Drill Steel Coupling
 - c. Lower Limit Switch Lever Arm
3. The mean time between failures was 16 holes.
4. The longest time between failures was 149 holes.
5. The Rod Changer did not reduce drill penetration capability: torque and thrust are equal to the capacity of a commercial manual bolter.
6. The Air Logic control system functioned for seven and a half months without malfunction - 3,000 rod changing cycles.
7. The upper clamps have operated for 2,000 cycles without failure.
8. The lower clamp has operated for 2,000 cycles with 3 failures.
9. The drill centralizer has operated for 2,000 cycles without malfunction.
10. The carousel gate has been operated for 25,000 cycles without malfunction. This includes a cycle test at IRRI prior to Phase V.
11. The drill bit life has increased by 300% over the comparable life of the drill bit on the manual bolter (see Section 15).
12. The air compressor system operated for 850 hours without evidence of malfunctions.

13. The modified hydraulic system has functioned for 7-1/2 months, 850 operating hours, 1,000 holes without malfunction.
14. The Rod Changer was approximately 50% slower than a comparable manual bolter.

3.2 Conclusions

The following is a list of conclusions based upon the test results:

1. The Rod Changer system successfully drilled holes in an underground environment.
2. The air logic control system can function in an underground environment when properly maintained.
3. The Rod Changer in the current configuration increases the drill cycle time.
4. An untrained miner can be taught to operate the equipment in less than 4 hours.
5. The Rod Changer increases the drill bit life by a minimum of 300%.
6. The Rod Changer will increase the operator safety during roof bolting.
7. The repetitive Rod Changer malfunctions are all related to "misalignment" caused by racking of the safety arm.
8. The carousel drive, centralizer and air limit switch mountings require redesign.
9. The Rod Changer can be developed to be competitive with the manual bolter yet increase operator safety.
10. The Rod Changer does not reduce the drill penetration rate since torque and thrust are maintained.
11. The Rod Changer could be used with hydraulic rotary percussive drilling as required in severe roof drilling conditions, i.e. sandstone or quartzite.
12. The Rod Changer can be adapted drill either 1" or 1-3/8" diameter holes.

3.3 Recommendations

1. The Rod Changer development program should be continued.
2. The Rod Changer design program should include:
 - a. Provisions for remote operation of the bolt bender.
 - b. Provisions for drilling with either rotary thrust or rotary percussion.
 - c. A more rigid mounting frame; such as a mast in lieu of the pantograph arrangement.
 - d. Provisions to convert from 1-3/8" to 1" diameter drills.
 - e. "Family" of drill steels in which the drill steel length is matched to the seam height to minimize the number of "rod changes".
 - f. An "automatic cycle" control feature.
 - g. Improved Centralizer, upper clamp arms, and lower clamp.
 - h. Greater capacity dust and water filters for the air logic supply system, including an automatic moisture dump valve.
 - i. A larger capacity dust collection blower to compensate for the losses across drill steel couplings.

4.0 Description of the Nicholas No. 81 Mine

The IRRI Low Coal Roof Drill was tested at the Nicholas Mine No. 81 which is owned and operated by the Bethlehem Mines Corporation, a division of the Bethlehem Steel Corporation. The mine is located in Drennen, West Virginia, Nicholas County, approximately 60 miles southeast of Charleston, West Virginia. No. 81 mine was originally opened in 1952 and operated until 1957 when it was closed. In 1974 Bethlehem reopened the mine.

The coal seam is part of the Big Eagle Seam, a high quality bituminous coal seam of varying thickness from 32 inches to 8 feet. The coal is high volatile bituminous "A" metallurgical grade with a low sulphur and ash content. The seam thickness in the area of the test section varied from 46 to 56 inches. The primary factors in the selection of the mine were:

1. The coal seam height - the 56 inch height was the upper design limitation for the Flex Drill vehicle.
2. The relatively safe mine conditions. During the field trial there were no roof falls within the mine.

The test mine is a drift mine entered by either walking approximately one half mile to the face or riding the personnel carrier. Because of the relatively short distance to the face of the mine, transportation of parts and supplies presented no problems. The mine main entry was higher than seam height, to afford sufficient clearance for laying track and the main belt haulage system.

When the Flex Drill field trial began the mine was operating only one section and developing other areas normally associated with the opening of a new mine. A second section was started during the period of the field trial. The two operating sections provided a total of eleven entries. The priority of developing sections and maintaining production levels often resulted in insufficient electrical power to permit operation of the test vehicle.

However, the mine supervision and all general mine personnel extended themselves and their equipment to accommodate the requests made by the Ingersoll-Rand Research test crew to complete the test.

The production mining cycle required a four foot mechanical roof bolt to secure the top. The roof surface pitched one to two degrees, and was comprised typically of 10-15 feet of sedimentary rock randomly interspersed with a 2-3 inch layer of precipitated limestone.

The floor conditions presented no major problems for the vehicle. The bottom was similar to the roof in that it had a slight negative pitch of one to two degrees.

The current production capacity was 1500 tons of raw coal netting 1000 tons of clean coal produced through a two shift, two section operation. The mine employed a total of 53 people. The typical section consisted of four entries, however, it was anticipated that a section will ultimately develop five entries.

The production goal was two million tons per year (2 mty) by 1982 by means of 24 unit shifts per day with an average of 12 sections operating per 8 hour shift. The estimated daily production within five years was 8300 clean tons per day on a two shift operation. A third shift will be utilized for maintenance operations as the mine develops.

5.0 Operator Adaptability

The favorable reception of the Flex Drill concept into the commercial market must occur from the mine operators, mine workers and labor unions. The mine operator's concern is to maintain current production levels and to increase productivity with new machines. The mine worker's concern is primarily safety plus vehicle reliability, speed and overall mine worthiness. The prototype design was less productive compared to the present bolter; however, a competitive cycle time is possible. The drilling and bolting of a hole from the rear of the vehicle is the most desirable aspect of Flex Drill Rod Changer concept.

After the completion of the field trial the Bethlehem miner assigned to the Ingersoll-Rand crew for safety requirements was trained to operate the vehicle. The miner was instructed in the machine operation as follows:

- a. Purpose of the vehicle
- b. Location of manual controls
- c. Normal sequence of control lever operation
- d. Corrective procedure sequence for cycle failure
- e. Normal machine sequence of operation.
- f. Observation of test drilling

With this training the miner developed sufficient competence to operate the bolter without assistance.

The major problems encountered with the vehicle were:

- a. Coupling of the drill steels in the load portion of the cycle.
- b. Coupling of the drill steels in the unload portion of the cycle.
- c. Boom raise and lower to reduce coupling problems.

It was shown that the miner was able to master the basic machine sequence in approximately five holes of operation and could complete the necessary coupling corrective actions within ten holes. The remaining cycles improved his

knowledge of the machine and further reduced the cycle time. The intangible factors affecting a miner's performance on a new bolting machine are as follows:

- a. Previous bolting experience
- b. Positive attitude
- c. Willingness to learn the machine

The training of the miner to operate the Flex Drill was not given serious consideration prior to the field trial. In the future, when a new machine is purchased the mine worker could be taught the operation of the vehicle by a combination of the following:

- a. A slide presentation
- b. Operator's handbook
- c. Introductory training period

The acceptance of air logic controls by the mine servicemen will ultimately depend on a modular concept similar to the electronic printed circuit board. It is doubtful that the mine servicemen will be able to troubleshoot air logic circuits within the mine environment; therefore, easily replaceable modules are desirable. In order for the mine technicians to service a vehicle with air logic, a service school covering the following points could be considered:

- a. Troubleshooting methods
- b. Assembly and disassembly of air logic components.
- c. Pamphlets for each miner to review at his leisure.

The above training would enable the mine servicemen to become familiar with the vehicle before working on the vehicle under adverse conditions.

6.0 Mechanical Modifications to Standard Lee-Norse Bolter

The IRRI Flexible Roof Drill was installed on a Lee-Norse TD1-29 Single Boom Roof Bolter, shown in Figure 5. In order to accommodate the new system, several changes were made to the basic vehicle frame and components.

6.1 Changes Made During Vehicle Fabrication at Lee-Norse

- a) The standard mounting frame was modified to allow correct positioning of the radius arms, which are part of the safety arm linkage. Relocation of these mounting pins allows the new safety arm to track in the desired path. Refer to drawing no. LN-09-99-03-01-073.
- b) The standard chassis frame was modified to allow clearance for the new safety arm. The modification consists of merely cutting material from the side chassis. Refer to drawing no. LN-09-99-03-01-078.
- c) The standard drill box mounting frame was modified to provide clearance for the upper clamp and centralizer arm assembly. The modification consists of merely cutting away material from the standard drill box support. Refer to drawing no. LN-09-99-03-01-079.
- d) The Lee-Norse cable reel was deleted from the IRRI bolter. The space occupied by the reel was necessary for mounting the air logic control box and air compressor. The cable was hung at the rear of the vehicle and is wound on two U-shaped brackets.

None of the above changes involved extensive modification of the standard vehicle. When done during vehicle construction, the additional cost to make the changes is small.

6.2 Changes Made to the Standard Vehicle at IRRI

- a) Radius Arms. The radius arms supplied with the bolter were replaced with IRRI design no. LN-09-99-03-01-033. This change involves only a dimension modification. Cost of the IRRI radius arm should be equal to that of Lee-Norse.
- b) Safety Arm. The safety arm supplied with the bolter was replaced with IRRI design no. LN-09-99-04-01-013. The new arm is designed to provide the proper mounting for the front chassis assembly, the pantograph stabilizer assembly and a modified rear tracking assembly. The new safety arm conforms to all MESA safety standards and provides for mounting a standard Lee-Norse safety canopy.
- b) Drill Clamp Assembly. The standard drill clamp and associated hydraulic components are no longer needed with the Flex Drill. The components mounted on the gear box were removed as an assembly and replaced with the IRRI design no. LN-09-99-03-01-027.

The above changes to the standard vehicle were made at IRRI prior to assembly of the Flex Drill components. They could very easily have been made at Lee-Norse during construction of the vehicle.

7.0 Air Logic System

7.1 Methods of Control

Automatic control of underground mining equipment can be accomplished by several methods:

- a) Electric control (solenoid operated hydraulic valves) using explosion proof components,
- b) Electronic control (microcircuit control of hydraulic valves) using intrinsically safe components,
- c) Fluidic control systems,
- d) Moving part air logic (MPAL) circuit control.

IRRI opted to use moving part air logic control of the Flex Drill for several reasons:

- a) MPAL systems are inherently free of explosion protection questions.
- b) The state-of-the-art has been advanced to the point where MPAL components undergo millions of operations with failure.
- c) A constant and reliable power supply system (an air compressor) can easily be mounted on the vehicle.
- d) Mechanism positions can be detected with air limit switches in the same way electrical limit switches are used.
- e) MPAL components are readily available for field repair.
- f) The system is relatively low in cost and provides extremely reliable operation.

The chief disadvantage in using explosion proof mechanisms is the excessive bulk and weight of protective boxes. The Flex Drill design could not provide sufficient room for these components.

The two main problems with using intrinsically safe components are:

1. A very stable and pure DC source must be provided. The use of 550 V-AC mine power with appropriate rectifiers, filters and regulators is questionable for this application.
2. IRRI and the mobile equipment industry in general have little experience with such systems.

Fluidic control systems were rejected due to the experience industry has had in general with such systems. Considerable development is necessary before true fluidic control of mechanisms can be considered comparable to other methods.

7.2 Description of the MPAL System

The MPAL circuit uses only six types of logic components: AND, OR and NOT gates, flip-flops and two types of timers. The function of each is identical to the comparable electrical component and will not be described here.

Basically, the circuit is designed so that a given function must be completed before the next can begin. Completion of a function is detected by a limit switch, which in turn activates the appropriate logic components, allowing initiation of the next function, whether it be under manual (push button) or automatic control.

The logic components then provide an air signal to the appropriate hydraulic valves. These, in turn, apply power to hydraulic cylinders which move the mechanisms.

Power for the logic system is provided by an oil-less, open crankcase piston compressor. Three stages of air/oil/dirt filtration are provided for maximum protection. The compressor operates against a constant 70 psig head. Pulsations are dampened with a small (1 pint) receiver. Power to the compressor is supplied by gear pump and motor driven off the main 40 HP electric motor. The compressor has proven exceptionally reliable in eight months of above and underground testing.

7.3 Sequence of Operations

The IRRI Flex Drill has three modes of operation:

- a) Load Cycle: semi-automatic operation used to drill the hole.
- b) Unload Cycle: automatic operation used to remove drill steels from the hole and store them in the carousel.
- c) Bolting: manual operation of the Bendix Roof Bolt Inserter.

Refer to Figure 6 for the layout of the operator's control panel.

To drill a hole, the operator selects "load" with the panel selector switch. Pressing the "feed" button swings the carousel over the drill, indexing a steel to the proper position as it moves. When the "couple" light comes on, the operator presses the "couple" button. The boom begins to raise and the drill rotates slowly. When the chuck has coupled with the steel, the boom stops moving and the drill ceases rotating. The "stow" light comes on. The operator then presses the "stow" button. The carousel moves back to its stowed position. The "drill" light comes on. The operator now has control over both drill rotation and boom thrust. He can now drill until the boom reaches the upper limit. At that point, manual upward boom thrust is shut off and the "lower" light comes on. The operator can now drop the boom to its lowest position. The "lower" light goes off, the air logic system is reset and the "feed" light comes on. If the operator desires to drill further, he repeats the above cycle with another drill steel until the hole is of the desired depth.

To remove a steel from the roof, the operator, after completing his last "load" cycle with the boom still at the roof, selects "unload" with the panel selector switch. The "retract" light comes on. The operator presses the "retract" button. The boom drops to two prescribed positions, the carousel rotates in to store the steel, returns to its stowed position and the boom drops to the lower limit. The "couple" light comes on. The operator presses the "couple" button. The boom raises and the drill rotates slowly. The drill couples to the next steel at the

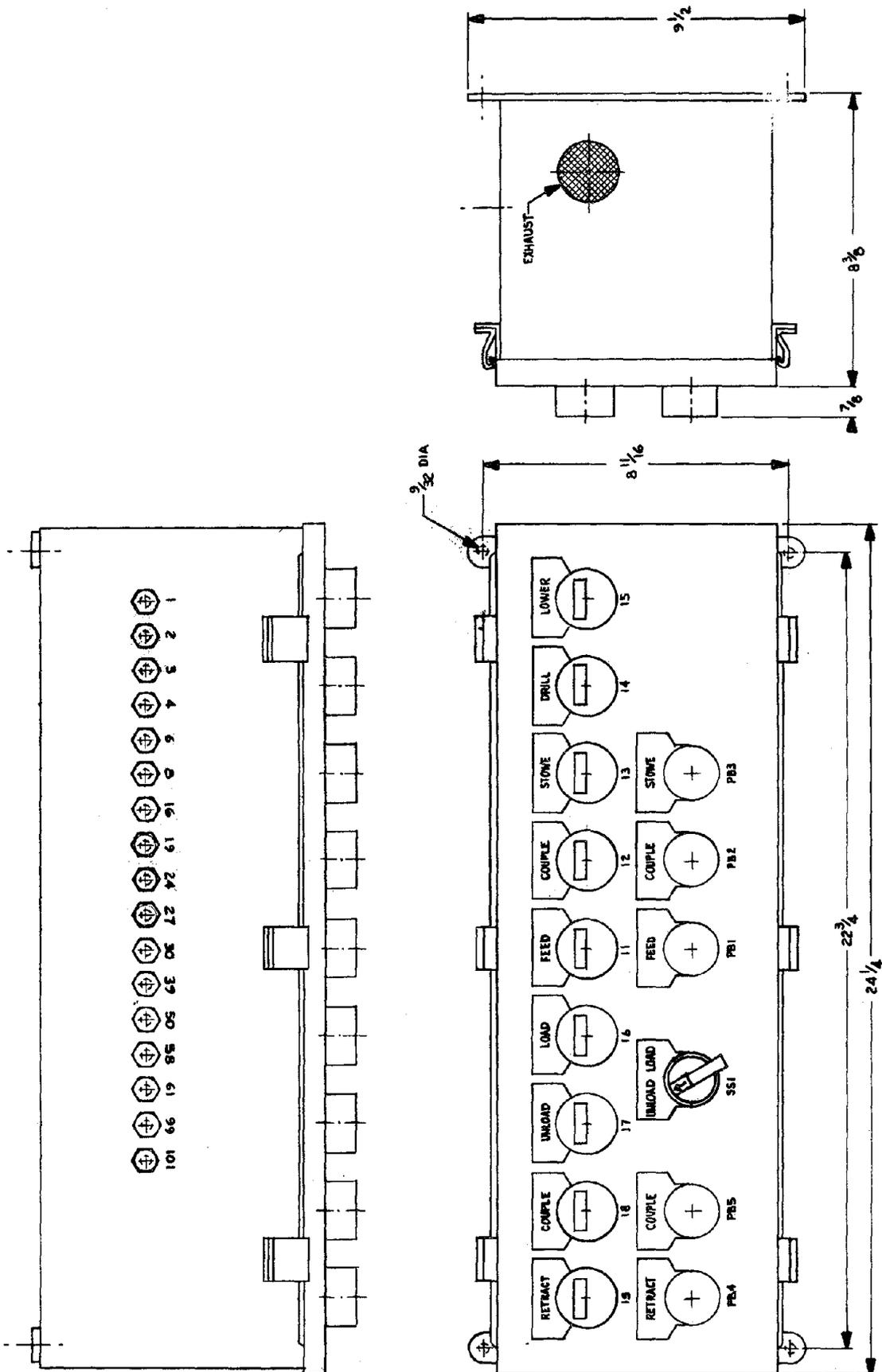


Figure 6: Operator's Control Box

roof. The "retract" light comes on and the operator repeats the above cycle until all steels are removed from the hole.

The bolting cycle is not under control of the air logic circuit and thus is not described here. Refer to Section 21.1 for details.

7.4 Summary of Test Results

The Flex Drill was tested both above and below ground. Approximately 500 holes (averaging 4 ft. in length) were drilled in each case.

At no point did the MPAL system experience a major failure. All problems associated with tripping limit switches were mechanical in nature and can be resolved by changes in these mechanisms.

There are two areas that must receive close attention when using an MPAL system:

- a) Every effort must be made to keep dirt and water out of the system. The system design must include more than "normally adequate" filtration to survive the mine environment. Changing and cleaning of filters on an established basis is essential.
- b) Servicing the MPAL circuit must be done by a competent mechanic who is well trained in troubleshooting and repairing the entire circuit.

7.5 Recommendations

- a) Short Range: The air logic circuit performed exactly as it was designed to. With proper alignment of the mechanical components of the Flex Drill, problems in reliable tripping of the air limit switches can be eliminated.

The "load" cycle can be made completely automatic with relatively little engineering effort in the circuit design. Such a revision will greatly reduce the number of operator controlled operations. This will reduce the overall cycle time appreciably.

There are several operations in both the "load" and "unload" cycles that are presently sequential in operation. It is possible to make them concurrent and further reduce cycle time.

In order to reduce the possibility of system contamination due to failure to adhere to a prescribed maintenance schedule, an air/moisture separator with automatic water dumping should be used in the compressor output line.

The plastic tubing used to interconnect all control boxes and limit switches should be completely protected from rock falls, mechanism contact, and operator abuse. Enclosing all lines in protective steel casing will greatly reduce the possibility of line damage and contamination.

- b) Long Range: The ultimate goal of the Flex Drill program is to allow completely automatic bolting of low seam coal mines. The operator must be removed from the area under unsupported roof. The air logic circuit should be redesigned along with the mechanical systems to allow complete drilling and storage of drill steels for any preselected hole depth. Roof bolting should be accomplished in the same manner: providing a magazine to store and dispense roof bolts for insertion into the roof. Air logic can accomplish the motions necessary for such a system. All of these operations can and should be carried out from the rear of the vehicle.

8.0 Carousel

8.1 Carousel Capability

The Bureau of Mines Contract to IRRI for development of a low seam roof bolting system called for the ability to drill an 8 foot hole in a 30 inch seam. The IRRI Phase I study recommended use of short lengths of drill steels to be coupled together to meet this requirement. Such a system requires a means of storing a number of drill steels, dispensing them upon demand and re-storing them after the hole is drilled. The result is the carousel shown in Figure 2.

The carousel stores six drill steel assemblies. Each steel can drill 18 inches; thus, a maximum hole depth is 9 feet. The carousel and resulting control circuit allow the use of from one to six drill steels. Thus, hole depths in increments of 1-1/2 feet from 1-1/2 feet to 9 feet can be attained.

The carousel concept has two additional features which are very desirable in underground operation:

- a) The operator can drill a hole longer than the length of the roof bolt. Federal regulations require that in any new section about to be bolted a hole one foot longer than bolt length be drilled to test the condition of the "top" where the anchor will seat.
- b) The IRRI Flex Drill can actually drill holes much longer than 8 feet. All the operator need do is insert additional lengths of steel by hand to reach the desired hole depth. In the underground test, a ten foot hole was drilled by this technique.

The carousel pivoting motion is accomplished by a hydraulic cylinder mounted to the pivoting arm and the front chassis. Cylinder operation is controlled by the air logic circuit.

Indexing of the drill steels is accomplished by a ratchet-pawl mechanism linked to a tie rod anchored to the front chassis. As the carousel moves, the tie rod forces the pawl to engage the ratchet. Thus, the internal shaft and plate assembly, which actually carries the drill steels, is rotated relative to the carousel body as the carousel

moves through its swing. The swing of the carousel, the length of the tie rod and the position of the pawl and ratchet are all designed and adjusted to insure the drill steels are in the proper positions at each end of the carousel swing.

The ratchet pawl mechanism functions in both the load and unload mode. In load, the steels are indexed as the carousel moves toward the drill. In unload, the reverse occurs: the drill steel is indexed to a stored position as the carousel swings away from the hole. Reversal of the double ended pawl is accomplished by a small short stroke hydraulic cylinder mounted on the ratchet housing. Operation of this cylinder is controlled by the air logic circuit.

8.2 Test Results: Aboveground

Both above and below ground testing of the carousel have proven its basic concept. Approximately 1000 holes averaging 4 coupling operations in both the load and unload modes have been drilled with a single carousel. In addition, considerable "dry" cycling of the carousel was done before drilling was attempted. Thus, the carousel has withstood approximately 10,000 operating cycles and is still fully operational.

Actual drilling has shown several weak areas in the design of the carousel. Each of the following areas can be improved with minor redesign and will result in a much improved system:

- a) Gate Mechanism: During the aboveground test, the original cam operated gate mechanism was judged unusable. A new gate design using an air logic controlled hydraulic cylinder pulling a flexible cable was added. Extensive testing of the gate resulted in the system shown in LN-09-99-03-01-081.

Originally, a common 1/16 inch diameter "motorcycle" cable was used. Failure of the cable occurred after approximately 100 operations. Substitution of a 3/32 inch aircraft quality cable solved all fatigue failure problems. A 10,000 cycle bench test was successfully completed. This system has undergone approximately 20-25,000 operations to date without failure.

A gate similar to the present design, but considerably longer should be used to insure proper seating of the drill steel in the carousel. The majority of carousel failures resulted from a steel being in a "cocked" position as the carousel rotated. Failure of the ratchet key occurred. An increased preload force by the gate on the steel should be used to insure proper seating.

- b) Ratchet Pawl Reversing Cylinder: The present mounting arrangement of the cylinder on the ratchet housing is not mine-worthy. The cylinder extends horizontally beyond the carousel housing. Any rock fall or operator abuse can easily damage it. Also, any excessive loads on the hydraulic lines supplying the cylinder can easily bend the cylinder

rod and associated components. The solution of these problems will require a remounting of the cylinder in an area not susceptible to abnormal loads. The linkage coupling the cylinder to the pawl must be redesigned.

- c) Slip Link: A fail-safe "slip link" must be incorporated at some point in the carousel drive train to prevent mechanism failure in the event of a system cycle mishap. Such a link, together with the limited amount of force the pivoting cylinder can apply, will considerably reduce downtime due to carousel failure.

- d) Shaft and Plates Assembly: The shaft and plates assembly which stores the drill steels should undergo a material and heat treatment review. Excessive wear of the plate cutouts where the steels are stored was noted throughout the tests although no failure occurred.

- e) Coupling Lubricator: A drill steel coupling lubricator should be part of the carousel design. Thorough testing of the coupling action revealed that a slight lubrication of the couplings greatly increased coupling reliability. A wiper type long life lubricator could accomplish this.

- f) Test Results - Underground: The underground testing substantially confirmed earlier predictions for life and wear in the carousel assembly. The two parts in the carousel assembly requiring replacement were the carousel ratchet key and the ratchet cylinder. The carousel ratchet key was replaced fifteen times while the ratchet cylinder failed once. The improvements to reduce or eliminate the carousel problems are contained in the recommendations.

8.3 Recommendations

a) Short Range:

The problem areas described above can be resolved with a minimum of design effort. No major revisions of the carousel design are required, and cost will be low. These revisions, coupled with the changes recommended in the other areas, will greatly improve the reliability of the existing system.

b) Long Range:

Eventually, a "family" of carousels should be developed to tailor carousel length to the mine height conditions. Use of a carousel with the fewest number of steels necessary to drill the desired hole depth will minimize cycle time and ultimately increase productivity. A universal carousel mount could be designed to allow "plug-in" of the optimum length carousel.

The long range redesign of the Flex Drill should use a drill mast in place of the current pantograph linkage drill boom. A carousel using a short stroke linear rather than angular motion could be incorporated in its design. Carousel movement can be minimized to reduce overall cycle time.

The Flex Drill vehicle height is limited by the carousel design. Future models might include a method of pivoting the carousel to a horizontal position to allow a decrease in overall vehicle height. Such a change would allow for easier tramming in a wide variety of coal seam heights.

9.0 Drill Steel Components

9.1 Design Parameters

The drill steels used with the Flex Drill were supplied by Kennametal, Inc., Mining Tool Group, Bedford, Pennsylvania. The components of the steels are:

- a) Drill bit: Standard Kennametal design 1-3/8, oval shank.
- b) Chuck: Standard Kennametal design, type VCH with male hexagonal pilot.
- c) Extensions: Standard Kennametal design, internal hexagonal configuration, types DEV-M-1159-1 and -2.
- d) Special VCH coupling: designed jointly by Kennametal and IRRI. Consists of a double male hexagonal pilot and barrel spring, type DEV-M-1159 (See Figure 7).

Refer to Figures 8 and 9 for details of both the "lead" and "middle" steel assemblies.

The drill steel was designed to satisfy the following:

- a. To transmit full drill torque, approximately 300 ft.
- b. To transmit full boom thrust (5500 lb.) without appreciable deflection.
- c. To provide adequate cross sectional area for dust collection.
- d. To minimize vacuum loss across couplings.
- e. To insure that drill steels could easily be coupled to the drill chuck.
- f. To provide a means of coupling drill steels together without human assistance.
- g. To insure that drill steels remain coupled while being removed from the drill hole.

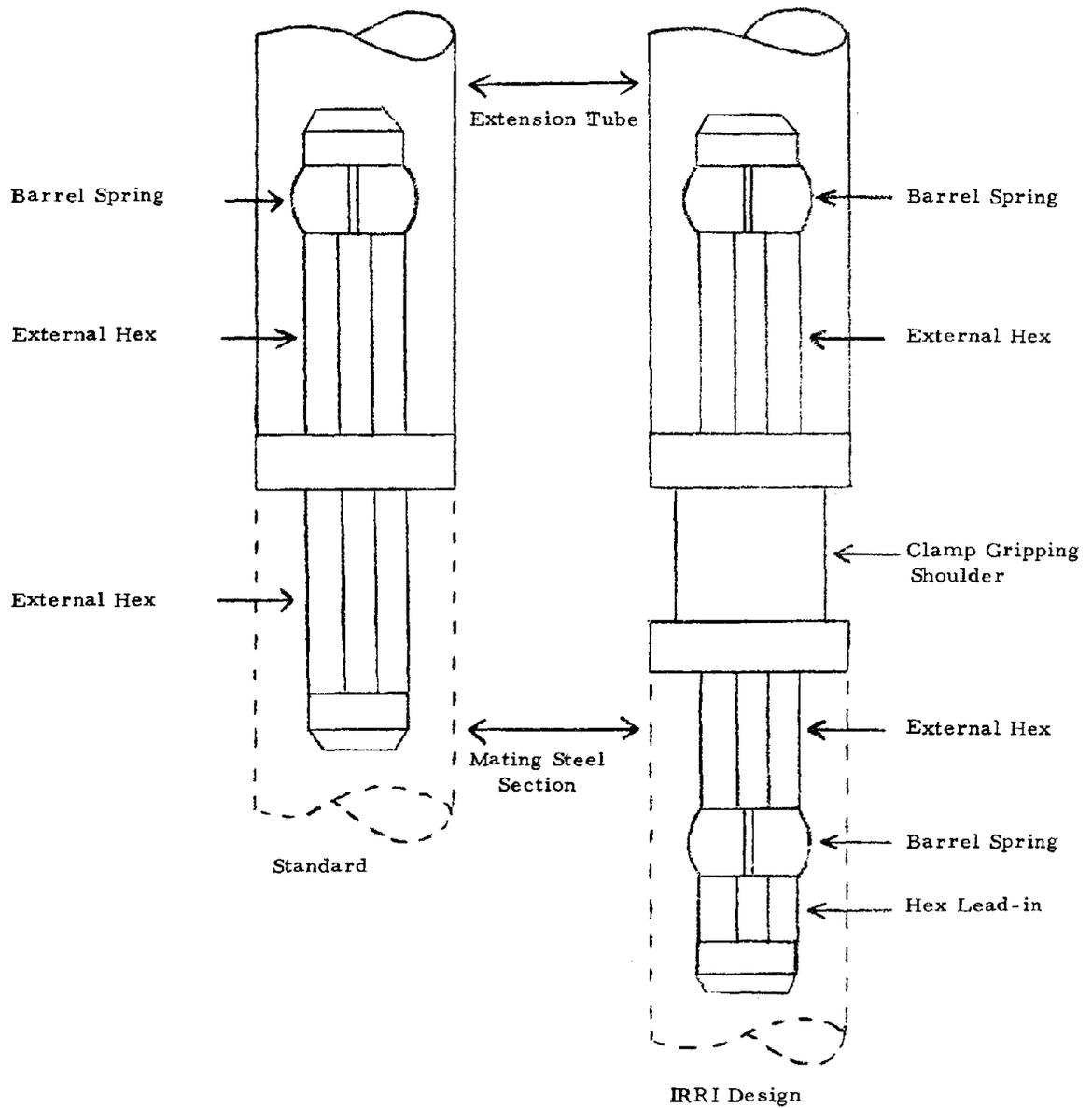


Figure 7: Comparison of Standard and IRRI-Designed Drill Steel Couplings

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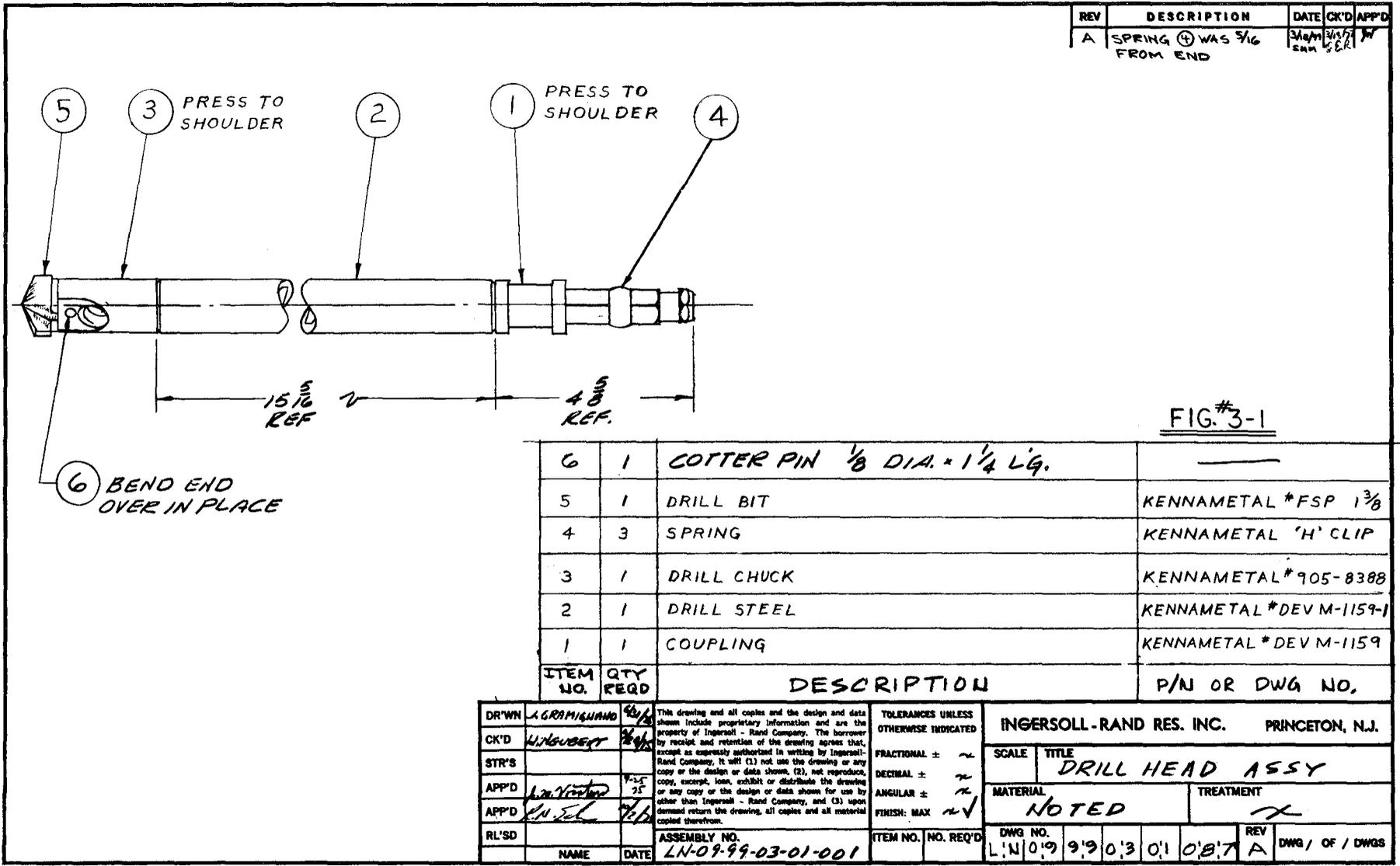


Figure 8: Drill Steel Coupling - Lead Steel

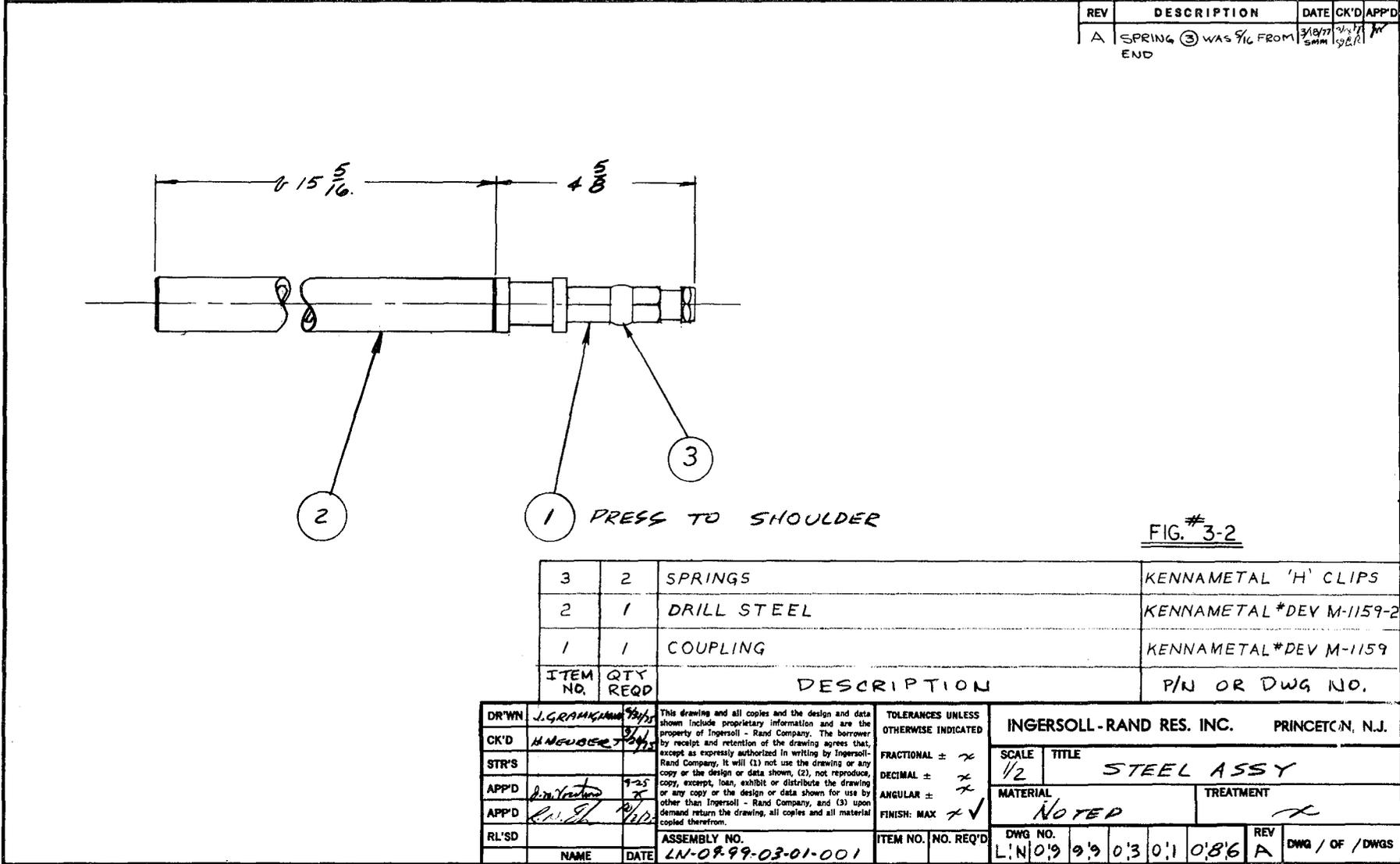


Figure 9: Drill Steel Coupling - Middle Steel

- h. To provide a bit and steel life and cost comparable to materials now in common use.

9.2 Test Results

There are four areas that have received very close attention in both design and aboveground testing:

- a. Coupling forces provided by barrel springs:

The springs must exert sufficient force on the mating drill steel to insure the integrity of the string while being removed from the hole, yet the force needed during coupling must be within the limits of the clamping arm design. Standard Kennametal barrel springs are used. Spring force is adjusted by changing the undercut spring retainer diameter: a larger diameter results in a larger coupling force. Tests were run at IRRI to determine the optimum coupling force. Refer to Appendix, Section 21.11 for details.

- b. Location of Spring on Male Hexagonal Lead-in:

As originally designed, the barrel spring was piloted into the female tube by a short circular chamfered section. Aboveground testing at IRRI revealed the need to provide a hexagonal lead-in before compression of the spring. With this design, the steels are not rotating relative to each other while the spring is being compressed. If relative rotation exists, excessive axial force is required to complete the coupling operation. Roughly 1/2 inch of hex lead-in is provided.

- c. Internal Hex Lead-In:

The female hexagonal tube requires a very generous lead-in to properly pilot and align the mating section. A 15° angle to the axis of the tube was machined on the "internal" base circle to minimize coupling forces.

d. Drill Chuck Design:

Aboveground tests at IRRI revealed the need to provide a modified hexagonal shape to increase coupling reliability. The standard hexagonal shape was modified to a "surface-drive" configuration, shown in LN-09-99-03-01-100A. Coupling reliability, with proper alignment, reached 98.5% in the aboveground test - (15 failures in 1044 attempts).

Incorporation of the above changes resulted in a very high coupling reliability when all mechanisms were properly aligned. However, misalignment was a consistent problem throughout the test phase.

Mechanism misalignment created the following problems with the drill string:

- a. Excessive wear of the male hex on the couplings.
- b. Inability to reliably couple steel sections together.
- c. Breakage of couplings.

To some degree, wear can be reduced by the selection of a more suitable coupling material and heat treatment. Also, periodic lubrication of the coupling greatly increases the probability of a successful coupling operation. In a redesign of the coupling these areas can be analyzed and a more effective, long-lasting coupling can be used.

Breakage of several couplings occurred only in the underground test. Five months of aboveground tests using a level floor and "roof" produced no coupling breakage. Uneven floor and roof conditions underground amplified the effects of misalignment and coupling breakage occurred. Again, eliminating misalignment will greatly increase reliability.

The coupling will wear with repeated use. With the current misalignment problems, a set of couplings were replaced after 150 four foot holes. However, replacement of the entire drill string is unnecessary. Merely replacing the coupling is required.

An important aspect of the economic analysis is the cost of the drill string. The "Rod Changer" drill string is standard Kennametal except for the modified drill steel coupling which will be approximately 20% more expensive than

a standard coupling, \$8.00 and modified \$9.60. One coupling is used per rod section in addition to the drill tube.

If one assumes two couplings required per four foot hole and a life cycle of 250 holes after improvements, the cost of couplings is \$19.20/250 holes or \$0.08/hole. This is an insignificant cost when one considers tha the cost per hole for the roof bit varies from \$1.38/hole to \$2.07/hole (Refer to Section 16) for manual bolting. Only the coupling need be replaced regularly, not the drill steel tube.

In conclusion, the majority of problems with the drill steel sections can be eliminated by insuring the proper mechanism alignment. Several changes should be made to the drill steels to further increase reliability. With large volume production, these steels will be competitive both in cost and performance with presently used drill steels.

9.3 Drill Steel Coupling Recommendations

The drill steel coupling wear can be reduced and the coupling life extended by incorporating the following modifications:

- a. Heat Treatment - The wear observed on the leading edges can be reduced by localized induction hardening of the primary contact area. The depth of the hardness should be .010 to .020 for the initial design and varied after evaluating several test couplings. The present material is 4140 steel with a hardness of Rockwell C40-45. To determine the proper hardness value for the least amount of wear before coupling failure, several drill steels with different hardness values should be obtained. It is recommended that several samples be hardened to "Rockwell C50 and 60;" with this range of hardness values the best heat treatment can be determined. To decrease the possibility of chipping the hardened surface edges, a low temperature should be applied to the hardened area. The temper may be applied only after several chipping failures have occurred so that the additional heat treatment will not be randomly applied without previous experience.

- b. Design - 1) Spring Location - a few recommendations with the coupling design should help extend the overall coupling life. The first would be the shortening of the hex lead-in between the leading edge of the steel and the groove machined for the barrel spring. The failures of the coupling encountered during the field trial were at the groove machined to accept the barrel spring. Moving the barrel spring closer to the end of the coupling (minimizing hex lead-in) will reduce the moment arm to the failure area. The reduction in the moment arm will allow for a higher force to be applied to the coupling for an equivalent moment.

The original drawing of the drill steel couplings supplied by Kennametal indicates a radius of 0.010 of an inch in the groove for the barrel springs. This radius reduces the high stress concentration at the sharp edge of

the coupling. The failures encountered in the field are with the couplings remachined by IRRI which involved cutting a new groove for the barrel spring. The small .010 radius was inadvertently omitted from the new drawing and not machined into the coupling.

10.0 Clamping Mechanisms

10.1 Upper Clamps

The upper clamping arms function in both the "load" and "unload" drilling modes to allow coupling of the drill steel sections. They provide the reaction force necessary to overcome the force of the barrel spring on the drill steel coupling.

The arms are subject to high bending loads in both drilling modes. Drill boom supply pressure was reduced to a safe level during automatic operating; however, under manual control, a careless operator could damage the arms.

The cutouts on the arms which contact the drill steels are subject to excessive wear. When the cutout portions become rounded, the drill steel tends to open the arms. This defeats their purpose, and cycle operation is interrupted.

Only one set of clamp arms was required for the underground test. Several modifications involving material, heat treatment and arm thickness were made as a result of the thorough aboveground test at IRRI. This testing was a key ingredient in the success of this mechanism underground.

The upper clamp mechanism should undergo a design revision to provide greater drill steel gripping force. A different linkage design providing a greater "mechanical advantage" would accomplish this. Improved material specification and heat treatment, when combined with a properly aligned system, will solve the remaining problems experienced with the upper clamp arms. See the Appendix, Section 21.4 for further details.

10.2 Centralizers

The centralizing arm assembly consists of a scissors type linkage designed to guide the drill steels vertically as the drill boom raises and lowers.

Because of misalignment of the various assemblies on the Flex Drill, the centralizers were subjected to loads exceeding design parameters. While the arms did not fail, high wear was experienced.

The centralizers were designed to keep the drill steel vertical during the drill cycle. They did not, however, provide sufficient force to keep the "lead" drill steel from walking on the roof surface when beginning a hole.

The centralizer arm assembly should be redesigned to provide more of a "stabilizing" action than it is now capable of. The linkage should provide more of a mechanical advantage to withstand greater side forces without deflection at the drill steel. A material and heat treatment review would eliminate wear problems. See the Appendix, Section 21.5 for further details.

10.3 Lower Clamp

The lower clamp functions only in the "unload" mode. Its functions are to draw the steel out of the hole and provide the unlocking force for the steel-to-steel couplings.

The lower clamp is mounted directly over the drill box. A U-shaped bracket, driven by a hydraulic rotary actuator, flips up and locks on the drill steel coupling in the drill chuck. With the upper steel held in the upper clamp arms, dropping the boom separates the two steels.

The lower clamp required minor repair during the field test. The bracket should be made heftier and more "mine-worthy" in construction. The portion contacting the drill steel should be hardened to reduce wear. See the Appendix, Section 21.6 for further details.

11.0 Safety Arm

11.1 Safety Arm Design

IRRI selected a Lee-Norse TD1-29 roof bolter for mounting the Flex Drill system. This bolter uses a pantograph linkage drill boom and a hydraulically positioned safety arm. The safety arm is jacked against the roof for support. The standard machine used a two point rear roller, hydraulic cylinder linkage and side radius arms to control operation of the arm.

The chassis containing the various Flex Drill components is mounted on the front of a redesigned safety arm. The chassis is held in a vertical position by a pantograph linkage connecting the chassis with the rear roller assembly of the safety arm.

The IRRI safety arm was designed according to all MESA specifications of loading, material and construction. Analysis of the structure was done using the computer program "Canopy" developed by Steven G. Sawyer and the Bureau of Mines. Certification of the T-1 steel used in the safety arm is in possession of IRRI.

Thorough testing of the Flex Drill has revealed that dependence on the operation of the safety arm linkage in the standard machine is not advisable for an automatic rod changing drill. Built-in tolerances and the inherent nonlinearity of the vertical movement of the safety arm cause frequent and unpredictable misalignment to occur between the drill box and the carousel. For these reasons, a drill mast system is the proper way to further develop the Flex Drill system.

11.2 Test Results

Thorough testing of the safety arm both above and below ground revealed two problems inherent in the safety arm linkage concept. Each of these was a major contributor towards almost every failure experienced in the test programs.

a. Rear Track Pivot Points

The two point roller guide used on L/N bolters is not meant to be a close fit, precision tracking mechanism. IRRI recognized this and provided close fit bushings to reduce sloppiness between rollers and tracks. Any play in this system is amplified by a factor of 5 to the front of the machine and shows up in racking of the chassis. Imprecise positioning of the chassis disrupts proper tripping of the air limit switches. Cycle failure results. IRRI's 3 point precision roller system is not compatible with the L/N installed tracks. These tracks do not provide parallel surfaces on which the rollers can ride. Thus, chassis alignment is unreliable and cycle failure can easily result.

b. Racking of the Safety Arm

As described earlier, one function of the safety arm is to provide temporary roof support in the area to be bolted. The roof in a coal mine cannot be considered level; thus, the load on the arm can be off-center at any point along the width of the arm. The jacking force acting along the distance separating this point and the lifting point creates a moment which "twists" the safety arm. Any such force disrupts the alignment between the safety arm and the drill boom, causing improper air limit switch operation. Cycle failure usually results.

The above conditions account for the majority of problems in the operation of the Flex Drill. These, combined with unpredictable tracking of the drill boom linkage, make it very difficult to produce repeatable operation. The test program was completed by installing two temporary field corrections:

- a. A spring loaded cable was installed between the front chassis and vehicle frame. This spring strut applied roughly 400 pounds over a two-foot moment arm to hold the front chassis in position. Thus, chassis position was more predictable at any safety arm height, and limit switches could be tripped with much greater reliability. This solution, however, is unacceptable as a final design.

- b. "Contact pads" were welded on each side of the safety arm to insure the pads, and not the front chassis, were loaded when contacting the roof. This eased the alignment problem, but was not totally acceptable. Safety arm jacking pressure was considerably reduced to decrease safety arm racking. A better solution is a 1 point contact pad positioned in line with the lifting cylinder. This would guarantee a minimum moment arm and considerably decrease safety arm racking.

11.3 Recommendations

a. Short Range

The alignment problems with the existing bolter can be reduced by a careful rework of the rear roller-track mechanism, and providing minimum moment arm for racking, as described earlier. The tracks on which the rear pivot bushings roll should be matched to the bushing assembly and accurately installed on the machine. If necessary, provisions can be made to use an adjustable track which can be welded in place when proper alignment is achieved. Incorporation of these two changes will greatly increase the overall reliability of the Flex Drill system.

b. Long Range

Accurate control of the Flex Drill depends on accurate alignment of the carousel, drill head and handling arms. Development of a rigid structure using the mast concept will improve overall system performance by eliminating virtually all alignment inaccuracies. Such a system should use a vehicle whose power rating and hydraulic system are fully compatible with the drilling module.

12.0 Hydraulic System

12.1 Circuit Design

The IRRI Flex Drill uses a Lee-Norse roof bolter as the basic vehicle. This bolter drives two 35 GPM and one 10 GPM hydraulic gear pumps from the shaft of a 40 hp electric motor. The pump functions are as follows:

- a. Each 35 GPM pump provides power to a tramming motor.
- b. One 35 GPM pump has a power take-off to provide power to the drill box.
- c. The 10 GPM pump powers the safety arm, drill boom, cradle, and drill clamp.

The Flex Drill system disrupts this arrangement very little. In addition to providing drill power, one of the 35 GPM pumps also powers the Bendix Bolt Inserter. The remaining Flex Drill circuits require power from only the 10 GPM pump. These functions are sequential and not concurrent so that the pump size is adequate.

The open center arrangement of the 35 GPM valves is unchanged, and thus these components contribute little heat to the system when idle.

The Flex Drill hydraulic circuit was designed by Fluid Circuits, Inc., Spring House, Pennsylvania. The original circuit and components left much to be desired, however, and considerable circuit modifications were made before a properly functioning drill could be tested. The circuit contains several areas which defy commonly accepted hydraulic practices. However, the circuit activates all mechanisms properly and no problems were encountered with hydraulic component operation. The need to modify certain areas was recognized too late in the program to redesign the system.

12.2 Test Results

The Flex Drill uses 10 air pilot operated and 6 manual hydraulic valves to perform all functions. Throughout eight months of rigorous testing, few problems were encountered with any of the components in the hydraulic system. These components include all pumps, valves, hoses, fittings and cylinders. At no time was the vehicle "down" due to a problem with the hydraulic circuit.

There were several areas in the original circuit design that required extensive modification before a workable machine was available.

- a. The safety arm circuit was provided with an adjustable pressure reducing valve to limit the force applied to the safety arm when contacting the roof.
- b. The boom stopping valve was provided with a check valve bypass to allow dropping the boom at the end of a load cycle.
- c. Speed controls were included on virtually every small cylinder to decrease excessive shocks and accelerations.
- d. A separate pump/motor combination was provided to drive the air compressor.
- e. A check valve/flow control was added to the safety arm lowering circuit to eliminate "chatter" as the arm dropped. This in no way affects the cylinder lock safety feature.
- f. The circuit providing power for both manual and automatic drill rotation as well as Bolt Inserter was completely revised. The resulting circuit uses several components which a properly designed circuit would not need. Considerably more heat than necessary is generated due to the compromise circuit.

At various places throughout the circuit, taps and connections for pressure gauges and flow meters were provided. In several instances they proved extremely useful in tracing circuit problems.

Steel braided hydraulic hose was used throughout the circuit except for the ratchet reversing cylinder where maximum flexibility was desired. This cylinder operates at low pressure (approximately 500 psi) and a nylon braid hydraulic hose was used. All hoses were reusable fittings for easy field servicing. JIC fittings (37°) are used on all new circuit components.

The compromise hydraulic circuit allows for the generation of excessive heat. Tank temperatures of 160°F were not uncommon. All areas contributing heat should be analyzed and designed to minimize heat generation in each phase of vehicle operation.

The circuit also suffers from an excessively long warm-up period. Most of the small volume cylinders use oil which is contained in the lines rather than being recirculated to the tank. As a result, an ambient underground temperature of 50°F will cause excessive sluggishness in some functions until the oil heats up.

12.3 Recommendations

The Flex Drill hydraulic circuit should be redesigned from the ground up. Such a redesign can minimize both drill cycle time and heat generation.

Providing "hard piping" in lieu of flexible hose where possible will result in a much cleaner package. Also, tubing will occupy considerably less space than hose.

All of the various relief valves, reducing valves, flow control valves, check valves, and pressure gage taps can be designed into one or two manifold packages. Considerable space savings would result.

14.2 Available Test Time

The factors reducing the holes/day ratio for the Flex Drill include the following:

- a) No electrical power
- b) No permanently assigned personnel
- c) Holidays
- d) Breakdowns
- e) Sick days
- f) Personal days
- g) MESA inspections

The time required for drilling the four hundred test

Date	No. of Full Working Days	Holes Drilled	No. of Partial Working Days
Oct. 20	1	17	--
Nov. 4 - 15	8	227	--
Nov. 22 - 23	2	91	--
Nov. 30	1	24	--
(Sub-Total)	12 days	359 holes	--
Oct. 22 (broken lower clamp)	--	3	1
Nov. 16 (broken lower clamp)	--	7	1
Nov. 18 (broken ratchet cylinder)	--	8	1
Oct. 19, 26 & 29 (ltd. electrical power available)	--	25	5
Totals	12	402	8

In evaluating the Flex Drill vehicle the eight ineffective work days are eliminated for the following reasons:

1. The small number of drilled holes completed.
2. Delays do not pertain to the Flex Drill concept.

The four hundred test holes were completed between October 18, 1976 and November 30, 1976. During this time period there were a possible twenty nine (29) working days based on a five day work week and excluding all holidays. Out of the total twenty-nine working days required for the test, nine working days were lost for the following reasons:

- a. 5 days - no electrical power
- b. 1 day - sickness
- c. 2 days - travel time
- d. 1 day - breakdown

Out of the total twenty nine working days required eight days were substantially reduced in effective working time for the following reasons:

- a. 5 days - limited electrical power
- b. 3 days - minor breakdowns

The three days substantially reduced for minor breakdowns were the result of the failures of the lower clamp and ratchet cylinder. The lower clamp repair required minor welding, but the entire day was spent waiting for a welder to become available. The failed ratchet cylinder could have been easily replaced, but no spare cylinders were available; the lack of spare parts extended the down time and reduced the available drilling time.

The graph shown in Figure 10 represents the number of holes drilled/day versus the total test time. The time between September 27 and October 18 was spent in the entry adjacent to the main entrance. The time after November 20,

1976 was used for additional drilling for a better evaluation of the Bendix Bolt Inserter and the drilling of holes at the face with the production crews.

During the completion of the four hundred holes the greatest time delays are shown by the blank space on the graph between November 1 and November 6, and the time between November 24 and November 30. The first major delay was a result of no electrical power. The second major delay was a result of holidays. The short columns indicate a series of minor problems or the availability of electrical power for only a short period of time. The longest columns indicate almost one half of the total test holes were completed in only a three day period. The future potential for the Flex Drill concept in competing against a standard bolter has not been fully evaluated.

During the completion of the eleven week field trial there were fifty three working days based on a five day work week. However, the first three work weeks were spent in an entry parallel to the mine entrance next to the belt head. The holes completed in this area plus the holes completed after November 30, 1976 were not considered or evaluated as part of the four hundred test holes.

The twelve days required to complete 359 holes represent only 22.6% of the total working days at the mine to complete 90% of the required 400 test holes. The graph in Figure 11 depicts the increase in the number of holes between November 4 and November 30 by the change of the slope on the test curve.

When electrical power was available, only minor breakdowns affected the number of holes completed. Figure 12 represents the Breakdowns vs. the Total Holes Drilled for the field trial. The large number of breakdowns occurred away from the entrance in the side entry as shown by the steep slope of the line on the left side of the graph. Continuation of the line indicates the relationship between the number of breakdowns and the test holes completed. This line indicates four breakdowns for the completion of one hundred test holes and was consistent for the entire four hundred hole test. The increase in slope after the test section can be attributed to working directly at the face of the mine under different seam height conditions. Figure 13 represents the Holes Between Failures vs. Failure Days for the field trial. This graph indicates how the minor breakdowns reduced daily production and reduced the Flex Drill effectiveness.

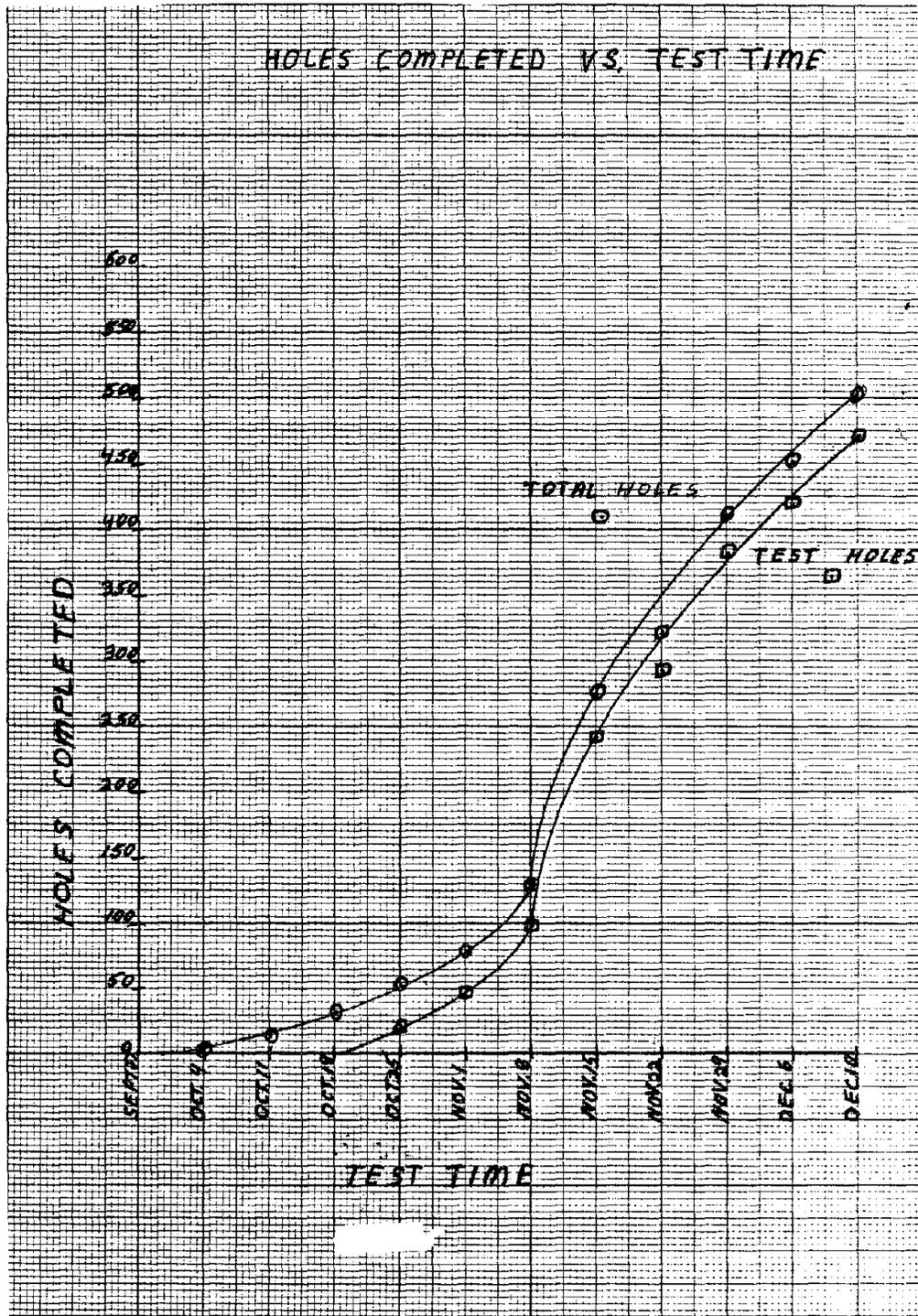


Figure 11: Graph - Holes Completed vs. Test Time

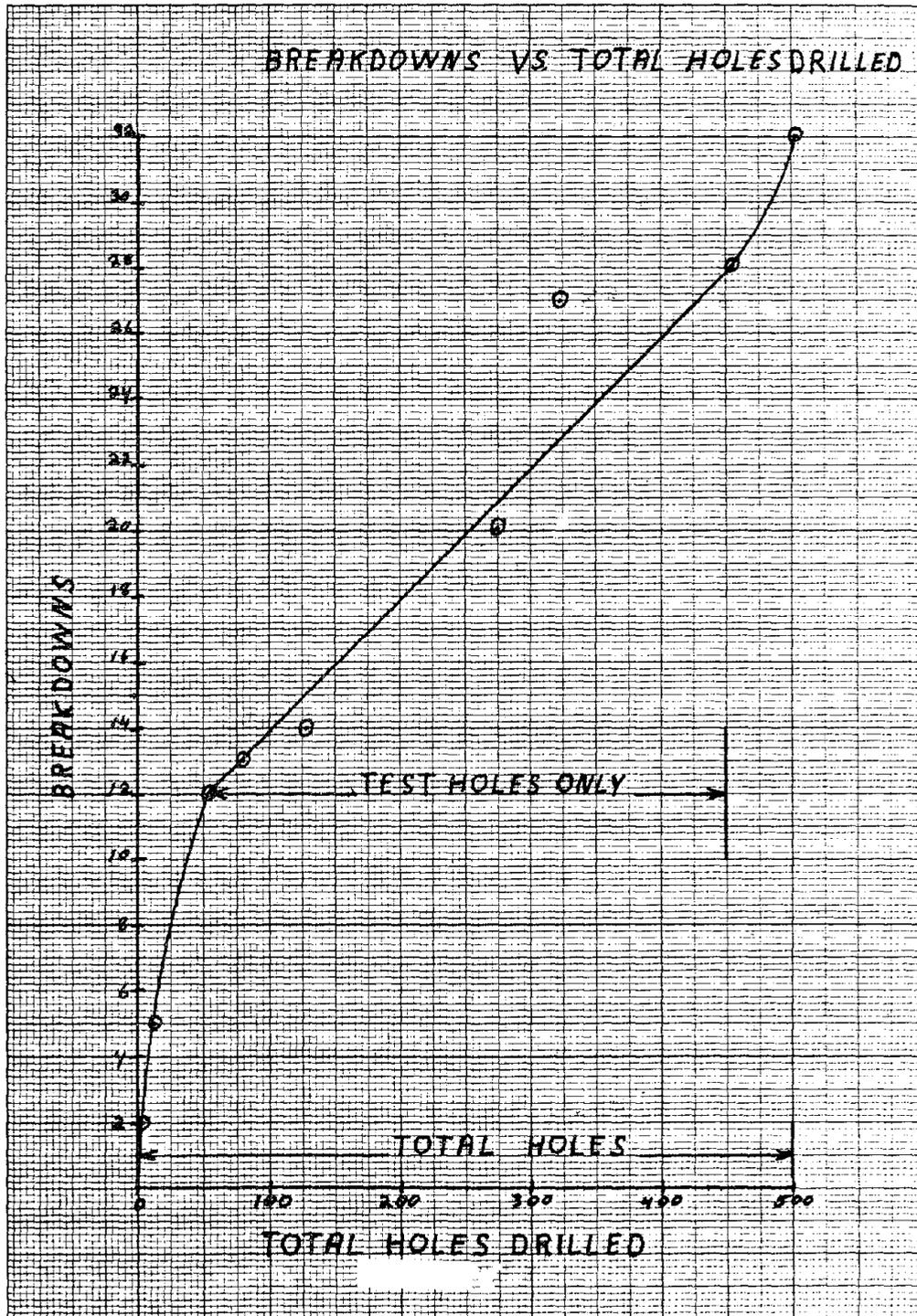


Figure 12: Graph - Breakdowns vs. Total Holes Drilled

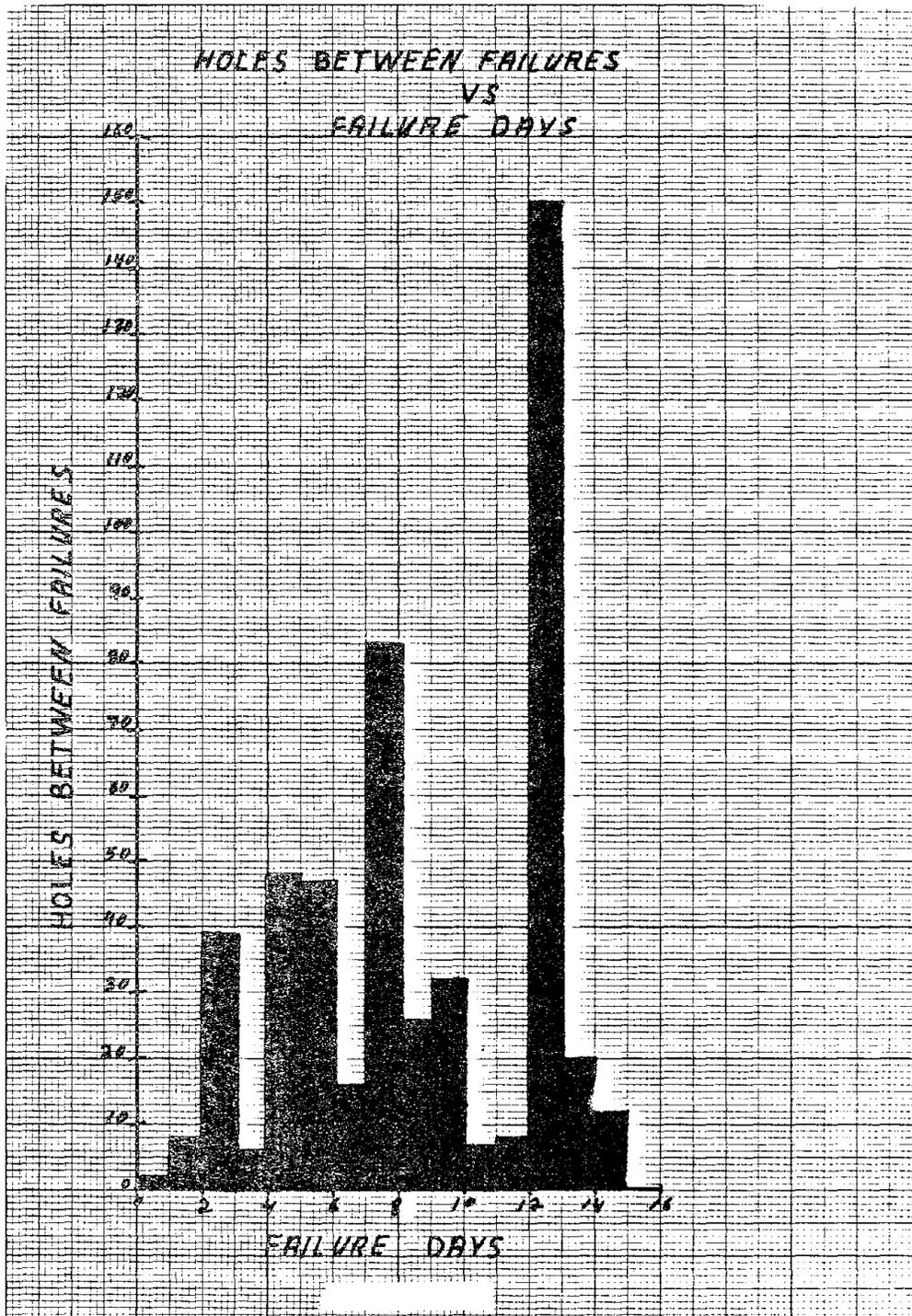


Figure 13: Graph - Holes Between Failures vs. Failure Days

The overall test results are best illustrated in Figure 14 where the daily hole completion breakdowns are listed. Figure 14 includes holidays, drill time, travel time, work time and lost time encountered in the field trial. The breakdowns listed in Figure 14 are defined in the malfunction area.

Sept. 27	Sept. 28	Sept. 29	Sept. 30	Oct. 1	Nov. 8	Nov. 9	Nov. 10	Nov. 11	Nov. 12
Start Up	Start Up	C. R. K. 1	Adjust & Test	Adjust & Test	C. R. K. 1 D. S. C. 2	C. R. K. 1	Top Plate Sheared	L. C. K. 1 L. S. L. A. 1	L. S. # 1 Broken
Daily Holes -	Daily Holes -	Daily Holes 2	Daily Holes -	Daily Holes -	Daily Holes 20	Daily Holes 16	Daily Holes 51	Daily Holes 32	Daily Holes 26
Total to date -	Total to date -	Total to date -	Total to date -	Total to date -	Total to date 118	Total to date 134	Total to date 185	Total to date 217	Total to date 248
Oct. 4	Oct. 5	Oct. 6	Oct. 7	Oct. 8	Nov. 15	Nov. 16	Nov. 17	Nov. 18	Nov. 19
Installed Upper Clamp Arms	Installed Upper Clamp Arms	C. R. K. 1 L. S. L. A. 1 D. S. C. 1	Sick Day	Adjust & Test	C. R. K. 2 Airlines 2	C. R. K. 2 L. C. K. 1 Broken lower Clamp	Sick Day	Ratchet Cylinder 1 Broken	No Power
Daily Holes -	Daily Holes -	Daily Holes 8	Daily Holes -	Daily Holes 2	Daily Holes 32	Daily Holes 7	Daily Holes -	Daily Holes 8	Daily Holes -
Total to date -	Total to date -	Total to date -	Total to date -	Total to date -	Total to date 275	Total to date 282	Total to date 282	Total to date 290	Total to date 290
Oct. 11	Oct. 12	Oct. 13	Oct. 14	Oct. 15	Nov. 22	Nov. 23	Nov. 24	Nov. 25	Nov. 26
Adjust & Test	Adjust & Test	Car Accident	Sick Day	Sick Day	Carousel Cover Broken	No Problems	Travel Day	Holiday	Holiday
Daily Holes 2	Daily Holes 15	Daily Holes -	Daily Holes -	Daily Holes -	Daily Holes 45	Daily Holes 46	Daily Holes -	Daily Holes -	Daily Holes -
Total to date -	Total to date -	Total to date -	Total to date -	Total to date -	Total to date 335	Total to date 381	Total to date 381	Total to date 381	Total to date 381
Oct. 18	Oct. 19	Oct. 20	Oct. 21	Oct. 22	Nov. 29	Nov. 30	Dec. 1	Dec. 2	Dec. 3
No Power	No Power (7 hours)	L. S. 1 Hyd. Hose 1 L. S. L. A. 2	Broken Lower Clamp Act.	Broken Lower Clamp Act.	Travel Day	Test Complete	Using Bendix Bolt Bender	Redesign Discussion	Blower Drive Belt Replacement
Daily Holes -	Daily Holes 3	Daily Holes 17	Daily Holes -	Daily Holes 3	Daily Holes -	Daily Holes 24	Daily Holes 7	Daily Holes 8	Daily Holes -
Total to date -	Total to date 3	Total to date 20	Total to date 20	Total to date 23	Total to date 381	Total to date 405	Total to date 412	Total to date 420	Total to date 420
Oct. 25	Oct. 26	Oct. 27	Oct. 28	Oct. 29	Dec. 6	Dec. 7	Dec. 8	Dec. 9	Dec. 10
Holiday	No Power (7 hours)	No Power (7 hours)	No Power (4 hours)	No Power (4 hours)	C. R. K. 1	C. R. K. 3	No Power (4 hours)	No Power	No Power
Daily Holes -	Daily Holes 4	Daily Holes 7	Daily Holes 7	Daily Holes 7	Daily Holes 19	Daily Holes 20	Daily Holes 12	Daily Holes -	Daily Holes -
Total to date 23	Total to date 27	Total to date 34	Total to date 41	Total to date 48	Total to date 439	Total to date 459	Total to date 471	Total to date 471	Total to date 471
Nov. 1	Nov. 2	Nov. 3	Nov. 4	Nov. 5	Legend: D. S. C. Drill Steel Coupling L. S. Limit Switch C. R. K. Carousel Ratchet Key L. C. K. Lower Clamp Key L. S. L. A. Limit Switch Lever Arm				
No Power	No Power	No Power	D. S. C. 1	No Problems					
Daily Holes -	Daily Holes -	Daily Holes -	Daily Holes 23	Daily Holes 27					
Total to date 48	Total to date 48	Total to date 48	Total to date 71	Total to date 98					

Figure 14: Test Completion Chart

15.0 Drill Bit Wear Analysis

The drill bit used on the IRRI Flex Drill was a Kennametal Model FSP - 1-3/8 with an oval shank. The IRRI Flex Drill drilled only with a finishing bit in lieu of the standard practice of using a starter bit for the first 12 inches of the roof bolt hole.

The IRRI test crew installed a total of 502 roof bolts during the underground test period. The majority of the holes were 6' in length with a few 8' holes and one (1) ten foot hole. During the test period a total of 73 drill bits were used. The subsequent review of the bits resulted in the bits being separated into three categories:

1. Worn (not resharpenable), no further drill life.
2. Bit prematurely replaced, additional drill life remaining, resharpenable.
3. Negligible wear - bit evidences negligible wear.

(This categorization has been supported by Kennametal engineering personnel.)

The following is a table showing the bit wear distribution:

Worn	10
Resharpenable	49
Negligible wear	14

During the underground test the IRRI crew had the opportunity to review the standard manual bolting in operation. The average drill bit life was between 2 and 3 holes per bit. The bits were generally broken or well worn when replaced. The manual bolting crew used bits supplied by VFR Wesson which sell for \$4.13 per bit or essentially the same price as Kennametal bits.

	<u>Manual Bolter</u>	<u>IRRI Flex Drill</u>
Bit Type	VFR Wesson 1-3/8"	Kennametal 1-3/8"
Bit Cost	\$4.13	\$4.08
Bit Life	2-3 holes/bit	7-14 holes/bit*
No. of Holes Drilled	Not Applicable	502
Hole Length	4'	6'
Bit Wear/Length	8 ft. of hole to 12 feet of hole	42 ft. of hole to 84 feet of hole
Bit Cost/Length	\$0.52/ft-0.34/ft.	\$0.10/ft-0.05/ft.
Bit Cost/4 ft. of hole	\$2.00 - 1.36	\$0.40 - 0.20

(* Total number of bits used 73, estimated that 37 bits could be reused.)

As noted in the chart, the bit wear and cost reflect a span between the maximum and minimum values. The most conservative value is based upon the total "73" bit usage and the optimistic value reflects the "37" bit usage (discounting the negligible wear and adjusting for the resharpened bits).

The preceding chart illustrates the bit cost related to a four foot hole since the manual bolter drilled a four foot hole and the rod changer a six foot hole. The IRRI Flex Drill has demonstrated superior bit life. This may be attributed to the following:

1. The omission of the "boost" control on the Flex Drill. The "boost" control provides additional hydraulic power (10 GPM at 1000 psi) to both the boom cylinder and the drill gear box. The manual bolting crew operated at maximum power during the drill cycles.
2. The increased use of semi-automatic cycle operations which decreases operator input during the drill cycle.

3. The opportunity for the drill bit to "cool" during the subsequent steel changing operation. The drill period of 10-15 seconds is followed by an inactive period of 30-40 seconds. It has been demonstrated in the use of alternate dust collection systems such as water-mist and water cooling that bit life increased by maintaining reduced operating temperature levels.

If one equates the cost of bits per shift per bolter drilling 100 holes, the Flex Drill could potentially save \$160 - \$180 per bolter shift. This assumes, however, that subsequent redesign of the bolter will reduce the cycle time to be competitive on a productive base.

16.0 Cycle Time Analysis

The rod changer has the ability to drill at the same rate as a standard roof bolter using conventional rotary thrust. It must be noted that the standard bolter and crew drilled a four (4) foot hole and installed a four foot bolt. The rod changer for purposes of test drilled a six (6) foot hole and installed a four (4) foot bolt.

At IRRI during the aboveground test evaluation the rod changer was timed at 3 minutes to drill a 5 foot hole in Indiana Limestone (10-14,000 psi compressive strength). This five foot hole required four drill sections. the IRRI test crew recorded 3 minutes and 40 seconds to drill a six foot hole during the field trial.

The drill penetration rate during the aboveground evaluation was measured at 4.5 FPM with a maximum drill torque of 273 ft-lbs. and a maximum drill thrust of 5,300 lbs. The standard Lee-Norse Roof Bolter TD-1-29, identical to the base model used with the rod changer (prior to modification), is capable of 6 FPM in similar strata. The drill penetration rate of the standard bolter in the mine was observed to be approximately 6 FPM.

The following is a chart summarizing the data discussed thus far:

Operation	IRRI Aboveground Test Data	Underground Test Data
to drill 6' hole	180 sec.	220 sec.
Rod Changer Drill Penetration Rate	4.5 FPM	4.5 FPM
Standard Drill Penetration Rate	----	6.0 FPM

A pro-rata of the rod changer time required to drill a four foot hole (3 drill sections) versus a six foot hole (4

sections is approximately 165 seconds, or 55 seconds per drill section).

220 sec. = 4 sections = 6 foot hole

165 sec. = 3 sections = 4 foot hole

The IRRI rod changer is capable of 6 FPM drill penetration rate although due to a much modified hydraulic circuit only 4.5 FPM was available during the test. Assume that 6.0 FPM is attainable:

Drill Penetration Rate	Actual Drill Time - 4 ft. hole
4.5 FPM	53 seconds
6.0 FPM	40 seconds
	—
	13 seconds difference

At 6 FPM a 13 second reduction in cycle time can be realized. Thus, the 165 second cycle time for a four foot hole can be reduced to 152 seconds.

165 sec. - 13 sec. = 152 sec. 4 foot hole, 3 drill sections
@ 6 FPM drill penetration rate

The rod changer as tested at the Bethlehem mine was designed for low seam operation. Hence a total of six sections of 18" effective drill length were used. It appears that a carousel properly matched for a given seam height would permit a drill steel with an effective length of 24 inches to be used. In this instance only two drill steels would be required for a four foot hole.

Referring to the time required for a 6 foot hole at 220 seconds, 80 seconds of the cycle was consumed by drilling the hole at 4.5 FPM. Thus, 220-80 seconds, or 140 seconds

total time was allocated for the mechanical manipulation of the four section drill string.

140 sec. divided by 4 = 35 sec. per drill section for manipulation.

Thus, the 152 second cycle postulated for a 4 foot hole could, with two drill sections, be reduced by 35 seconds.

$$152 - 35 = 117 \text{ seconds}$$

Thus, through minor design changes in the carousel length and an increase in the drill penetration rate to standard performance the cycle time was reduced by 47%. This cycle time reduction from 220 seconds to 117 seconds was achieved through basic changes without projecting cycle time analysis and reduction. The mechanical manipulation is projected to be reduced by 30% or approximately 10 seconds per drill section. Thus, it appears feasible that a four foot hole could be drilled by the rod changer in less than 100 seconds utilizing two steels. The rod changer demonstrated the ability to drill and bolt 51 holes on the best, trouble-free day.

17.0 Discussion of Malfunctions

Prior to the beginning of the field trial two possible types of malfunctions were considered for the Flex Drill vehicle. The first was defined as "major Malfunctions" which could have affected the duration and quality of the test; the second type were classified as "minor malfunctions" which affected only daily production.

A major malfunction is defined as any of the following: (1) a failure that required removing the Flex Drill vehicle outside of the mine for repair. (2) a broken weldment in the Lee-Norse chassis, (3) a defective or destroyed Lee-Norse cylinder, pump, or tramming components. (5) a minimum downtime of 5 days. (6) a complete disassembly or removal of the air logic control box.

A minor malfunction is defined as failure or interruption in the test cycle which could:

1. result in a repair/replacement requiring up to 8 hours to correct.
2. be repaired through minor machining (hand drill) or welding within the test area.
3. require replacement of the carousel ratchet drive key.
4. require replacement of a hydraulic hose.
5. require replacement of an air limit switch.

Throughout the aboveground test period malfunctions occurred and were addressed through equipment redesign and/or adjustment. However, a persistent problem which remained throughout Phases IV and V was the inability to maintain limit switch adjustment due to racking of the safety arm.

The following is a list of the minor malfunctions that occurred during the test:

Time Lost/Failure (hr.)	Description of Failure	No. of Failures
2/3	Carousel Ratchet Key	15
.10	Readjustment of No. 2 Limit Switch Lever Arm	15
2	Lower Clamp Key	2
4	Broken Ratchet Cylinder Rod	1
1/4	Broken Drill Steel Coupling	5
1-1/2	Ruptured Hydraulic Hose	1
1	Replaced and Adjusted Blower	1
1-1/2	Replaced Lower Clamp Rotary actuator	1
8	Repair Lower Clamp Weldment	1
4	Repair Ratchet Carousel Cover	1
3	Installing New Upper Clamp Arms	1
1/2	Service Air Inlet Filters	20
1	Limit Switch Replacement	2

A complete discussion of the above minor malfunctions and their solutions will be described in the appropriate section in the Appendix.

18.0 Bendix Roof Bolt Inserter

The USBM contract to IRRI for the development of a Low Coal Roof Drill required the use of the Bendix Roof Bolt Inserter in conjunction with the Flexible Drill. IRRI was supplied the assembly, installation prints and operation manual by Bendix.

A bolt inserter was shipped to IRRI during the Phase IV, fabrication effort, by the USBM. The intent of this combination package was to provide a complete system for low coal drilling and bolt insertion; however, the two units were not fully compatible.

The bolt inserter is useful in low seam mines in which longer than seam height bolts must be installed; however, in the test mine only four foot bolts were required in a four foot seam.

Use of the inserter for automatic bolt insertion from the rear of the vehicle is possible. Two areas requiring review are the automatic loading of a bolt into the inserter and guiding the bolt into the hole after it exits the inserter. The advantage for performing the bolting operation from the rear of the vehicle lies in the safety accorded the operator.

the use of the inserter design with a commercial machine depends on the following requirements:

- a. Bolt insertion speed
- b. Operator safety
- c. Bolt insertion and torquing time
- d. Mine worthiness
- e. Remote operation
- f. Remote loading with the bearing plate and flat washer attached to the bolt.

After reviewing the installation prints and operation manual, the inserter was attached to the Flex Drill chassis. It was mounted so that the assembly pivoted from the right side of the chassis over the center of the drill chuck.

Loading of the inserter was time consuming. The problems experienced with operation of the bolt inserter were as follows:

- a. Bolt loading
- b. Accessibility to the inserter
- c. Hole alignment
- d. Jamming of the bearing plate in the inserter
- e. Jamming of the flat washer in the inserter
- f. Releasing a partially installed bolt from the inserter
- g. Excessive bolt bending and torquing time.

Initial attempts to use the inserter resulted in a substantial increase in total cycle time, which prevented the proper evaluation of the IRRI Flex Drill system.

Because of the limitations encountered in adapting the inserter to the Flex Drill chassis, it was evaluated after the required 400 holes were completed. This allowed IRRI to devote more time to develop the bolt loading parameters. A successful demonstration of the inserter was conducted for the Bethlehem Research personnel.

A description of the difficulties encountered with the inserter and suggested solutions are included in the Appendix.

19.0 Low Coal Drill System Evaluation

This section of the report is devoted to a critique of the IRRI Drill concept. The criteria for evaluation are those features which were set forth in the letter from R. Burnham to R. Eakin dated May 10, 1976, a copy of which is included in the Appendix, Section 21.17.

Following is a list of evaluation criteria, each of which is to be judged on a 10 point scale. The standard roof bolter will be used as a comparative base to establish the scale factor for a similar feature on the Low Coal Drill, i.e.: item (c), practicality of the rotation and thrust system. This is the same system used on the current bolter and thus has been judged a 10.

a) Ease of fabrication - (10)

The IRRI design package reflects the current "state-of-the-art" manufacturing techniques used by the Lee-Norse Co. and other mining equipment manufacturers. The fabricated items are constructed with low carbon steel plate except where bearing loads or excessive wear was anticipated. Many of the clamp arms would ultimately be machined from castings to facilitate fabrication and reduce the cost of the piece.

During the design phase many contacts were made with Lee-Norse to determine their "standard" approach in material selection. In addition a design review was conducted at IRRI with members of the Lee-Norse engineering staff.

There are no components in the IRRI Low Coal Drill package that require manufacturing processes beyond those methods currently employed by the coal equipment manufacturers.

b) Repairability - (10)

The ease of repairing the system is coincident with the initial design philosophy, i.e.: readily available steel plate and bearing material are used. There are no long lead castings or forgings in the design.

c) Practicality of rotation and thrust system - (10)

The rotation and thrust system is the same system currently used by the Lee-Norse production roof bolters which had wide industry acceptance. The much modified hydraulic system of the IRRI Drill resulted in less hydraulic power to the rotation and thrust system. This condition will be corrected by revising the hydraulic circuit.

d) Cuttings and dust removal on suppression system - (8)

The dust collection system is the same system employed on the standard roof bolter. The performance has degraded with pressure drop through each steel coupling; however, no difficulty was experienced with this system in the test. It is recommended that the standard blower be replaced with another model having an increase in vacuum level to compensate for the loss due to coupled sections.

e) Total System Simplicity - (6)

The IRRI concept has been additive to the standard bolter and complexity has increased primarily through the need and use of a control circuit. The mechanical componentry, upper clamp arms, lower clamp arms, carousel, and centralizers, are not complicated in design and require no adjustment unless disassembly is required. The air logic system resulted in the addition of an air compressor, air controlled hydraulic valves, and air logic box. System trouble-shooting would require the services of trained maintenance personnel.

f) Adaptability of a 30" coal seam - (7)

The rod changer system can be readily adapted to function in 30" coal, but not drilling an 8' foot bolt hole. An 8' drill string in low coal (30") would require in excess of 6 coupled sections, and total drill time will not be competitive with current production machines.

Any length hole can be accommodated by design within the 30" seam by increasing the number of drill steels required

to form the drill string. However, the practicality of a string in excess of six (6) sections becomes doubtful because of time required to couple and uncouple the steels.

g) Simplicity of Maintenance - (8)

The initial design concept as described in (a) addressed both fabrication and maintenance. All pivot points for the new pivoting links incorporate sleeve bearings with grease fittings which is common practice with the Lee-Norse products. The upper clamps, lower clamp, centralizers, and carousel assembly do not require adjustment unless disassembly is required.

The additional maintenance requirements added pertain solely to the air logic system:

1. Air compressor inlet filter - filter change.
2. Air logic system main filter and moisture separator - periodic discharge of moisture and cleaning the filter elements. In subsequent designs an automatic moisture drain valve will be added to decrease maintenance frequency.
3. In line air logic filter - This filter located within the control box was difficult to service. However, a large capacity filter with a moisture drain valve will be incorporated on the subsequent system.
4. Air Limit Valves - The air limit valves used to sense specific motion and travel required frequent adjustment - the frequency was aggravated by the tendency of the safety arm to rack as noted in Section 11.1(b). A proper installation would require switch adjustment only if jamming occurred.

h) Simplicity of Adjustments - (10)

The adjustments, although frequent (as noted in Section "g" above) required no special tools or fixtures.

i) Vulnerability to damage from mine environment and other mine equipment - (8)

The design attempted to foresee and protect from all potential mine environment damage. Protective covers were added to the carousel, the handling arms and the air logic control lines. No mine falls or crumbling was experienced during the test so it is difficult to assess the worth of the covers.

Air limit switches and a rotary actuator were broken during the vehicle tramping operation. This area will be corrected in subsequent redesign.

j) Compatibility with other mining equipment - (10)

It is as compatible as the standard roof bolter since the base vehicle is a production bolter. The modifications have not created any unique operating, power, or maintenance problems.

20.0 Economic Summary

In addressing the cost impact of the IRRI "Rod Changer System" the following are assumed:

1. Only the increased cost of the IRRI Rod Changer System will be analyzed.
2. A marketable "Flex Drill System" must have an integrated package of drill and automatic bolt inserters.

The contract guidelines requested that a direct comparison be made of the productivity of the IRRI Rod Changer compared to a conventional roof bolter. In the test data described in this report specific productivity comparisons would be invalid for the following reasons:

1. The test mine only required four foot roof bolts.
2. The test bolter "Rod Changer" was test drilling a 6 foot hole with four rod sections not 4' hole.
3. The test bolter crew had no previous underground experience prior to the mine test.
4. The carousel length had not been properly matched for a productive 4 foot seam, i.e.: effective drill rod length - 2 feet.

It should be noted that Rod Changer system required no changes in the following areas:

1. Projected life cycle of the bolter amortized over a ten year period.
2. Operating crew requirements - no additions.
3. Bolter electrical power consumption.

It is probable that an improvement can be realized in the following areas:

1. Drill Bit Life (minimum 300% increase, see Section 16.0).
2. Operator Safety
 - Through reduction in lost time accidents.
 - Through reduction in litigation settlements.
3. Increased productivity even if the Rod Changer matches the cycle time of the manual bolter. The reduction in fatigue coupled with consistent repeatability should result in increased production over extended periods.
4. Reduction in temporary jack placement operations. If the remote drilling is proven reliable and acceptable relaxation in jack setting practice may be considered by MES2A.

The base vehicle for the contract addition Rod Changer was a Lee-Norse TD 1-29 production vehicle which has the following selling price:

Base Bolter	\$29,900
500 ft. Trailing Cable	1,100
Safety Arm	2,800
"Fly swatter" (Safety Arm)	450
Operator's Canopy (Cab)	1,200
Auger Clamp and wrench	670
	\$35,120

However, the IRRI "Rod Changer" deleted the auger clamp and wrench since the lower clamp and centralizer serve a similar function.

The "Rod Changer" required the following additions to the vehicle:

1. Air Logic Control System
 - a. Air Supply - Air Compressor
 - b. Air Logic Control Box and Panels
2. Safety Arm Front Chassis (to which the Rod Changer mechanism is mounted) and the pantograph arm.
3. Rod Changer Mechanism is comprised of:
 - a. Upper clamp arm assembly
 - b. Lower clamp arm assembly
 - c. Centralizer
 - d. Carousel assembly
4. Modified hydraulic circuit.

It is projected that a single arm roof bolter equipped with the "Rod Changer" similar to the tested configuration could be sold for \$50,000. This does not reflect the cost of the Bendix Bolt Inserter with an automatic feed system. This reflects an increase of 28% in the purchase price to an operating mine. Capital depreciation would still be taken over a ten year period reflecting an increase of 28% depreciation per year.

The addition of the control system (a must for remote operation) alone increased the selling price by 16%.

It is recognized through published reports that the roof bolter is a highly reliable piece of underground equipment with an average mean time between failures of approximately 900 holes. The rod changer system will require more frequent maintenance to assure clean air filters and proper air limit switch adjustment. The coal mine operators are beginning to address preventative maintenance programs and the increased maintenance proposed for the rod changer

should have no significant effect on operating costs of the roof bolter.

An important aspect of the economic analysis is the cost of the drill string. The "Rod Changer" drill string is standard Kennametal except for the modified drill steel coupling which will be approximately 20% more expensive - standard coupling, \$8.00, and modified \$9.60. One coupling is used per rod section in addition to the drill tube.

The coupling life in the Phase V test was approximately 250 holes versus approximately 1,500 holes for a manual operation. The coupling life for a rod changer can be increased to the 900 hole life cycle with changes in the alignment of the steel as described earlier in Section 16.0.

If one assumes two couplings required per four foot hole and a life cycle of 250 holes (Phase V), the cost of coupling is $\$19.20/250$ holes or $\$0.08/\text{hole}$. This is an insignificant cost when one considers that the cost per hole for the roof bit varies from $\$1.38/\text{hole}$ to $\$2.07/\text{hole}$ (Refer to Section 16.0) for manual bolting. Only the coupling need be replaced, not the drill steel tube.

21.0 APPENDIX

21.1 Bendix Roof Bolt Inserter

Several conditions contributed to the ineffective utilization of the bolt inserter and are listed as follows:

a) Loading Difficulties:

Insufficient information on the correct placement of the bolt into the inserter lengthened the learning period associated with the inserter. The placement of the bolt into the bolt inserter was illustrated in the manual; however, no dimensions were forwarded. A trial and error method was then undertaken to load the bolts. The loading of the bolt inserter was completed by the best method preventing interference with the head of the Flex Drill vehicle. If the bolt anchor was placed forward of the swing arm clamp, interference with the head of the vehicle was inevitable. The bolts were placed in the bender as close to the swing arm clamp as possible with varying results.

Before the bolt was loaded into the inserter, the bearing plate was placed near the anchor so that the bolt would not have to be stopped once it was fed through the inserter. This caused occasional problems with the bearing plate becoming caught in the swing arm clamp or interfering with the head of the machine. It was not possible to predict beforehand if the bolt would clear all the mechanisms.

b) Bolt Inserter and Hole Alignment

The inserter was rigidly mounted to the right frame of the vehicle. This permitted swinging the inserter over the center of the chuck and placing the bolt directly under the drilled hole when leaving the bender. The forward swing position of the bender was adjusted by the clevis attached to the swing cylinder. This adjustment, combined with the fixed radius of the swing arm attached to the inserter, provided accurate positioning of the inserter.

However, when the bolt cleared the inserter it had difficulty finding the drilled hole in the roof. The bolt

appeared to exit the inserter and swing arm clamp rollers at slightly different angles for no discernable reason and jam against the roof of the mine instead of entering the drilled hole. Once the jammed bolt was against the roof there was no way to maneuver the bolt into the hole.

To alleviate the problem the vehicle was trammed to allow the bolt to enter the hole. If this failed, the safety arm was lowered a small amount to free the bolt from the roof and the bolt was manually bent into position. Once the bolt was manually started in the roof the bolt inserter was activated to continue inserting the bolt into the roof.

c) Jamming of Flat Washer

The flat washer is located at the end of the bolt and bears against the bearing plate when the bolt is secured to the roof. The washer was initially placed at the end of the bolt when being loaded into the bolt inserter. However, when the bolt was being drawn through the inserter the washer vibrated down the length of the bolt and jammed against the reaction roller, stopping the head of the bolt from passing the roller. It was also possible for the washer to slide past the roller and jam against the drive wheel causing the drive wheel to slip and chatter on the bolt. At this point it was necessary to pry the washer from the roller and determine if the partially installed bolt could be completely inserted and torqued to specifications. If not, the bent bolt had to be removed from the roof and the machine.

d) Releasing the Bolt from the Inserter

After the bolt was inserted into the roof a problem was encountered with swinging the bolt inserter away from the bolt and back into its stowed position. The bolt caught on the clamp arm swing or the drive rollers when the bolt inserter was swung out of the way. This caused the bolt to bend near its head and possibly prevent it being secured to the roof. If the clamp arm failed to release the bolt properly, the operator freed it before returning the bolt inserter to the stow position. The jamming of the bolt was the result of friction between the drive wheel, clamp arm rollers and the bolt.

e) Cycle Time with Bendix Bolt Inserter

The cycle time for the Bendix Bolt Inserter included the following motions:

1. Index bolt inserter under the drilled hole.
2. Release clamp arm swing.
3. Place bearing plate on bolt; place flat washer against bolt; place bolt into bolt inserter.
4. Engage clamp arm swing (insert bolt).
5. Release clamp arm swing.
6. Retract bolt inserter.
7. Place torquing adapter in drill chuck.
8. Torque bolt to specified setting (120-160 ft-lbs).
9. Remove torquing adapter from chuck and prepare to drill next hole.

The production equivalent for inserting a bolt consisted of the following motions:

1. Insert pre-bent bolt into hole.
2. Manually straighten bolt and insert into the roof.
3. Place torquing adapter into drill chuck.
4. Torque bolt to specified setting (120-160 ft-lbs).
5. Remove torquing adapter from drill chuck and prepare to drill next hole.

The use of the bolt inserter increased the cycle time of the vehicle so that it became impossible to evaluate the effectiveness of the Flex Drill concept. On various occasions attempts to measure the time required to insert a bolt were undertaken. No accurate data could be obtained because of the difficulty in loading the inserter and torquing the bolt into the roof. A conservative estimate for use of the bolt inserter with no malfunctions is to add two minutes to the total cycle time with the Flex Drill vehicle. Because of the additional cycle time the bolt inserter was used primarily for demonstration purposes and only occasionally during the underground test.

It is felt that correct loading of the bolt is the most important factor to insure successful operation of the bolt inserter. In an attempt to determine the best method to loading the bolt inserter, the following attempts were used with no appreciable success.

Hole #1

(1st attempt) The first method used to load the bolt inserter was that shown in Figure 15. With this method the anchor was flush with the end of the bolt. The bearing plate had a slight friction fit with the anchor which resulted in the plate being 3/4 of an inch from the end of the bolt. The bolt was also placed against the drive wheel as shown in the catalog. The problem encountered with these conditions was jamming the anchor between the drive wheel and swing arm clamp when starting the bolt. This was a result of the anchor and bearing plate being too close to the drive wheel. Consequently, the bolt was bent improperly and insertion in the roof was impossible.

(2nd attempt) The second configuration attempted is shown in Figure 16. With this arrangement the anchor was passed through the bearing plate and the bearing plate was held with light pressure against the clamp arm swing. The bolt was fed through the bolt inserter properly, but missed the drilled hole and jammed against the roof, causing the drive wheel to slip and chatter on the bolt until the drive wheel was stopped. The safety arm was then lowered to free the bolt from the roof. The bolt was then manually aligned under the hole until it was properly inserted and successfully torqued to specifications.

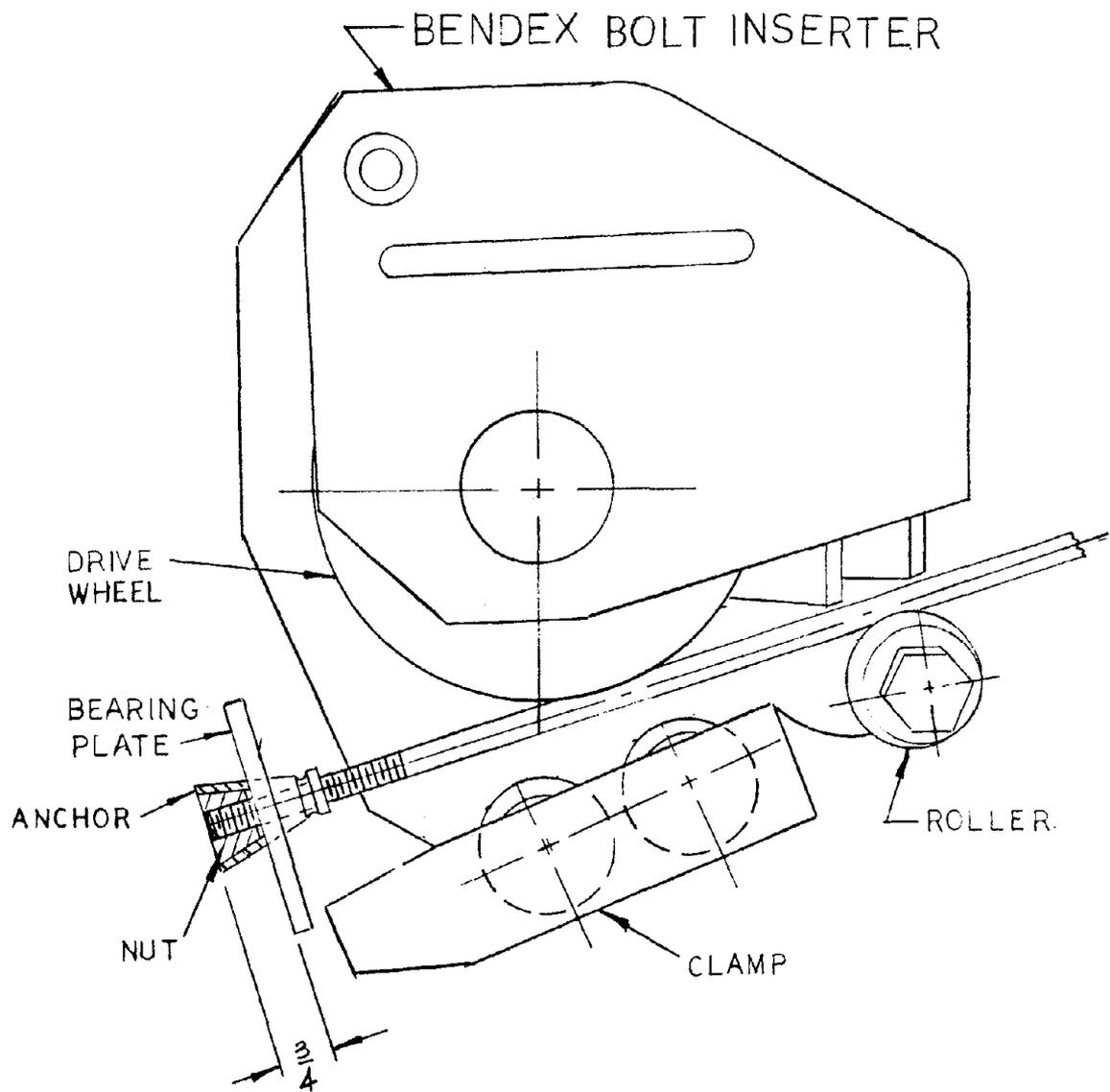


Figure 15: Bendix Bolt Inserter Loading Method No. 1

IRRI 7.70.02.1

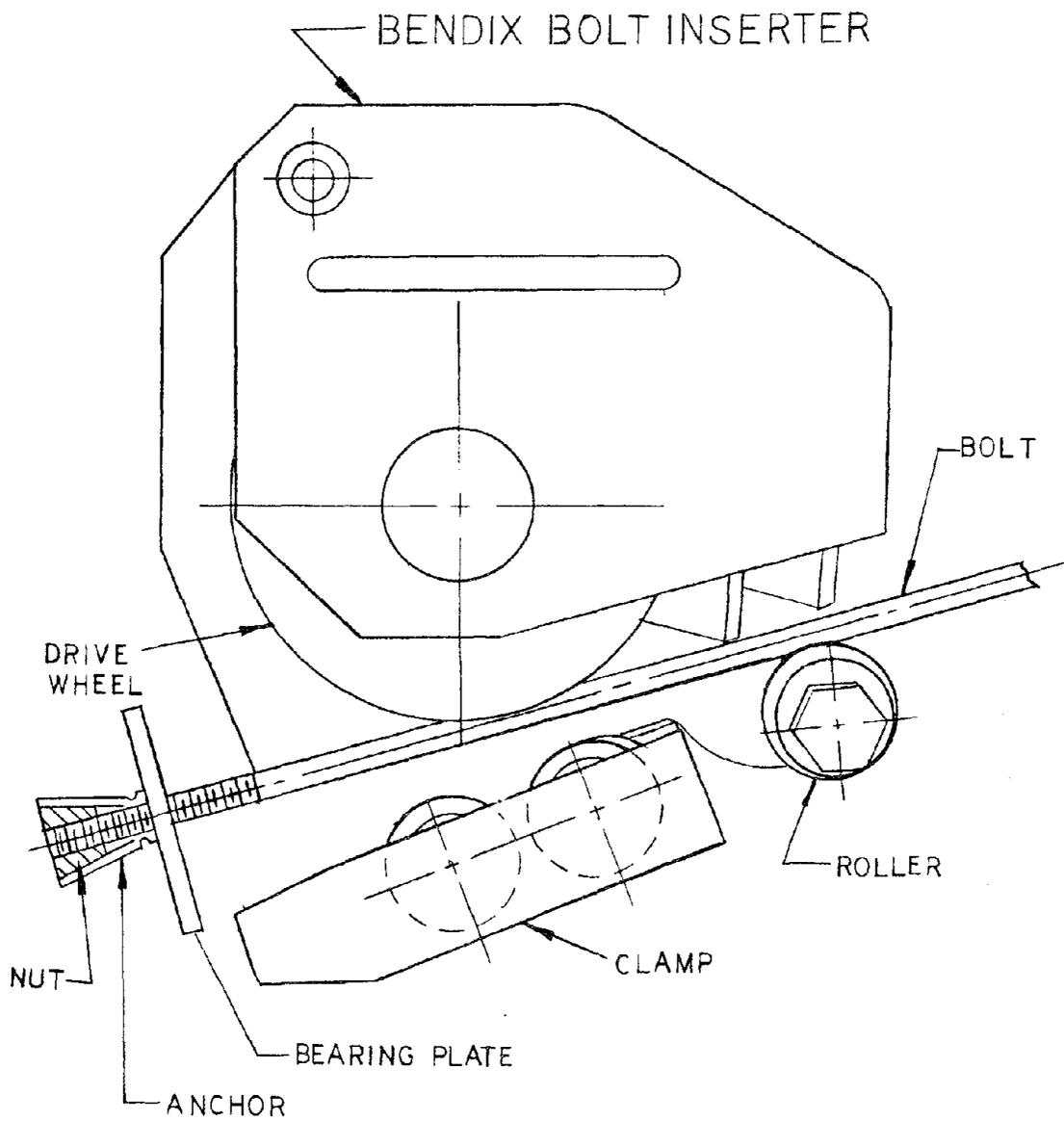


Figure 16: Bendix Bolt Inserter Loading Method No. 2

IRRI 7:7 0/0 8:3

Hole #2

(1st Attempt) After the second hole was drilled the third configuration as shown in Figure 17 was attempted. This method permitted placing the bearing plate 2 inches from the end of the bolt and allowed 2 inches of the anchor assembly to extend into the clamp arm swing assembly. The bolt was held against the drive wheel as shown in the bolt inserter manual. The bolt passed through the bolt inserter without problems, but failed to find the drilled hole in the roof. This bolt was again manually aligned into position under the hole and secured to the roof in the usual manner.

Hole #3

(1st attempt) The third hole was drilled and the configuration shown in Figure 17 was applied to loading the bolt into the inserter. The bolt was flush with the anchor and approximately half way in the clamp arm swing. The first attempt failed when the bolt jammed against the drive wheel and was severely bent, freeing it from the wheel. It was discarded and a new bolt placed in the bolt inserter.

(2nd attempt) The second attempt failed when the bearing plate wedged between the drive wheel and swing arm clamp. This malfunction destroyed the bolt as it was being freed from the bolt inserter and the bolt was discarded.

(3rd attempt) On the third attempt, the bolt was placed into the inserter as shown in Figure 18. The significant differences between Figure 17 and Figure 18 were the placement of the bearing plate in the notched area of the anchor and bringing the bearing plate flush back to the swing arm clamp. The nut was adjusted flush with the end of the bolt and the expandable section of the anchor slid back to the stops on the bolt. With this configuration the bolt found the drilled hole without problem and the bolt was secured in the prescribed manner.

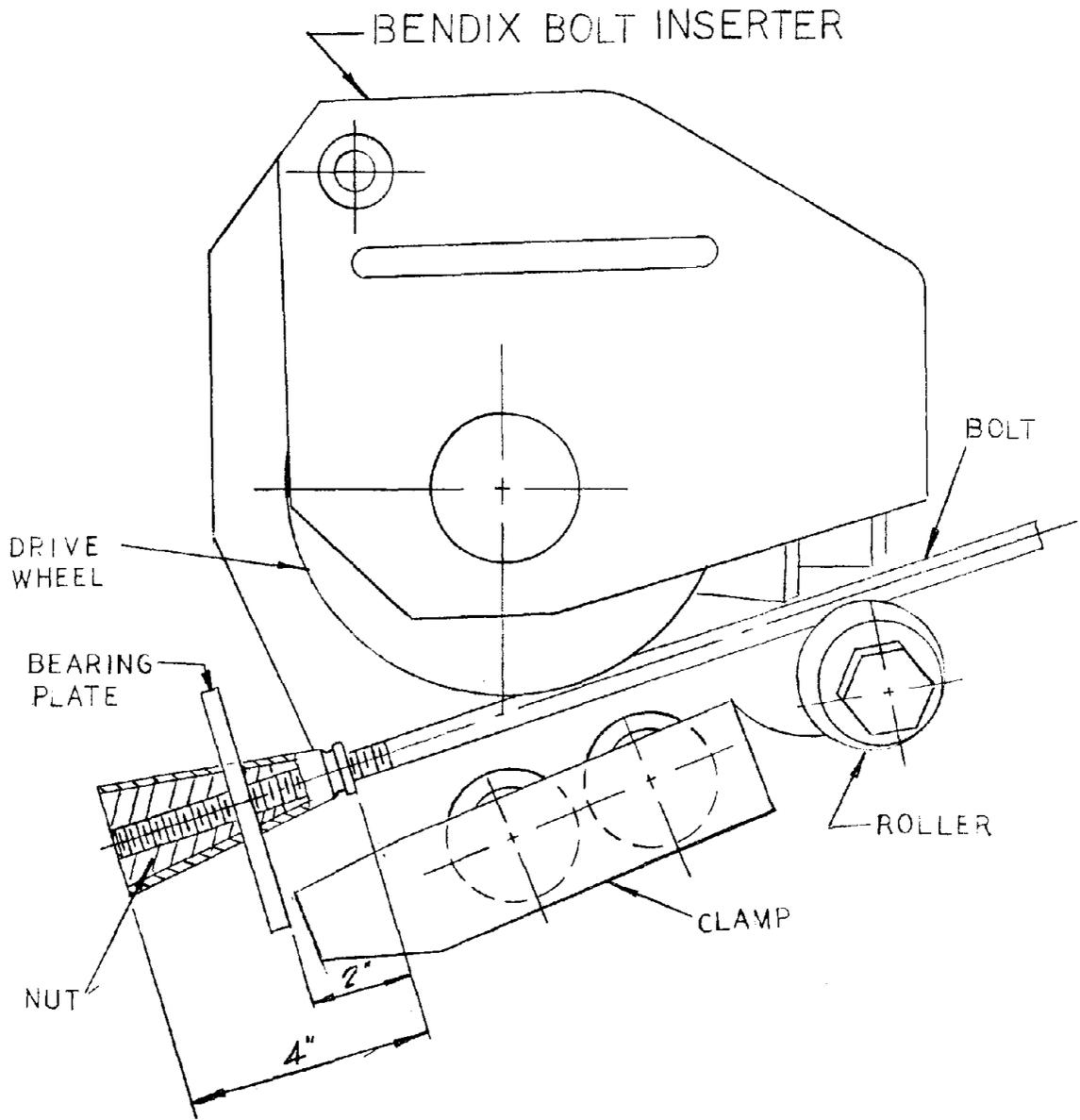


Figure 17: Bendix Bolt Inserter Loading Method No. 3

IRRI 7,700'8'0

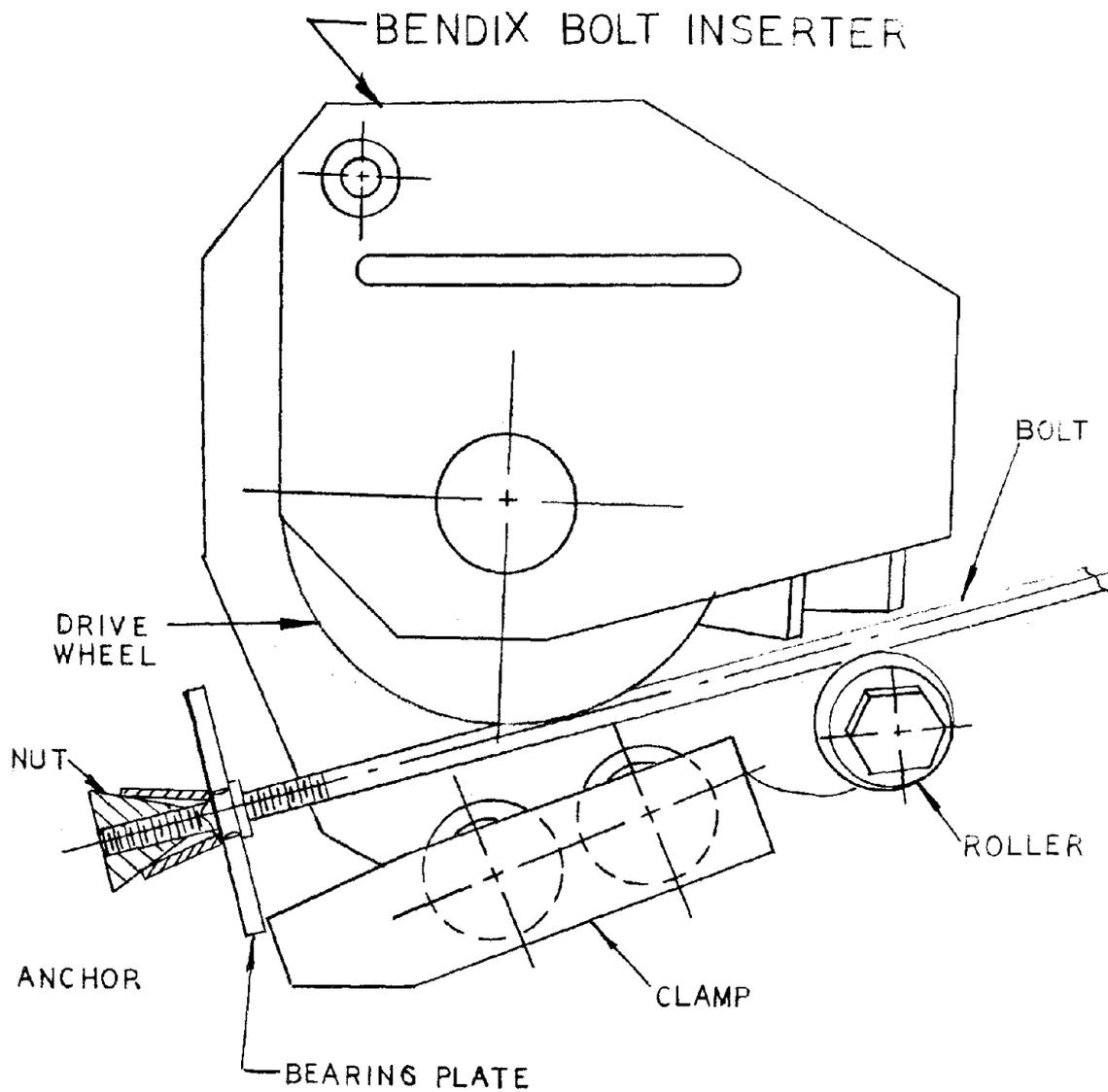


Figure 18: Bendix Bolt Inserter Loading Method No. 4

Hole #4

(1st attempt) The fourth hole required four attempts before a bolt was properly installed in the roof. On the first attempt, the bolt was loaded in the bolt inserter as shown in Figure 18. The bearing plate rested in the notch of the anchor flush against the swing arm clamp. The end of the nut was flush with the bolt. The first attempt failed when the bolt slipped through the swing arm clamp and jammed against the gate opening cylinder. This bent the bolt into a useless configuration.

(2nd attempt) In the second attempt the bolt was placed in the inserter in the same manner. A malfunction occurred when the bearing plate jammed between the drive wheel and the swing arm clamp. This second attempt also resulted in the destruction of the bolt.

(3rd attempt) The third attempt failed when the washer slipped down the length of the bolt and jammed against the drive wheel. Then the bolt was freed from the bender it was discarded.

(4th attempt) The fourth attempt on the fourth hole was successful in bolting the roof. The same insertion configuration was used as on the previous attempts. In summary, three out of four attempts failed. The final attempt proved a bolt could be successfully inserted. However, reliability is low.

Hole #5

(1st attempt) For the fifth hole the configuration was changed into that of Figure 19. The significant difference was in placing the bolt on the swing arm rollers instead of the drive wheel. The bearing plate was placed in the notch of the anchor and the bolt was flush with the end of the nut. The bolt again missed the hole. The vehicle was repositioned until the bolt was under the hole. The bolt was then secured to the roof.

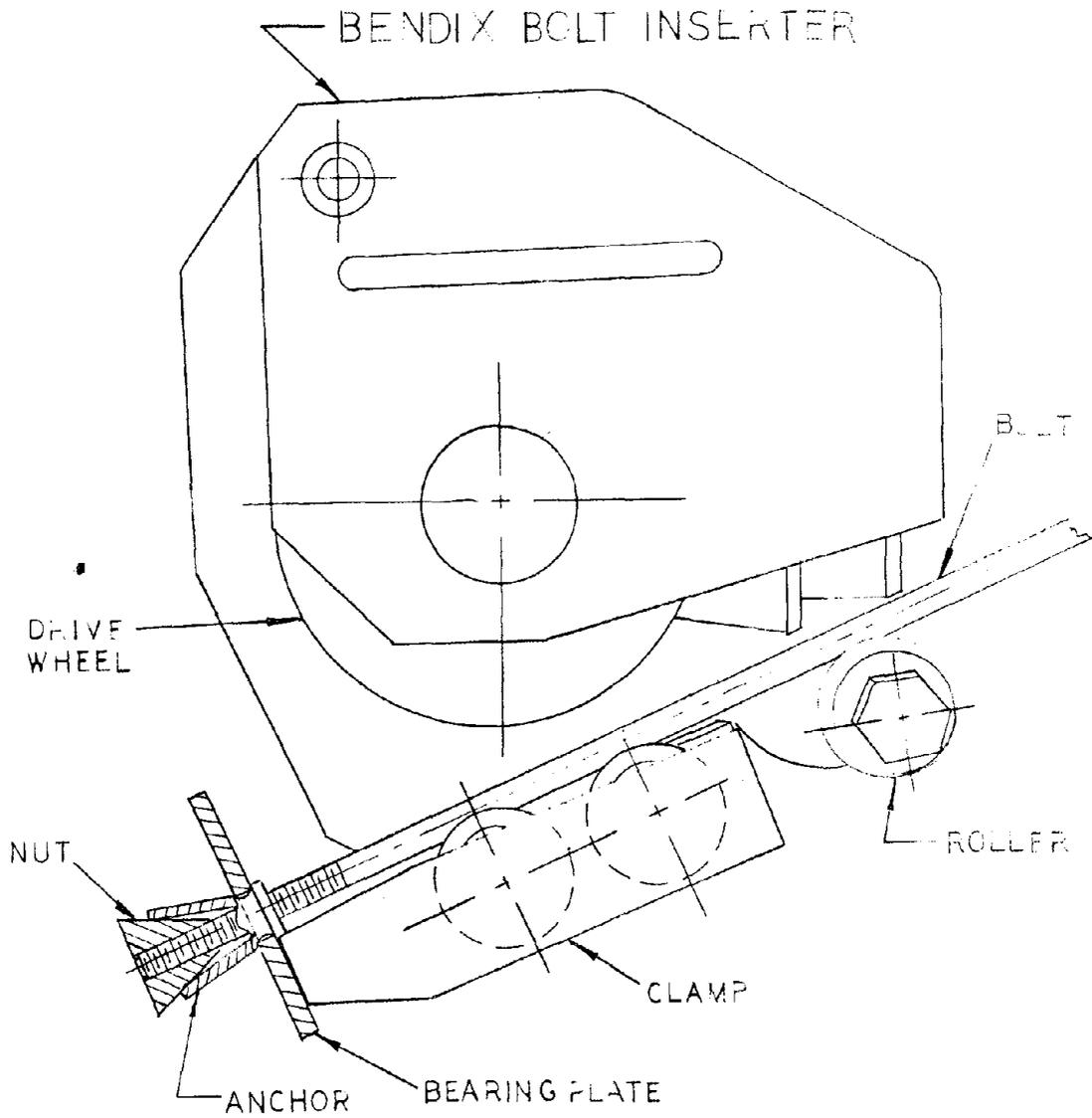


Figure 19: Bendix Bolt Inserter Loading Method No. 5

IRRI 717 010179

Hole #6

(1st attempt) The sixth hole was begun by placing the bolt in the inserter as shown in Figure 19. The important difference between Figure 19 and Figure 18 was the placement of the bolt on the swing arm rollers rather than pressing it against the drive wheel rollers. The bolt missed the drilled hole after running through the bolt inserter. Trimming the vehicle and bending the bolt secured the hole and only one bolt was used to secure the top.

Hole #7

(1st attempt) The completion of the seventh hole required three bolts before properly securing the roof. The first bolt was placed in the bolt inserter as shown in Figure 19. The bolt rested in the clamp arm swing rollers. The first attempt failed when the bolt jammed against the ratchet cylinder. It was therefore discarded.

(2nd attempt) In the second attempt, the bolt was fed through the inserter and into the hole with no apparent problems. However, it would not release properly from the bolt bender. When the bolt was freed from the inserter it had been bent close to the head of the bolt, but it was used to complete the bolting of the hole.

The completion of these seven holes required approximately six hours work, in which fourteen attempts and fourteen bolts were required to secure the roof. The mandatory use of the bolt inserter would have required extending the field trial time and the destruction of many more bolts than could be accepted. While using the bolt inserter the bolting of one entry per shift would not have been possible. The problems encountered using the bolt inserter attached to the vehicle in the present design excluded it from working at the face. It is felt that a significant number of loading combinations were undertaken to determine the best method of operating the bolt inserter. However, no combination showed a significant improvement over the others. Every method attempted presented numerous problems in the bolting cycle and, therefore, the bolt inserter was used very little throughout the test.

21.2 Lee-Norse Vehicle Modifications at Mine Site

The modifications made to the chassis before shipment to the mine site will not be recorded in this section; however, the Lee-Norse chassis was slightly modified at the mine site to adapt it to the conditions underground. The first modification involved changing the tires from 8:50-14 to 8:25-15. This gave the vehicle an additional six inches of ground clearance. The change to a larger tire size resulted in an interference with the oil filler cap for the hydraulic sump. The oil filler cap was relocated near the operator's cab and permitted filling of the tank from the same area. The auxiliary air tanks which are used in the event of compressor failure were removed from the vehicle as a precaution when the tires were changed, but were never installed or needed on the machine for the remainder of the field trial. The air tanks would allow for the completion of one hole in the event of compressor failure. The practice of bringing air tanks in and out of the mine so that a bolter can maintain productivity is undesirable due to the distance involved in transporting the tanks.

The next modification involved the top cover on the vehicle protecting the electric motor and other components. A small section of the cover was cut and bent so that the oil temperature gauge could be read at all times. The temperature of the hydraulic oil was a major concern during the test. The maximum temperature of the oil was 175°F and the ambient temperature averaged 50°F. It was feared that the high temperature would ruin the seals on the various hydraulic components. No such problems occurred during the test, but a circuit redesign should definitely provide for increased oil cooling. It was necessary to add oil once during the field trial. The oil added appeared very heavy; however, it was the standard oil Bethlehem used for their hydraulic mining equipment. Because of the conditions in the mine it was impossible to add oil to the vehicle without introducing more dust into the system.

When considering the redesign of the hydraulic system, the routing of the hydraulic lines from the chassis to the head should be changed so that the lines are kept to the "outside" front of the head on the vehicle. This would eliminate most interference problems and allow for larger actuators. The air and hydraulic lines around the operator's station could be better relocated under the protective tray. This would allow for easier servicing of the vehicle.

The boom cylinder should be prevented from leaking down once the vehicle is turned off. This would facilitate tramming the vehicle by people unfamiliar with the machine. This correction would eliminate the need for an unfamiliar operator to manipulate the controls (other than tramming) on the vehicle to raise the boom up to a position suitable for tramming.

The last consideration for the hydraulic circuit in general would be the "Warm up Time". This was determined to be approximately twenty minutes with an ambient mine temperature of 55°F using constant drill rotation and occasional boom operation. The warm up time affects those areas of the work cycle where time delays are preset and no limit switches are used.

21.3 Air Logic System

The pneumatic and air logic components of the machine presented no major problems during the entire Flex Drill program. The air system had the highest reliability of any system on the vehicle. To maintain this reliability it was necessary to perform daily maintenance on the two air filters and one air dryer. The inlet filter to the air compressor required changing only once and that was a precautionary measure not prompted by any symptoms of equipment malfunction. The second filter serviced was the Aro miniature filter located within the control box directly before the moving part air logic components. On one servicing this filter was filled with water and highly contaminated from the air supply line.

After disassembling the filter it was found that two steel components (a "flat washer" and "compression spring") were pitted and corroded. The filter element should have been replaced at this time, but there were no spares available; therefore, the existing element was cleaned and placed back in the filter. The Aro filter and other miniature filters should be reviewed for their construction and application in the mine environment during the next phase of the contract.

The third component of the air filtration system is the Dollinger "filter dryer". This component required daily maintenance involving removal of the filter elements from the machine, discharge of the water collected, purging the air lines of water, replacing the elements when possible and reassembly of the filter dryer. This procedure could not be done by a production bolting crew, but would be considered a maintenance operation performed by an electrician. The problem can be corrected by replacing the Dollinger filter-dryer with a dryer fitted with an automatic drain. The automatic drain (dump) reduces the opportunity for water to contaminate the moving part air logic components. The water accumulated in the system is a result of water condensing from the cool, ambient air of the mine and not any water that the vehicle may have trammed through.

The servicing of the air filters on the next machine should be given more consideration. The inlet air filter and Dollinger filter should be located so that they can be serviced without removing the top protective covers of the machine. The Dollinger filter-dryer, if retained on the vehicle, should be mounted in a vertical position (it was not possible with this machine) to allow for easy draining

of the filter. The air filter, downstream of the Dollinger filter-dryer should be visible outside the moving part air logic box and physically protected from the mine environment. This extra effort will ensure servicing of the filter when required, but protecting it from the mine environment at all other times. Because this filter is the main component before the moving part air logic components it is impossible to positively protect the filter and the downstream line from contamination while it is being serviced.

With the present design of the air logic box "cover" it was impossible to fully open the box for servicing in low seam mines. This situation required one person to hold the cover up while another person serviced the air logic. This would be unacceptable for a production machine. However, it must be noted that the current air logic box was intentionally selected oversized to allow prototype rework additions and substitutions. One possible solution would be a cover that hinges in two areas and then folds out of the way. This would allow a partial opening of the box for servicing if necessary. The air logic box should be positively pressurized to keep dust from entering the box. Once dust or dirt has entered the air logic box there is no effective way of removing it. All lines entering and exiting the box should be bulkhead mounted so that easy removal of the total box could be accomplished. Also, the removal of one contaminated line without affecting the entire air logic package could be accomplished. The quick disconnect fitting with an internal check valve would be the best fitting available for this application. The acceptance of air logic in the mines would be greatly improved if the air logic components were mounted on panels labeled for machine function. If trouble developed in a specific area of the machine sequence, the labeled panel could be easily removed for reference to a pneumatic schematic.

In this design and future designs the operator's air logic box will be the area most susceptible to breakage and contamination. The dust from the drill motor, dust box, rock dusting and the floor all combine to make the operator's box difficult to protect from the environment. Because of the heavy dust concentration in the area it was necessary to wipe the indicator lights a minimum of once a day. This operation could be done while the hole was being drilled, but illustrates the environment that the air logic system must work in. The push buttons used on the operator's panel engaged 3-way switches with the exhaust port venting inside the operator's box. The operator switches were bulkhead mounted through the box with a rubber

gasket for positive sealing. With this operator button there was not a chance for dust to enter the system. The push button valve itself was mounted inside the operator's box where the venting of the exhaust was into a controlled area. The operator's box was considered sealed and in future designs should be positively pressurized at all times.

The indicator lights were positively sealed against the front panel and they too exhausted inside the operator's box. The selector switch mounted at the front of the control panel provided little opportunity for dust to enter the system because of its construction and the way it was mounted to the control panel.

The third area of concern for the air system involved limit switches deployed throughout the vehicle. The Aro limit switch proved to be a very durable and reliable switch. The actuation of all the limit switches was the weakest part of the air control system. Because of excessive boom movement the actuations of limit switches 2, 3, 4 and 5 were unreliable during the field trial period. The actuation of the switches was not considered an air logic or pneumatic problem, but rather a mechanical problem affecting the operation of the air system. There were no limit switch failures due to a stocking spool, or a defective part that affected the moving part air logic.

The most common problem was the destruction of the limit switch #2 lever arm which occurred five times during the field trial. The lever arm was broken during a malfunction in the unload portion of the drill cycle. The malfunction was caused by a portion of the boom, other than the cam itself, striking the lever arm. The premature actuation of the switch by the boom resulted in the carousel extending early and deflecting the boom into the lever arm, causing it to break. The problem was effectively eliminated by grinding and filing those parts of the boom that came into contact with the limit switch.

The complete destruction of limit switch #2 occurred twice while the vehicle was in the mine. The first accidental destruction occurred when the vehicle was trammed into the mine. This again was caused by movement between the head and the boom during tramping. The second destruction of limit switch #2 occurred in the same manner. Under both circumstances the switch was broken by miners unfamiliar with the vehicle. The adjustments on limit switch #2 included moving the switch forward or back, a limited left and right movement, and an up or down movement

of the limit switch lever arm. However, the total boom movement was greater than all of the adjustments for the limit switch. The replacement time for the entire switch takes approximately an hour if a spare valve and fittings are available.

The two air lines connected to limit switch #2 were severed once during the field trial. This was a result of a hot drill bit on the lead steel accidentally falling on the air lines when the lead steel was being changed. The 1/4" pneumatic lines are made from polyethylene material (polyflo) which was not designed to withstand high temperatures. The two 1/4" lines and all air logic lines in the future machines should be protected with a steel enclosure to prevent any damage. The repair of these two 1/4" lines required approximately 15 minutes to complete due, in part, to the fact that spare tubing and fittings were carried on the machine. A concern of the Bethlehem Steel Research Group was the contamination of the small pneumatic lines with dust or water. Their feeling and experience indicated that the pneumatic line problems began at the end of four months and only increased with time. The Bethlehem people also felt that once a line was contaminated in the mine there was no effective way of cleaning the line and no easy method of replacing a damaged or contaminated line on the vehicle. It is certain that both of these conditions will arise in any future field trials, and a procedure should be recommended for cleaning and replacement of the lines before a future field trial.

Another problem associated with the pneumatic lines is the fatiguing and/or cracking of the plastic fittings used in securing the plastic lines to the hydraulic valves. The hydraulic valves reached a temperature in excess of 120°F because of the heat from the hydraulic oil. This high temperature will result in the failure of a pneumatic connection if not corrected. A possible solution is (a) change from the polyflo fitting to another type (LEGRI), (b) the use of impolene ferrules rather than polyethylene, or, (c) keep the hydraulic oil approximately 120°F.

The #3 limit switch was mounted on the carousel frame and its function was to control the upper limit of the boom travel. The lever arm adjustment was a source of constant aggravation when the seam height approached fifty-six inches. The switch was never destroyed and only one lever arm was replaced during the field trial. The excessive boom movement from left to right affected the correct actuation of the switch in all seam heights; however, the boom would stop within the allowable limits a high percentage of the

time so that the cycle would continue without interruption. One possible relocation for the switch would be from a modified upper cover plate. The plate would have a maximum upper limit mechanical stop for the boom with the limit switch mounted from the stop. This would ensure the boom stopping at the same locations each time, in either the load or unload portion of the drill cycle.

The #4 limit switch was mounted in the frame of the vehicle and was activated by the boom arm raising and lowering. The function of this limit switch was to signal the boom lower limit stop position. This switch was never destroyed and never required a lever arm replacement during the field trial. When the vehicle was on level ground or on a slightly slanted up-grade, the switch actuation presented no difficulties. However, when the vehicle was positioned on a slight downhill grade the switch lever arm required constant adjustment and could never be satisfactorily adjusted for all possible mine floor conditions. If the switch was actuated prematurely the carousel would retract at the incorrect time. This often resulted in the drill steel jamming in the carousel and consequently shearing the ratchet key. Drill cycle failure also occurred when the boom dropped completely to the ground and the lever arm did not rotate sufficiently to actuate the limit switch. The simplest corrective action taken at that time was to manually actuate the switch, which would in turn retract the carousel. In future designs the relocation of the limit switch to the head of the vehicle, where the boom travel is not critical, should alleviate the problem.

The last limit switch for discussion is limit switch #5. The function of this switch is to open the centralizer when the boom has risen to a set height. The switch was destroyed once because of interference with the boom just prior to or during tramming. The switch actuation presented no problem during the field trial and did not malfunction when signaling the moving part air logic. The switch location and operation can be considered relatively trouble-free for the overall field trial.

Limit switch #3 presented no problem with actuation, position, or operation during the entire field trial. The switch's function was to establish the position of the carousel as either extended or retracted during the work cycle. The switch was mounted on the carousel arm and was not affected by the excessive boom movement.

When discussing the air limit switches, there were no major unsolvable problems associated with them. The Aro

switches employed in the machine are definitely mine-worthy. If mounting in future designs is considered the severest drawback, a suitable solution could definitely be found.

There were two minor problems encountered with the air logic system. The first, and of no real consequence to the machine operation, was the shifting of the "drill" indicator light. This malfunction occurred between the completion of a hole and the start of the next hole. The feed button was depressed for the cycle to start. At this time the drill indicator light would appear and then "shut off". This malfunction occurred only with the first drill steel and not in the remaining three.

The second minor problem encountered with the air system was during the unload portion of the cycle. This problem also occurred with the first drill steel only, and had little effect in the operation of the machine. The problem appeared after the retract button was depressed and the boom was accelerating downward. It would strike limit switch #2, but fail to stop in the proper location. This failure to stop could be attributed to a malfunction with the air logic or a questionable adjustment with limit switch #2, and was difficult to determine because it was intermittent. Because of the adjustment problem involving the limit switch it is felt that the problem poses no significant drawbacks with the air system.

In future designs a selector switch on the pneumatic panel that would separate the bolting operation would be beneficial. This mode of operation would ensure that the upper clamp arms would open before inserting a bolt and allow the eventual automatic bending and insertion of a bolt. This would allow pneumatic control of an automatic placement of the bolt in a bolt bender (Bendix) or other design, bending of the bolt, inserting the bolt into the roof, and torquing the bolt to a pre-determined setting. In the field trial the air logic system was modified to make it possible for the lower clamp to remove the bolt torquing adapter from a bolt head after torquing. The operation could be easily included as a revision to the air logic circuit, but isn't recommended because of the simple mechanical solution of the problem. Because the actual automatic bolt bending cycle in future machines is undetermined at this point the air logic circuit could be easily added as a separate module controlled by the selector switch previously discussed.

The on-board air compressor was a dual piston, oil-less Rego model 9722. Power to the compressor was applied

whenever the dust collector blower was activated. Thus, the Flex Drill had the advantage that drilling could not begin unless the dust collection system was operational. The compressor worked against a constant 70 psig head provided by a fixed relief valve. The choice of the Rego unit was an excellent one since it provided more than adequate air to the logic system for six months of above and belowground testing.

21.4 Upper Clamp Arms

The upper clamp arms are located above the centralizers on the front of the machine as shown in Figure 20. The vertical location of the clamp arms is important for utilizing the maximum possible drill steel length. The upper clamp arms are required for the proper coupling and uncoupling of the drill steels. During the load portion of the cycle (2nd cycle latch) the upper clamp arms (1) support the drill steels already in the hole, (2) allow the next drill steel to index under the supported drill steels and couple with the supported drill steels. In the unload portion of the cycle the upper clamp arms support the remaining drill steels in the hole while the first drill steel is being uncoupled and placed in the carousel.

a. Forces Applied to Upper Clamp Arms

The upper clamp arms receive high bending loads during the load and unload portions of the cycle. During the unload cycle the clamp arms are subjected to a downward force resulting from the uncoupling of the drill steels. They are forced upward when the drill chuck couples with the drill steels remaining in the hole. The vertical forces applied to the upper clamp arms during unload may separate the clamp arms and cause the interruption of the automatic cycle. If the upper clamp arms should separate, the boom lift and drill rotation controls must be jockeyed to complete the coupling sequence.

At times, one upper clamp arm was required to carry the entire coupling load because a hole started at an angle to the drill chuck. At this point a severe deflection of the upper clamp arm would result which could destroy the clamp arms. Because this condition may happen at any time the upper clamps should be constructed to withstand these loads.

b. Wear on Upper Clamp Arms (Area "A", Figure 21)

The area of the upper clamp arms that contacted the drill steel coupling became worn and ineffective with the completion of the four hundred holes. The sharp edges on the clamp arms are required for effective coupling of the drill steels for the load and unload modes.

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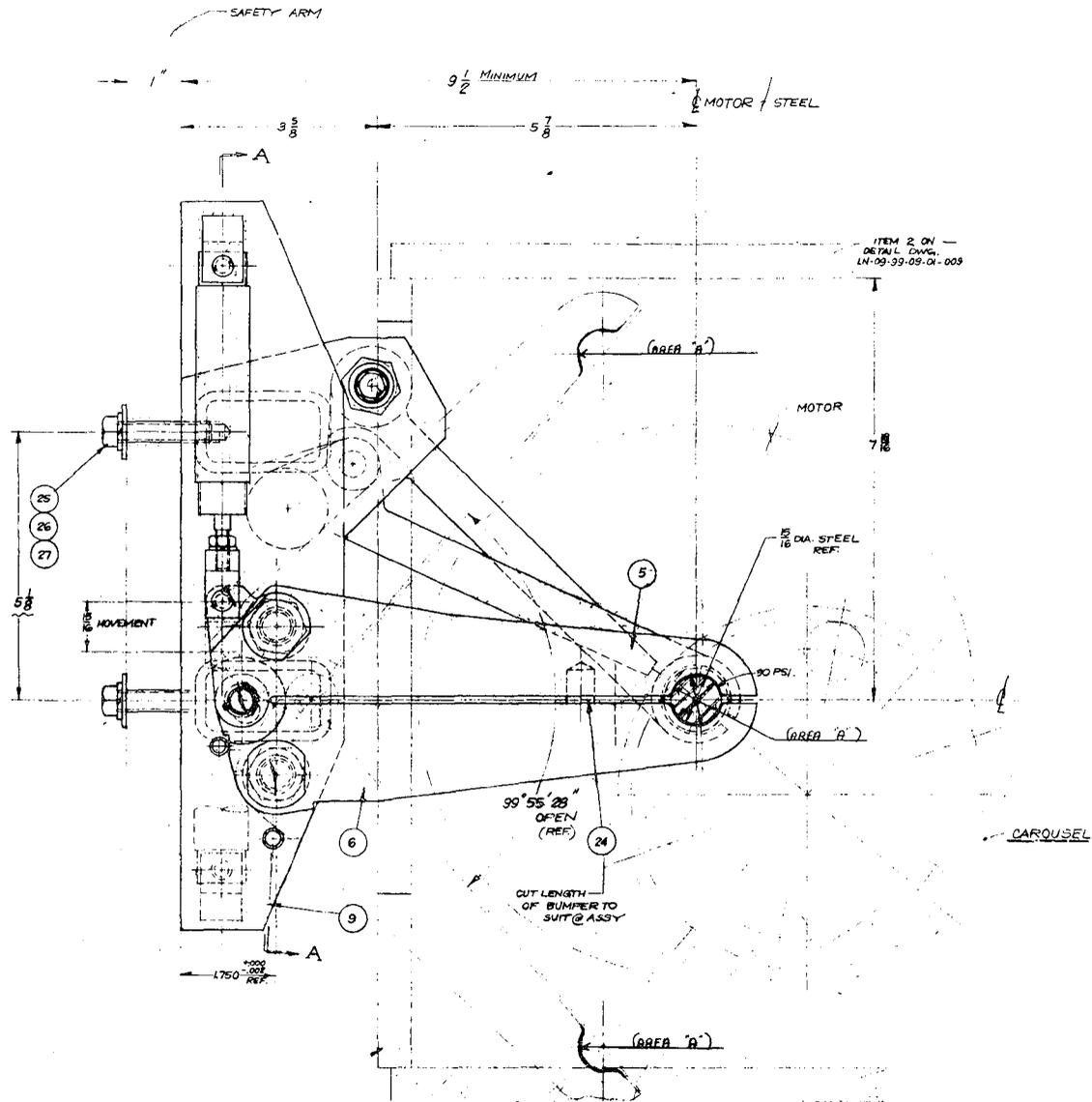


Figure 21: Upper Clamp Arms and Centralizer - Top View

When the edges are rounded off, problems are evident in the unload mode. The drill chuck engages the drill steels supported by the upper clamp arms and separates the arms before coupling can be completed. When the arms have separated, coupling becomes impossible and can be completed only through manual operation of the boom.

As a result of these problems, a sharp edge on the upper clamp arms must be maintained by either heat treating the steel edges or making hardened inserts that could be bolted into the clamp arms.

c. Maintenance of Upper Clamp Arms

The upper clamp arms used throughout the test were installed on the vehicle during the second week of the field trial so that an accurate evaluation of the arms could be made. The installation of the new arms required a slight grinding of the arms and thrust washers, and a fitting replacement for the hydraulic hose. When the new arm installation was complete the tolerances were as required by the drawings.

The problem encountered most often with the arms was the loosening of the retaining nuts securing the arms. (Area "C", Figures 20 and 22). The nuts required tightening daily to assure coupling and minimize additional problems. The loosening of the nuts can be attributed to two factors: (1) the wear of the body that the pints rest on as shown in Figures 20 and 22, Area "A" (counter sunk area), (2) the stretched pin shown in Figure 22 (item 3). An inspection of the flat washer, arms and upper body surface showed no signs of wear or distortion in the area surrounding the pivot points as shown in Figures 20 and 22. The clearance for the parts shown was 0.002 after the nuts were tightened sufficiently. Once the shoulder of the pin wore, the upper clamp arms would bind and become inoperative. With this design it was possible to overtighten the nuts which resulted in binding or the arm or loosening of the nut. This in turn increased coupling problems.

There was a rubber bumper cemented into one upper clamp arm which was designed to eliminate the banging of the clamp arms during closing. However, the bumpers lasted approximately one month and did not cushion the impact as desired. It is not recommended for future designs.

d. Upper Clamp Recommendations

It is believed the upper clamps could be redesigned to withstand a total uncoupling load on one arm plus a manual override of the second cycle latch without breaking any parts connected to or associated with the upper clamp arms. With this accomplished, the replacement of the existing upper clamp hydraulic cylinder with a cylinder approximately four times larger in area would make the separation of the arms very difficult. The increased cylinder diameter will help prevent the forcing of the arms apart in the unload coupling operation. The design of new upper clamp arms would be enhanced with a greater mechanical advantage designed around new bearings selected for optimum thrust against the boom. With these features, in addition to being mounted directly at the top of the vehicle, the upper clamp arms would be mine-worthy in construction, operation and appearance.

21.5 Centralizers

The centralizers are located at the front of the vehicle underneath the upper clamp arms as shown in Figures 20 and 22. The centralizer's first requirement during the load and unload operations is to align the drill steel with the carousel. A second and equally important function is to guide the drill steel into vertical alignment with the drill chuck when starting a new hole. The operation of the centralizers was one of the most reliable features of the Flex Drill during the field test.

a. Wear and Maintenance

The centralizers became worn on the inside surface (area A, Figure 23) that contacted the drill steels during drill rotation. The wear on this surface was never sufficient to affect the operation of the arms, but could be eliminated with the proper heat treatment of the arms or with the selection of an alternate material. Maintenance on the centralizer arms consisted of tightening the arms (Area B, Figure 20) six times during the field trial. On one occasion during the field trial the centralizer arms were bent because of an interference with the boom. However, the arms were re-straightened on the vehicle with no associated downtime.

b. Operation of Centralizers

The operation of the arms was controlled through the moving part air logic actuated by air limit switch #5. On one occasion damage to the switch caused the centralizers to become inoperative until the switch could be replaced. The air logic circuit was designed to allow the centralizer arms to open in case of malfunction, preventing damage to the arms by boom movement.

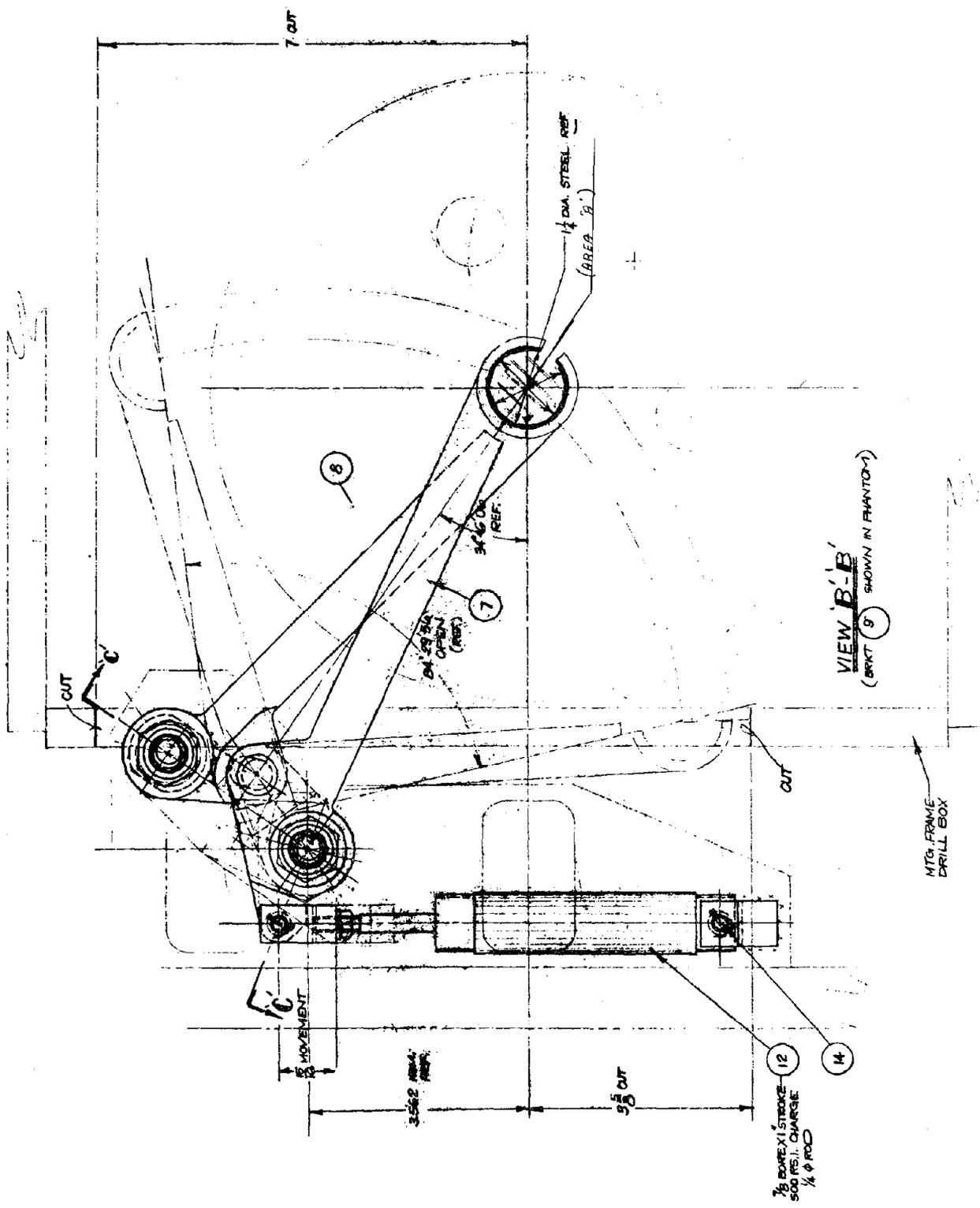


Figure 23: Centralizer - Top View

c. Centralizer Function

The centralizers successfully maintained drill steel alignment with the carousel, but could not assure vertical alignment of the drill steel during initial penetration of the steel into the roof.

The unevenness of the roof frequently caused the drill steel to "walk" along the roof before penetration began. The centralizers were unable to restrain this walking. The single most important consideration in drilling holes with this vehicle was that the initial penetration be vertical in relation to the drill chuck. A hole that started misaligned vertically relative to the drill chuck continued to create problems with the steel to steel coupling operations.

d. Recommendations

Because the initial hole straightness is an important consideration it is recommended that in future designs the arms be designed as stabilizer arms rather than centralizer arms. The stabilizer arms must be designed to withstand the full boom thrust without a permanent deformation. Also, they must maintain the vertical alignment of the drill steel regardless of roof condition. The redesign of the centralizer assembly would include an increase in cylinder size for higher closing forces, a new bearing design to distribute the higher boom thrust loads, and a re-evaluation of the mechanical advantage used in the present design.

A cam actuation of the stabilizer arms would allow simplification of the air logic circuit; however, space limitations may prevent this design change.

21.6 Lower Clamp

The lower clamp is located on the drill box near the drill chuck. It is used in the unload cycle. The primary functions of the lower clamp were to pull the drill steels out of the drilled hole and to separate the last drill steel from the remaining drill steels in the hole. The secondary function was the removal of the bolt torquing adapter from the roof.

The hydraulic rotary actuator pivots the lower clamp around the drill steel coupling "undercut" so that high tensile loads are not directed through the actuator, but through the clamp. The difficulties encountered with the lower clamp were the result of human error and machining problems not related to design. The problems with the lower clamp extended the length of the field trial, but did not affect the machine cycle time. The lower clamp required attention on three occasions as listed below.

a. Sheared Lower Clamp Key

The Woodruff key is located on the shaft of the rotary actuator and connects the lower clamp pivoting member to the actuator as shown in Figure 24, Area A. The key was sheared twice during the field trial, but was easily replaced and working within an hour. There was excessive clearance between the Woodruff key and the keyway in the lower clamp pivoting member. The increased clearance between these parts was due to a machining error and was partially responsible for the key failure.

The key sheared when the carousel struck the lower clamp during the indexing cycle. This interference with the carousel was a major factor for the key failures and was caused by an improper boom stopping position. The interference was minimized by adjusting the boom stop position and filing the leading edges of the lower clamp. The elimination of the interference can be accomplished in future redesigns of the lower clamp and carousel.

b. Replacement of Rotary Actuator

The second problem with the lower clamp resulted when a miner operator, unfamiliar with the machine, inadvertently destroyed the rotary actuator during tramming. The replacement of the actuator required two hours for removal and installation, but two days for the procurement of the actuator. The possibility for destruction will not be present in future models because a protective plate will be incorporated which will surround the rotary actuator and much of the lower clamp, precluding inadvertent damage.

The lower clamp assembly could be improved by:

1. Testing the welds on the fork and pivot assembly.
2. Heat treating the area of the lower clamp that contacts the drill steels to reduce the amount of wear on the front edge (Area B, Figure 24).

These improvements combined with the positive actions of the actuator would ensure reliable lower clamp performance.

21.7 Carousel

The carousel assembly mounts directly off the head of the vehicle. The present design allows for six drill steels with a maximum length of 18" to be stored in the carousel. During the field trial the carousel required many minor adjustments and often prevented a higher hole/day production rate than was achieved. Although the carousel required a great deal of attention it is the key component for commercial acceptance of the design.

a. Replacement of Carousel Ratchet Key

The replacement of the carousel ratchet key was required throughout the field trial. The key was replaced fifteen times during the length of the field trial, and is shown in Area "B" of Figures 25 and 26. The shearing of the ratchet key was a result of a drill steel jamming between the housing weldment and the shaft and plate assembly. The possibility of jamming the carousel could never be totally eliminated. However, in a redesign, the frequency will be substantially reduced along with the associated lost drilling time. The time required to replace a sheared key was approximately thirty minutes in lost drilling time, but was very disruptive to the work schedule.

b. Damage to the Ratchet Cylinder

The ratchet cylinder was located in the upper portion of the carousel as shown in Figure 25, Area C. This cylinder was vulnerable to roof falls and was only slightly protected by the carousel cover. The ratchet cylinder was trunnion mounted to the carousel because of space and alignment design considerations. The cylinder was designed to engage the ratchet pawl mechanism to the ratchet for the load and unload modes of operation.

The failure of the cylinder occurred at the end of the cylinder rod. The cylinder rod failed at the threaded portion of the rod adjacent to the drilled hole for pinning the rod. The cylinder replacement time, had one been available at the mine, would have been approximately two

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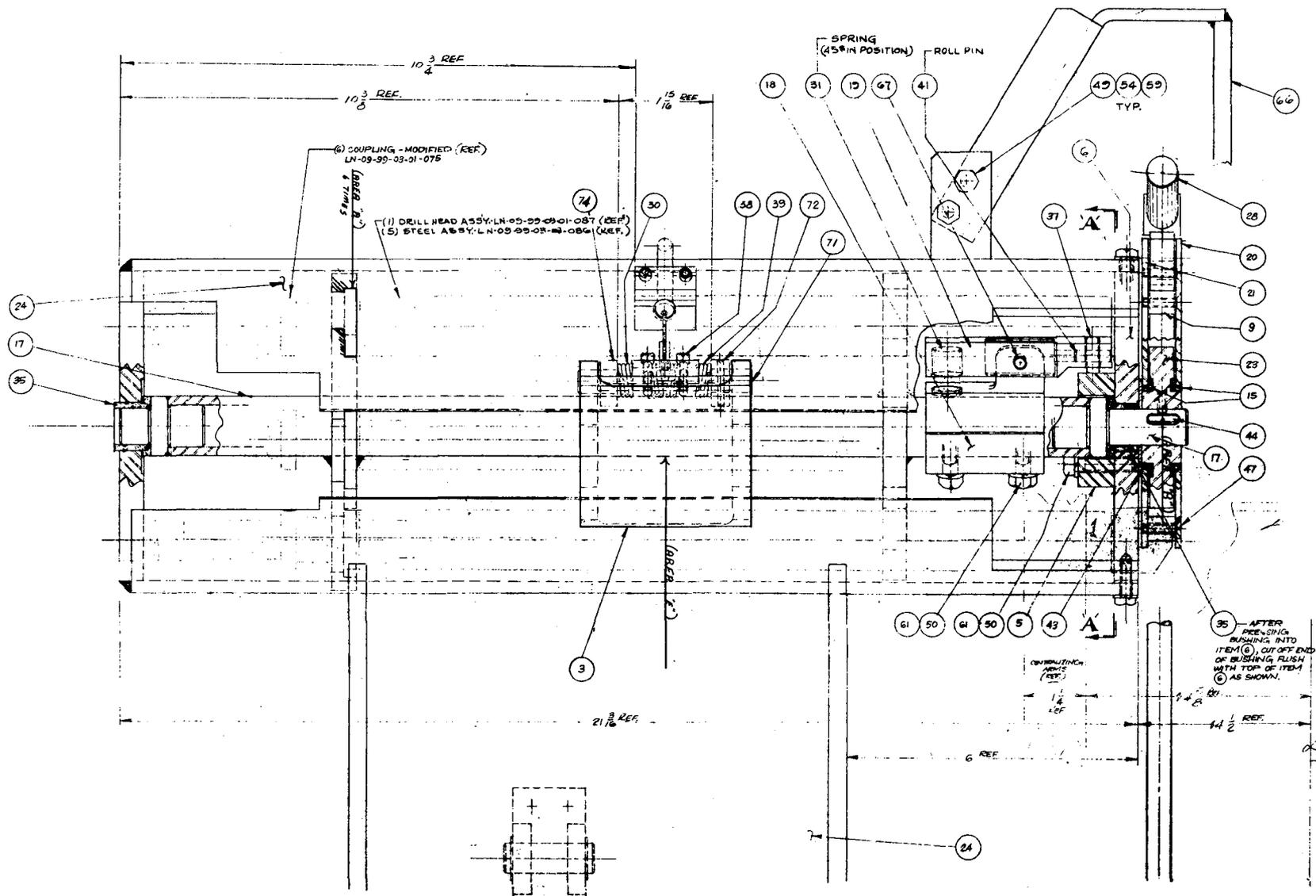


Figure 25: Carousel Assembly

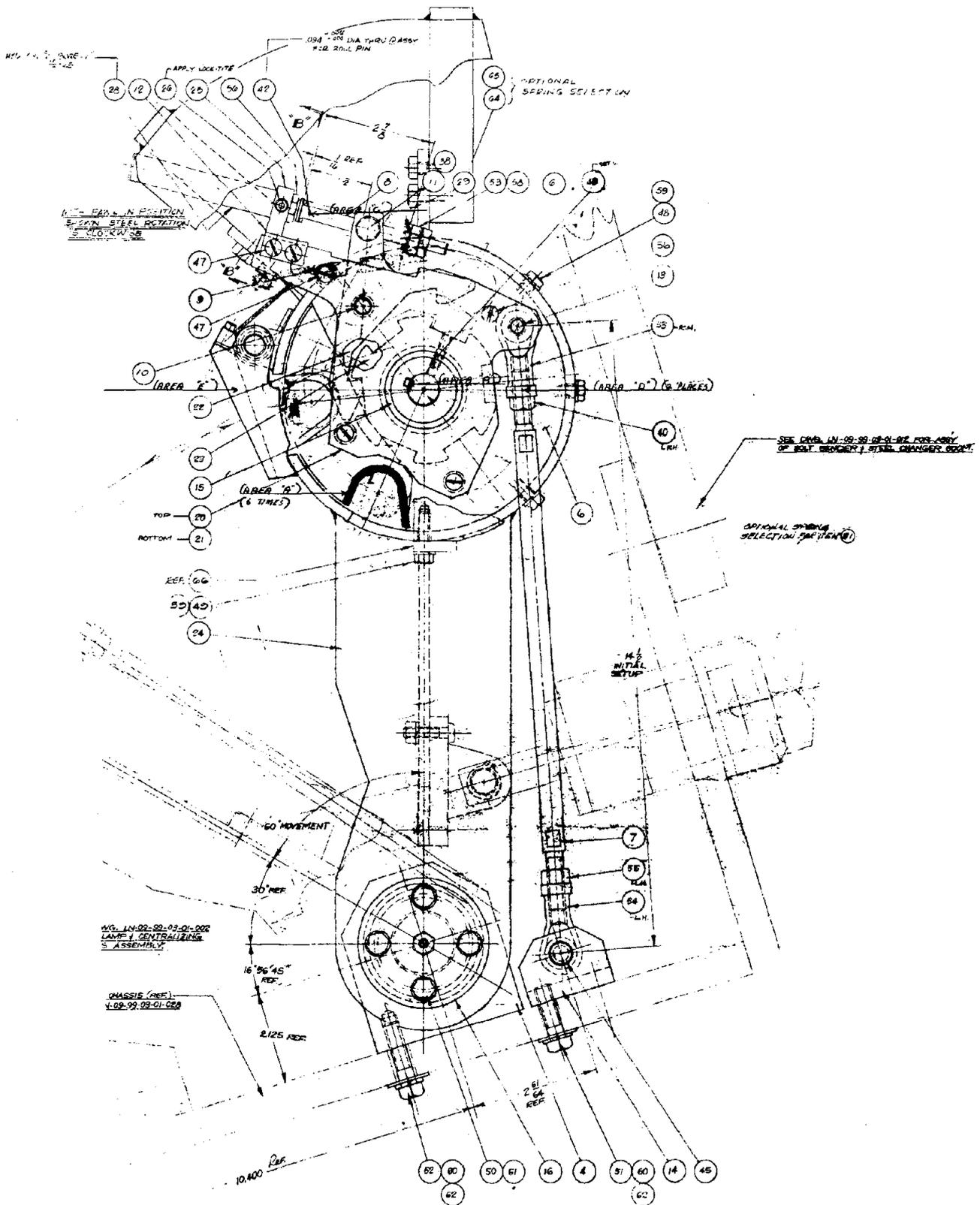


Figure 26: Carousel Ratchet Drive Assembly

hours. However, with this cylinder it was necessary to call the Ingersoll-Rand Research Center and request they procure a cylinder from the local vendor. The vendor responded by supplying a cylinder rebuilding kit within a few days. This underlies the importance of dealing with responsible vendors and the need for adequate parts supply. The actual lost time attributed to the failure of the cylinder was approximately one half day during the field trial. While the new parts were being shipped the original cylinder rod was modified so that the field trial could continue. The modifications consisted of drilling the shortened cylinder rod and pinning it to the existing bolt. The ratchet mechanism was then adjusted for proper operation.

c. Maintenance and Wear of the Carousel

Maintenance performed on the carousel was limited to replacing the ratchet cylinder and the carousel ratchet keys. The accumulation of dust and dirt in the ratchet mechanism presented no immediate problems, but would require cleaning over an extended period of time. The pawl and ratchet mechanism was lubricated on several occasions to insure smooth operation.

The areas of the carousel subject to wear were the lobes on the shaft and plate assembly shown in Figure 25, Area A. If the wear was allowed to continue, malfunctions caused by the drill steels jamming the carousel would become frequent. The wear in this area had no effect on the cycle time, but would ultimately increase the downtime. A simple heat treatment of this area would increase the wear resistance and reduce the problem.

d. Improvements and Recommendations

Several areas of improvement involve relocating the carousel, redesign of the ratchet and gate mechanisms. An explanation and description of each of the above follows:

1. Relocation of Carousel - The relocation of the carousel from the left side of the machine to the right side of the machine would allow the operator to monitor the loading and unloading of the

carousel. With the operator monitoring the relationship between the carousel and drill steels, the jamming problem would be avoided and the ratchet key would not have to be replaced. The redesign and relocation of the carousel may also include pivoting the carousel to a horizontal position for tramming and to a vertical position while bolting. A design implementing this configuration substantially reduces the cycle time and therefore enhances the early commercialization of the Flex Drill concept. The pivoting of the carousel allows for a substantially longer drill steel which reduces the number of drill steel couplings required for every hole drilled. The length of the carousel will ultimately permit this design to be competitive with a conventional roof bolter.

2. Redesign Top Plate and Ratchet Mechanism - The ratcheting mechanism could be improved by a modification of the top plate which will permit a visual inspection of the ratchet and pawl engagement. Another improvement would be the permanent attachment of the spacers to the top plate which would incorporate captive screws as the fasteners. An additional improvement could be the drilling and tapping of four holes in the ratchet; this would allow a quick removal of the ratchet for replacement of the ratchet key. The use of captive screws and a one piece top plate reduces the possibility of losing the smaller parts in the dark and dusty mine environment. The four tapped holes in the ratchet would make the removal of the ratchet possible with the least number of tools. The only tools required for replacement of the key would include an allen wrench and a screw driver.

3. Carousel Gates - The carousel gate as shown in Figure 25, Area E, is a system that worked flawlessly throughout the field trial. The cable actuation of the gates was positive and received preventive maintenance just twice during the field test.

It was never necessary to adjust the cable and it showed no sign of wear throughout the test. The

end of the cable attached to the head of the vehicle was protected by a dust shield, but close inspection of the cable indicated dust entered the protective cover. However, no serious effects of the dust were encountered with the cable during the test. Before the test was started underground, a hydraulic flow control valve was removed from the pressure port of the directional control valve because of erratic operation in controlling the gate cylinder. This change had no affect on the cycle time of the machine and only increased its reliability.

A change that would improve the reliability of the carousel operation would consist of a longer gate to hold the drill steels in the carousel. The longer gates would reduce any tilting or cocking of the drill steels in the carousel. Another improvement would provide a gate preload against the carousel. The preload would press the gate tight against the carousel, thus preventing the drill steels from falling out of the carousel during indexing. Both improvements for the redesign of the gate would impart a positive force and direction to the drill steels while indexing the carousel and replacing the drill steels into the carousel.

21.8 Drill Steel Coupling

A total of 16 couplings were used during the field trial for the completion of five hundred and two holes. Five couplings were replaced on the drill steels used to complete the field trial. These five couplings were broken under mine operating conditions considered normal for production bolting crews. The coupling breakage occurred when starting the hole into the roof at an angle with the drill chuck. When a slightly crooked hole was started in the roof the resultant bending stresses applied to the coupling would shorten the drill steel life. If the drill angle with the chuck exceeded approximately 3° the possibility of coupling failure was greatly increased. The uneven roof conditions are the causes for the beginning of all "crooked holes" which shorten the coupling life. To prevent exceeding the 3° drill angle with the drill chuck the redesign of the centralizers into stabilizer arms (as stated in the centralizer section of the report) would be required.

The first drill steel coupling was sheared on October 6 while the test vehicle was still in a side entry near the entrance to the mine. This failure was a result of the high seam height, rough top, and the improper drill angle with the chuck. The second drill steel coupling was sheared on November 4 in the production area of the mine while completing the four hundred holes. The third and fourth failures occurred on Monday, November 8. With an exceptionally bad penetration angle in one hole, both of the two drill steel couplings were sheared at apparently the same time in adjoining drilling steels. The fifth coupling failure occurred on November 30, for apparently fatigue factors rather than a bad penetration angle.

There were eleven drill steels used for drilling the holes and two drill steels required for the bolt torquing adapter. The eleven drill steels used for drilling were each subjected to a different number of cycles during the completion of the test. However, on Friday, November 12, the original five drill steels and couplings were replaced after approximately 275 holes. The coupling and drill steels were replaced due to difficulties in coupling the drill steels during the second cycle latch period of the load cycle and the coupling time period in the unload mode. The coupling problems resulted from wear on the leading edges of the coupling engagement area by rolling over the metal at the points of the external hex.

The coupling problems were overcome by manual operation of the boom and drill rotation until coupling was completed. The factors affecting the coupling life are the number of holes drilled, drill penetration angle, coupling to chuck alignment, material and heat treatment of the steel. The life of the first five drill steel couplings was approximately two hundred seventy-five holes. The exact number cannot be specified because of broken couplings and only four out of the six possible couplings in the carousel were used for drilling a hole. All of the coupling problems encountered were resolved through manual movements; however, this increased the total cycle time. The life of the drill steels used in the production bolters was approximately 1500 holes, with no additional drill steel coupling parts required. The drill steels supplied to the Flex Drill would equal the present production bolters, but the couplings would require a number of changes. The acceptable life for the drill couplings will have to be determined after more testing is completed.

21.9 Dust Collection System

The Flex Drill vehicle uses the same dust collection system as a standard Lee-Norse RD-1-29 roof bolter. With this design the blower is belt driven by the electric motor. The hose connecting the blower, the dust box and the drill chuck is flexible and allows for full boom movement.

In order to determine the vacuum for the dust collection system in the Flex Drill, several vacuum test measurements were taken and the results are the following:

Place Measured	LN Roof Bolter	Flex Drill (w/o insert)	Flex Drill (w/ insert)
At drill chuck	14" Hg	14" Hg	14" Hg
1 - drill steel	*	11" Hg	13+" Hg
4 - drill steels	*	10" Hg	12+" Hg

* not available for test.

The first test conducted determined the vacuum at the drill chuck for both vehicles. The following tests on the Flex Drill vehicle revealed a loss of vacuum at the chuck. A steel insert was fabricated and pressed into the drill chuck. The vacuum loss was less than 1" of Hg at the modified surface drive chuck. The loss of vacuum for the string of 4 drill steels was less than 2" of Hg.

The significant differences between the Flex Drill vehicle vacuum system and the standard bolter vehicle vacuum system were:

- a. The decreased inside diameter of the drill steel coupling
- b. The number of drill steel couplings required for completing the drill hole (4 vs. 1).

In order to compensate for these restrictions and losses the blower could be replaced with a larger size and its speed increased. The modification to the drill steel coupling or overall drill steel lengths are covered in Section 10.

21.10 Rod Changer Boom Analysis and Design

The rod changer concept, as designed, requires the drill to travel in a straight line co-linear with the axis of the carousel. The machine, as modified for Phase V testing, is to operate in seam heights from 48 inches to 60 inches. The drill couples with the drill steel approximately 24 inches from the roof and must travel to the roof in a straight line. To insure these conditions it was necessary to analyze the boom and the safety arm of the proposed bolter vehicle, the Lee-Norse Top Dog TD-1-29. Modifications could then be made, if necessary, to obtain proper motion.

The boom and the safety arm are classified as slider crank mechanisms. The drill and carousel are mounted on the coupler link of their respective mechanisms so the path of the drill and the location of the carousel was modeled by the coupler point curve of a slider crank mechanism. A computer program (included) was written to analyze this mechanism. Variables in the computer program are the crank length, the slope of the slider, the eccentricity of the slider, the initial position of the crank, the final position of the crank, and the step size of the crank rotation. (See Figure 27 for the nomenclature). With these variables the current mechanism as well as an infinite number of changes could be modeled. The computer program, when supplied with the above variables, calculates the position of the coupler point c (see Figure 27), in x and y coordinates for each position of the crank. These points are then plotted to provide a visual representation of the coupler point travel. Since the point m (see Figure 27) on the TD-1-29 roof bolter is 24 inches above the ground, the y-axis range of interest is from 0 inches to 36 inches. The carousel axis is parallel to the x-axis.

The existing boom mechanism was analyzed on the program. The following is a list of the initial conditions:

Crank MA = 25.25 inches

Coupler AB = 35.25 inches

Trace Length BS = 90.25 inches

Initial Angle of Trace Length = 0°

Slope of Slider = 0°

- MA = CRANK LENGTH
 AB = COUPLER LENGTH
 BC = TRACE LENGTH
 α = INITIAL ANGLE OF TRACE LENGTH
 β = SLOPE OF SLIDER
 E = ECCENTRICITY
 ψ = ANGLE OF COUPLER LENGTH
 θ = ANGLE OF CRANK

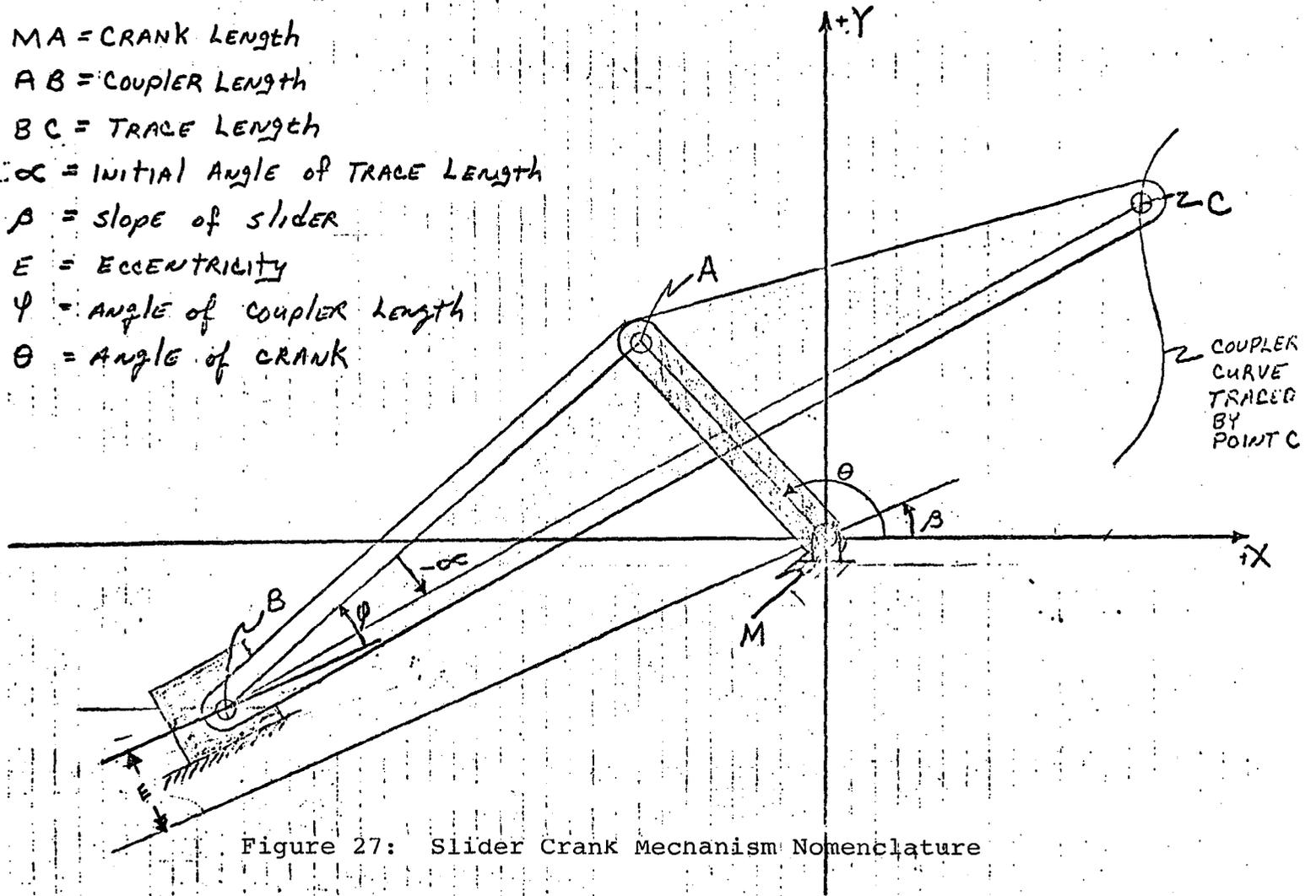


Figure 27: Slider Crank Mechanism Nomenclature

Eccentricity of slider = 0 inches

The maximum variation of the coupler point curve (see Figure 28) from the x axis is .2883 inches. This deviation is too great for the rod changer mechanism, therefore a design modification to the boom is necessary.

The existing safety arm was analyzed with the computer program. The initial conditions were:

Crank MA = 30.353 inches

Coupler AB = 35.833 inches

Trace Length BS = 95.756 inches

Initial Angle of Trace Length = -4.536°

Slope of Slider = 0°

Eccentricity of Slider = 0 inches

The variation of the coupler point curve (see Figure 29) from the x-axis is 1.1553 inches. This deviation is too great for the rod changer design so the safety arm must be re-designed.

After many substitutions of the variables in the computer analysis a useable solution was found and is shown in Figure 30. The safety arm mechanism had the following dimensions:

Crank MA = 25.25 inches

Coupler AB = 35.25 inches

Trace Length BC = 90.25 inches

Initial Angle of Trace Length = 0°

Slope of Slider = 0°

Eccentricity of Slider = .500 inches

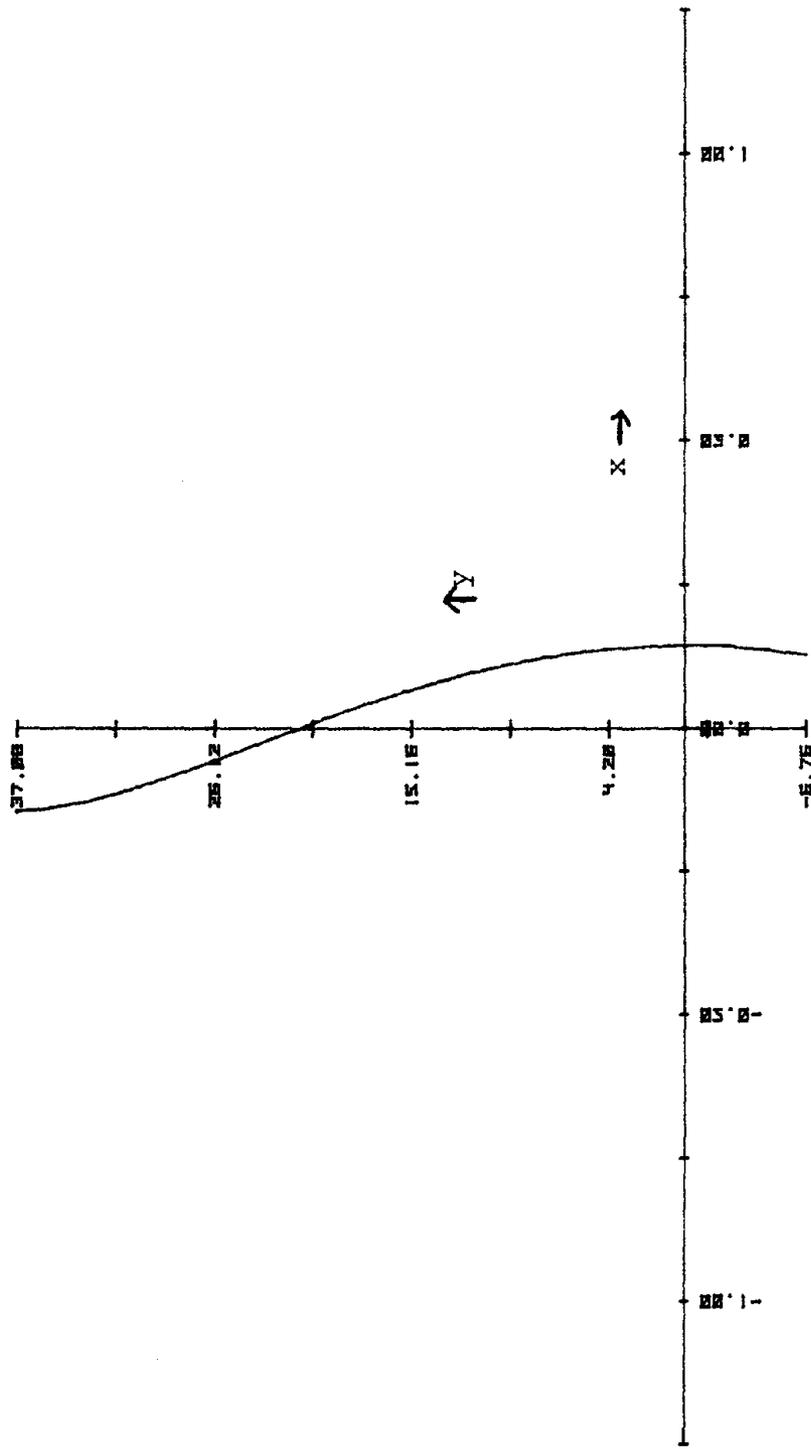


Figure 28: Point Curve Original Boom

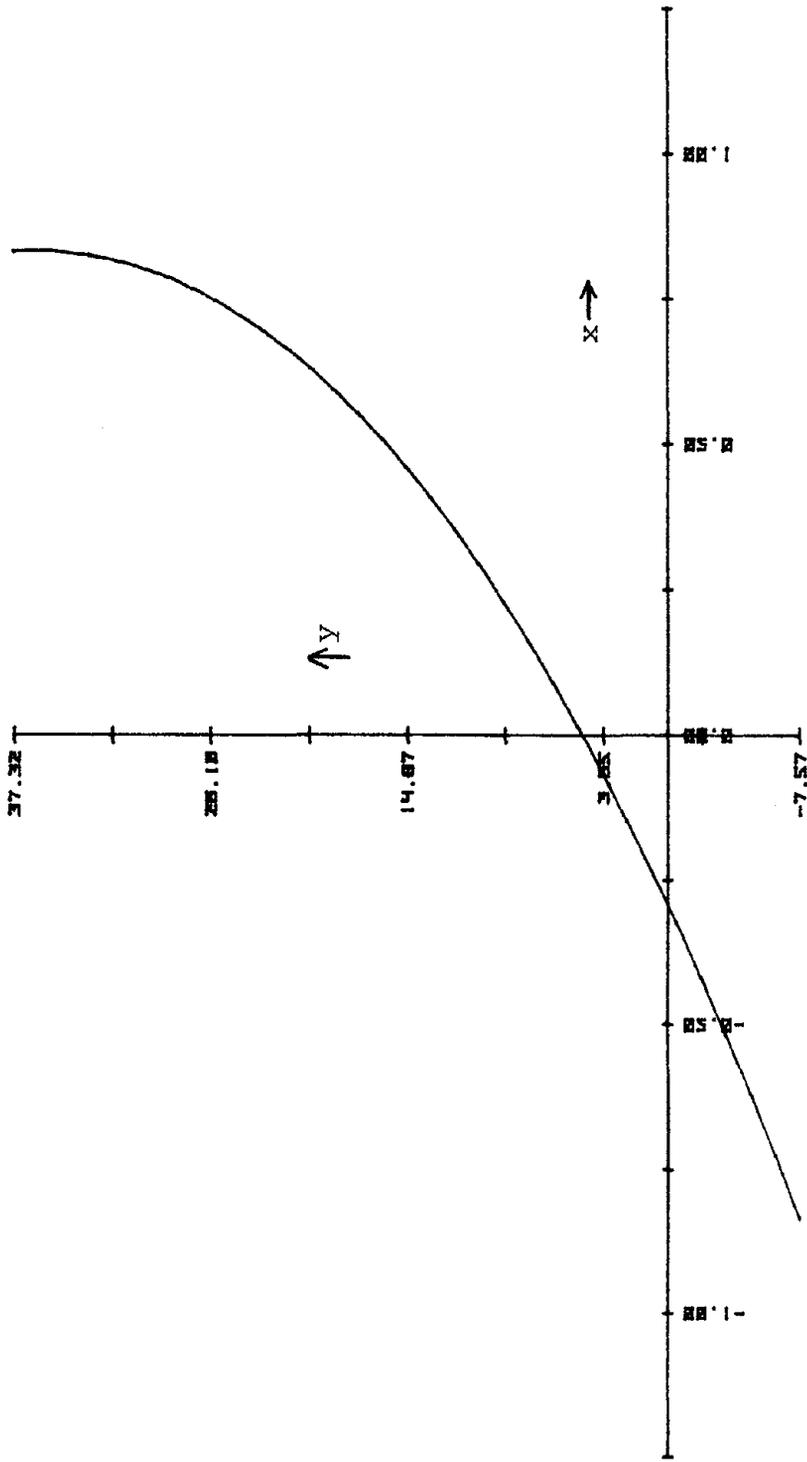


Figure 29: Point Curve Original Safety Arm

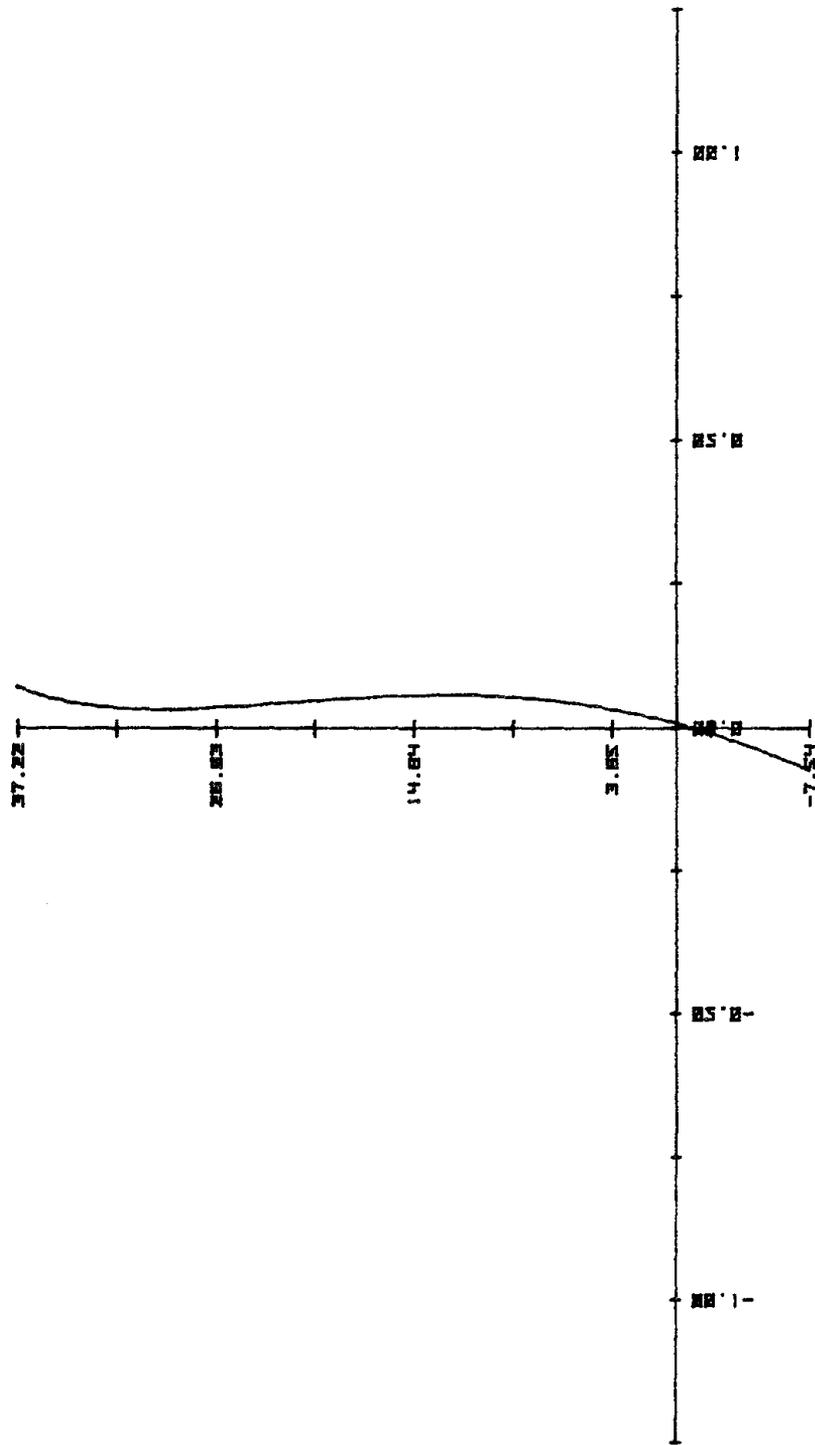


Figure 30: Point Curve Modified Mechanism

This mechanism has only .0504 inches of deviation from the x-axis in the operating range of the rod changer drill. This is an acceptable variance from the straight line desired and can be used. The mechanism is identical to the existing boom mechanism with the exception of the .5 inch eccentricity. By making the safety arm identical to the boom and by lowering the point m (see Figure 27) on the TD-1-29 chassis by .5 inches the design change can easily be made.

Computer Program for analysis of Boom & Safety Arm Travel of the Lee-Norse Top Dog Roof Drills.

The program is stored at IRRI Computer facilities on tape for use on the Hewlett Packard model 9830 mini computer on tape 17, file 3, in Basic Language.

```
100 DIM XS 256 , YS 256
105 N=0
110 PRINT
120 PRINT
130 PRINT "ANALYSIS OF COUPLER POINT CURVES FOR A SLIDER
        CRANK MECHANISM"
140 PRINT
150 PRINT
160 DISP "CRANK LENGTH"
170 INPUT L1
180 PRINT "CRANK MA = L1"
190 PRINT
200 DISP "COUPLER LENGTH"
210 INPUT L2
220 PRINT "COUPLER AB=" L2
230 PRINT
240 DISP "TRACE LENGTH"
250 INPUT L3
260 PRINT "TRACE LENGTH BS =" L3
270 PRINT
280 DISP "INITIAL ANGLE OF TRACE LENGTH"
290 INPUT 0
300 PRINT INITIAL ANGLE (DEG) OF TRACE LENGTH = 0
310 PRINT
320 0 = 0 *PI/180
330 DISP "SLOPE OF SLIDER"
340 INPUT B2
350 PRINT "SLOPE OF SLIDER = B2"
360 PRINT
370 B2 = B2 *PI/180
380 DISP "ECCENTRICITY"
390 INPUT E
400 PRINT "ECCENTRICITY"
410 PRINT
420 DISP
430 INPUT T1
440 DISP "INPUT-FINAL POSITION (DEG) "
```

```

450 INPUT T2
460 DISP "INPUT-STEP SIZE (DEG)"
470 INPUT T3
480 PRINT THETA CX CY"
490 PRINT
500 X1 = 1E + 99
510 Y1 = 1E + 99
520 X2 = 1E - 99
530 Y2 = 1E - 99
540 J = 0
545 N = N + 1
550 FOR I = T1 to T2 STEP T3
560 J = J + 1
570 T + I *PI/180
580 REM FNA (X) = ARCSIN (X)
590 DEF (FNA (X) = ATN (X/SQR (1 - X * X + 1E - 99))
600 T4 = (L1/L2) * SIN (T-B2) - E/L2
610 P1 = FNA (T4)
620 S = - (L2 *COS (P1) - L1 *COS (T-B2)
630 S1 = S *COS (B2)
640 S2 = S *SIN (B2)
650 E1 = E *COS (PI/2 - B2)
660 E2 = E *SIN (PI/2 - B2)
670 X {J} = S1 - E1 + L3 *COS (P1 + 0 + B2)
680 Y {J} = S2 + E2 + L3 *SIN (P1 + 0 + B2)
690 PRINT " " "I" "X{J}" "Y{J}"
700 IF X1 X {J} THEN 720
710 X1 = X {J}
720 IF X2 X {J} THEN 740
730 Y1 = Y {J}
740 IF X2 X {J} THEN 760
750 X2 = X {J}
760 IF Y2 Y {J} THEN 780
770 Y2 = Y {J}
780 NEXT I
790 REM SET UP PLOT PROGRAM
800 IF Y1 0 THEN 820
810 Y1 = 0
820 IF Y2 0 THEN 840
830 Y2 = 0
840 Z1 = (X2 = X1)/2
845 IF N # 1 THEN 1180
850 Z2 = (Y2 = Y1)/2
860 IF (X2 - X1) 2 THEN 910
870 SCALE -1.25, 1.25, Y1-0, 1 *Z2, Y1 + 0, 9 *Z2
880 A5 = 7/10
890 X AXIS 0, 0.25, -1.25, 1.25
900 GOTO 940
910 SCALE 1.25 *(X1 - Z1), 1.25* (X2 - Z1), Y1-0, 1* Z2, Y1
+ 0.0 *Z2
920 A5 = 7/10

```

```

930   X AXIS 0, (X2 - X1)/10, X1 - Z1, X2 - Z1
940   Y AXIS 0, Z2/10, Y1, Y2
950   H5 = 1
960   LABEL (*, H5, 1.3, 0. A5)
970   FOR Y5 = Y1 to Y2 STEP Z2/5)
980   PLOT 0, Y5, 1
990   C PLOT 0, Y5 1
1000  LABEL (1010) Y5
1010  FORMAT F7.2
1020  NEXT Y5
1030  LABEL (*, H5, 1.3, PI/2, A5)
1040  IF (X2 - X1) >2 THEN 1120
1050  FOR X5 = -1 to 1 STEP 0.5
1060  PLOT X5, 0, 1
1070  CPLOT -6, -0.3
1080  LABEL (1090) X5
1090  FORMAT F5.2
1100  NEXT X5
1110  GOTO 1180
1120  FOR X5, 0, 1
1130  PLOT X5, 0, 1
1140  C PLOT -6, -0.3
1150  LABEL (1160) X5
1160  FORMAT F5.2
1170  NEXT X5
1180  REM PLOT ANALYSIS DATA
1190  FOR K = 1 to J
1200  PLOT X {K} - Z1, Y {K}
1210  NEXT K
1220  PEN
1225  GOTO 160
1230  END

```

21.11 Coupling Force Tests

1. Purpose: to determine the average forces necessary to couple and uncouple various coupling-drill steel combinations.

1. Couplings Tested:

- a. External hex: spring retainer diameters of 0.720, 0.725 and 0.730 inch.
- b. Internal hex: Two pieces with "across the flats" (avg) measurements of 0.817 and 0.809 inch.

3. Test Setup: A heavy duty tensile testing machine was used to compress and pull apart the coupling pair at constant rates. Three readings were taken of each pair under both coupling and uncoupling.

4. Results: See Figures 31 and 32

5. Conclusions:

- a. Coupling forces resulting from sliding friction after spring compression are higher than the corresponding uncoupling forces.
- b. The largest forces occur during coupling as the barrel spring is compressed. For each retainer diameter, the approximate forces are:

0.720 diameter - 50 lbs.

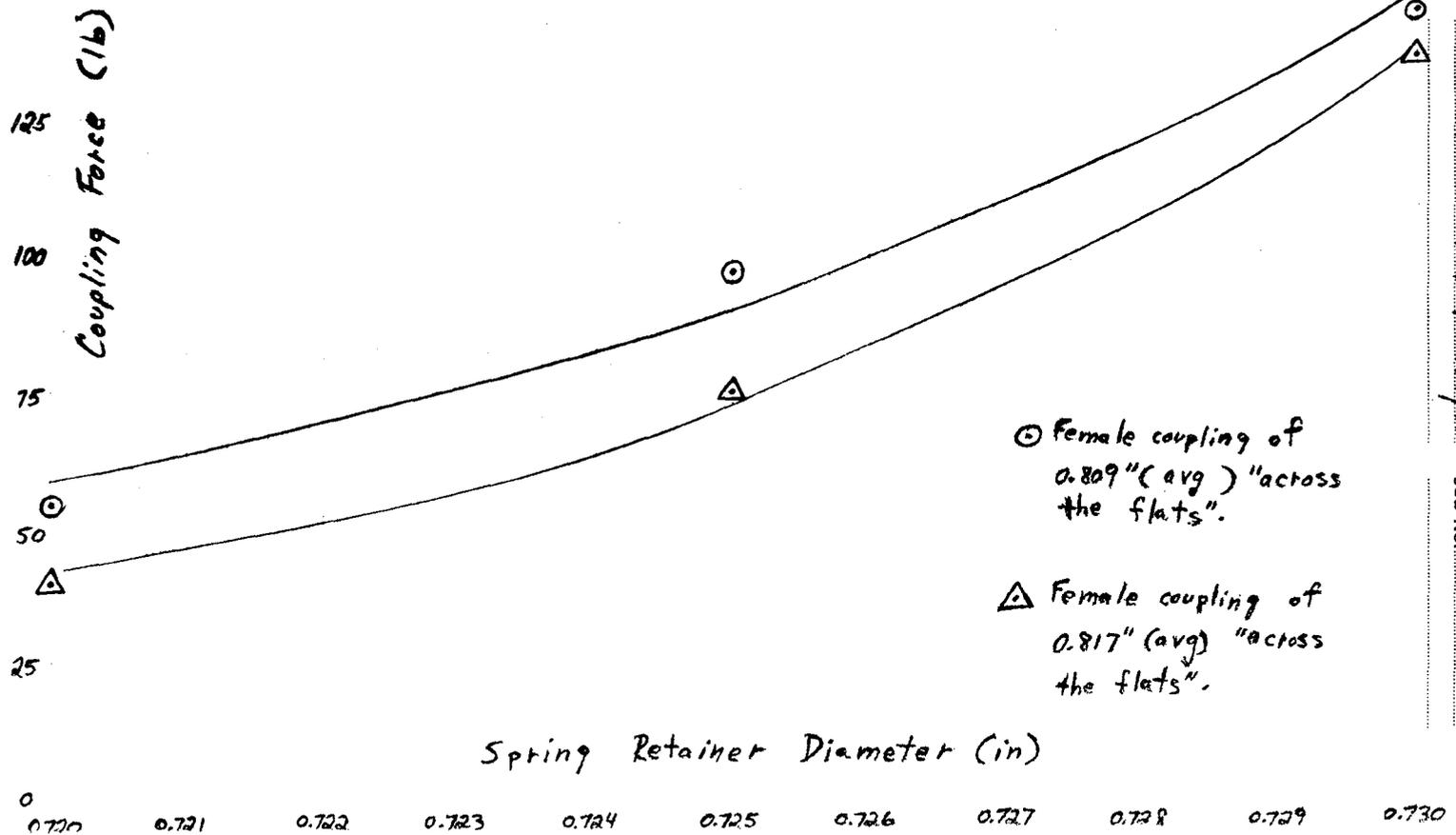
0.725 diameter - 85 lbs.

0.730 diameter - 140 lbs.

- c. All loads were gradually applied. Any shock loading during coupling will increase these forces.

Figure 31: Coupling Force vs. Spring Retainer Arm

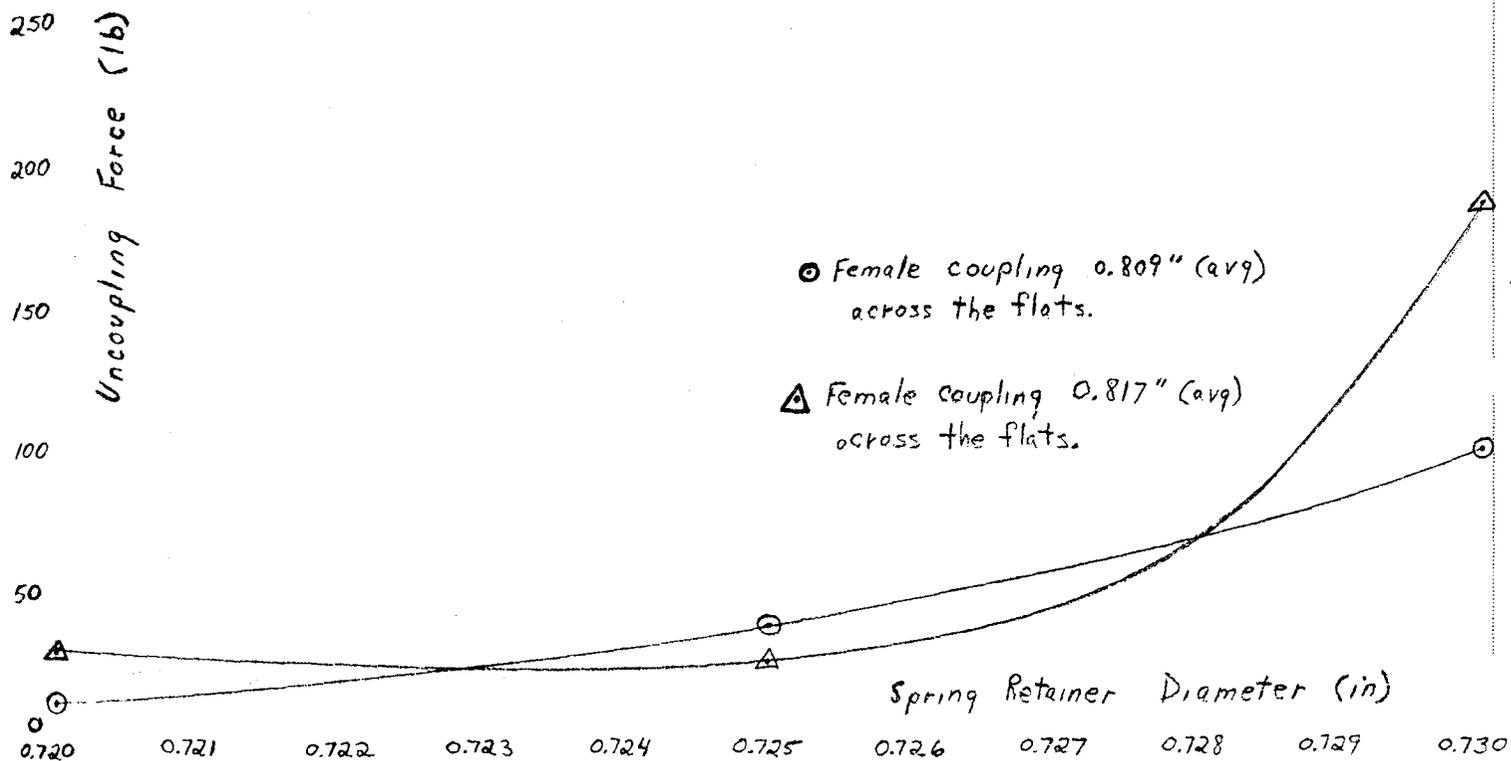
Coupling Force as a Function of Spring Retainer Diameter
using
Two Female Couplings (sleeves)



BY: T. Wall DATE: 6-20-75 SUBJECT: LN-09-99-03
CHKD. BY: DATE: FLEX Drill Rod Changer
SHEET NO. 1 OF 1
JOB NO.

Figure 32: Uncoupling Force vs. Spring Retainer Arm

Uncoupling Force as a Function of
Spring Retainer Diameter
using
Two Female Couplings (sleeves)



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BY: T. Wall DATE: 6-23-75 SUBJECT: LU-09-99-03
CHKD. BY: DATE: FLEX Drill Rod Changer
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JOB NO.

- d. Variations in data in the coupling tests within a given pair resulted from non-linear initial line-up of the coupling pair.

21.12 Safety Arm Racking Test

A pressure gauge was installed on the safety arm raising cylinder. A wooden 2 x 4 was placed on each side of the safety arm. The carousel was swung over the drill chuck so that a steel was in line with the axis of the chuck. 3 tests were run, with the following results:

1. Full hydraulic pressure applied to safety arm with both 2 x 4's contacting tower. Result: no appreciable disalignment of chuck and steel, limit switch tripped properly.
2. 2 x 4 used on left side of arm only. Could not read less than full pressure due to pilot operated check in line. Full pressure caused steel to move roughly 3/8" from chuck. Coupling under this condition would be very difficult. Limit switches tripped properly.
3. 2 x 4 used on right side only. Again, could only read full pressure of 1250 psig. Under these conditions, chuck-steel misalignment was roughly 1/4". Coupling would be difficult but fairly reliable. Limit switches tripped properly.

Due to the possibility of misalignment occurring when contacting an uneven mine roof, a pressure reducing or relief valve should be included in the safety arm circuit. Adjusting for the maximum permissible roof pressure should result in an acceptable amount of misalignment.

21.13 Drill Torque and Thrust Tests

The Flex Drill was tested for torque and thrust capability by installing a series of strain-gages in an actual drill steel.

The gages were connected by a "wrap-around" cable approximately 50 feet long. The far end of the cable was connected to a multi-track voltage recorder and a storage oscilloscope.

The two time - varying voltage signals were stored on magnetic tape. A printout of the tape was later made and analyzed.

The test consisted merely of wrapping the cable around the steel and drilling a hole. As the drill rotated, the cable unreeled. When the end was reached, the cable wrapped around the steel in the opposite direction.

With constant drill penetration, the following results were obtained:

Thrust	5100 - 5500 lbs.
Torque	212 - 272 ft/lbs.

Details of strain-gage installation, calibration curves and recorder print-out are provided in Figures 33, 34 and 35.

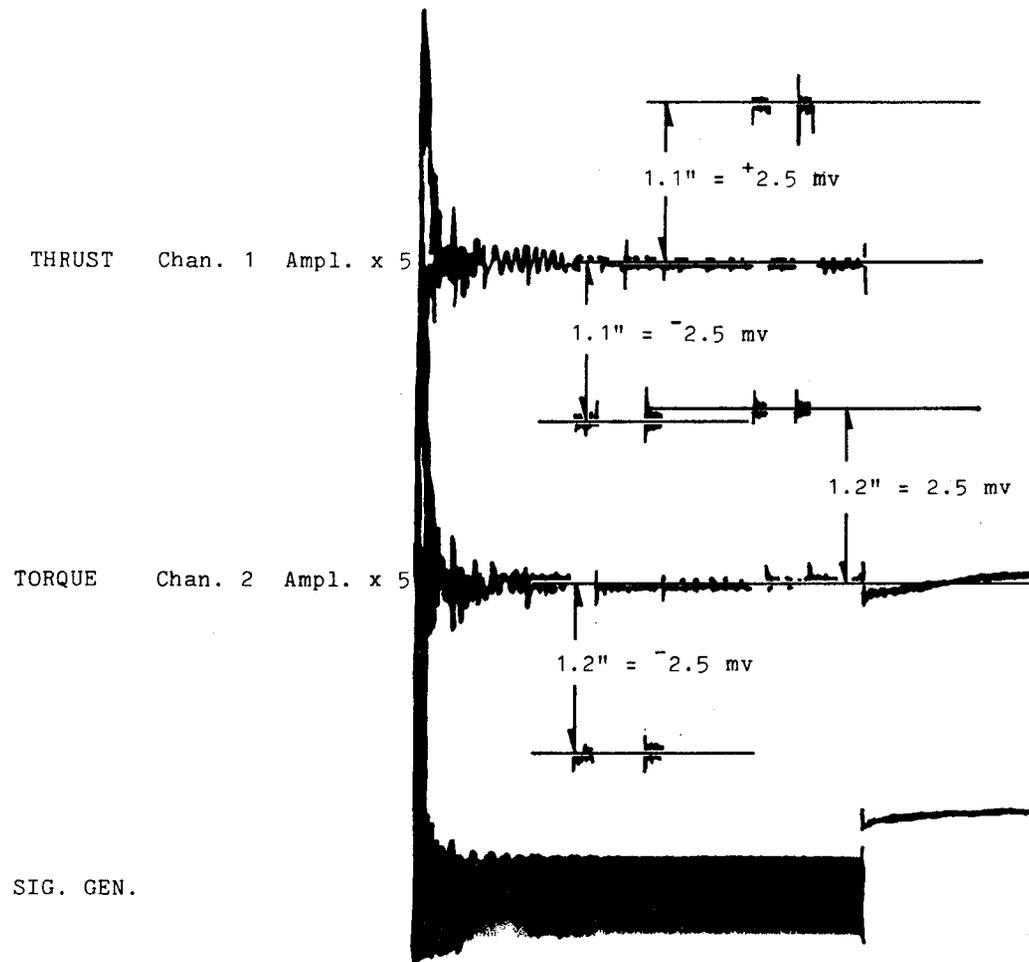


Figure 33: Torque and Thrust Calibration

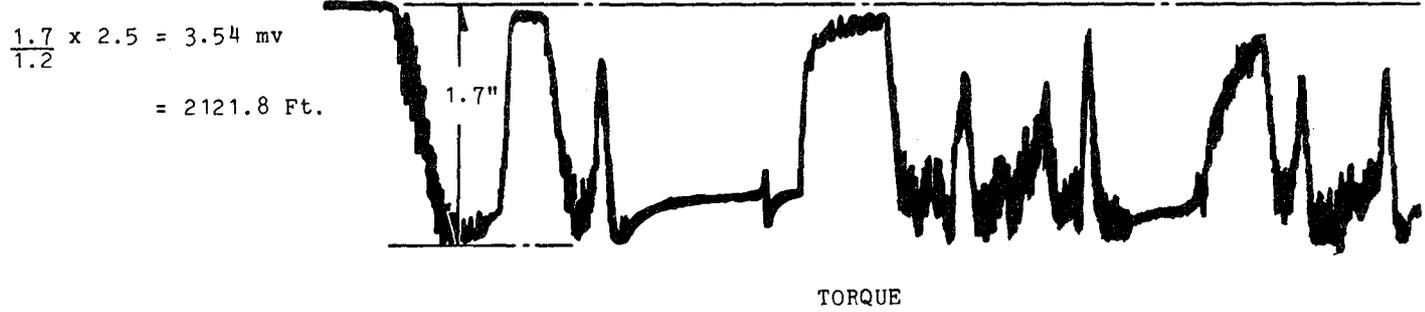
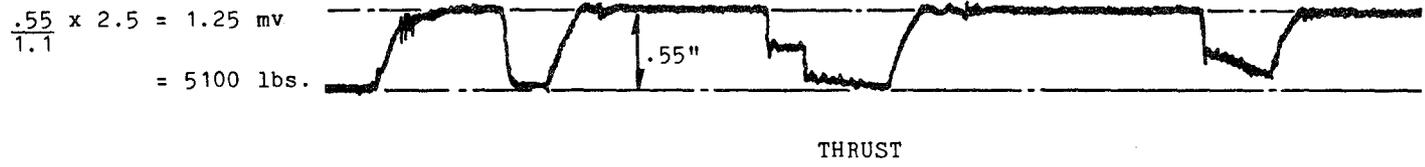
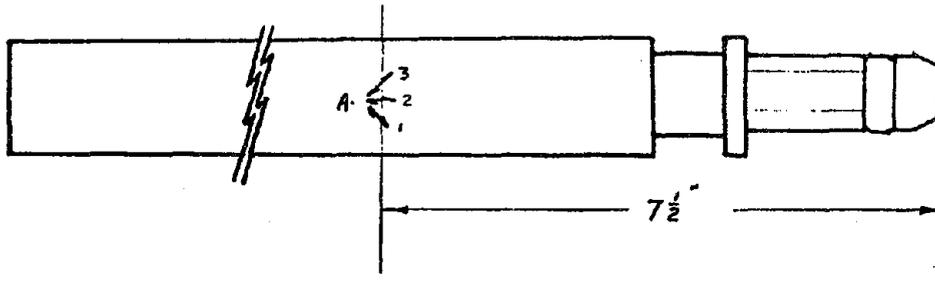


Figure 34: Torque and Thrust Test Results

JOB- LN 09 99 04
 ENG- T. WALL
 TECH: D'ANGELO
 DATE: 4-20-76



*INSTRUMENTATION of STEEL ROD
 for FLEX-ROOF BOLTER*

GAGE TYPE - WA-06-062RB-120
 RES. - $120.0 \pm 0.4\% \Omega$
 LOT - ZA21AD131
 G.F.
 1 - $2.045 \pm 0.5\%$
 2 - $2.075 \pm 0.5\%$
 3 - $2.045 \pm 0.5\%$
 Nominal - $2.06 \pm 1.5\%$

ADHESIVE - BR 610
 CURE - 345°F 1HR.

NOTE -

RESISTORS ON THRUST BRIDGE
 WERE CONNECTED ON CABLE -
 USED 1RC 120.0 Ω M \pm 1%

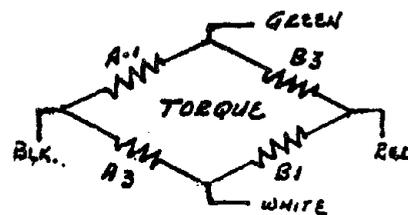
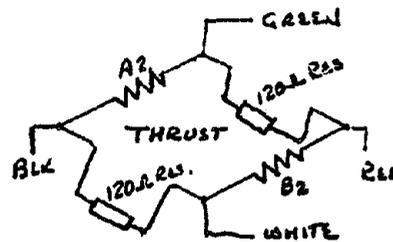


Figure 35: Instrumentation of Steel Rod

21.14 Air Compressor Bench Test

Recommendation:

- Hydraulic:
1. Substitute 5/8" ID hose and fittings for the previous 3/8" ID hose and fittings. This substitution will reduce both oil foaming and back pressure.
 2. Submerged the hydraulic motor return line. This will reduce oil foaming.

- Mechanical:
1. Use a 6.0" diameter, sheet metal pulley on the hydraulic pump. The pump will have a speed of 1600 RPM, giving a compressor start-up speed of approximately 2000 RPM, and running speed of 1700 RPM. These speeds will also be sufficiently above stall speeds.

- Air Supply:
1. Use a 1 quart air accumulator to dampen pulsations (provided).
 2. Use a relief valve set for 70 psig (provided).
 3. Use an intake manifold to reduce compressor noise (provided).

Test Results:

The next four pages contain the test data.

5/17/76
 LN-05-92
 S.R.L.

(Boil-wärmer ^{nyce. pump} &
 Jammet air compressor test
 Rpus

Texaco Kando C' oil

	Hyd. pump	Compressor	(P _{oil}) Compressor oil	(°F) Hyd oil
{ 5/8" I.D. inlet pump line	1460	1840	70	-
{ 3/8" I.D. disch. line →	1370	1700	70	-
3/8" I.D. inlet pump line	Oil return line { 1460	1780	45	-
	submerged in oil tank, → { 1460	1780	60	-
	line { 1460	stalled Comp.	70	-
Oil dumping in top of tank	1460	Comp. stalling Oil is foaming	60 - 40	-
5/8" I.D. inlet pump line	Oil return line → 1460	1820	70	-
	submerged in Oil tank, { 1460	1800	70	-
	{ 1460	1860	0 unloaded	
	After 30 min. { 1460	Comp. stalling	70 - 60	12
	{ 1560	Comp. stalling 1900	70	
Changed to W ₂ B oil	1550	1920	70	
	Pump lost prime			

(Borg-Warner hyd. pump
+
Lammert Air Compressor test)

TEXACO RANDO 'C'

ARS	TIME	R p m s		P s i g		of TEMP
		Hyd. pump	COMPRESSOR	Hyd. pump	Comp. AIR	
	11:30	1460	1750	-	70	-
5/8" I.D.	11:40	1450	1720	-	70	-
OIL LINES	11:45	1450	1705	-	70	-
3/8" I.D. oil	11:50	1450	1700	-	70	108
RETURN LINE	11:53	1450	1690	-	70	111
TO TANK	start	1460	1840	500	0	90
	1:40	1450	1760	750	70	92
	1:53	1450	1730	750	70	109
	2:02	1450	1680	750	70	-
	2:10	1450	1650	750	70	-
	2:17	1450	1620	700	70	130
	2:20	1450	1600	700	70	130
	2:25	1450	1600	700	70	130
Comp. disch.	2:30	1420	1580	700	70	130
PIPE TEMP - 173°F	2:35	1420	1550	700	70	140
AIR TANK TEMP. - 150°F	2:37	1400	1510	700	70	141
Comp. Cyl.	2:40	1370	Comp. stall point		70	141
HEAD TEMP - 200°F	2:45	1540	1700	720	70	14
	2:50	1540	1660	720	70	14
stop	2:55	1540	1645	720	70	14
start	3:45	1540	1800	720	70	12
INSTALLED	3:50	1420	1630	700	70	15
5/8" I.D. oil	4:13	1420	1580	680	70	15
RETURN LINE TO TANK.						

Oil in tank doesn't foam when return
line is submerged in the oil.

Temperature:

(After one hour of operation)

Compressor Cylinder Head : 200-225 °F

Compressor Discharge Pipe: 173 °F

Air Accumulator: 150-175 °F

Air Discharge 115 - 125 °F

(All temp. approx.)

Pulley and Belt Selection:

$$\frac{\text{Pulley}}{\text{RPM}} \times \frac{\text{Distance}}{\text{Rev.}} = \frac{\text{Distance}}{\text{Min}}$$

$$1420 \times 6.75 \pi = 1600 \times d \pi$$

$$d = \frac{1420 \times 6.75}{1600} = 5.99''$$

⇒ 6" pulley

Belt:

$$\text{Belt Length} = 2C + \frac{\pi}{2} (D+d) + \frac{(D-d)^2}{4C}$$

D = dia. of large sheave

d = dia of small sheave

C = center to center distance

$$\text{Belt Length} = 2 \times 9 + \frac{\pi}{2} (6.75 + 6.0) + \frac{(6.75 - 6.0)^2}{4 \times 9}$$

$$= 38.04'' \approx 38''$$

7/2
5/20/76

Air Compressor Bench Test
Supplement

Belt Length:

Due to mounting change, the center to center distance
is now:

$$\text{Min} : 9.625''$$

$$\text{Avg} : 10.25$$

$$\text{Max} : 10.875$$

$$\text{Belt Length} = 2C + \frac{\pi(D+d)}{2} + \frac{(D-d)^2}{4C}$$

$$\begin{aligned} \text{Min Belt Length} &= 2 \times 9.625 + \frac{\pi(6.75+6.0)}{2} + \frac{(6.75-6.0)^2}{4 \times 9.625} \\ &= 39.29'' \end{aligned}$$

$$\begin{aligned} \text{Avg. Belt Length} &= 2 \times 10.25 + \frac{\pi(6.75+6.0)}{2} + \frac{(6.75-6.0)^2}{4 \times 10.25} \\ &= 40.54'' \end{aligned}$$

$$\begin{aligned} \text{Max. Belt Length} &= 2 \times 10.875 + \frac{\pi(6.75+6.0)}{2} + \frac{(6.75-6.0)^2}{4 \times 10.875} \\ &= 41.79'' \end{aligned}$$

Ruzinski
6/1/76

gorsell-Rand Research Inc.

BOX 301
PRINCETON, N. J. 08540

Date January 30, 1976

To Lee-Norse, Charleroi

Sub: Certification of the Rod-Changer's
Safety Arm (LN-09-99-04)

Attn: Mr. Gopal Shah

Dear Gopal:

In response to my recent communications with you, I have performed three new analyses of the rod-changer's safety arm. They are attached to this letter.

The first analysis used the original safety arm model as described in my letter to C. Holvenstot dated June 24, 1975, except that members 19 and 20 are now T-1 steel. The loads applied were provided by your CANOPY analysis of the fly-swatter (Ref: letter from G. Shah to R. Frick dated December 16, 1975). Note that opposite placement of the fly-swatter on the rod-changer precluded changing the moment signs.

In the second analysis, the safety arm was remodeled to incorporate the following changes:

1. The bolt-bender, carousel, and front plate were removed as previous analyses indicated that their effect was negligible.
2. Any member under the $\frac{1}{2}$ inch plate was modeled as a solid rectangular section with $\frac{1}{2}$ inch added to its height.
3. Member 13 was divided in half to reflect the extra $\frac{1}{4}$ inch thickness along half its length due to our modifications.
4. Member 38's (now 31 and 33) height was increased to a more realistic value.

This new model more accurately represents the safety arm and is also simpler. Note that members 7 and 28 just yield, and 8 is on the verge of yielding.

The following model changes were made for the third analysis:

1. Members 28 and 29 were increased in width by $\frac{1}{4}$ inch to reflect a plate that could be welded to these members.
2. In previous models, joints 7 and 8 (the radius arm pin connections) were restrained from rotation in the \hat{z} direction. This restraint was relaxed since rotation can occur because the pin connection is at the bottom of the safety arm, the internal pin tolerance allow motion, and the radius arms can twist. (The actual condition is between these two extremes).

Yielding did not occur in this analysis.

Mr. Gopal Shah
Page 2
January 30, 1976

Prompt response on this certification request is desired because the rod-changer is nearing completion. We are prepared to add a $\frac{1}{8}$ inch plate to members 28 and 29 if you feel that it is essential. Also, I have not received the necessary blue-prints in order to modify our present fly-swatter canopy to satisfy the law (Ref: letter from G. Shah to R. Frick dated December 16, 1975). Of course, if we receive a redesigned fly-swatter, as you have indicated that we might, then modifications to our present one would be unnecessary.

Your continued cooperation and assistance is appreciated.

Sincerely,

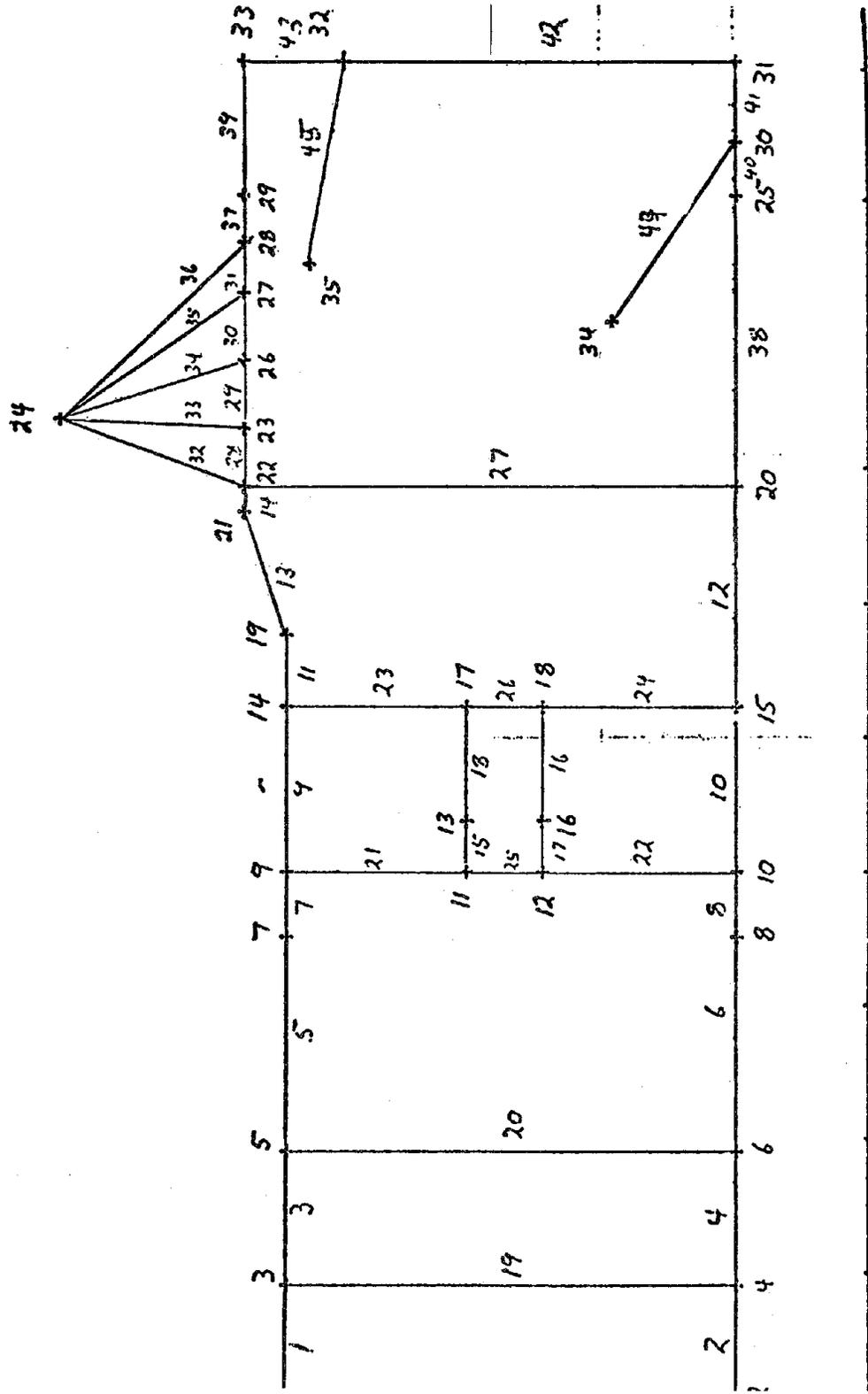


Ray Frick

RF:BB

Attachments: CANOPY Analyses

cc - W. A. McGahan - Letter only
B. R. Banerjee - Letter only
R. N. Eakin - Letter only
D. Holtkamp - Letter only
B. Self
T. Wall ✓



FLEX-DR BCLT LOADS ON SAFETY ARM DUE TO COPY CANOPY.C/ULATED 12
 MINES INFO. CIRCULAR 8546/1972

MEMBER STRESSES								
MEMBER	SIGMAX	SIGAX	TAUMAX	SIGMAX	SIGMA1	SIGMA2	VMISES	VMISES/YIELD
1	6875.	-.	315.	6875.	6889.	-14.	6896.	.07
2	5519.	-.	315.	5519.	5537.	-18.	5546.	.06
3	14532.	-.	204.	14532.	14535.	-3.	14536.	.15
4	11444.	-.	204.	11444.	11447.	-4.	11449.	.11
5	26541.	-.	-307.	26541.	26545.	-4.	26547.	.27
6	21165.	-.	-307.	21165.	21170.	-4.	21172.	.21
7	47094.	3.	17741.	47097.	53032.	-5935.	56235.	.56
8	37795.	1.	-1067.	37795.	37825.	-30.	37840.	.38
9	50164.	3.	5728.	50167.	50813.	-646.	51139.	.51
10	15956.	1.	2150.	15956.	16241.	-285.	16385.	.16
11	-60639.	-6.	39066.	-60645.	19130.	-79776.	90864.	.91
12	5356.	8.	2384.	5364.	6270.	-906.	6768.	.07
13	-59774.	-12.	15865.	-59787.	3949.	-63736.	65799.	.66
14	23320.	213.	23858.	23533.	38368.	-14835.	47554.	.48
15	-30820.	-.	-11110.	-30820.	3587.	-34407.	36334.	.36
16	-13172.	5.	6138.	-13167.	2417.	-15585.	16923.	.17
17	-20552.	0.	1371.	-20552.	91.	-20643.	20688.	.21
18	-30821.	-8.	12106.	-30829.	4186.	-35015.	37284.	.37
19	212.	-.	-1214.	212.	1325.	-1113.	2114.	.02
20	976.	-.	-1608.	976.	2169.	-1193.	2952.	.03
21	-36059.	0.	-972.	-36059.	26.	-36085.	36099.	.36
22	-2483.	-.	-9454.	-2484.	8294.	-10777.	16562.	.17
23	-30151.	-7.	-16842.	-30158.	7527.	-37685.	41958.	.42
24	-2131.	-7.	-6588.	-2138.	5605.	-7743.	11610.	.12
25	-20348.	-.	-3471.	-20348.	576.	-20923.	21217.	.21
26	-13758.	-8.	-18604.	-13767.	12953.	-26720.	35040.	.35
27	73280.	102.	-4811.	73381.	73695.	-314.	73853.	.74
28	11911.	99.	7857.	12010.	15894.	-3884.	18151.	.18
29	10434.	113.	566.	10547.	10577.	-30.	10592.	.11
30	6004.	134.	968.	6137.	6287.	-149.	6362.	.06
31	4734.	159.	-2332.	4893.	5826.	-933.	6345.	.06
32	-17413.	-85.	-1338.	-17498.	102.	-17600.	17651.	.18
33	5333.	72.	-1301.	5405.	5702.	-297.	5856.	.06
34	4984.	57.	-78.	5041.	5042.	-1.	5042.	.05
35	4970.	77.	590.	5047.	5115.	-68.	5150.	.05
36	-1903.	-131.	177.	-2034.	15.	-2049.	2057.	.02
37	11343.	35.	-18289.	11378.	24843.	-13465.	33660.	.34
38	-8028.	-27.	11340.	-8055.	8007.	-16062.	21230.	.21
39	1657.	8.	-6626.	1664.	7510.	-5846.	11597.	.35
40	-1612.	-8.	4033.	-1620.	3304.	-4924.	7171.	.22
41	-1742.	-6.	2731.	-1748.	1994.	-3742.	5043.	.15
42	-1724.	-9.	-2929.	-1733.	2188.	-3922.	5362.	.16
43	733	0	2679	733	2310	2110	733	.07

FLEX-DRILL BOLT LOADS ON SAFETY ARM DUE TO CANOPY CANOPY CALCUL
BUMINES INFO. CIRCULAR 8546/1972

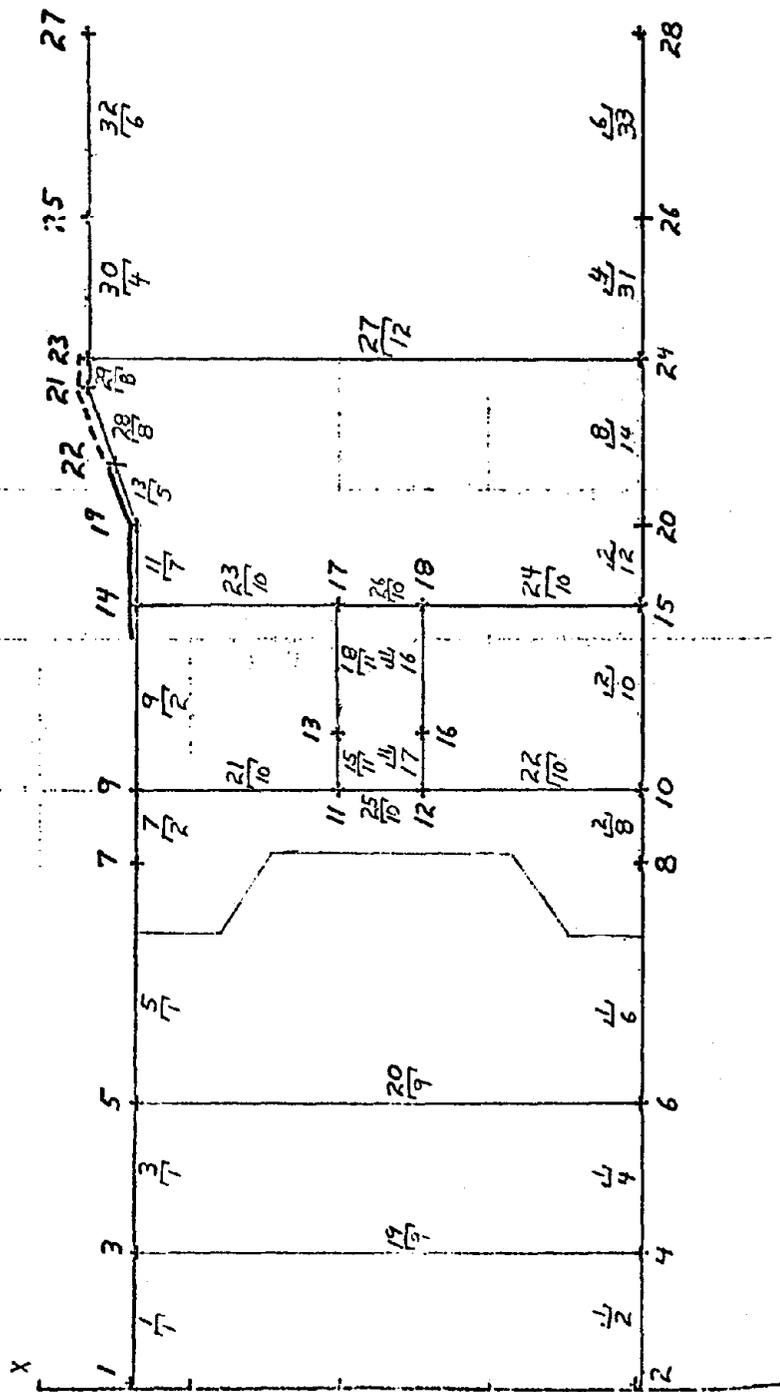
TOTAL LOAD IN X DIRECTION IS 0. POUNDS
TOTAL LOAD IN Y DIRECTION IS-10006. POUNDS
TOTAL LOAD IN Z DIRECTION IS 0. POUNDS
MAXIMUM LOAD THAT CANOPY CAN SUPPORT IN X DIRECTION IS 0. POUNDS
MAXIMUM LOAD THAT CANOPY CAN SUPPORT IN Y DIRECTION IS-11012. POUNDS
MAXIMUM LOAD THAT CANOPY CAN SUPPORT IN Z DIRECTION IS 0. POUNDS
WEIGHT OF CANOPY IS1079. POUNDS

FLEX-DRILL (LN-09-99-04) REMODELED SAFETY ARM FOR 9000LB POINT LOADS
BUMINES INFO. CIRCULAR 8546/1972

CANOPY CALCULATED 0

TOTAL LOAD IN X DIRECTION IS 0. POUNDS
TOTAL LOAD IN Y DIRECTION IS-18000. POUNDS
TOTAL LOAD IN Z DIRECTION IS 0. POUNDS
CANOPY CANNOT SUPPORT ABOVE LOADS WITHOUT YIELDING
MAXIMUM LOAD THAT CANOPY CAN SUPPORT IN X DIRECTION IS 0. POUNDS
MAXIMUM LOAD THAT CANOPY CAN SUPPORT IN Y DIRECTION IS-17967. POUNDS
MAXIMUM LOAD THAT CANOPY CAN SUPPORT IN Z DIRECTION IS 0. POUNDS
WEIGHT OF CANOPY IS 566. POUNDS

$\frac{5}{17}$ or $\frac{1}{5}$ means 5th member with section properties #1



Flex-Drill Safety Arm

FLEX-DRILL (LN-09-99-04) REMODELED SAFETY ARM FOR 9000LB POINT LOADS
 BUMINES INFO. CIRCULAR 8546/1972

CANOPY CALCULATION 01/72

MEMBER STRESSES								
MEMBER	SIGMAX	SIGAX	TAUMAX	SIGMAX	SIGMA1	SIGMA2	VMISES	VMISES/YIELD
1	16829.	-.	2087.	16829.	17084.	-255.	17213.	.17
2	16694.	-.	-2129.	16694.	16961.	-267.	17096.	.17
3	-35260.	-.	3031.	-35260.	259.	-35519.	35649.	.36
4	34995.	0.	-2995.	34995.	35250.	-254.	35378.	.35
5	-64824.	-.	5480.	-64824.	460.	-65284.	65515.	.66
6	64211.	0.	-5189.	64211.	64628.	-417.	64837.	.65
7	-91157.	-.	6202.	-91157.	420.	-91577.	91787.	.92
8	90259.	0.	-5873.	90259.	90639.	-381.	90830.	.91
9	-80455.	-3.	-286.	-80458.	1.	-80459.	80459.	.80
10	-79904.	-1.	1876.	-79905.	44.	-79949.	79971.	.80
11	66996.	3.	11053.	66998.	68774.	-1776.	69680.	.70
12	-82172.	-3.	4304.	-82175.	225.	-82400.	82512.	.83
13	71458.	5.	-27772.	71463.	80987.	-9524.	86144.	.86
14	-89250.	-4.	4943.	-89253.	273.	-89526.	89663.	.90
15	26802.	6.	-10763.	26806.	30594.	-3787.	32653.	.33
16	-24856.	-4.	-3765.	-24860.	558.	-25418.	25701.	.26
17	-24851.	-1.	9420.	-24852.	3167.	-28019.	29730.	.30
18	26820.	9.	3586.	26829.	27300.	-471.	27539.	.28
19	1806.	0.	136.	1806.	1817.	-10.	1822.	.02
20	-4684.	-.	106.	-4685.	2.	-4687.	4688.	.05
21	-47596.	-2.	13862.	-47597.	3743.	-51340.	53310.	.53
22	-45362.	-1.	-13394.	-45362.	3660.	-49022.	50951.	.51
23	37376.	7.	-8098.	37383.	39062.	-1679.	39928.	.40
24	35569.	6.	7711.	35575.	37174.	-1600.	37999.	.38
25	-43238.	-2.	939.	-43240.	20.	-43261.	43271.	.43
26	35924.	6.	83.	35930.	35930.	-.	35930.	.36
27	-13710.	-19.	-417.	-13729.	13.	-13741.	13748.	.14
28	61436.	5.	-30546.	61441.	74042.	-12602.	81081.	.81
29	52747.	3.	623.	52750.	52758.	-7.	52761.	.53
30	66432.	254.	0.	66686.	66686.	0.	66686.	.67
31	66432.	254.	0.	66686.	66686.	0.	66686.	.67
32	50102.	240.	0.	50341.	50341.	0.	50341.	.50
33	50102.	240.	0.	50341.	50341.	0.	50341.	.50

FLEX-DRILL (LN-09-99-04) REMODELED SAFETY ARM FOR 9000LB POINT LOADS
BUMINES INFO. CIRCULAR 8546/1972

CANOPY CALCULATION

TOTAL LOAD IN X DIRECTION IS 0. POUNDS

TOTAL LOAD IN Y DIRECTION IS-18000. POUNDS

TOTAL LOAD IN Z DIRECTION IS 0. POUNDS

MAXIMUM LOAD THAT CANOPY CAN SUPPORT IN X DIRECTION IS 0. POUNDS

MAXIMUM LOAD THAT CANOPY CAN SUPPORT IN Y DIRECTION IS-19611. POUNDS

MAXIMUM LOAD THAT CANOPY CAN SUPPORT IN Z DIRECTION IS 0. POUNDS

WEIGHT OF CANOPY IS 568. POUNDS

Ingersoll-Rand Research Inc.

BOX 301

PRINCETON, N. J. 08540

Date October 14, 1976

To R. Eakin

Att:

Sub: Field Trial Flexible Roof Drill - 1st Week
Week of: 9/27/76 to 10/1/76

The field trial began with the IRRI personnel of R. Eakin, T. Wall, A. Sanders, and D. Rich meeting with the Bethlehem mine representatives, S. Terry and W. Johnson. At the time it was decided to change the pneumatic tires on the Flex Drill vehicle from 8:15 - 14 8:50 - 14 ; this change in tires gives the bolter approximately 3 1/2" of additional ground clearance. The mine personnel had installed the correct cable connection for their electrical system prior to the meeting with IRRI. Because of the change in tires, IRRI removed the two emergency air supply bottles for safety purposes. Later in the day, IRRI personnel bought the following mining equipment: helmets, brass tags, mining belt, leg bands, knee pads, and coveralls.

On Tuesday, IRRI requested the Bethlehem personnel to change the location of the oil fill cap because of interference with the new oversized tires that were installed the previous day; this was accomplished Tuesday night. The flex drill was then started and inspected for any damage during shipment. At this time, the gate cylinder flow control valve was removed from the circuit because of erratic operation.

Wednesday morning, the hydraulic and pneumatic systems were checked and adjusted for entry into the mine. New upper clamp arms were installed on the vehicle chassis, but required additional machine work and were returned to IRRI in New Jersey. At this time, drill rotation pressure was checked and set at 1250 psi. The vehicle was then trammed into mine entry #4 where drilling and bolting were attempted. However, the conditions were not within the range specified for the operation of the vehicle. The mine height exceeded the 54" maximum height by approximately two feet. In addition to the extremely high seam height, the floor pitched both forward and to the right and consisted of approximately 8" of mud. The roof further complicated matters by chipping and therefore, deflecting the drill steel at a severe angle. Under these severe conditions, four holes were drilled and two bolts secured. The vehicle was then trammed out of the mine for further adjustments. The bolt torque setting was determined to be 150 ft/lbs, which was within the

120-160 ft/lb range required in actual mining conditions. At this time, the relief valve setting (valve #42) was decreased to 250 psi so as to increase the amount of torque available at the drill motor.

On Thursday morning, it was observed that the boom operation was not correct. The boom malfunction was during unload section of the cycle. The boom malfunction occurred in that it would not drop to the lower boom position. After considerable trouble shooting, it was determined that limit switch #5 was out of adjustment due to the high seam height, and the new vehicle height. The switch was then adjusted to its proper position. Because of the improper boom operation, a carousel ratchet key was sheared and replaced.

On Friday, the severe mine entry conditions were reviewed with the Bethlehem mine personnel where it was decided to lay header and half headers in the path of the vehicle which resulted in a lower seam height and firmer floor support. Although this was an improvement, the overall mine conditions were very severe. Before tramping back into the mine all limit switches were adjusted again and the machine sequence of operation was corrected and repeated many times. The vehicle was then trammed back into the mine and two holes were drilled and bolted.

At the end of the first week, a total of six holes were drilled and four bolted.

Dean E. Rich

Dean E. Rich

10/14/76:sls

cc: R. N. Eakin
T. J. Wall
F. O. Olsson
J. Bevan

Ingersoll-Rand Research Inc.BOX 301
PRINCETON, N. J. 08540

Date October 14, 1976.

To I.R.R.I.

Att: R.N. Eakin

Sub: Field Trial Flexible Roof Drill -
10/4/76 to 10/8/76 - 2nd Week

On Monday of the second week, the boom thrust pressure was reduced from 1250 psi to 950 psi by adjusting relief valve # 6. The new upper clamp arms were installed on the vehicle, but required further machining. The upper clamp flat washers were brought to a G.E. machine shop where the left washer was ground down to a thickness of .098" and the right washer ground down to a thickness of .102". The bolt tightening adapter was also machined to accept a typical bolt head and bolted directly to a drill steel.

The machining was completed on Tuesday morning and the parts picked up at that time. The upper clamp arms were then custom fitted to the vehicle, which resulted in an absolute perfect fit of the arms. It was also necessary to replace a hydraulic hose assembly at the fitting on the upper clamp cylinder. The machine was then cycled where it became necessary to replace two carousel ratchet keys. All limit switches were adjusted again.

Before tramping into the mine, the machine operation was successfully demonstrated to Ed Kasor, the Bethlehem District Manager.

On Wednesday the vehicle was then trammed back into mine entry #4, where 8 holes were drilled and bolted. In accomplishing this, 1 limit switch arm (switch # 2), 1 carousel ratchet key and one lead steel coupling were broken and replaced. The problem of shearing the above parts was a direct result of working in a seam height above design specifications. After returning to Charleston, a Sears & Roebuck steel shed was ordered to store the repair parts shipped with the vehicle.

As a result of the high seam height, a new upper plate cover with boom limiting stops was constructed as a possible solution. This was accomplished on Thursday and produced

Att.: R.N. Eakin

further unloading problems. It was also necessary to replace one limit switch striker arm.

Friday resulted in the modification to the upper cover plate, but has not produced the desired results of better upper boom limit control. The bolter drilled and bolted 2 holes.

At the end of the week, Bethlehem recognized the design limitations of the vehicle and agreed to move the vehicle to the mining face as soon as possible. The 2nd week resulted in a total of 10 holes drilled and bolted and a 2 week total of 16 holes drilled and 14 bolted. All the bolts used in the field trial test to date have been four ft. in length.

D.E. Rich
D.E. Rich

cc: R. N. Eakin
T. J. Wall
F. O. Olsson
J. Bevan

Ingersoll-Rand Research Inc.

BOX 301
PRINCETON, N. J. 08540

Date November 4, 1976

To R. N. Eakin

Sub: Flex Drill Field Trial - 10/11/76 to
10/15/76 - 3rd Week

Att:

On Monday, October 11, 1976, the machine was trammed into a **breakthrough** off to the right of entry # 4. In this area the **roof was very flat**, and after trammng the vehicle onto half headers, it was operating in the **designed and specified seam height** of 48" to 54". At this time, **all limit switches** were checked for proper adjustment and vacuum at the drill chuck was recorded.

With one drill steel in the chuck, the vacuum was measured at 11" of Hg. Further testing revealed an approximate loss in vacuum of 1/2" Hg, per steel coupling. With 4 drill steels connected to the **chuck**, the vacuum was recorded at approximately 10" of Hg. The vacuum was measured with the use of an automotive vacuum gauge calibrated in inches of Hg. The test results are recorded in the following table:

<u>Drill Vacuum Measurements</u>			
<u>Conditions</u>	<u>Standard L-N Bolter</u>	<u>Flex Drill (without insert)</u>	<u>Flex Drill (with insert)</u>
at chuck	14" Hg.	≈ 14" Hg.	≈ 14" Hg.
1-drill steel	----	≈ 11" Hg.	≈ 13 ⁺ " Hg.
4-drill steels	----	≈ 10" Hg.	≈ 12 ⁺ " Hg.

Because of approximately 3" of Hg. loss at the drill chuck, a wooden chuck insert was fabricated; however, after a few trial couplings, the wood was destroyed. A second chuck insert was fabricated from the head of a standard bolt and pressed into the chuck. The steel insert has not shown any excessive wear in the subsequent one hundred couplings completed and shows no loss of vacuum at the chuck. The vacuum with the Flex Drill vehicle appears to be the same as a normal Top Dog roof bolter. With the remaining time in the day, 2 holes were drilled and 2 bolts inserted with the Bendix Bolt Bender without a problem or failure.

On Tuesday, October 12, 1976, the Flex Drill was successfully demonstrated to the Bethlehem Research Division personnel. The demonstration consisted of drilling one hole and inserting one bolt using the Bendix Bolt Bender. The cycle time, air logic, and machine motion were of utmost importance to the Bethlehem personnel.

The bolt torque was set at 150 ft.-lbs., but was not repeatable. It was also apparent that the bolt torquing adapter would have to be pried from the bearing plate on the roof after torquing of the bolt. To complete this, an air limit switch was added to the control panel. When the air switch was activated, the lower clamp engaged the torque drill steel adapter in the chuck. The lowering of the boom pulls the adapter from the roof and the switch was deactivated. The adding of the air limit switch involved running one air line from the control box to the air switch and from the air switch to the air pilot port of the lower clamp directional control valve (#30). During the rest of the day, a total of 15 holes were drilled and bolted.

On Wednesday, October 13, 1976, the IRRRI personnel were in an automobile accident and no work was done. On Thursday and Friday, October 14 and 15, no holes were drilled and bolted because only 1/2 the IRRRI personnel were available at the mine. However, during this time the drill steel torque adapter was shortened by 5", a coupling welded to the drill steel to prevent separation between the drill steel and coupling and 5 vent holes drilled in the drill steel to stop the dead heading of the blower while the drill steel adapter was in the chuck during torquing.

The total number of holes drilled and bolted for the week was seventeen with nothing significant broken or replaced. It should be recorded that both the centralizers and upper clamp arm pins cannot be tightened without binding their arms. This indicates wear in the frame (body) acting against the shoulder of the screw and should be reviewed prior to a redesign.

D. E. Rich

D. E. Rich

DER:sls

cc: R. N. Eakin
D. E. Rich
T. J. Wall

Ingersoll-Rand Research Inc.

BOX 301
PRINCETON, N. J. 08540

Date November 2, 1976

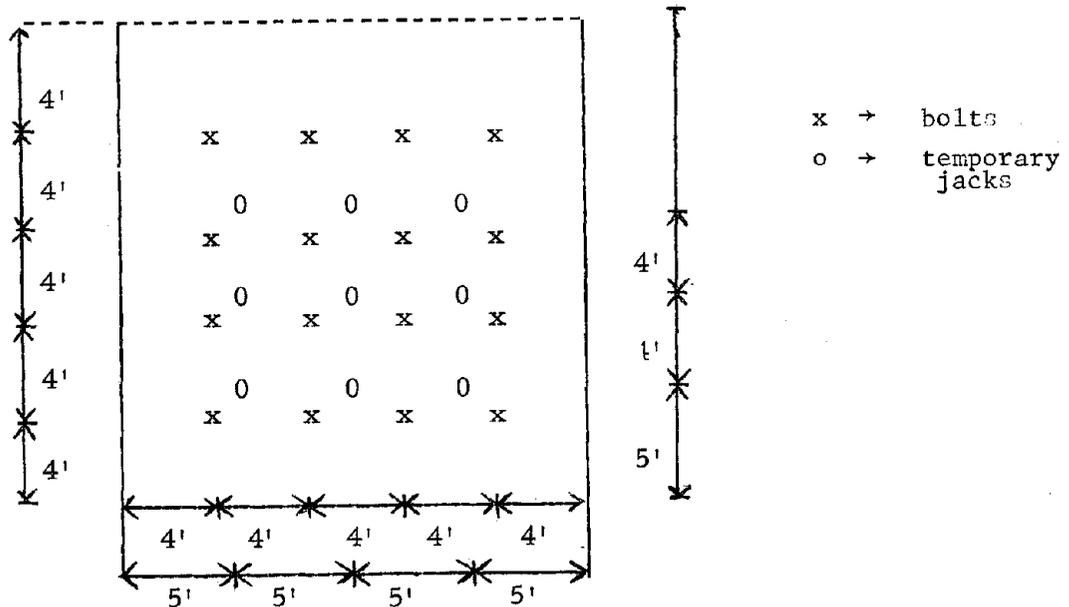
To R. N. Eakin

Sub: Flex Drill Field Trial - 10/18/76 to
10/22/76 - 4th Week

Att:

On Sunday, October 17, 1976, the Flex Drill vehicle was moved into the production area of the mine. When the IRRI personnel arrived Monday morning, October 18, 1976, the vehicle was not connected to the electrical power source and it was necessary to wait the entire day for a decision by the Bethlehem personnel as to how this would be accomplished. At this time, all available electrical connections were being used for the production mining equipment. The Bethlehem mine is in the process of becoming a full production mine and the addition of the Flex Drill vehicle was being integrated into their plans whenever possible.

IRRI personnel took advantage of this lost drilling time by observing the in-cycle drilling operation (with a Lee-Norse bolter) by the Bethlehem personnel. At this time, the correct procedure for marking the roof and setting the temporary roof jacks was observed. The Bethlehem mines prefer to drill and bolt working from left to right. The left to right procedure affords the bolter personnel maximum protection in the unsupported roof area. The temporary roof jacks are set approximately 5 feet forward from the previous bolts set and 5 feet from each other. There are a total of nine temporary jacks set before drilling begins. The Bethlehem personnel also demonstrated the present method for bending, inserting, and straightening the roof bolt.



On Tuesday, October 19, 1976, electrical power was connected to the vehicle and the machine inspected for any possible damage when the vehicle was brought into the production area of the mine. At this time, it was noted that the head tensioning spring had been over stretched and required adjustment for proper operation; the cam lever on lower limit switch #2 was also replaced. After checking the machine operation, the vehicle was trammed close to the face in entry #1, and 3 holes were drilled and bolted. It was then decided to wait for the continuous miner to mine entry #1 so the in-cycle test of the program could begin.

On Wednesday, October 20, 1976, the IRRI personnel found that the machine had been trammed during the second shift on Tuesday, October 19, 1976, and limit switch #2 (3 function switch) had been completely destroyed and ripped from the machine. The limit switch was replaced and adjusted so that a successful demonstration was presented to J. Bevan, K. Kerby, and other Bureau of Mines personnel. With the consent of the Bureau of Mines personnel, additional holes were drilled and bolted close to the in-cycle face of the mine, but not directly at the face. There were a total of 17 holes drilled and bolted during the day. While the 17 holes were being completed, a hose connected to valve #32 (centralizer directional control valve) separated from its hose fitting, the fitting was the reusable type and appeared to be improperly assembled. An additional lever arm (limit switch # 2) was broken and replaced. The lever arms are broken during the unload cycle when the extra movement of the boom jams and breaks the lever arm.

On Thursday, October 21, 1976, the IRRI personnel found that the machine had again been trammed during the second shift of the previous day. This tramping of the machine resulted in the destruction of the lower clamp actuator. After conferring with the mine personnel, it was evident that the damage done was unintentional. The IRRI personnel in Princeton, NJ, located 2 rotary actuators and made arrangements for the items to arrive Friday evening. The IRRI personnel at the mine attempted to make a temporary repair on the rotary actuator.

The repaired rotary actuator was bolted back on the machine Friday morning, October 22, 1976. The repair lasted 3 holes before failing. It was then decided to wait and replace the damaged actuator with the new component on Tuesday morning.

The conditions at the face of the production area of the mine appear very safe. The roof and floor appear to be fairly flat with the face area quite wet.

D. E. Rich
D. E. Rich

DER:sls

cc: D. E. Rich
T. J. Wall
R. N. Eakin

Ingersoll-Rand Research Inc.

BOX 301

PRINCETON, N. J. 08540

Date Nov. 2, 1976

To

Sub: Flex Drill Field Trial - 5th week

Att: R. Eakin

Monday, October 25, 1976 was a holiday for the Bethlehem Mine people and therefore the mine was closed accordingly.

On Tuesday, October 26, 1976 there was no electrical power outlet available for approximately seven hours. However, during the remaining time available, 4 holes were drilled and bolted. These holes were completed during a demonstration for two M.E.S.A. representatives. The M.E.S.A. people were accompanied by Bethlehem Mine personnel and had no comments or questions concerning the vehicle.

Wednesday, October 27 was similar to the previous day in that power was only available during lunch break and a demonstration was conducted at that time. The demonstration was presented to R. Dorman, J. Nichols and R. McKibbon from the Bureau of Mines office in Spokane, Washington. The demonstration consisted of drilling and bolting seven holes in a 52" seam height. The only problem encountered was the alignment of L.S. #2 with the boom in the unload mode during the completion of one hole. The Bureau personnel had no immediate comments or suggestions.

The next day, Thursday October 28, the vehicle had electrical power for approximately half the day. The vehicle was trammed to a section of the mine where an overcast was to be created. At this location seven 8 foot holes were drilled and bolted. In this area the seam height was approximately 56" which caused minor alignment problems with the limit switches and the shearing of one ratchet key in the carousel.

The following day was only a four hour drilling and bolting period. This resulted in a quantity of six eight foot bolts and one ten foot bolt being secured to the roof. These bolts were placed in the same area as those on Thursday, but there were no component failures or complications.

The total number of bolts secured during the week are as follows:

<u>Quantity</u>	<u>Demonstration</u>
11	4 ft. bolts
13	8 ft. bolts
<u>1</u>	<u>10 ft. bolts</u>
25 bolts	total

Sincerely,

Dean E. Rich

Dean E. Rich

DER/ms

cc: R. Eakin
T. Wall
R. Worthen

Ingersoll-Rand Research Inc.

BOX 301
PRINCETON, N. J. 08540

Date 12/21/76

To R. Eakin

Att:

Sub: Flex Drill Field Trial - 6th Week
Nov. 1, 1976 to Nov. 6, 1976

On Monday, November 1, 1976, there were no electrical power connections available for the flexdrill vehicle. The loss of power may be attributed to the fact that the number of mining personnel was doubled with the opening of a new mine section and an unusual amount of dead work associated with the operation of the mine. The Bethlehem mine personnel were also changing from their temporary mine site quarters to their presently uncompleted permanent buildings.

The IR personnel took advantage of this time to accomplish the following related work:

- (a) The 3 air filters on the vehicle were inspected. The air compressor inlet filter showed no change; the Aro M.P.L. filter appeared the same; however the Dollinger air filter had approximately 8 oz. of water in the container and the elements were very moist. All the elements were therefore changed and the water removed from the container.
- (b) The vehicle was greased.
- (c) A four foot bolt was bent in a tight circle to assist the Bendix Bolt Bender when it fails to insert a bolt in the drilled hole.
- (d) The bracket on the head of the vehicle where L.S. #2 (3 position switch) mounts were cut and filed to allow for greater adjustment of the switch. The area on the boom frame close to the cams for L.S. #2 was filed approximately 3/16 of an inch to allow for greater clearance of the limit switch lever arm in the event of a malfunction. This area was where the lever arm on L.S.#2 jams and breaks during a malfunction.

- (e) A new type limit switch arm was constructed from a hack saw blade to test for a more versatile arm.
- (f) Sears & Roebuck Co. was called to confirm the arrival of the steel tool shed.

On Tuesday, Nov. 2, 1976, there was no electrical power available for the machine. The IR personnel purchased the steel tool shed from Sears & Roebuck Co. and the spare parts and fittings were stored in a dry, secured area.

On Wednesday, Nov. 3, 1976, the expense accounts and weekly reports were completed during the day because of the lack of electrical power.

Thursday, Nov. 4, 1976, was the first full day electrical power was available to the flexdrill vehicle. During the day, a total of 20 four foot bolts and three eight foot bolts were drilled and secured to the mine roof. The bolts were placed in an area away from the face, but with the same roof and floor conditions as the face. With the completion of the twenty three holes 1 drill steel coupling was sheared, however no lever arms were broken and no serious problems developed.

The following day, Friday, Nov. 6, 1976, twenty seven four foot holes were drilled and bolted. This day represented the first time no parts were broken and there were no delays in the bolting program.

The total number of bolts secured for the sixth week was as follows:

47 four foot bolts
3 eight foot bolts

Sincerely,

Dean E. Rich

Dean E. Rich

DER/ms

Ingersoll-Rand Research Inc.

BOX 301

PRINCETON, N. J. 08540

Date 12/21/76

To R. Eakin

Att:

Sub: Flex Drill Field Trial - 7th Week
Nov. 8 - Nov. 12, 1976

On Monday, Nov. 8, 1976, a total of 20 four foot holes were drilled and bolted in an area near the face. During the completion of the holes two drill steel couplings were broken. The couplings broken were in the same hole which was started in the top at an angle of approximately 10 degrees from vertical. In future designs the centralizer arms will have to hold the drill steel vertical (≈ 3 degrees) so that the hole will be started straight regardless of roof or floor conditions. The strength and rigidity of the centralizers will ultimately determine the life of the drill steel coupling. In addition to the drill steel coupling being broken, one ratchet key was also replaced. The ratchet key was sheared when a drill steel was improperly placed in the carousel during the unload cycle.

On Tuesday, Nov. 9, 1976, a total of sixteen holes were drilled and bolted. The sixteen holes were primarily completed by Earl W. Stollinger. He was a certified miner for the Bethlehem mine who was assigned to the IR representatives for safety purposes. The IR personnel instructed Mr. Stollinger for approximately 20 minutes on the purpose of the test, location of the manual controls and the sequence of operations of the machine. With this brief explanation of the machine, the drilling and bolting of the test holes was continued by Mr. Stollinger where the first holes completed presented the most problems. During the load cycle an apparent problem was the failure of the second and subsequent drill steels to couple with the drill steels already in the hole (2nd cycle latch). Familiarity with the machine using drill rotation and boom "raise and lower" eliminated the majority of these problems. In the unload sequence of the cycle Mr. Stollinger had the greatest amount of difficulty with the coupling of the second and subsequent drill steels with the chuck. The coupling problem was due to the fact that the boom was not stopping in the exact position every time. Improper coupling would prevent the lower clamp from pulling the drill steel from the hole. Mr. Stollinger was able to detect the possible coupling problem within a few cycles and then made the necessary boom movement corrections to complete the unloading of the drill steel.

It is the opinion of the writer, IR representative Dean E. Rich, that E.W. Stollinger was able to master the basic machine sequence in approximately five holes and could complete the necessary coupling correction actions within ten holes. The remaining holes improved his knowledge of the machine and further reduced his cycle time. Mr. E.W. Stollinger has had no previous bolting experience and it is believed that this test was a good indication as to the difficulty in training miners on this design.

The problems encountered with the machine during the day was the replacement of the dust bag; the repair of the gasket in the dust collector door; and the replacement of one ratchet key.

On Wednesday, Nov. 19, 1976, a total of fifty one holes were drilled and bolted. All of the bolts installed were four foot in length and thirty one of the bolts were completed in a three hour increment. The mine conditions were ideal for bolting with this machine. The top and bottom were flat and dry with a seam height of 52". The oil temperature reached 175°F with this steady operation of the machine which exceeds all reasonable limits for the oil. With the completion of the fifty-one holes the top plate being held by six bolts was sheared from the machine. Two of the broken bolts were extracted from the head which allowed the plate to be reinstalled in the machine.

The next day, Thursday, Nov. 11, 1976, the air filters for the system were inspected prior to beginning the installation of the bolts. At this time it was discovered that the Dollinger filter elements were moist and had to be changed and the filter cartridge had 8 oz. of water in the bottom. It was also discovered that the Aro filter inside the M.P.L. box was full of water and other contaminants. The filter bowl spring, and flat washer were removed and thoroughly cleansed. The spring and washer were corroding severely so that the Aro filter should be re-evaluated before being used again. After the air filters were reassembled, the installation of the bolts was begun again. At this time one lever arm on L.S. #2 was broken and the lower clamp key was sheared. At the end of the day a total of thirty two holes were drilled and bolted all four foot lengths.

The last day of the week, Friday, Nov. 12, 1976, twenty six four foot holes were drilled and bolted. The mine conditions were characterized by a flat top inclined slightly down and very dusty. At this time it was necessary to change drill steels because of wear on the coupling engagement edges after approximately two hundred and seventy five holes. The parts

broken during the day were the centralizer open limit switch and the carousel upper cover weldment plate. The upper cover was welded and placed on the machine and the limit switch replaced. With the completion of the twenty six holes the oil reached a maximum temperature of 165°F and the air filter elements remained dry.

The total number of bolts for the week was one hundred forty five four footers; and the total number of test bolts completed at two hundred forty three.

Sincerely,

Dean E. Rich

Dean E. Rich

DER/mes

Ingersoll-Rand Research, Inc.

lex 84-3301

BOX 301
PRINCETON, N. J. 08540

Area Code 609
921-9103

January 6, 1977

Robert N. Eakin
Supervising Engineer

Subject: Flex Drill Field Trial
8th Week - Nov. 15 - 19

The 8th week was started by drilling and bolting thirty-two four foot bolts on Monday. The drilling and bolting was begun after the Aro air filter was checked for water and other air contaminants, however, nothing was found. The Dollinger air filter dryer did have water in the bottom which was removed and the filter reassembled. In addition to the maintenance performed on the air filters two ratchet keys in the carousel assembly were replaced and the two air lines connected to limit switch # 2 were repaired. During the completion of the thirty-two holes the oil temperature reached a maximum of 160 °F with an average cycle time of 4-1/2 minutes per bolt.

The most significant problem was the first apparent air logic problem. The malfunction was the "drill" indicator light appearing when the load "feed" button was depressed. The problem appears with the first drill steel only, or when the "feed" button was depressed for the first time after going from the unload mode to the load mode. The indicator light malfunction has no affect on cycle time or machine performance. The second air logic problem was intermittent but did affect the machine performance. The problem was encountered in the "unload" mode with the first drill steel to be retracted from the top. After the retract button was depressed the boom lowered but would not stop when it struck L.S. # 2; the boom would drop to the ground without affecting the air logic. For the second and subsequent drill steels the boom would stop in the desired location and the cycle would continue without air logic problem. It is felt that this problem was a result of an air logic malfunction rather than limit switch alignment problems because it happened with the first drill steel only and happened quite regularly.

On Tuesday, November 16, 1976 a total of seven holes were drilled and bolted before the lower clamp was broken. The lower clamp was broken at both welds and had to be rewelded and ground into position. An additional part broken during the completion of the seven holes included two carousel ratchet keys. It was also necessary to drain water from the Dollinger filter prior to drilling the holes.

The next day, Wednesday, November 17, 1976 only half of the IR representatives were available for work; which precluded the drilling of any additional holes.

The following day, Thursday, November 18, 1976 another eight holes were drilled and bolted before the ratchet cylinder rod sheared. The shearing of the ratchet cylinder rod was considered a major problem at the time because of the lack of spare parts available. The IR representatives contacted several vendors in the area but were unable to locate a cylinder. The home office in New Jersey was contacted whereupon the original vendor was requested to supply additional cylinder rods for the broken cylinder. The remainder of the day was spent procuring roll pins to make a temporary fix on the broken cylinder.

The last day of the week, Friday, November 19, 1976 the repaired cylinder was reinstalled on the machine and the carousel cover was welded to repair a defective initial weld. This repair work was completed by noon but because of a power failure in the mine it wasn't tested until Monday, November 22, 1976.

The total number of holes completed during the eighth week were forty-seven four footers.

Sincerely,

Dean E. Rich

Dean E. Rich

DER:np

cc: R.N. Eakin
T. Wall
D.E. Rich
F.O. Olsson

Ingersoll-Rand Research, Inc.

Telex 84-3301

BOX 301
PRINCETON, N. J. 08540

Area Code 609
921-9103

January 6, 1977

Robert N. Eakin
Supervising Engineer

Subject: Flex Drill Field Trial
9th Week - Nov. 22 - 25

The ninth week was begun by adjusting the repaired carousel ratchet cylinder. The cylinder was adjusted four times before it worked reliably. The carousel cover was also reinstalled at this time; but the repaired cover was welded in an incorrect position which bent the ratchet cylinder during indexing. After discovering the interference the cover was discarded for the remainder of the test. A new brass bearing was also added to the carousel however, it had the wrong dimension which required filing and fitting of the part until it could be used on the machine. Another time consuming effort was the moving of the power cable to a new power source. However, after the above adjustments were complete a total of forty-five holes were drilled and bolted during the day.

On Tuesday, November 23, 1976 a total of forty-six holes were drilled and bolted. The problems encountered were the adjustment of the ratchet cylinder twice and the straightening (rolled over) of the blower motor drive belt twice. The drive belt was very loose yet the machine was still able to perform adequately.

The IR representatives didn't return to the mine on Wednesday because of early flight reservations from Charleston. The remainder of the week was the Thanksgiving holiday.

The total number of holes completed during the first two days of the week was ninety-one four footers.

Sincerely,

Dean E. Rich

Dean E. Rich

DER:np
cc: R.N. Eakin
T. Wall
D.E. Rich
F.O. Olsson

Ingersoll-Rand Research, Inc.

ex 84-3301

BOX 301
PRINCETON, N. J. 08540

Area Code 609
921-9103

January 6, 1977

Robert N. Eakin
Supervising Engineer

Subject: Flex Drill Field Trial
10th Week - Nov. 20 - Dec. 3

Monday, November 29, 1976 was spent as a traveling day returning from the Thanksgiving weekend. The replacement ratchet cylinder rods were picked up at the airport.

The next day, Tuesday, November 30, 1976 a successful demonstration was presented to Donald Hamilton and Thomas Dougherty of FMC Corporation. Both men have had previous mining machinery experience and appeared pleased with the demonstration presented. After the demonstration the drill steel coupling adapted for tightening the bolts broke and was replaced with a new one. The ratchet cylinder required adjustment an additional time. However, during the course of the day twenty-four holes were drilled and bolted which completed the contract requirements.

On Wednesday, December 1, 1976 an additional seven holes were drilled and bolted. The bolting was performed using the Bendix Bolt Bender exclusively. The Bendix Bolt Bender presents numerous problems for the operator and a definite lengthening of the cycle time. The different combinations available to the operator in inserting the bolt in the bender and the problems encountered with each method are attached with sketches to this report.

The following day, Thursday, December 1, 1976, R. Eakin and G. Kellinger from the IR Research Center came to observe the problems of the Flex Drill with the Bendix Bolt Bender and to compare the Flex Drill with a production bolting vehicle. The significant problem areas discussed for the redesign included the upper clamps, lower clamp, centralizers, carousel drill steel length and the Bendix Bolt Bender. A total of eight holes were drilled and bolted by all the personal involved in the discussion. The redesign of the Flex Drill will require an improvement in all of these areas. A major improvement for reducing cycle time would be the lengthening of the drill steel and the carousel. At this time it was decided to tram the vehicle using a clearance of only 9" from the top and 9" from the bottom as a guide for determining the maximum carousel length. After tramping the vehicle in different areas of the mine it was determined that this clearance would not be enough for all conditions encountered. With this knowledge a pivot motion on the carousel will have to be considered.

The remaining time of the week was spent replacing the blower drive belt and observing a Lee-Norse bolter and continuous miner.

There were thirty-nine holes drilled and bolted for the week.

Sincerely,

Dean E. Rich

Dean E. Rich

DER:np

cc: R.N. Eakin
T. Wall
D.E. Rich
F.O. Olsson

Ingersoll-Rand Research, Inc.

Telex 84-3301

BOX 301
PRINCETON, N. J. 08540

Area Code 609
921-9103

January 6, 1977

Robert N. Eakin
Supervising Engineer

Subject: Flex Drill Field Trial
11th week - Dec. 6 - 10

At the beginning of the eleventh week the vehicle was trammed to the face of entry number one of section # 1. These conditions were the same as encountered in previous areas except for the water present. The seam height was 48" with a slight downhill grade. The water present would accumulate to six or eight inches within a couple of hours which would then have to be pumped to a different section of the mine. The water presented problems only for the operator and not for the machine. Under these conditions nineteen holes were drilled and bolted which completes one box-cut for the Bethlehem mine bolting team. In addition to bolting the present position enabled the I-R representatives to set temporary jacks and mark the roof as observed in previous weeks. There were many alignment problems encountered because of tramping the vehicle to the face; however, these were resolved and the only part replaced was one carousel ratchet key.

On Tuesday, December 7, 1976 a total of twenty holes were drilled and bolted which completed the bolting of one box-cut of coal. The presence of six inches of water presented additional problems for the bolting crew. The normal problems encountered were the replacement of three carousel ratchet keys and the adjustment of limit switch # 5. The setting of the temporary jacks, marking the roof, and moving the portable sump pumps make the bolting procedure significantly more difficult.

The following day, Wednesday, December 8 there was no power available in the mine for five hours while the power source was moved closer to the face. When power was restored to the machine a demonstration was conducted for T. Cushman and H. Phillips of the Lee-Norse Company. The demonstration was conducted to present the machine concept; advantages and disadvantages to the Lee-Norse representatives for their evaluation.

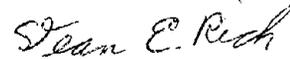
Thursday, December 9, 1976 the Flex Drill vehicle was trammed to the belt tailpiece and prepared for leaving the mine. This included removing the tool box from the mine and leaving the remaining bolts that were on the machine in the mine. The work to be done outside the mine was the construction of a new plywood box for the return trip to New Jersey. The materials

for a new box were supplied by the Bethlehem mine for which they will have to be reimbursed.

The last day of the week, Firday, December 10, 1976 was spent getting ready to return to New Jersey.

The total number of holes drilled and bolted for the week was fifty-one four footers.

Sincerely,



Dean E. Rich

DER:np

cc: R.N. Eakin
T. Wall
D. Rich
F. Olsson

Ingersoll-Rand Research, Inc.

Telex 84-3301

BOX 301
PRINCETON, N. J. 08540

Area Code 609
921-9103

January 6, 1977

Robert N. Eakin
Supervising Engineer

Subject: Flex Drill Field Trial
12th Week - Final Week
December 13 - 17

On Monday, December 13, 1976 the IR representatives packed the spare parts and accumulated items associated with the field trial into the wooden box and made final arrangements for the truck the following day. The Bethlehem mine personnel assured the IR representatives that the Flex Drill vehicle would be brought out of the mine that evening.

On Tuesday, December 14, 1976 the truck from S.H. Trucking was there at 8:00 A.M. ready for loading. The Flex Drill vehicle was brought out of the mine at approximately 9:00 A.M. and loaded on the truck at that time. The spare parts crate was also loaded on the truck at this time along with a few rock samples requested by IR. The truck was loaded and the paper work completed by 10:30 A.M., at which time the field trial was finished. IR representatives returned to New Jersey.

Sincerely,

Dean E. Rich

Dean E. Rich

DER:np

cc: R.N. Eakin
T. Wall
D.E. Rich
F.O. Olsson



United States Department of the Interior

BUREAU OF MINES

EAST 315 MONTGOMERY AVENUE
SPOKANE, WASHINGTON 99207

21.17 Copy of Letter from R. Burham to
R. Eakin - 5/10/76

May 10, 1976

Mr. Robert Eiken
Ingersoll-Rand Research, Inc.
Box 301
Princeton, NJ 08540

Dear Mr. Eiken:

In order that the Government may perform an equitable evaluation of all systems developed under the Flex-Drill Program, the USBM has compiled the following evaluation criteria for this purpose. Items in sections 1, 2 and 3 will be rated on a 10-point scale and weighted according to their relative importance in the program.

1. Systems Engineering

- A. Ease of fabrication
- B. Repairability
- C. Practicality of rotation and thrust system
- D. Cuttings and dust removal or suppression system
- E. Total system simplicity
- F. Adaptability to a 30" coal seam
- G. Simplicity of maintenance
- H. Simplicity of adjustments
- I. Vulnerability to damage from mine environment and other mine equipment
- J. Compatibility with other mining equipment

2. Performance Evaluation

- A. Typical optimized drilling speed vs. rock properties
- B. Total drilling system performance including bit life
- C. Effectiveness of dust and cuttings removal system
- D. Comparison of flex-drill/bolt system to conventional drill/bolt system
- E. Operational and repair record during test of drill system
- F. Total system performance (drill, RBI, bolt torquer)
- G. Adjustments (mechanical and operator) required



3. Cost Analysis

- A. Cost comparison of bolted holes (Flex. vs. Conventional)
- B. System cost comparison (Flex. vs. Conventional)
- C. Repair cost comparison (flex. vs. Conventional)

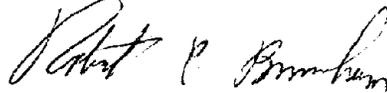
4. Geological Data

- A. Strata definition and description
- B. Entry description
- C. Roof and floor description

Contractors will be expected to provide sufficient data in their final reports pertaining to the above criteria to allow the United States Bureau of Mines evaluation committee personnel to make proper evaluation of each system.

Any questions concerning the above should be directed to the undersigned.

Sincerely,



Robert E. Burnham
Technical Project Officer

FLEXIBLE ROOF DRILL FOR LOW COAL

VOLUME II

(This volume covers work performed from June, 1977
through May, 1979.)

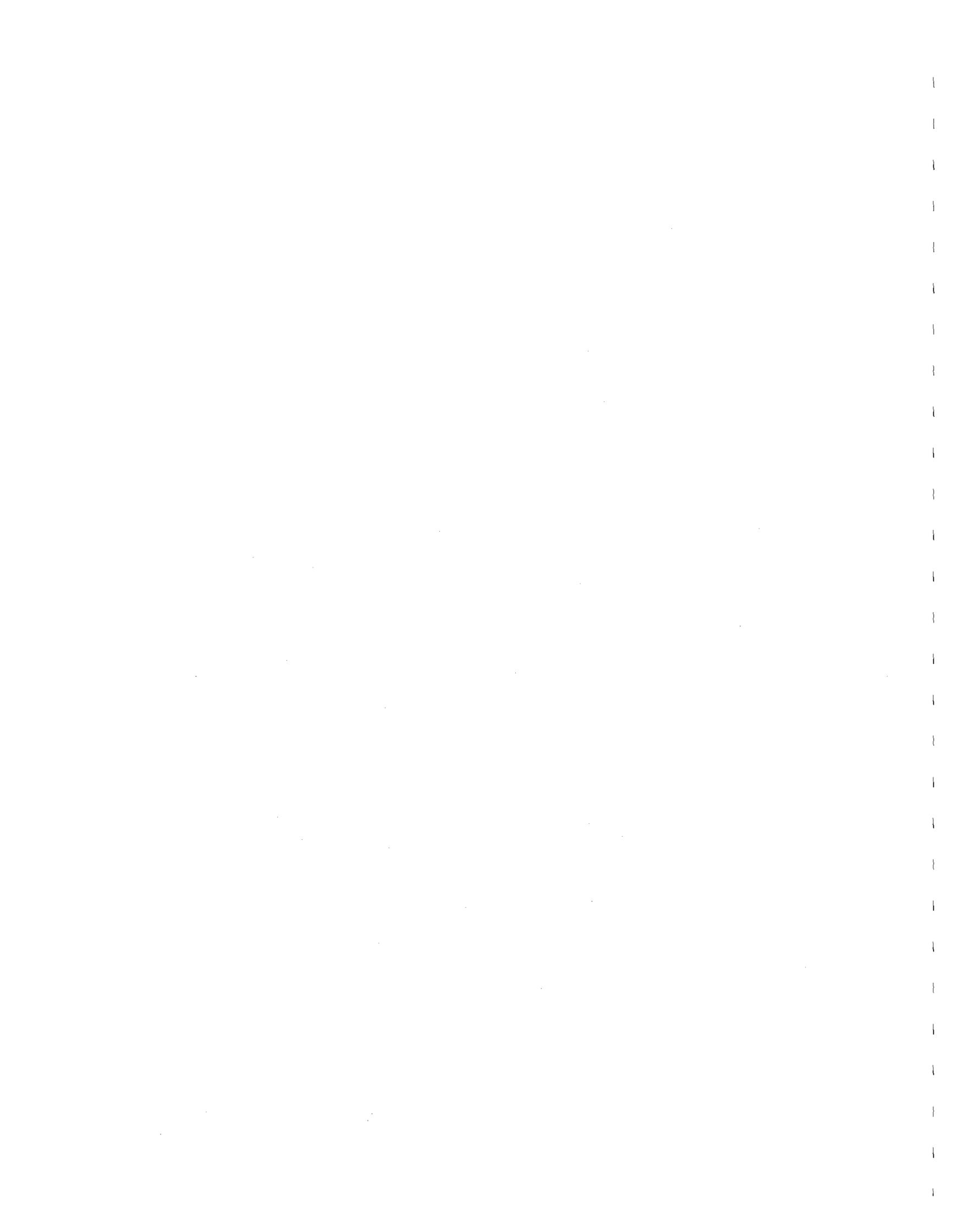


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ABSTRACT

Volume II of the Final Report for DOE Contract No. ET-74-C-01-9090 discusses a modification carried out on the Flexible Drill for Low Coal to improve its performance following an underground test.

Nine areas of the Flex Drill design were modified to improve the operational reliability of the drill. In addition, a drilling simulator was constructed so that extended testing of the drill could be performed. Forty-five hundred eight-foot holes were simulated with this device. In addition, fifteen hundred five-foot holes were drilled in limestone during testing.

At the close of the scheduled testing, the drill assembly had exceeded the reliability level of 935 holes between failure. One interval was 926 holes, and the machine had completed 3810 holes without a failure at the close of testing. Other good runs were 402 holes and 522 holes. The 3810 hole level is 25 times the longest run achieved in earlier underground testing with the original machine.

During the course of the program, several investigations important to later rod changing drill programs were conducted. The double socket chuck design was made useful through this program. The reliability of the modified drill string it uses was explored. The drill steel coupling design was developed through testing.

EXECUTIVE SUMMARY

In the final report for United States Bureau of Mines Contract HO242063, dated June 1977, is a set of recommendations for improving the Flexible Drill Assembly developed under that Contract by Ingersoll-Rand Research, Inc., for the Bureau. This report, Volume II of the Final Report, is a summary of the redesign work initiated by the Bureau in response; and the operational testing performed to verify the improvements. Ingersoll-Rand Research, Inc. (hereafter termed IRRI) was the contractor for this later work as well. Responsibility for administration of this contract has transferred to the Department of Energy and the Contract Number has changed. The current contract number is ET-74-C-01-9090.

Nine areas of the Flex Drill design were modified to improve the operational reliability of the drill. In addition, a drilling simulator was constructed so that extended testing of the drill could be performed. Forty-five hundred eight-foot holes were simulated with this device. In addition, fifteen hundred five-foot holes were drilled in limestone during testing.

At the close of the scheduled testing, the drill assembly had exceeded the reliability level of 935 holes between failure. One interval was 926 holes, and the machine had completed 3810 holes without a failure at the close of testing. Other good runs were 402 holes and 522 holes. The 3810 hole level is 25 times the longest run achieved in earlier underground testing with the original machine.

During the course of the program, several investigations important to later rod changing drill programs were conducted. The double socket chuck design was made useful through this program. The reliability of the modified drill string it uses was explored. The drill steel coupling design was developed through testing.

In spite of the great improvement made in the machine's reliability, it must be recognized that the Flexible Roof Drill in its current form does not promise to be a commercial machine.

First, the machine will be more expensive than a conventional manual roof bolter. The automatic systems insure that. Second, the machine is slow to bolt. This

problem is not easily corrected in the Flexible Drill. The Flexible Drill uses six drill steel sections to drill an eight-foot hole. This results in an excessive number of coupling and uncoupling operations. With the limitations of the current air logic system, these operations take too much time - on the order of 30 seconds for each steel used. The current system cannot be speeded up without significantly enlarging the air logic system. This conclusion was verified in testing. The hydraulic system is capable of operating the mechanisms faster than the air logic can control them. These findings are being incorporated in another ongoing DOE program to develop a fully automatic roof bolter module, so Flex Drill's experience will not be lost, even if the machine is not commercialized.

Third, the machine in its current form offers few safety advantages to its operator. True, the operator does not now handle turning drill rods, but he is still required at the drill, under dangerous top. It remains for later programs to build upon the success of Flex Drill to produce a complete remotely-operated roof bolter, where the operator is no longer exposed to a dangerous roof condition. Flex Drill has demonstrated that such a machine is possible utilizing the rod changing concept.

1.0 Introduction

In the final report for United States Bureau of Mines Contract H0242063, dated June 1977, is a set of recommendations for improving the Flexible Drill Assembly developed under that Contract by Ingersoll-Rand Research, Inc., for the Bureau. This report is a summary of the redesign work initiated by the Bureau in response; and the operational testing performed to verify the improvements. Ingersoll-Rand Research, Inc. (hereafter termed IRRI) was the contractor for this later work as well. Responsibility for administration of this contract has transferred to the Department of Energy and the Contract Number has changed. The current contract number is ET-74-C-01-9090.

Nine areas of the Flex Drill design were modified to improve the operational reliability of the drill. In addition, a drilling simulator was constructed so that extended testing of the drill could be performed. Forty-five hundred eight-foot holes were simulated with this device. In addition, fifteen hundred five-foot holes were drilled in limestone during testing.

At the close of the scheduled testing, the drill assembly had exceeded the reliability level of 935 holes between failure. One interval was 926 holes, and the machine had completed 3810 holes without a failure at the close of testing. Other good runs were 402 holes and 522 holes. The mean time between failure for all Flex Drill testing done was 242 6-foot holes.

An overall view of the Flexible Drill is given in Figure 1 on the following page. The drill is shown by itself, and in position for testing under the simulator and a limestone block.

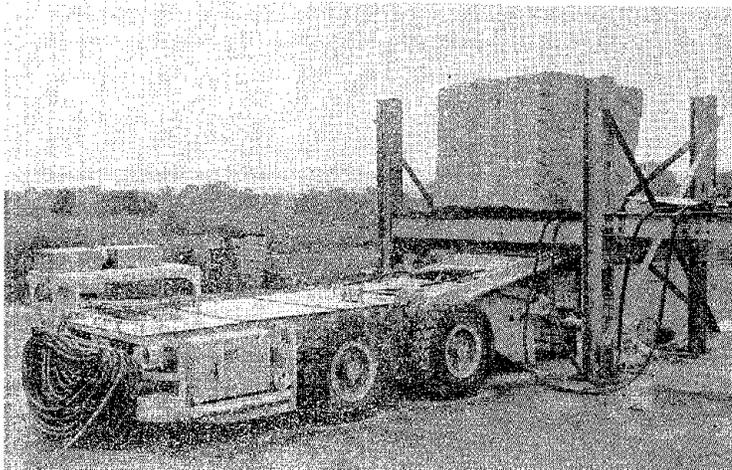
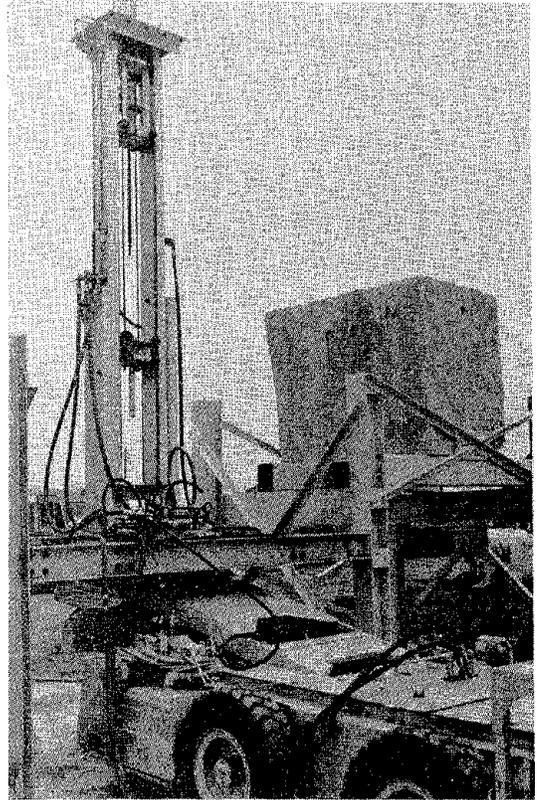
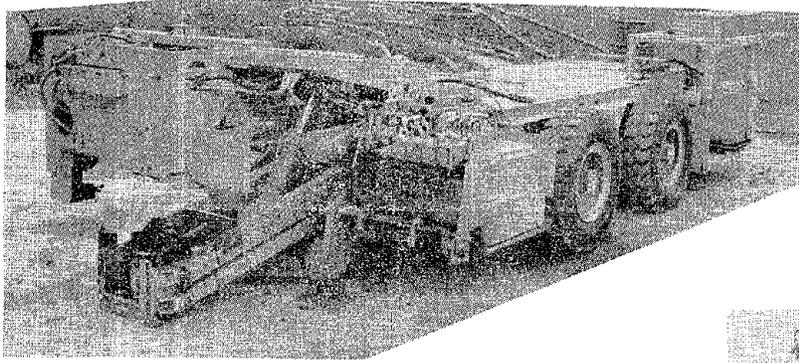


Figure 1. The Flexible Drill.

2.0 Design Revision

The nine areas of the Flex Drill improved in this contract were:

- Chassis Mounting
- Centralizer
- Upper Clamp
- Air System
- Hydraulic System
- Carousel
- Lower Clamp
- Drill Chuck
- Drill Steel Couplings

The reasons for each improvement and the resulting equipment changes will be described individually.

2.1 Chassis Mounting

The Flexible Drill is a modification of a currently used underground boom-type roof drill. The semi-automatic steel handling system which is the heart of Flex Drill was adapted to a Lee-Norse TD1-29 roof drill.

In the course of the underground testing, several problems showed up which can be traced to this marriage. First, the chassis which carries the steel changer replaced the roof contact ring of the safety arm. The chassis was held in a vertical position by a parallelogram link which derived its orientation from the original safety arm slides of the roof bolter. The original design proved to be unable to maintain a consistent chassis orientation to the drill box during

underground drilling. Since the chassis would not maintain its relationship with the drill box, the limit switches cammed by the drill box would not operate reliably. The drill steels could not be presented over the drill box consistently either, so coupling was unreliable.

The field fix employed was to add a large spring to bias the mounting to a known position. Problems to be corrected were identified as excessive clearance in the safety arm slide track and a flexing parallelogram link.

The figures below show the changes as installed. Figure 2 shows a front view of the right hand side of the chassis installed on the safety arm. The drill box rests on the ground below the chassis. The lower extension of the chassis protects a limit switch which was modified for more reliable operation. The parallelogram link is in evidence to the left of the chassis extending back to the slide area. The link was strengthened to reduce flexing and now has a T-shaped cross-section.

Figure 3 shows the slider block which replaced two rollers in the slide assembly orienting the chassis. The rollers were originally 8 inches apart and they have been replaced by a 15-inch slider block. The 15-inch block thus cuts the angular misalignment potential of the slide roughly in half for the same operating clearance. Reducing the slide's misalignment reduced the chassis' angular misalignment potential by the same amount.

Figures 4 and 5 show the revised lower limit switch conceived to overcome the unreliable switching noted in the field. The limit switch of Figure 4 is now mounted on a spring loaded movable block with a cam roller mounted in front. The cam roller rides on the cam shoe shown in the foreground of Figure 5. With this arrangement the limit switch is always in the same relative position to its trip lugs, regardless of the precise fore and aft location of the drill box to the chassis. The success of the slider parallelogram changes appears to have made this change somewhat redundant for its original purpose, but its inclusion simplifies the adjustment of the drill. Now, the parallelogram link can be adjusted to get the best steel coupling location and the limit switch set points are not affected.



Figure 2. Chassis mounting changes.

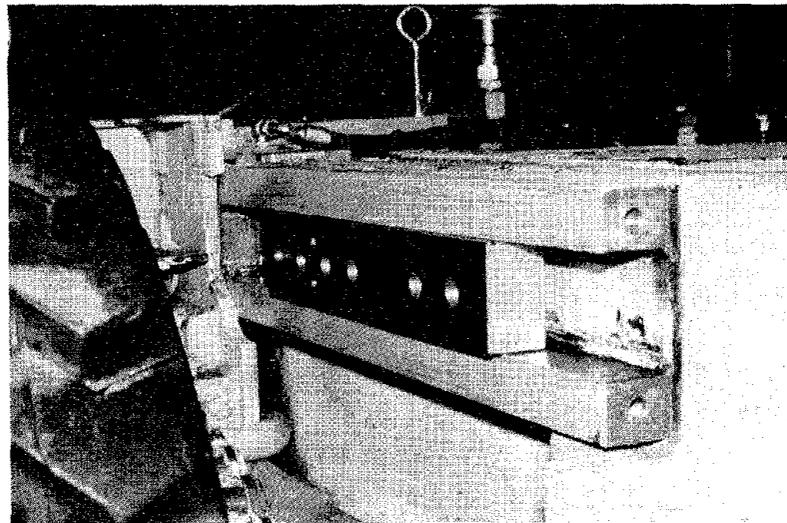


Figure 3. Slider block installation.

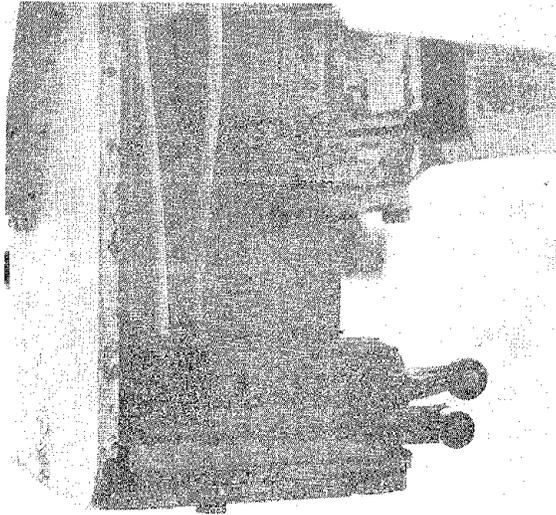


Figure 4. Modified lower limit switch.

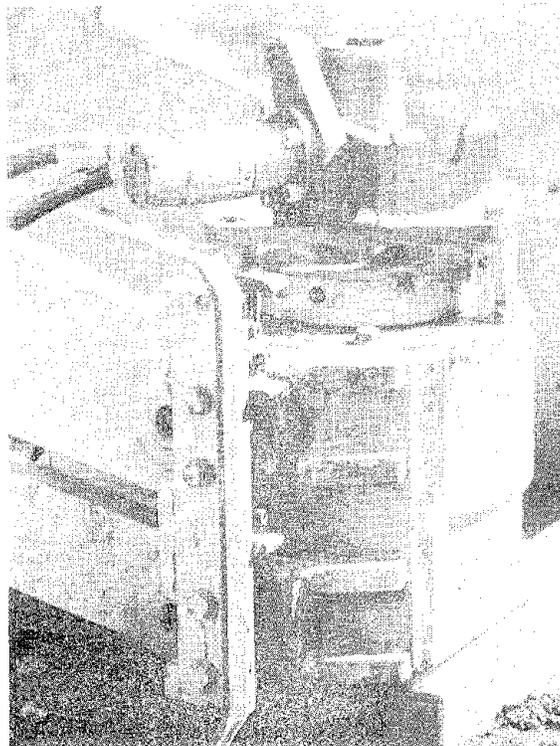


Figure 5. Cam shoe for limit switch.

A problem noticed during the underground testing was that the chassis contacted the roof on one side before the other, owing to irregularities in the roof. When this happened the safety arm twisted, throwing the chassis out of alignment with the drill box. The correction is shown in Figure 6. A single roof contact point in the center of the safety arm now touches the roof. Off-center loads do not occur, so twisting is no longer a problem.

A third mounting problem that developed during testing concerned the limit switch which indicates that the drill box is down below the working area of the carousel. The original cam operating the switch was too short, so that it was possible to lower the drill box too much and cause a logic malfunction. A longer, stronger cam was designed to prevent this override from occurring. The result is shown in Figure 7.

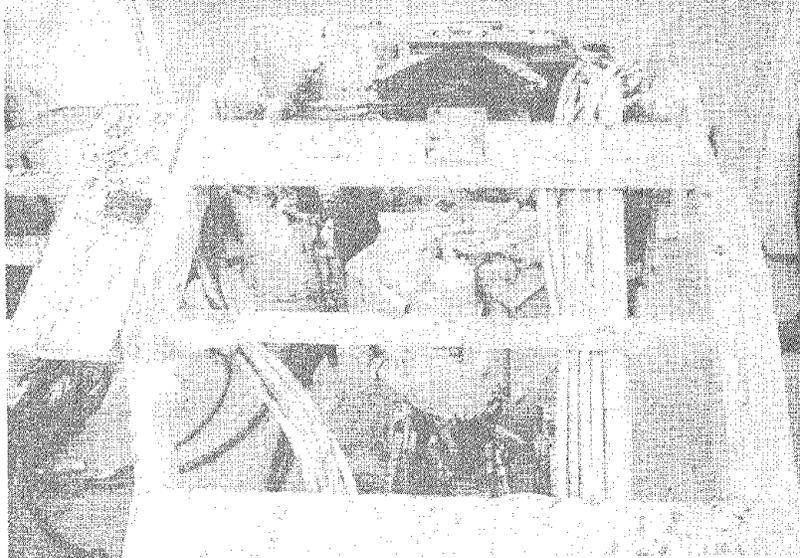


Figure 6. Revised roof contact pad.

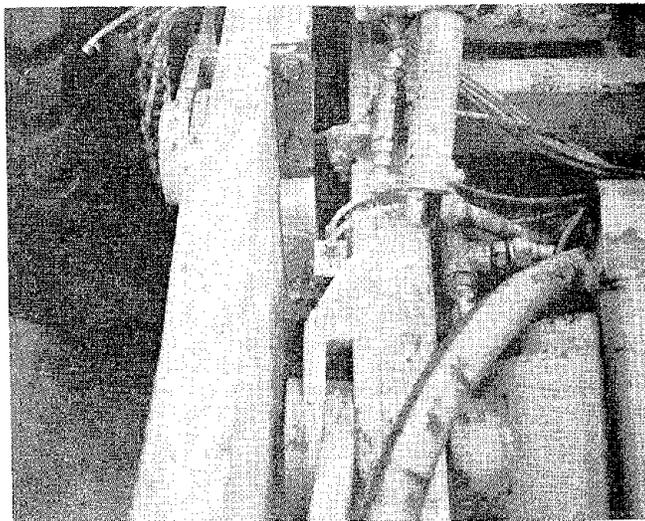


Figure 7. Revised cam for limit switch.

2.2 Centralizer and Upper Clamp

Two major areas improved were the centralizer and upper clamp designs. The two will be discussed together as they are inseparably linked in the machine. Underground experience with these parts revealed that the centralizer and upper clamp both opened up under drilling conditions. The centralizer's being unable to resist the sideloads of starting a hole resulted in off-axis holes, which was hard on drill steel couplings and made joining steels difficult. The upper clamp would score the drill couplings and perhaps cause a malfunction in the drill cycle when it was forced open during drilling. The centralizer also showed unacceptable wear rates on the surfaces that contacted the drill steel. Neither the centralizer nor upper clamp designs were capable of resisting the thrust of the drill box during an operating malfunction. The arms were originally pivoted on cantilevered pins which would not remain tight under repeated bumping by the drill box. In addition, the upper clamp arms broke under the strain.

Figures 8, 9, and 10 show the mechanical changes made to the upper clamp and centralizer assembly. In Figure 8, the new unpainted assembly is in the foreground, with the original assembly in the rear. The mount assembly is considerably strengthened. Provision has been made to hold the upper clamp and centralizer arms within a saddle, rather than carry them on cantilevered pins as before. Figure 9 shows the original mounting of the upper clamp and the new saddle mount quite clearly. The original mount could not take the loads encountered and had to be tightened up repeatedly. The special tube fittings that were fabricated for the new larger centralizer cylinder also show in the figure. These low-height fittings were needed to allow the drill box to pass the cylinder.

Figure 10 shows a view from the bottom of the two assemblies. The bottom edge of the original assembly corresponds to the notch in the new assembly. To increase the centralizer's lockup force, the mount assembly was extended through the Flex Drill chassis approximately 2-inches. This enabled a toggle link assembly to be included which materially increased the sideforce which the centralizer can resist.

Figures 11 and 12 show the strength improvements made in the individual upper clamp and centralizer arms. Figure 11 shows the original upper clamp arm and its replacement. The failure of the original arm is evident in the cracks around the pivot hole. The new arm does away with a bushed pivot hole to gain additional bending strength around the hole. In addition, the new arm is slightly thicker for a strength improvement. Figure 12 contrasts the new larger centralizer arms with the original design. The revised centralizer arms and upper clamp arms were nitrided to improve wear resistance during drilling.

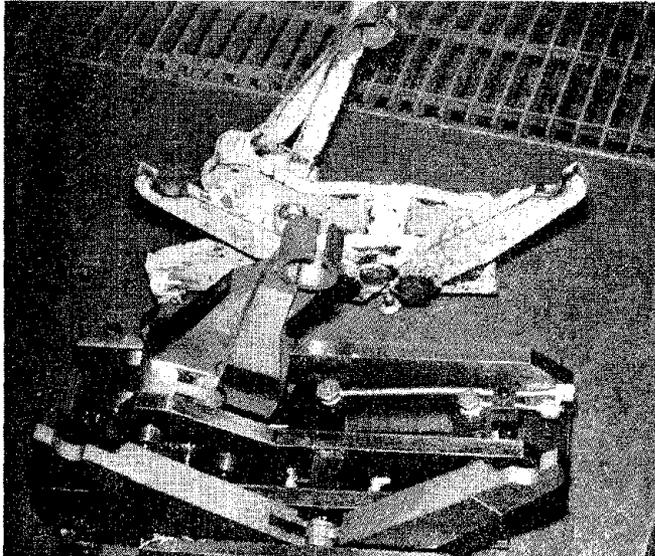


Figure 8. Upper clamp and centralizer assembly.

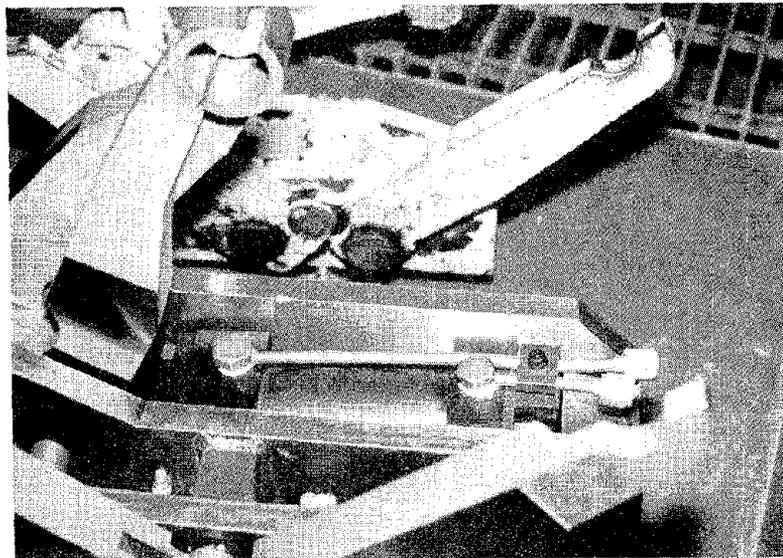


Figure 9. Cylinder detail.

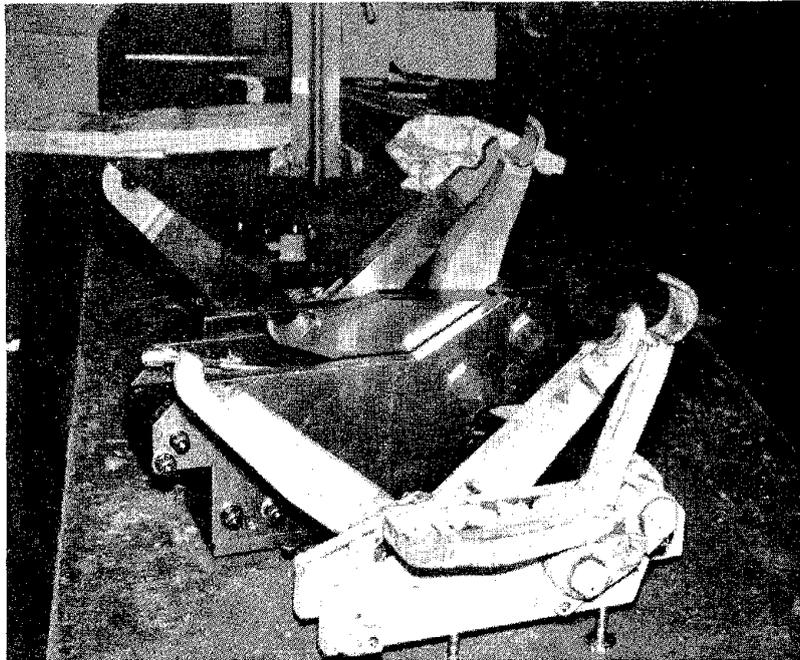


Figure 10. Bottom view of upper clamp and centralizer.

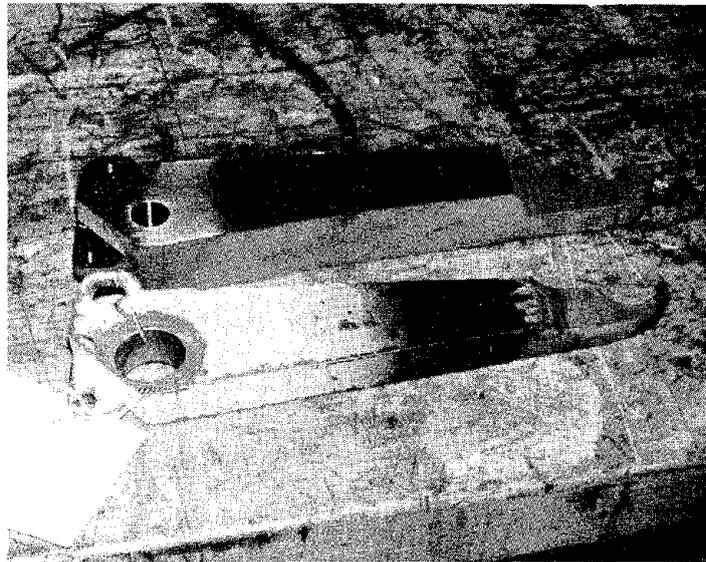


Figure 11. Upper clamp arms.

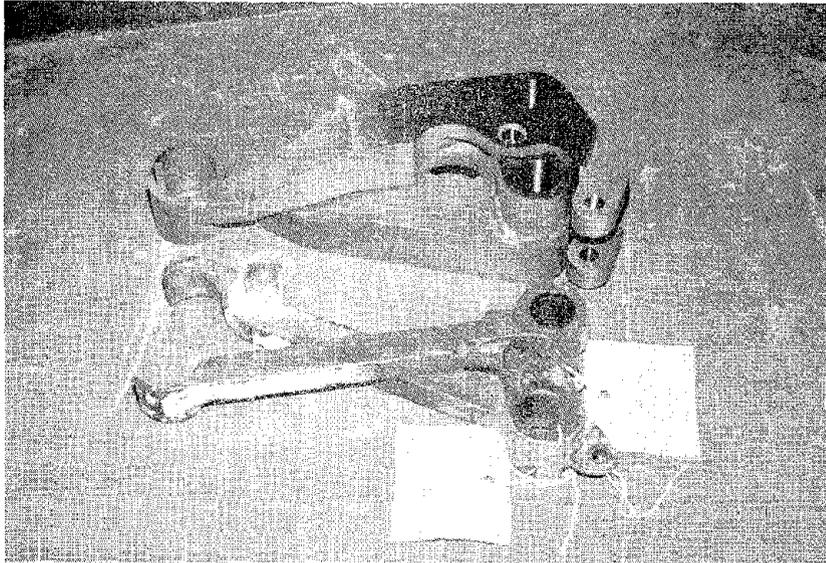


Figure 12. Centralizer arms.

2.3 Air System and Hydraulics

The changes to the fluid power systems of Flex Drill will be discussed together. The work on the air system was mainly of a cleanup nature. No changes to the air logic were needed. Underground testing had revealed the need for a moisture disposal method more efficient than taking off a filter housing once a day. The hydraulic system was modified to utilize an unloader valve and accumulators in place of the relief valve originally used. The marriage of Flex Drill to an existing roof bolter had resulted in a fixed-volume gear pump feeding oil to a closed-center valvebank. A relief valve provided pressure control for the valve stack. The net result was an unnecessary heat load on the hydraulic oil.

Figures 13, 14, and 15 demonstrate the air logic area at the rear of the machine. This area was extensively modified during the rebuilding process. In Figure 13 several things are evident. To the left rear of the area is the unloader valve for the new accumulators which were installed in the machine. Next to that, in the center of the machine, is the new air filter assembly. This filter assembly consists of two particle entrapment filters and a microlescer filter designed to remove oil. All three of the filters are fitted with automatic float drains to prevent moisture accumulation. On the right of the area under the plumbing is the air compressor assembly. This assembly was not changed in the redesign. To the left lower corner of the figure is the new air logic box. This box is considerably reduced in size from the original assembly.

Figure 14 is a view of the air logic area toward the front of the machine. On the right of the area is the air logic box. In the center of the picture is the flow control valve which was installed to control the drill rotary speed during automatic coupling.

Figure 15 shows the reduction in size which was achieved by repackaging the air logic unit. The air logic unit was originally contained in the large box sitting on the machine to the right. In the modification process, all of the logic was accomodated in the smaller box to the left. The two 11-inch square

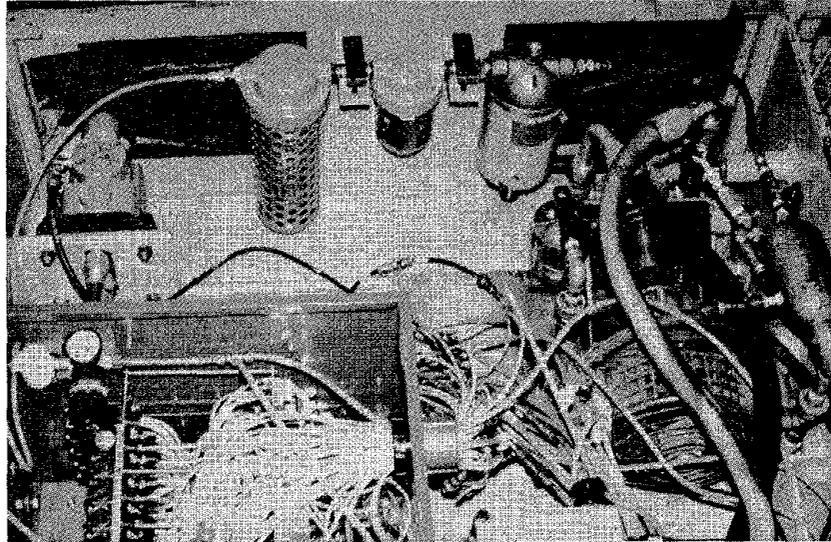


Figure 13. Air logic area.

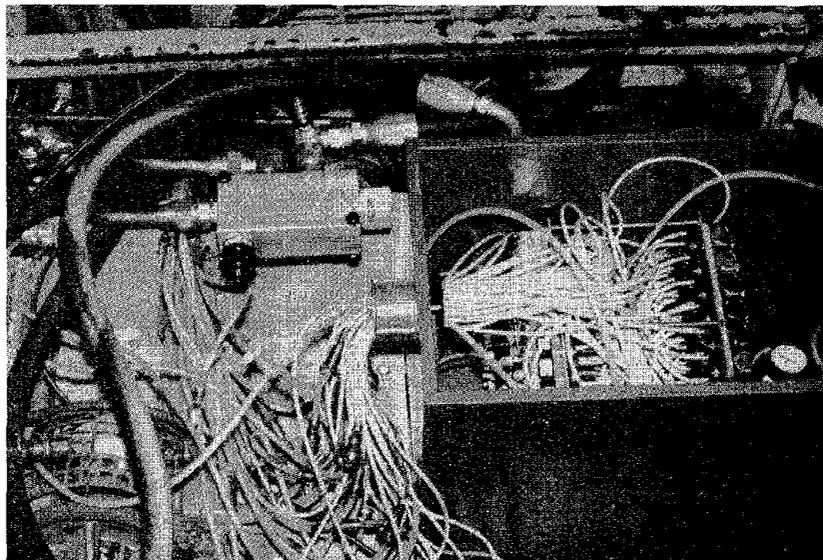


Figure 14. Air logic area.

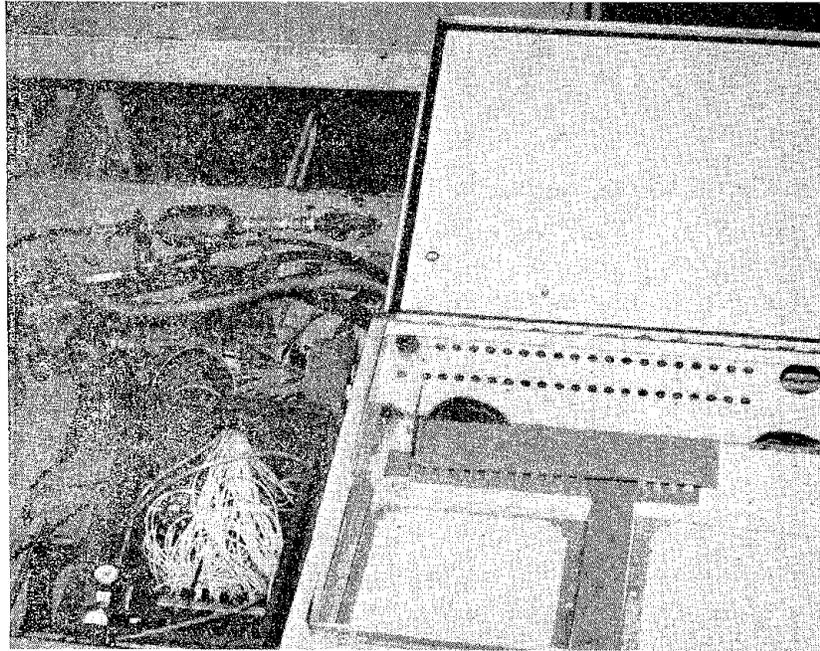


Figure 15. Packaging for the air logic.

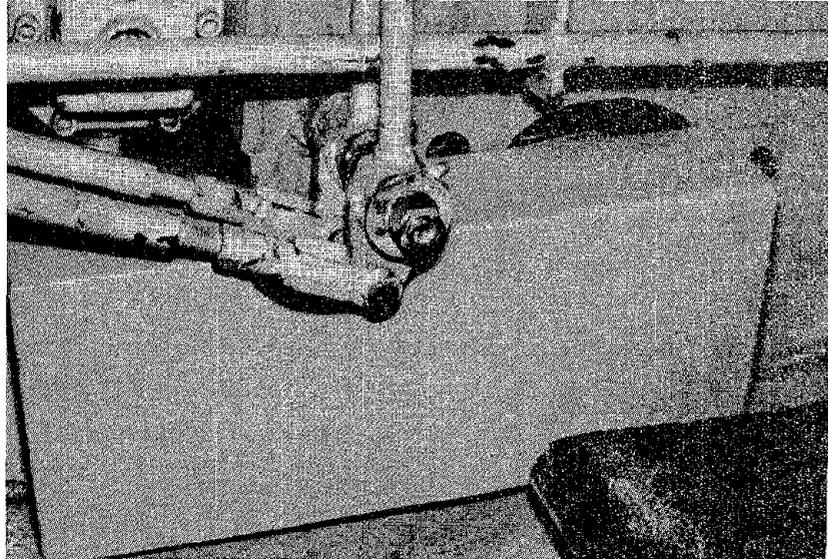


Figure 18. Accumulator guard.

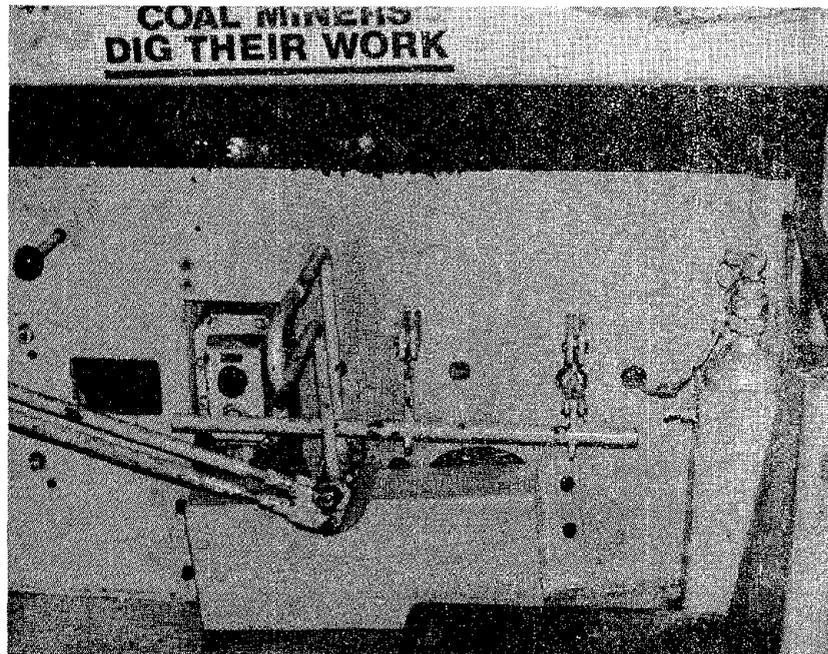


Figure 19. Operator's tram station and accumulators.

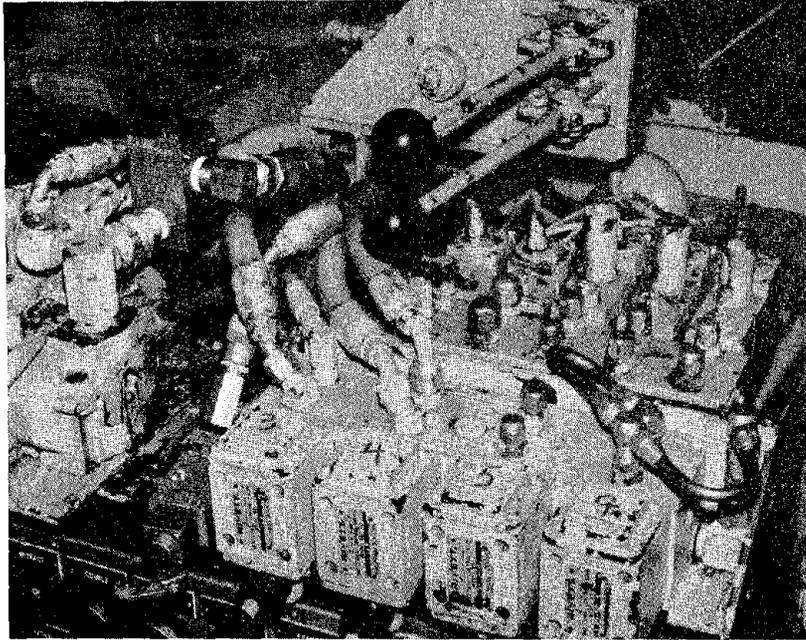


Figure 20. Hydraulic accumulator
dump valve (valve 9).

left of the picture is the pressure relief valve which was installed for the automatic couple cycle. This relief valve controls the torque which the drill box can develop during coupling.

2.4 Carousel

Several aspects of the carousel assembly caused trouble during the underground testing. The carousel was rotated by a two-directional ratchet mechanism. The hydraulic cylinder controlling the direction of rotation of the ratchet proved delicate in testing. A more rugged shifter design was needed. The ratchet rotator proved vulnerable to jams during steel changing. If a steel did not seat properly in the carousel it would block the rotation of the carousel and shear the key connecting the rotator to the carousel shaft. Repair was a time consuming operation requiring disassembly of the ratchet. The design of the carousel gate contributed to the likelihood of a steel jam occurring. It was short in length and allowed the steels to slip out of position easily.

Figures 21, 22, 23 and 24 show changes made to the carousel assembly in the redesign work. Figures 21 and 22 show two views of the linear clutch assembly added to the ratchet actuator arm of the carousel assembly. The function of these parts is to limit the force which can be applied to the ratchet assembly at the top of the carousel. In the original machine no provision for limited force was provided. Lacking this, a common occurrence was the shearing of the drive key holding the shaft and plates assembly in the carousel. In the new design the force which can be applied to the ratchet and ultimately to the drive key is limited by the linear clutch assembly.

This new assembly consists of two parts, a radial leaf spring and a piston. The radial leaf spring has knobs on the end of each leaf which seat in a groove of the piston. As an axial force is applied to the piston, the knobs are forced up out of the groove. As soon as the knobs ride on the outer surface of the piston, the axial force required to move the piston relative to the leaf spring drops to a low value. The assembly then lengthens or shortens as needed to protect the drive key. The force applied to the ratchet is limited in this instance to approximately 350 lbs. static force. In the original assembly the hydraulic cylinder driving the carousel could apply approximately 800 lbs. of force to the ratchet through the ratchet actuator link.

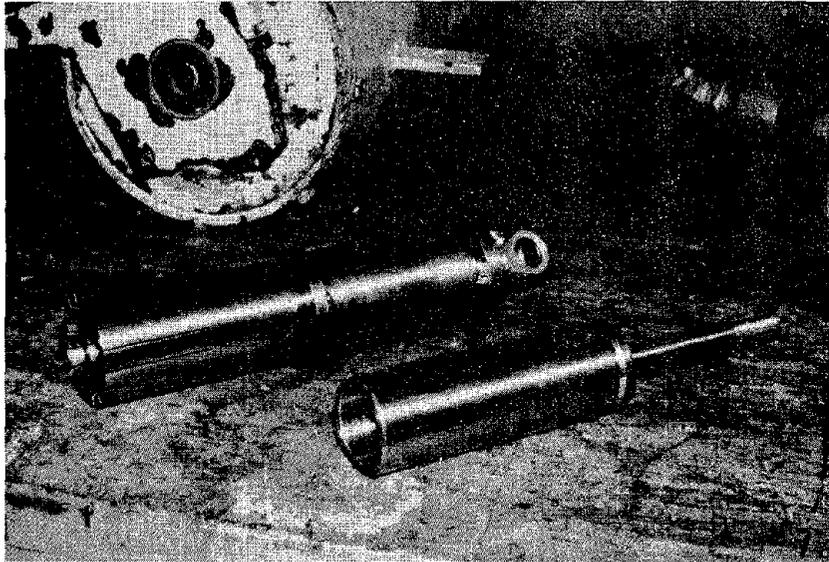


Figure 21. Linear clutch assembly.

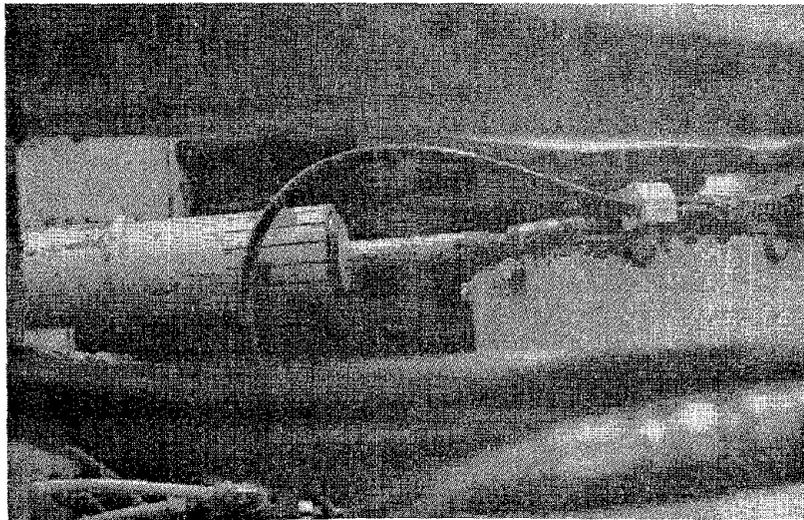


Figure 22. Linear clutch installed.

Figure 23 shows the new shifter for the ratchet assembly. The original ratchet shifter is shown at the bottom of the picture. It was a double-acting hydraulic cylinder which shifted the ratchet in either direction as required by the logic of the machine. This design proved fragile during the lifetime of the original Flex Drill assembly. The new assembly utilizes a spring to drive the ratchet to one position and a cylinder and cable arrangement to pull it back to the second of its two positions. One position is used to load the carousel and the other is used in unloading the carousel. The combination of a hydraulic cylinder operating through a flexible cable assembly was tested in the original Flex Drill as an actuator for the gate, and proved very satisfactory.

Figure 24 shows the extended gate designed for the carousel. The gate is roughly twice as long as the original and serves to restrain the steels in the pockets of the carousel at the load-unload position. The longer gate does not allow the steels to jam against the carousel housing as readily as before.

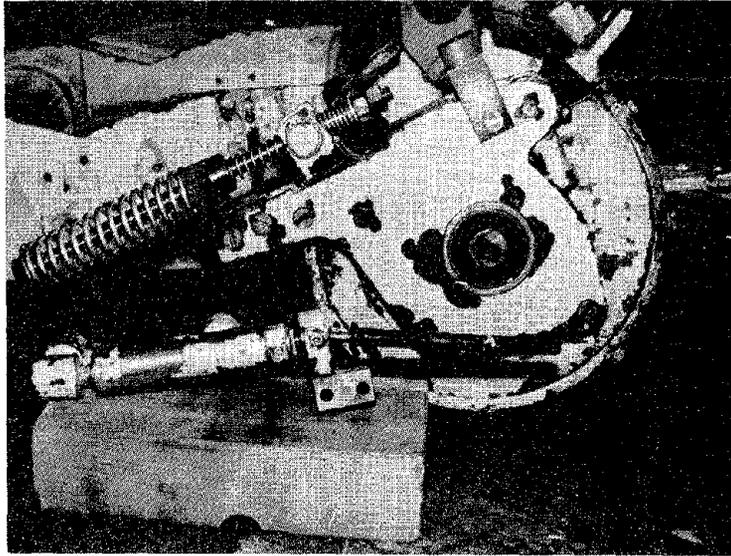


Figure 23. Ratchet shifter.

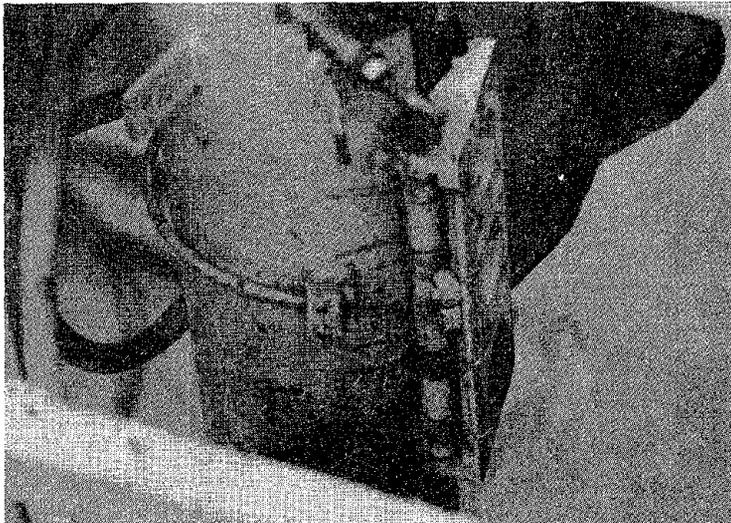


Figure 24. Carousel gate.

2.5 Lower Clamp

The lower clamp was reported as needing minor repairs during the underground testing. It was thus not a high priority item for correction. Operational testing of the revised Flex Drill was begun before it became apparent that the lower clamp was a very troublesome area. The test results to be discussed later include a number of lower clamp failures. The existing clamp was finally replaced by a strengthened version in the course of testing.

The improved clamp was derived from a design developed for DOE Contract No. ET-77-C-01-9113, a mechanical anchor bolter module. Substituting this design in Flex Drill allowed an early assessment of its reliability - valuable spinoff for the overall goal of developing a reliable, hands-off bolter. The revised lower clamp is shown in Figure 25. The concept of operation remains unchanged from the original; a fork rotates through 90 degrees to engage the groove on the drill steel when required. The strength of the parts is improved to extend the clamp's life.

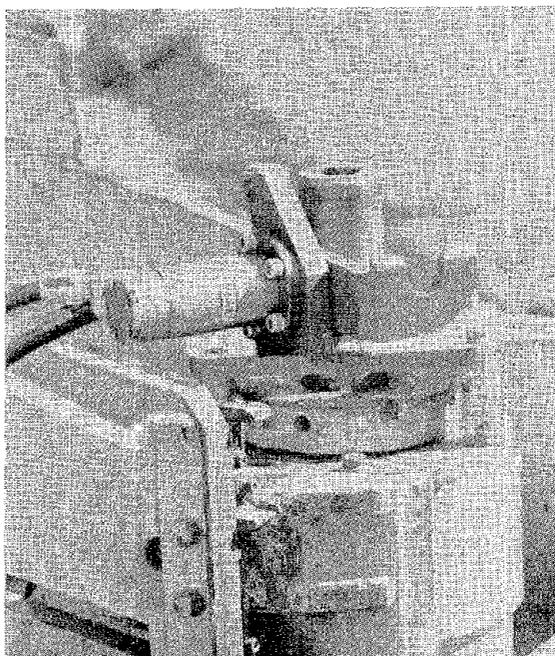


Figure 25. Revised lower clamp.

2.6 Drill Chuck and Drill Steel Couplings

The drill string has gone through several evolutions during the course of redesign and testing. Based on the underground experience, the drill steel couplers were redesigned to incorporate the changes found necessary for reliable coupling. Chief among these was the provision of a hex driving section to engage the surface drive chuck before the compression of the retaining spring (barrel spring) began. If this surface was not provided the coupling could not be engaged reliably. The coupling configuration used in the underground test is shown in Figure 26. The right end of the coupling in the photograph is the engaging end. The left end is permanently wedged in a steel. The original barrel spring location near the end of the coupling was unsatisfactory so all of the existing stock of couplings were modified to move the spring up and provide the hex driving section below it. This coupling modification worked well in the underground test. More couplings were needed to conduct the 6000 hole reliability test, so the coupling design was changed to that shown in Figure 27. The most significant change in this coupling is the removal of the pilot diameter below the hex driver. The pilot diameter resulted from reworking existing couplings and was judged unnecessary. This coupling was used in the early reliability testing. It was not entirely satisfactory, as the hex driving section was rapidly rounded off in service.

Figure 28 shows the coupling change which evolved when the rounding-off was noticed. A larger hex driving section delays the rounding-off almost indefinitely. A second very significant change is shown in this coupling as well. The two ends of the coupling are symmetrical and can be interchanged. Now when an engaging end wears out, the coupling can be reversed, exposing a new coupling. The preferential engagement of the barrel springs necessary for reliable uncoupling is secured by shimming the spring at the permanently engaged end of the coupling. The result of this change is to reduce the effective cost of a drill coupling almost in half for any given wear life. Before, all wear occurred on a single engaging end. Now two ends can be worn out before the coupling is replaced.

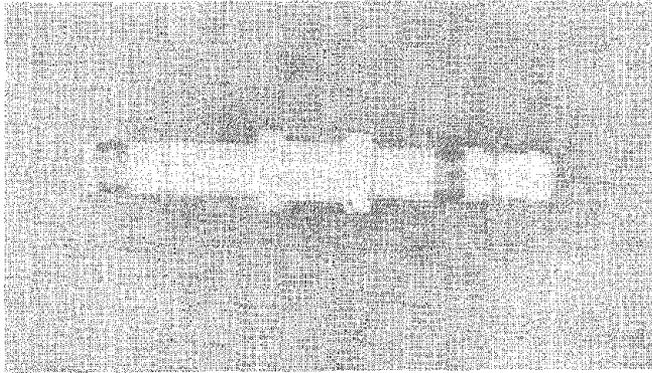


Figure 26. Drill steel coupling used during the underground test.

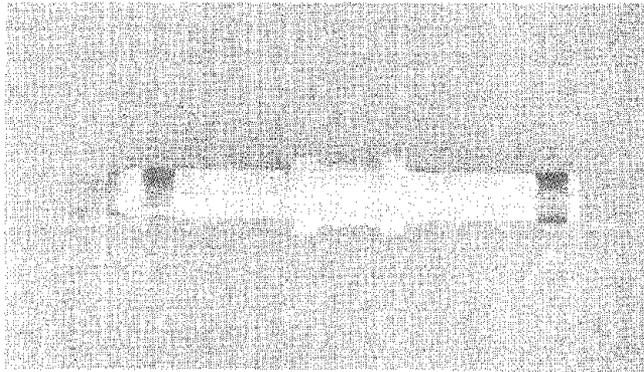


Figure 27. Interim drill steel coupling.

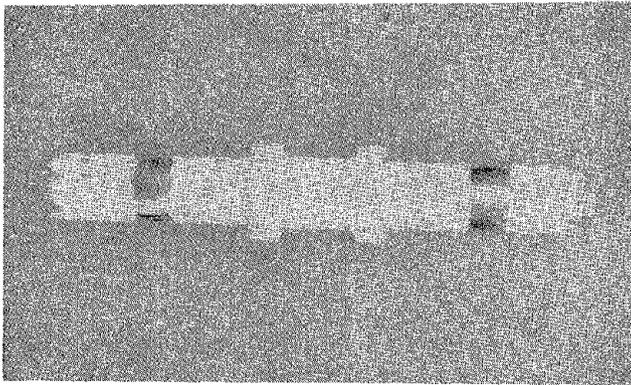


Figure 28. Current drill steel coupling.

One additional improvement has been made to the drill couplings which allows one of the Flex Drill's original design features to be utilized as intended. The original Flex Drill chuck was a double socket design, with a 1-1/8 inch square socket sitting above the hex surface drive socket needed to engage the drill steel couplings. The idea of this was to use the square socket to tighten a roof bolt after drilling the hole with the hex socket. When this was tested, the chuck would not seal the vacuum dust collection system because the 1-1/4 inch diameter drill couplings bridged the 1-1/8 inch square socket and allowed the chuck to leak. As a stop-gap measure, the square socket was filled with a piece of metal and not used during the underground testing. A separate bolting wrench was made up with a drill coupling to tighten bolts with the hex socket.

This modified chuck was in use when reliability testing commenced. Design studies on a fully-automatic mechanical anchor bolter module made the combination chuck a very attractive proposition, however, so concepts were sought to overcome its limitations. Simply reducing the steel coupling diameter to 1-1/8 inches was rejected because only a very thin shoulder or bearing surface would exist between the steels and the couplings. Finally the idea of slipping a 1-1/8 inch diameter ring up onto the coupling was proposed. The ring would span the width of the square socket and seal the chuck. It had the same thin bearing shoulder, to be sure, but it was inexpensive and easily replaced. Rings with a hex hole were fabricated from old drill steel and installed with a combination chuck. The drill string has performed satisfactorily in this configuration.

3.0 Drilling Simulator Test Stand

Simulated drilling of the 6000 hole reliability test of the modified Flexible Roof Drill was conducted on a hydraulic simulator designed, built, and installed at the Ingersoll-Rand Research Center.

The structure consists of welded and/or bolted channel and I-beam frame members which support a vertical slide mechanism. A trolley assembly rides inside the slide which carries a hydraulic pump to supply torque resistance. The trolley is also attached at its top end to a vertically mounted cylinder which provides thrust resistance. The thrust cylinder in conjunction with the torque pump provide reactions to simulate the conditions which normally occur during drilling an actual hole in a rock formation. The test stand with Flex Drill in place is shown in Figure 29. The reaction trolley is at the top of the slideways in the photograph. The thrust reaction cylinder extends out of the photograph above the slideways approximately 10 feet.

A second trolley can be seen in roughly the middle of the slideways. Its purpose is to support the lengthening drill string as "hole drilling" proceeds and prevent the string from buckling laterally under the thrust load.

To use the simulator, the Flex Drill is trammed to a position directly beneath the trolley mechanism of the tower. The bitted lead steel of the machine is replaced by an extra intermediate steel. The drive sub of the test stand carries a drill steel coupler which can be mated to the intermediate steel. The drill chuck of Flex Drill is aligned by moving the machine so that the drill column is vertical, then the safety arm is positioned against the frame structure of the test stand. With the drill column alignment established, Flex Drill can be cycled in a manner identical to that used in drilling an actual hole in rock.

Equivalent loads being applied during the simulation are derived from the calibration charts shown in Figure 30. Three hydraulic pressures, P_1 through P_3 can be read from the test stand. Their relationship to drill box torque and drill box thrust is detailed in the charts.

The hydraulic schematic of Figure 31 gives an indication of the pressure readings being used for the above charts. The relief valves located near each pressure gauge provide

the means of regulating each pressure. Operating oil for the test stand is obtained from Flex Drill. During drill simulation low pressure charge oil (P_1 pressure) is obtained from the return line of the drill box. No other oil is needed during the simulation cycle. To recover the trolley for another cycle, however, a high pressure oil source is needed. One of Flex Drill's tram circuits is diverted through a manual valve on the test stand for this purpose. Manual valve number 3 in Figure 31 lowers the reaction trolley to start another simulation.

Valve number 8 in Figure 31 is a refinement added after startup of the simulator. Because of the relief valve creating pressure P_2 , all coupling of drill steels was being done under full drilling torque. The drive sub or drill string already in the "hole" could not turn freely as is the case in a real drilling sequence during coupling. Valve 8 removes the restriction of the relief valve to approach the actual condition. This eliminated the premature coupling wear that was occurring with the simulator. The rounding off of the hex tip mentioned in the discussion of the drill steel couplings was extremely rapid before adding this valve. Couplings needed at least deburring after 5 to 10 holes. Sometimes the coupling was ruined. With the valve, the coupling problem disappeared.

Drill cycle times for Flex Drill on the simulator averaged between 4 and 4-1/2 minutes. This included the total time to load and unload 6 steels, with no time allowed nor needed for tramming while simulating.

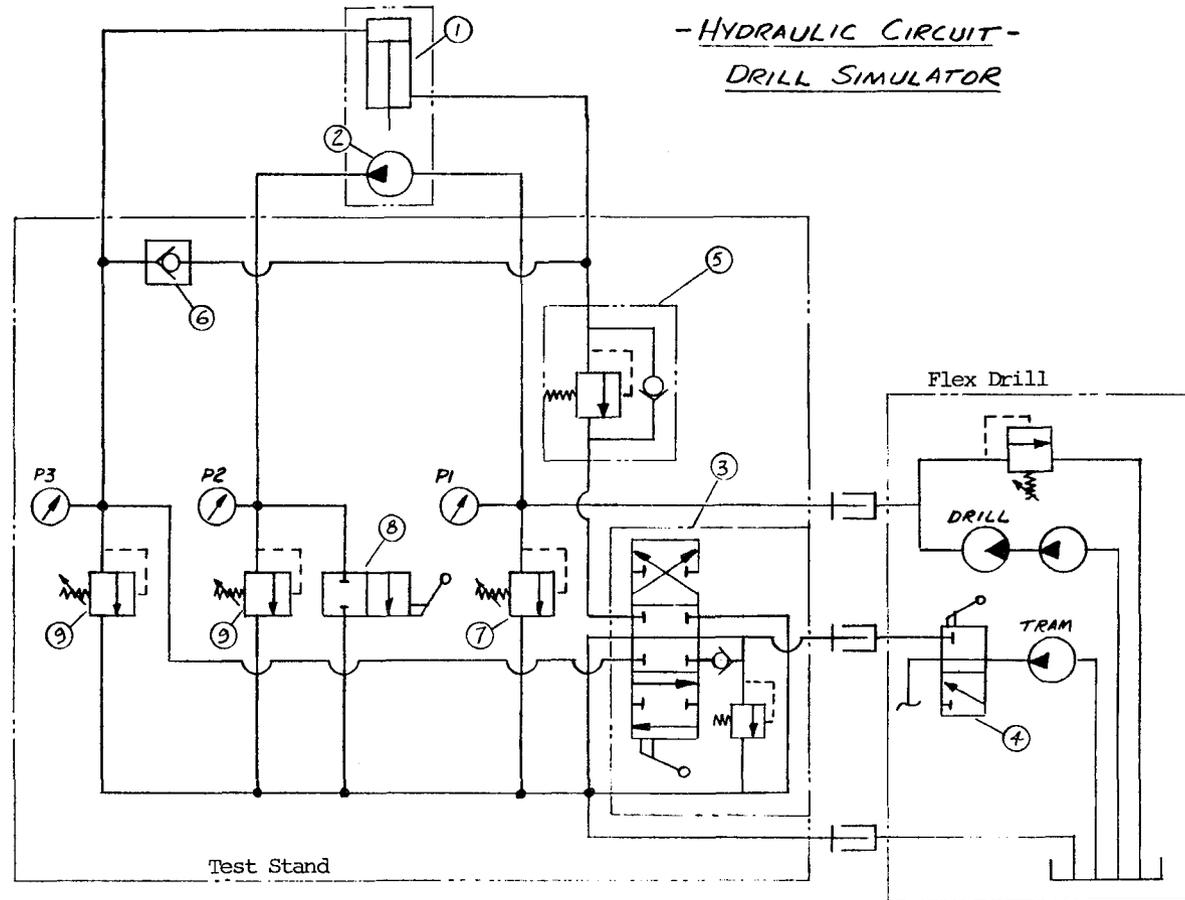


Figure 31. Drilling simulator hydraulic schematic.

4.0 Flex Drill Test Program

A major portion of the effort devoted to this program was spent in verifying that the reliability of the Flex Drill was indeed improved as a result of the redesign work. Fifteen hundred five-foot holes were drilled in Indiana limestone, and forty-five hundred eight-foot holes were simulated with the test fixture developed under this program. The limestone blocks were mounted on a test stand constructed during earlier phases of the Flex Drill Program.

4.1 Testing Progress Summary

The summary of testing progress is shown in Figure 32. The failure description is briefly given in column 5 of the figure. Columns 1 and 2 show how many actual holes five-foot long and simulated holes eight-foot long had been drilled when the failure occurred.

The difference in hole depths was due to the availability of limestone blocks 5 x 5 x 4 feet in size. Four drill steels were used to drill through the 5-foot blocks. Using the simulator, on the other hand, all six steels in the carousel could be used each time to simulate an 8-foot depth. The two kinds of testing can be "normalized", and summarized together when the operating cycle of Flex Drill itself is considered. All the logic and all the mechanisms of Flex Drill are exercised each time one steel is loaded into the hole and unloaded. There is no logic involved with hole depth, for instance. It is logical to say that a 5-foot hole takes 4 machine cycles, and an 8-foot hole takes 6 machine cycles, if a machine cycle is defined to be all of the operations necessary to load one steel into the hole and unload it. This is the information contained in columns 3 and 4 of Figure 32.

Adopting the convention of machine cycles, the time between failures for the bolter as a whole is conveniently expressed as machine cycles in Column 11. The "normalized" time between failures can then be computed for any desired hole depth by dividing the

Figure 32. Testing Progress Summary.

Actual Holes @ Failure	Simu. Holes @ Failure	Actual Machine Cycles	Simu. Machine Cycles	Failure Description	Actual Holes @ Last Failure This Part	Simu. Holes @ Last Failure This Part	Actual Machine Cycles Between Failures	Simu. Machine Cycles Between Failures	Machine Cycles Between Failures of Same Part	Bolter TBF Machine Cycles	Bolter TBF 6-Foot Hole
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)	(11)	(12)
5	0	20	0	Centralizer Grip Broke Found Toggle Link Broken	0	0	20	0	20	20	5
7	0	28	0	Air Logic Freeze-Up	0	0	8	0	28	8	2
13	0	52	0	Boom Limit Switch Freeze-Up	6	0	24	0	24	24	6
17	0	68	0	Boom Hit Carousel-Bent Lower Mount & Sheared Bolts	0	0	16	0	68	16	4
33	2	132	12	Lost Boom Pressure Unloader Valve Clogged	0	0	64	12	144	76	19
33	35	132	210	Boom Hit Bolt Bender, Sheared Bolts on Mount	0	0	0	198	342	198	49.5
70	44	280	264	Loose Set Screw in Carousel Ratchet	0	0	148	54	544	202	50.5
77	44	308	264	Lost Air Pressure Replace Compressor	0	0	28	0	572	28	7
112	44	448	264	Lost Air Pressure Replace Motor	77	44	140	0	140	140	35
116	44	464	264	Ratchet Bearing Broke	0	0	16	0	728	16	4
214	44	856	264	Lower Clamp Bent Rep'd	0	0	392	0	1120	392	98
270	80	1080	480	Lower Clamp Replaced	214	44	224	216	440	440	110
306	262	1224	1572	Problem Coupling - Shimmed Drill Motor	0	0	144	1092	2796	1236	309

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Figure 32. Testing Progress Summary.

Actual Holes @ Failure	Simu. Holes @ Failure	Actual Machine Cycles	Simu. Machine Cycles	Failure Description	Actual Holes @ Last Failure This Part	Simu. Holes @ Last Failure This Part	Actual Machine Cycles Between Failures	Simu. Machine Cycles Between Failures	Machine Cycles Between Failures of Same Part	Bolter TBF Machine Cycles	Bolter TBF 6-Foot Hole
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)	(11)	(12)
351	262	1404	1572	Lost Air Pressure Shifted to Shop Air	77	44	180	0	2404	180	45
381	262	1524	1572	Lower Clamp Shear Key	270	80	120	0	1536	120	30
442	262	1768	1572	Centralizer Grip Broke	5	0	244	0	3320	244	61
479	262	1916	1572	Lower Clamp Broke	381	262	148	0	392	148	37
511	262	2044	1572	Lower Clamp Welded to Shaft	479	262	128	0	128	128	32
537	264	2148	1584	Tighten Set Screws in Ratchet	70	44	104	12	3188	116	29
537	310	2148	1860	Lower Clamp Broke	511	262	0	276	392	276	69
675	566	2700	3396	Centralizer Grip Broke	442	262	552	1536	2756	2088	522
841	717	3364	4302	Upper Clamp Leader Sheared Synchronizer Ears	0	0	664	906	7666	1570	392.5
921	717	3682	4302	Ratchet Bearing Broke (Solid Replacement Used)	116	44	320	0	7258	320	80
1001	932	4004	5592	Centralizer Opens Slow Adjust Cushion Rates	0	0	320	1290	9596	1610	402.5
1001	975	4004	5850	Ratchet Cable Rod Broke	0	0	0	258	9854	258	64.5
1001	1035	4004	6210	Set Screws Loose in Ratchet - Welded	537	234	0	360	6662	360	90
1001	1189	4004	7134	Torsion Spring on Gate Broke	0	0	0	924	11138	924	231

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Figure 32. Testing Progress Summary.

Actual Holes @ Failure	Simu. Holes @ Failure	Actual Machine Cycles	Simu. Machine Cycles	Failure Description	Actual Holes @ Last Failure This Part	Simu. Holes @ Last Failure This Part	Actual Machine Cycles Between Failures	Simu. Machine Cycles Between Failures	Machine Cycles Between Failures of Same Part	Bolter TBF Machine Cycles	Bolter TBF 6-Foot Hole
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)	(11)	(12)
1001	1249	4004	7494	Lower Clamp Broke	675	566	0	360	5402	360	90
1001	1320	4004	7920	Upper Clamp Broke	841	717	0	426	4258	426	106.5
1090	1467	4360	8802	Repaired Upper Clamp Broke	1001	1320	356	882	1238	1238	109.5
1235	1988	4940	11928	Repaired Upper Clamp Broke	1090	1467	580	3126	3706	3706	926.5
1371	1988	5484	11928	Replaced Shaft and Plates Assembly	400	0	544	0	19012	544	136
1458	1988	5832	11928	Centralizer Grip Broke	675	566	348	0	11664	348	87
1500	4500	6000	27000	Test Terminated	--	--	168	15072	--	15240	3810

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machine cycles between failures by the number of machine cycles necessary to drill that depth. Column 12 summarizes the bolter time between failure as the number of 6-foot (4 steel changes) holes that could be completed before failure.

In the Sections IV. B through IV. H, the performance of individual systems will be assessed using the machine cycle technique.

4.2 Air Logic System

The MPAL (moving part air logic) system was chosen for Flex Drill basically because it provides extremely reliable operation. This proved to be true during the reliability testing of the machine.

During the early testing of the drill, failures in the air logic system were due to freezing moisture in the line. Testing was being done outside during the months of January and February when the temperature stayed consistently below freezing. A portable space heater was requisitioned which helped keep most of the components above freezing. When warmer weather began to set in, the space heater was no longer required. Limit switches also required a great deal of adjustment during this time because of the temperature changes from tramming the machine in and out of the cold weather.

Once the temperatures stayed above freezing and the limit switches no longer required constant adjustments, the part of the system which required attention was the compressor. Power for the logic system is provided by a oil-less open crankcase piston compressor, operating against a constant 70 psig head. Three stages of air/oil/dirt filtration are provided for maximum protection along with a small one pint receiver to dampen pulsations. Power to the compressor is supplied by a gear pump and motor combination driven off the main 40 HP electric motor.

The continuous duty cycle was too much for the compressor. It continually stalled. At one point, the compressor motor was replaced with the next larger size, but the problem continued. Two compressors were used during the tests both of the same capacities, with each one being rebuilt once during the tests to hopefully improve its operation.

Finally it was decided to bypass the compressor and use "shop air" as the power supply for the MPAL system. Repair parts delivery was extremely slow. The MPAL system itself was the chief test item, and that would be properly tested with the alternative power supply.

It was concluded at the completion of the testing that the compressor capacity would have to be increased for a reliable system. In addition, a larger receiver

should be included. Although the testing was completed, an air unloader valve was added to the compressor so that it works against the head only when there is the requirement to do so and not continually as before. By reducing the duty cycle, reliability of the system should be improved. Preliminary operation has indicated that the valve is performing its function as anticipated, however, no cycle time has been accumulated to prove its reliability.

The performance record of the air logic system is shown in Figure 33.

PART DESCRIPTION: Air Logic System

Failure Description	Actual Holes	Sim Holes	Actual Cycles	Sim Cycles	ΔAC	ΔSC	Σ Cycles Bet Fail	TBF 6' Hole
Freeze-Up in Lines	7	0	28	0	--	--	28	7
Boom Limit Switch Froze Portable Heater Acquired	13	0	52	0	24	0	24	6
Lost Air Pressure	60	44	240	264	188	264	452	113
Lost Air Pressure Replaced Compressor	77	44	308	264	68	0	68	17
Lost Air Pressure Replaced Compressor Motor	112	44	448	264	140	0	140	35
Lost Air Pressure, Shifted to Shop Air	365	262	1460	1572	1012	1308	2320	580
Test Completed	1500	4500	6000	27000	4540	25428	29968	7492

Figure 33. Air Logic System Performance

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4.3 Upper Clamp Assembly

The function of the upper clamp arms is to provide the necessary reaction forces required to couple and uncouple steels. They are located at the front of the mechanism above the centralizers at the top most portion of the assembly.

The original arms could be damaged by the thrust of the drill box. During assembly, the new, strengthened arms successfully resisted the full boom thrust. The test wasn't intentional, but it was informative. During the reliability testing, another problem surfaced.

A failure of the upper clamps was observed after 1582 total holes had accumulated. The upper clamp mechanism is comprised of two clamps, a leader and a follower, which transmit their motion through a sliding pin arrangement to clamp the steel. The leader houses the pin, while the follower contains the pin slot. The failure occurred through the material section which houses the pin. An examination by the metallurgical lab at the IRRI Center revealed it to be a fatigue failure (approximately 15,500 cycles) and also diagnosed the propagation of cracks in the follower adjacent to the pin slot. Since only about 1/3 as many cycles were obtained in the underground testing, this potential failure was not observed in the original design. The nature of the failure is demonstrated by Figure 34.

The performance record of the upper clamp assembly is shown in Figure 35. A number of the failures recorded are failures of weld-repaired clamps. As such, they contributed to Flex Drill failures which are not truly indicative of the reliability of the parts as designed. The failures were not edited out in assessing the overall reliability of the Flex Drill. There is some justification for doing so.

A new set of upper clamps was designed and installed during the testing. The material sections both perpendicular and parallel to the failure areas in both the leader and the follower were increased. The redesigned clamps accumulated a total of 15,324 operational cycles without a failure. The clamps were disassembled and examined at the completion of the testing. With the exception of some slight wear at the clamping areas, the leader and follower appeared to be in very good condition. There were no hairline fractures beginning to take affect as was noticed on the previous set of clamps.

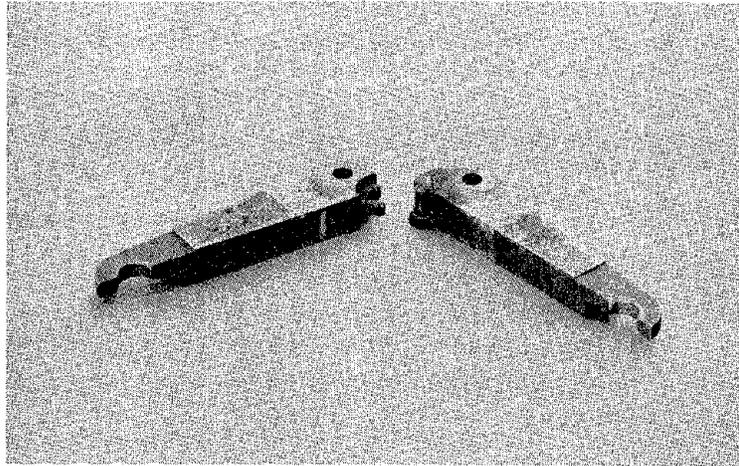


Figure 34. Upper clamp failure.

PART DESCRIPTION: Upper Clamp

Failure Description	Actual Holes	Sim Holes	Actual Cycles	Sim Cycles	ΔAC	ΔSC	Σ Cycles Bet Fail	TBF 6' Hole
Disassembled and Cleaned	129	44	516	264	0	0	780	195
Leader Sheared Ears - Replaced	841	717	3364	4302	3364	4302	7666	1916
Sheared Ears on Replacement Leader	1001	1320	4004	7920	640	3618	4258	1065
Repaired Replacement	1090	1467	4360	8802	356	882	1238	310
Repaired Replacement	1235	1988	4440	11928	580	3126	3706	927
Installed New Clamps	1500	3223	6000	19338	1060	7410	8470	2117.5
Test Completed	1500	4500	6000	27000	0	7662	7662	1915.5

Figure 35. Upper Clamp Performance

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4.4 Centralizer

The centralizer is located at the front of the chassis directly beneath the upper clamp arms. Its function is to keep the drill steels vertical while:

- a. providing sufficient force to keep the lead or bitted steel from walking along the roof surface when starting a new hold.
- b. aligning the drill steels when loading and unloading the carousel.
- c. aligning the drill steels for coupling to the drill string in the hole.

The performance record of the centralizer is shown in Figure 36.

The centralizer is essentially a scissors type mechanism, which is activated through a force multiplying toggle link to clamp around a steel section. A failure discovered during the early stages of testing revealed a sheared stationary pin on one of the toggle links. This was attributed to the fact that the toggle was permitted to go beyond center thereby causing the pin to see excessively high forces. This problem was quickly remedied by installing an adjustable stop on the toggle to prevent it from going beyond center. No further failures were reported with this mechanism.

Another problem which occurred frequently during the early stages of testing was the shearing off of the welded "C" shaped grip at the ends of the centralizers. This problem was due mostly to sequencing difficulties. If the boom was traveling at too great a speed, the centralizers would not open fast enough to allow clear passage of the drill motor. When this happened, the tips of the centralizers would break off having seen the full thrust of the boom. The centralizer arms themselves, however, did not bend from this force. The grips were welded back on and put back into service. Since this collision occurred a few times afterward without the grip breaking, the early failures could possibly have been attributed to poor welds. As a future solution to this problem, the grip and the arm could be formed as a single piece eliminating the weld, or a replaceable bolt-on type grip could be provided at the end of the arm.

PART DESCRIPTION: Centralizers

Failure Description	Actual Holes	Sim Holes	Actual Cycles	Sim Cycles	Δ AC	Δ SC	Σ Cycles Bet Fail	TBF 6' Hole
Boom Hit Centralizer (left) "C" on Up Path. Also Found Sheared Stationary Pin on Toggle Link	5	0	0	0	--	--	20	5
Left Centralizer "C" Part Sheared Off, Rewelded	442	262	1768	1572	1748	1572	3320	830
Right Centralizer "C" Part Sheared Off, Rewelded	675	566	2700	3396	932	1824	2756	689
Adjusted Cylinder Cushion On Centralizer to Open Faster	1001	932						
Right Centralizer "C" Part Sheared Off, Rewelded	1458	1988	5832	11928	3132	8532	11664	2916
Test Completed	1500	4500	6000	27000	168	15072	15240	3810

Figure 36. Centralizer Performance

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4.5 Lower Clamp Assembly

The lower clamp assembly is located on the drill box adjacent to the drill chuck. The primary function of the lower clamp is to pull the drill steels out of the drilled hole and to separate the last drill steel from the remaining drill steels in the hole. The secondary function of the clamp is the removal of the bolt torquing adapter from the roof.

The lower clamp was one area of the Flex Drill concept which was redesigned during the reliability testing to improve the systems reliability. The performance record of the lower clamp is shown in Figure 37.

a. Original Design Version

The lower clamp mechanism was originally designed using a rotary hydraulic actuator and a U-shaped bracket. The actuator would pivot the clamp allowing it to engage the undercut on the drill steel coupling. The actuator had internal stops to position the clamp in its actuated and stored position.

The major problem encountered with this design in testing was the shearing of the woodruff key on the actuator shaft. This would happen when the bracket was struck or forced to stop before the internal stops on the actuator were reached. Stress calculations revealed that the key could not withstand the full force generated by the actuator under a maximum pressure condition.

Another problem occurred with the U-shaped bracket breaking in half across the cut out portion which engaged the coupling. It was not strong enough for the loads encountered.

PART DESCRIPTION: Lower Clamp

Failure Description	Actual Holes	Sim Holes	Actual Cycles	Sim Cycles	Δ AC	Δ SC	Σ Cycles Bet Fail	TBF 6' Hole
Lower Clamp Bent	260	44	1040	264	0	0	1304	326
Replaced Lower Clamp, Old One Worn too Badly for Repair	270	80	1080	480	40	216	256	64
Sheared Key on Actuator	381	262	1524	1572	444	1092	1536	384
Lower Clamp Broke in Half, Rewelded	479	262	1916	1572	392	0	392	98
Lower Clamp Worked Loose, Welded to Shaft	511	262	2044	1572	128	0	128	32
Clamp Broke in Half	537	310	2148	1860	104	288	392	98
Clamp Broke in Half	1001	1249	4004	7494	1856	5634	7490	1873
Installed New Clamp	1500	3223	6000	19338	1996	11844	13840	3460
Test Completed	1500	4500	6000	2700	0	7662	7662	1915.5

Figure 37. Lower Clamp Performance

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b. Redesigned Version

The clamp was redesigned with two major criteria kept in mind. One that the shaft key should be strong enough to withstand the force generated by the maximum relief pressures, and two that the clamp be enlarged to resist the actual loads encountered.

The rotary actuator was replaced with a gerotor type hydraulic motor with no internal stops. The mounting bracket was redesigned so that it would provide physical stops for the clamp in both the stored and actuated positions. The U-shaped bracket was made thicker and wider, and hardened along the wear surfaces to help improve its life. The motor shaft was modified to accept a larger key dictated by stress calculations.

The redesigned clamp was installed on the machine for the last 1277 holes of the simulated drilling. This allowed it to accumulate a total of 7662 machine cycles before testing ended.

4.6 Carousel Assembly

The components of the carousel assembly which failed or malfunctioned during the testing were basically the ratchet-pawl assembly and the detent bearing assembly. The performance record is shown in Figure 38.

- a. Ratchet-Pawl Mechanism - Indexing of the drill steels is accomplished by a ratchet-pawl mechanism linked to a clutch-type tie rod anchored to the front chassis. As the carousel moves, the tie rod forces the pawl to engage the ratchet. Thus, the internal shaft and plates assembly which carries the drill steels, is rotated relative to the carousel body as the carousel moves through its swing. The swing of the carousel, the length of the tie rod, and the position of the pawl and ratchet are all designed and adjusted to insure that the drill steels are in the proper positions at each end of the carousel swing. One of the biggest problems during the testing was keeping all of the components within adjustment. Backlash of any type could not be tolerated. The set screw which locked the key that fastened the ratchet to the shaft would continually work loose. This would cause the key and keyway to move and increase the clearances. The additional backlash would cause the mechanism to go out of adjustment and malfunction.

As a means of locking the ratchet to the shaft in a much more positive way than a key and setscrew, the ratchet was welded to the shaft. This method would normally not be practical since it meant that disassembling the unit was much more difficult, however, it did eliminate the problem of continual adjustments. Another means of attachment other than a key would have to be considered.

PART DESCRIPTION: Carousel

Failure Description	Actual Holes	Sim. Holes	Actual Cycles	Sim Cycles	ΔAC	ΔSC	Σ Cycles Bet Fail	TBF 6' Hole
Replaced Detent Bearing	5	0	20	0	—	—	20	5
Boom Hit Carousel Bent Lower Mount and Sheared Mounting Screws	17	0	68	0	48	0	48	12
Carousel Would Not Ratchet Found Loose Set Screw	70	44	280	264	212	264	476	119
Replaced Detent Bearing	116	44	464	264	184	0	184	46
Tighten Set Screw in Ratchet	537	264	2148	1584	1684	1320	3004	751
Replaced Detent Bearing (Used Solid Bearing)	921	717	3684	4302	1536	2718	4254	1064
Ratchet Cable Rod Broke	1001	975	4004	5850	320	1548	1868	467
Set Screw Loose in Ratchet, Could Not Repair, Welded	1001	1035	4004	6210	0	360	360	90
Lower Torsion Spring on Gate Broke, Replaced All of Them	1001	1189	4004	7134	0	924	924	231
Replaced Shaft and Plates Assembly	1371	1988	5484	11928	1480	4794	6274	1569
Test Completed	1500	4500	6000	2700	516	15072	15588	3897
Figure 38. Carousel Performance								

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- b. Detent Bearing Assembly - The detent bearing assembly is located on the detent bracket which is housed within the carousel beneath the ratchet and pawl mechanism. Its function is to lock the carousel into its new position each time it is indexed whether loading or unloading. This is accomplished by having a spring loaded lever pushing a single row radial ball bearing along a cammed surface, which has detents or locking positions corresponding to each position of the carousel. This bearing was replaced three times during the testing. It could not withstand the high forces that were generated whenever there was a misalignment or malfunction of the carousel. The bearing was finally replaced the fourth time with a solid roller bearing. This bearing finished the remaining drilling without a failure accumulating 579 actual holes drilled and 3783 simulated holes or 25014 machine cycles.
- c. Gate Mechanism - The new gate mechanism worked exceptionally well during the realibility tests. Only once did the mechanism require any maintenance. The lower torsion spring broke causing the gate to lose some of its preload force. This loss of preload would occasionally allow a drill steel not to be fully seated in the carousel. Since the lower spring had to be replaced, it was decided that it would be just as easy to replace the remaining three springs while the unit was disassembled. The gate was rebuilt using the new springs and installed back into the chassis. The cable did require a minor adjustment at that point. The gate worked successfully beyond that point.
- d. Shaft and Plates Assembly - The shaft and plates assembly sorts and carries the drill steels into position within the carousel. No failures occurred with this portion of the carousel during testing, however, some excessive wear was noticed along the cut out areas of the plate which make contact with the drill steels. Although this wear did not cause the carousel to malfunction at any particular hole, a new shaft and plates assembly was installed about midway through the testing. The second shaft and plates assembly was nitrided for improved wear resistance.

The excessive wear problem can be controlled by selecting the proper material and heat treatment specification for the cutout area of the plates. A method was designed which allows for the cutout section of the plate to be replaceable item of the shaft and plates assembly, however, this design change was not implemented because the testing was near completion, and a significant number of cycles could not have been accumulated on the new design for evaluation.

4.7 Hydraulics

Bit Condition Survey

Broken/Cracked Carbide	Useful	Undamaged
1 Yellow	21 White	36 White
13 White	1 Blue	1 Brown
4 Blue	1 Brown	3 Blue

(Cannot be Resharpended) (Can be Resharpended) (No Drilling Damage
Gage Corner Dull or Gage Corner Intact
Edge Dull.) and no Chips or Fla

Figure 40. Bit Condition Survey

The average performance of this group of bits, 5.8 holes per bit or 29 feet per bit is conservative because an operational decision was made to run no bits beyond the sixth hole. The incidence of drill box stalling and stuck steels rose drastically on the few bits allowed to go beyond six holes. Penetration rate also suffered in the opinion of the drill operators. In the interest of timely test completion, all bits were taken out of service after six holes.

As Figure 40 shows, 36 of the bits reported could evidently have gone beyond the 6-hole level easily, as careful inspection shows no appreciable loss of cutting edge. On the other hand, 34 of the bits which reached the 6-hole level were damaged or in need of sharpening. Thus, the rise in failures and loss of penetration which led to the decision to discard all bits after 6 holes is explained. All bits used in the bit condition survey were commercial production FSP 1-3/8-inch roof bits purchased from Kennametal, Inc., Bedford, Pennsylvania. No endorsement is expressed or implied for these bits by IRRI or the Department of Energy.

It is doubtful that the results obtained in this test should be directly compared to those reported in June of 1977 under the Flex Drill Program. The relative difficulty of drilling underground (strata hardness and abrasiveness) has not been established with respect to the Indiana limestone used for this test. Nonetheless, the 29-foot average developed here is significantly

greater than the 8 to 12-foot average maintained by manual bolting machines in the mine where Flex Drill demonstrated a 42 to 84-foot average.

b. Couplings

In drilling 6000 hole cycles, a total of 33,000 steel changes (in and out - one machine cycle), 49 couplings were put into service. Of those, 6 were similar to that shown in Figure 26. This is the coupling used in the underground test. Thirty-five couplings of the style shown in Figure 27 were consumed. Eight of the current coupler, shown in Figure 28, were used.

The major reason for coupling replacement was failure to couple. The 35 couplings of the style of Figure 27 were generally discarded for this reason. Two of this type broke in the spring groove and two were lost in a clogged hole. The other 31 suffered rounding off of the hex lead section and were replaced. (See the discussion in Section II-F, for a further description.) The 6 couplings similar to those used in the underground test were not worn out. They were used to keep testing underway until the couplings of the current type arrived. The current design couplings were put into service when 1146 actual holes had been drilled and 1497 simulated holes had been run. Since that point, eight couplings have been employed. One broke through the spring groove. One was replaced because it would not mate with all the drill steels in the string - the hex was slightly oversize. The other six were in use at the close of testing. These eight couplings served for 354 actual holes and 3003 simulated holes, or 19434 steel changes. It cannot be said with certainty that any one coupling experienced all 3357 holes, but it is highly likely. Four couplings are used to drill an actual hole and all six in the carousel are used during a simulation. No effort was made however to keep each coupling in the same spot in the carousel all the time. It is possible that the couplings were rotated in such a manner that no one coupling was used in every hole. It is correct to say that the average life of the current coupling is at least 19434 steel changes divided by 8 couplings, or 2429 steel changes, based upon the above information. For the Flexible

Drill, this is a good measure of coupling life, for the coupling operation of each steel change creates more stress and wear on the coupling itself than drilling does.

c. Drill Steel

Experience with the commercial drill steel used in the Flexible Drill is good. At no time was a drill steel replaced because of a failure due to wear. One steel was bent while using the simulator, but this was due entirely to operator abuse and misalignment of the trolley mechanism while the simulator was being debugged. The short lengths of the steels undoubtedly contribute to their longevity.

Fifteen-degree internal bevels are added to the ends of each drill steel to assist in coupling. These are susceptible to wear over time, but the time to wear out is beyond the length of this test. Similarly, the internal hex of the tubing itself should wear under the drill torque. Again, the life of this surface is beyond this test.

d. Double Socket Chuck

The double socket chuck originally designed for Flex Drill finally proved its utility in the course of the test program. The addition of 1-1/8 inch diameter rings on the couplings to bridge the area of the square socket and seal the drill string to the chuck made the design work. This success is very significant to future rod changing drills - most notably the mechanical anchor bolter module, DOE Contract Number ET-77-C-01-9113.

e. Drill Torque and Thrust Data

Drill torque and boom thrust readings were made during the reliability testing. The tests were conducted by installing a series of strain gages on an actual drill steel. The gages were connected to a wrap around cable approximately 50

feet long. The far end of the cable was connected to a multi-track recorder and a storage oscilloscope. The two signals were stored on magnetic tape from which a print-out was made and analyzed. The tests consisted of allowing the cable to wrap around the steel as the hole was being drilled. Approximately 10" of constant penetration was achieved while wrapping the cable.

During one rock sample, 3860 lbs. of thrust and 180 ft-lbs of torque were observed. Another rock sample showed peaks of 4885 lbs. of thrust and 221 ft-lbs. of torque. Examples of the recorder print-out are provided in Figures 41 and 42.

In addition to torque and thrust readings for actual rock drilling, data was collected while operating on the simulator. The same method employed for rock drilling was used on the simulator. The print-outs showed that 3863 lbs. of thrust and 200 ft-lbs. of torque were developed while simulated drilling was being performed. (These settings are of course variable by adjusting the relief valves of the simulator.)

Thrust loading on the simulator was also measured under static conditions. This was done to measure the thrust output of the pantograph type linkage used to raise and lower the drill head. The drill head was placed at various heights off the ground in a dead locked position. The thrust exerted through the drill steel was recorded. As anticipated, varying thrust values were obtained for varying heights of the drill box. Refer to Figure 43. The curve that was drawn was obtained from a linear regression curve fit analysis.

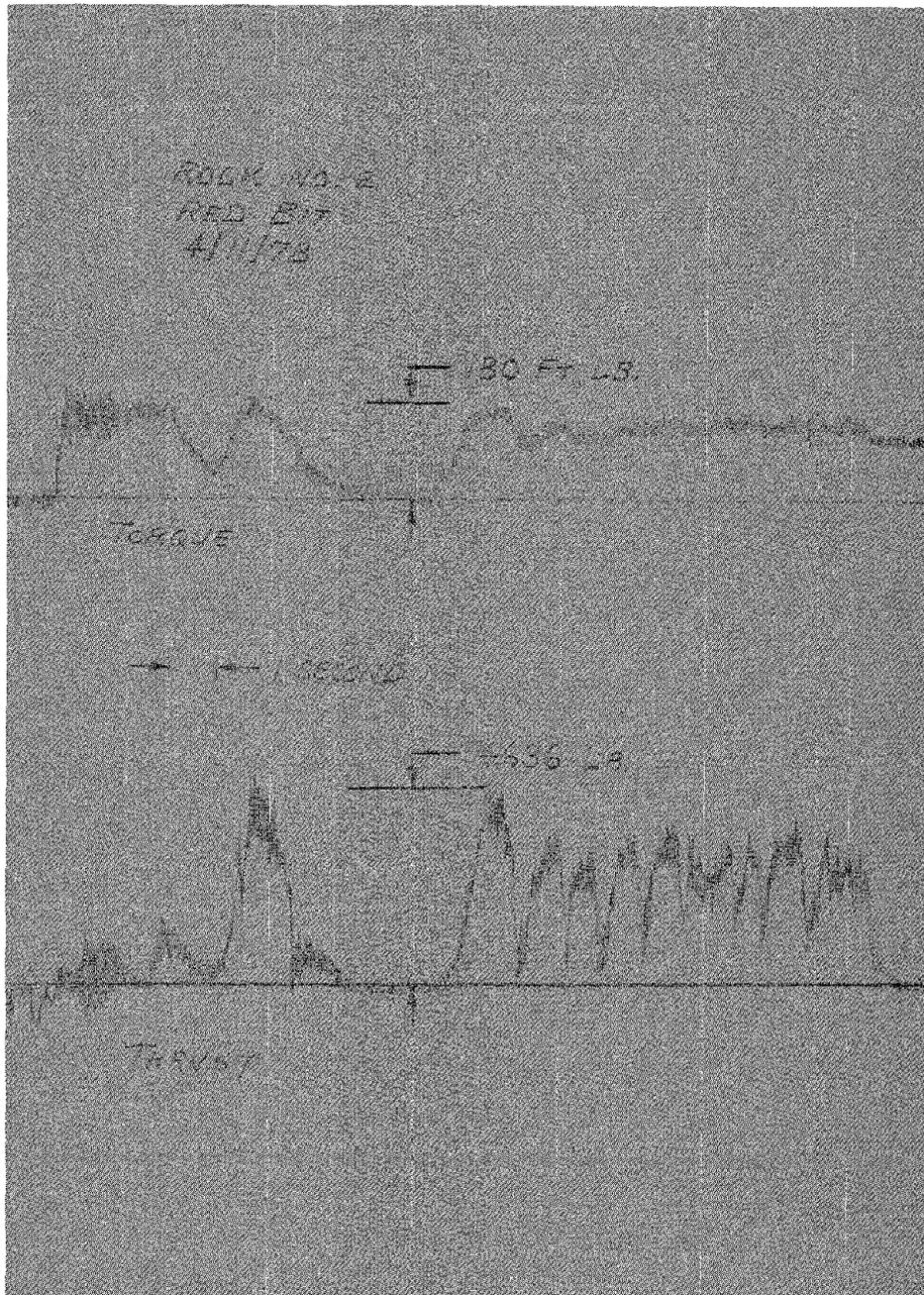


Figure 41. Drill torque and thrust data.

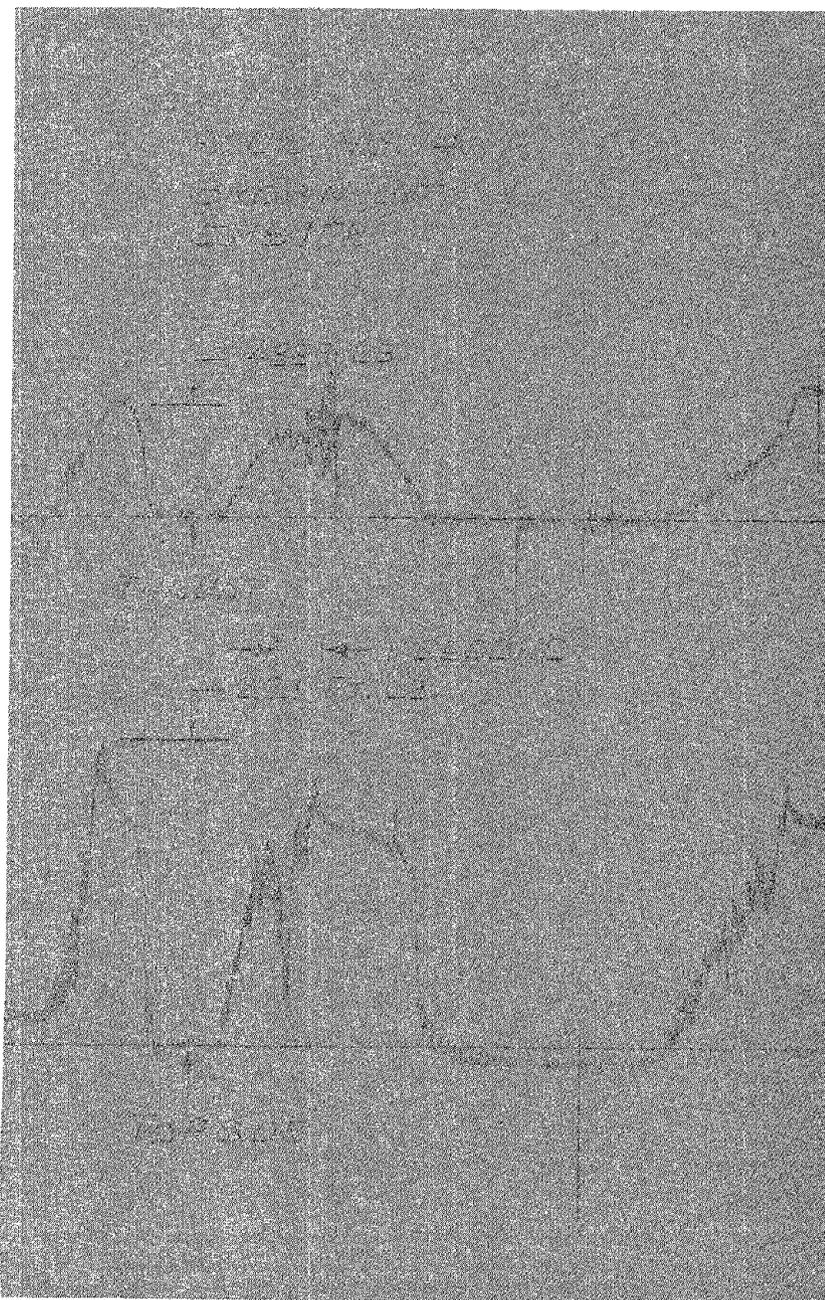


Figure 42. Drill torque and thrust data.

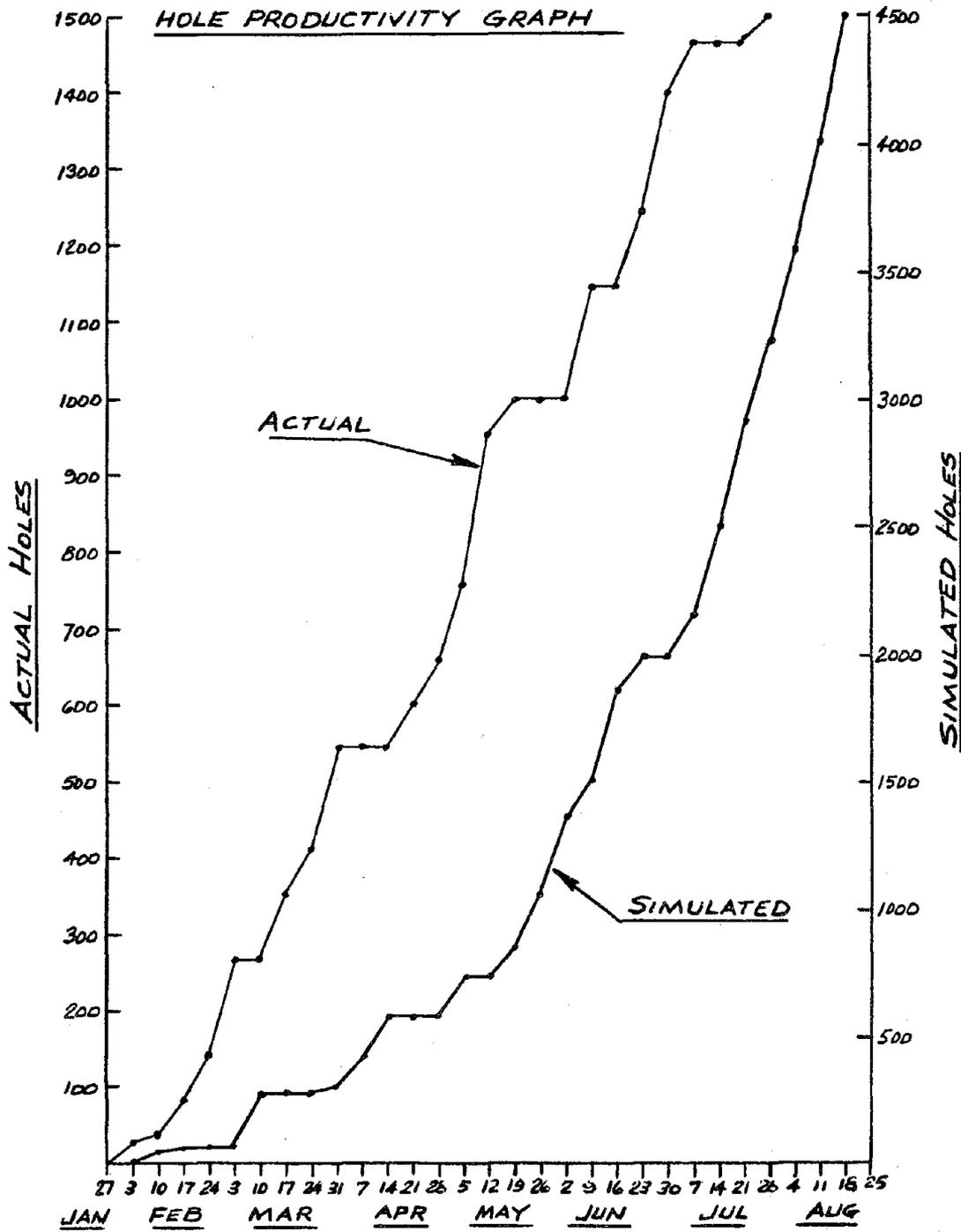
operating the mechanisms faster than the air logic can control them. These findings are being incorporated in another ongoing DOE program to develop a fully automatic roof bolter module, so Flex Drill's experience will not be lost, even if the machine is not commercialized.

Third, the machine in its current form offers few safety advantages to its operator. True, the operator does not now handle turning drill rods, but he is still required at the drill, under dangerous top. It remains for later programs to build upon the success of Flex Drill to produce a complete remotely-operated roof bolter, where the operator is no longer exposed to a dangerous roof condition. Flex Drill has demonstrated that such a machine is possible utilizing the rod changing concept.

Testing Productivity

In undertaking testing programs, it is always difficult to estimate the amount of time required to conduct a test. A productivity graph has been prepared for the recently completed test program which may aid in planning further drilling programs utilizing the rock stands and the drilling simulator. It is included on the following page.

FLEX DRILL



Hydraulic System Performance

The next several pages comprise the correspondence received from Oklahoma State University regarding the data read from two statistical analog monitoring units installed on Flex Drill. Inconsistencies in the data cannot be resolved with the size of the sample available. The information is presented without comment, beyond the marginal notes on the pages.