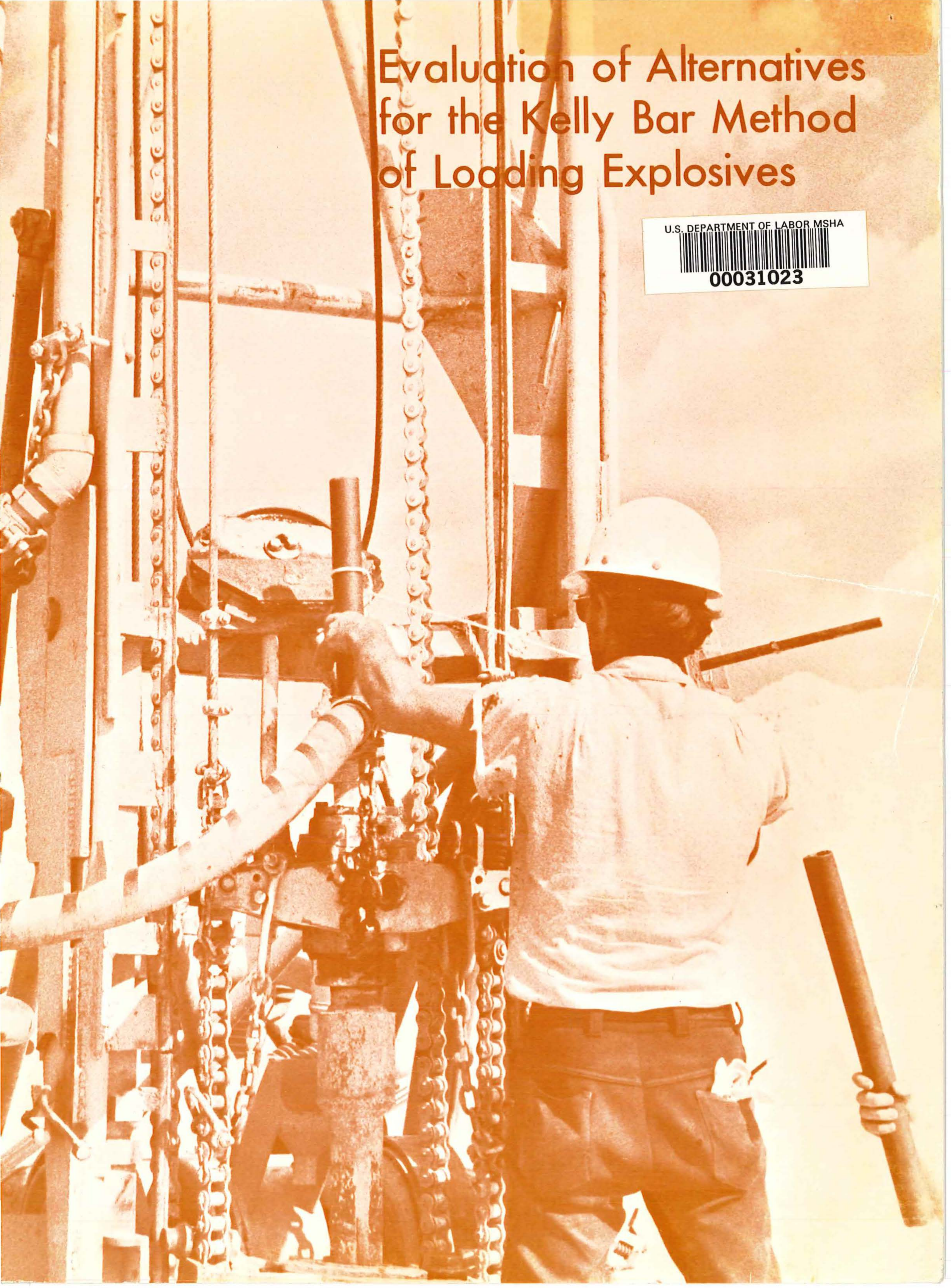
Evaluation of Alternatives for the Kelly Bar Method of Loading Explosives



OFR 1978-22

EVALUATION OF ALTERNATIVES FOR THE KELLY BAR METHOD OF LOADING EXPLOSIVES

Prepared for

UNITED STATES DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

by

ENGINEERING CONTRACTORS ASSOCIATION OF SOUTH FLORIDA, INC.
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FINAL REPORT

CONTRACT NO. J 0177005
EVALUATION OF ALTERNATIVES FOR KELLY BAR METHOD

(Date Submitted)
MARCH 1978

50272-101

REPORT DOCUMENTATION PAGE

REPORT DOCUMENTATION PAGE	1. REPORT NO.	2.	3. Recipient's Accession No.

4. Title and Subtitle
EVALUATION OF ALTERNATIVES FOR THE KELLY BAR METHOD OF LOADING EXPLOSIVES.

5. Report Date MARCH 1978 DATE PUBLISHED

6.

7. Author(s) DONELSON A. ROBERTSON
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8. Performing Organization Rept. No.

9. Performing Organization Name and Address
ENGINEERING CONTRACTORS ASSOCIATION OF SOUTH FLORIDA, INC.
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10. Project/Task/Work Unit No. N/A

11. Contract(C) or Grant(G) No.
(C) J-0177005
(G)

12. Sponsoring Organization Name and Address
U.S. BUREAU OF MINES
DEPARTMENT OF INTERIOR
2401 E STREET NW
WASHINGTON, D.C. 20241

13. Type of Report & Period Covered
FINAL
10/26/76 - 3/31/78

14.

15. Supplementary Notes
PREPARED BY PHILIP R. BERGER & ASSOCIATES AS SUBCONTRACTOR TO ENGINEERING CONTRACTORS ASSOCIATION OF SOUTH FLORIDA, INC.

16. Abstract (Limit: 200 words) Includes a review of the state-of-the-art of explosives loading in the crushed stone and cement industries of South Florida and the development of a field test program for alternatives. Emphasis is on the loading of small diameter (2 1/4" or less) explosives. Test results are evaluated for operational and economic feasibility. Details are described of an explosives loading method which involves placing cardboard tubes with external couplings through the hollow drill steel and then loading explosives into the tube casings after the drill rig has moved off. This method eliminates the hazards peculiar to loading explosives through the kelly bar.

The alternative method can be performed by existing drill rigs with or without minor modification. With modification, improvement in yield per blasthole and/or primary crushing rate provides opportunity to offset some or all of the additional cost.

17. Document Analysis a. Descriptors Explosives, loading, kelly, drilling, blasting, blastholes, casing.

b. Identifiers/Open-Ended Terms

c. COSATI Field/Group

18. Availability Statement	19. Security Class (This Report) UNCLASSIFIED	21. No. of Pages
	20. Security Class (This Page)	22. Price

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

FOREWORD

This report was prepared by Philip R. Berger & Associates as subcontractor to Engineering Contractors Association, prime contractor under U.S.B.M. Contract No. J 0177005. The contract is authorized by law and executed pursuant to Public Law 89-577 (80 Stat. 772) and funded pursuant to Department of Interior and Related Agencies Appropriation Act, 1977 (P.L. 94-373, 90 Stat. 1043), dated July 31, 1976. It was administered under the technical direction of the Pittsburgh Mining and Safety Research Center with Mr. Richard W. Watson as the Technical Project Officer. Mr. P.B. Schultz was the contract administrator for the Bureau of Mines, and Mr. Joe Fiedorek provided liaison with MESA.

This report is a summary of work recently completed as part of this contract during the period from October 20, 1976 to January 20, 1978. This report was submitted by the author on March 31, 1978.

Reference to specific brands, equipment, or trade names in this report is made to facilitate understanding and does not imply endorsement by the Bureau of Mines.

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CHAPTER 1

EXECUTIVE SUMMARY

EXECUTIVE SUMMARY

Loading explosives through the kelly bar is standard procedure for placing the charge in blastholes in South Florida. This is because the blastholes do not remain open after the drill steel is removed. Loose sand and rock collapse into the borehole and prevent open hole loading.

Several accidents involving injuries or fatalities have occurred and interim safety rules were adopted by industry and MESA representatives until alternative methods of loading explosives could be explored. This cost sharing project was initiated to search out and evaluate alternatives to loading explosives through the kelly bar.

STATE OF THE ART

In order to document what was being done, what had been done, and what had been tried, an intensive review of the state-of-the-art was made.

Every firm that uses the kelly bar method in mining in South Florida was contacted. Visits to the operations were made in most cases. Non-mining practices were included to a lesser degree but were necessary because contract drillers frequently work in both endeavors. Suppliers and service industries were also interviewed.

Detailed information on number and type of drill rigs, kelly bar manufacture, blast design, loading methods, drilling rates, and previous experimentation was collected.

From these data, the specific hazards that needed to be eliminated were defined. This background information also indicated that some parties had developed feasible alternatives for the large diameter (5" I.D.) kelly bars using paperboard tubes and bulk slurry. This system was not acceptable at all locations though, because ground vibration problems with neighbors prevented the use of the large charge weight.

Other limitations to the kelly bar method are addressed which affect management control of costs and scheduling, and place limitations on capital expenditure decisions.

UNIT TESTS OF ALTERNATIVES

Test proposals for alternatives were solicited by mail from 255 parties with possible interest. An informational meeting was held to give additional background and this was attended by 105 persons.

Four proposals were received and two were accepted. These offered to place paperboard tubes through the small diameter (2 1/2" I.D.) kelly bars into the blastholes. Bulk slurry was then to be pumped into the tube casings after the drill rig had completed drilling and moved off the site. These tests were contracted for as units in which the contractor provided the drilling, explosives, and labor in a package, and are referred to in the report as unit tests. These were conducted in February, 1977, and indicated that the pumped slurry technique is not feasible as a small diameter alternative.

Other unit tests were conducted under more adverse geologic conditions which involved loading 2" diameter packaged products into 2.493" ID resin-treated paperboard tubes. This required a slightly larger kelly. These tests indicated that the holes could be stabilized, the yields increased and that time required for tube placement and loading was reasonable. It was decided to concentrate on this approach as the most likely to provide an acceptable alternative for small diameter kelly bar rigs.

A total of 17 tests at three different sites were conducted in this unit test part of the project.

3" I.D. KELLY BAR PROGRAM

The resin-treated paperboard tube was selected because of its previous performance, availability, price, storage characteristics, easy handling and placement, stability in water filled holes, and the lack of contamination in finished rock products.

A test plan was developed with the concept of placing these tubes with a 2.493" ID in the blastholes through a 3" ID kelly bar. The kelly bar would be custom-made, other necessary rig conversions made, and testing would determine if holes could be drilled with this size bar without using an inner core breaker bit. Different packaged products would be loaded into the tube casings after drilling was completed.

The test program emphasized broad geographic coverage in order to test the capability of the tube casings to stabilize the blastholes under a wide variety of geologic conditions.

Optimization of blast design would be attempted at each location insofar as permitted by the limited number of shots and the operator's willingness to permit experimentation.

A total of 76 blasts were made at twelve different locations in this part of the project. Over 12 miles of hole were drilled and stabilized with the tube casings, and 87,514 pounds of explosives were used.

Rock jams in the bit occurred 10% of the time but can be reduced by driller's experience at each site.

Drilling and loading efficiency with this system was about 64% as efficient as drilling and loading through a 2 1/2" ID kelly bar, including the time spent in clearing rock jammed bits and redrilling holes. Tube casings can be placed in holes drilled with either air or water flush but water is preferable.

Tube casings placed in test holes to test their endurance were still open to full depth with full diameter nine months later.

Tube casings placed in close proximity to side of blasts have a 70% chance of survival if at a distance of 20 feet. Survival probability is much poorer for tube casings placed behind the shot.

Dynamite has a wide tolerance for poor design and is not as susecptible to pre-compression as slurry. It performed well in most tests with good results unless initiated from the top only.

Packaged slurries are safer to store, handle and use, but they require more sophistication and experimentation to develop a successful design. Once the successful design is established, the packaged slurries offer a better opportunity to increase yields and lower costs because they generally contain about 25% or more weight energy per pound than the dynamites commonly used in South Florida, based on manufacturers data.

The need for buffers (barriers of shot rock between the blast and open water) is probably related to the unequal distribution of explosives when loaded through the kelly bar. Tests at eight different locations without buffers indicated that when tube casings are used there is no excessive throw into the water.

POST SHOT ANALYSIS

Digging and processing results at several locations were analyzed. Observation of digging provides little usable information other than the operator's subjective opinion. Too many variables are involved for reliable treatment.

Processing data on the other hand can be handled with relatively sophisticated statistical methods yielding quantities which serve as inputs to economic analysis.

Results indicated that bulk slurry in small diameter tubes causes a serious decline in primary crushing rate.

Dynamite or packaged slurry in tube casings, using the same load factor as with loading through the kelly bar, improved the hourly rate of production significantly where the feed hopper was fed with a front end loader. Although yield per blasthole remained constant, the increased production rate lowered overall production cost. This offers an attractive economic incentive at those plants where the primary crusher is currently being operated at full capacity.

Alternatively, yields can be increased which causes the total cost per hole to be divided by a larger number of cubic yards. Increases in yield ranging upwards to 29% were made with no adverse effect on production rate and with an indication that uniformity of plant feed was improved. Depending on the degree of optimization already being practiced, operators can probably increase yields from 15% to 50% with the use of tube casings and careful experimentation.

ECONOMIC ANALYSIS

The additional cost of placing the tube casings through the kelly and then loading the holes after the drill rig has moved off depends on:

(1) the efficiency of the operators blasting design when loading through the kelly, and

(2) his efforts to optimize his blasting design when utilizing the tube casings.

Seven cases are analyzed to provide specific economic comparisons for the following:

(1) Bulk slurry in small diameter casings versus 2" diameter explosive cartridges loaded through the kelly bar.

(2) 2" diameter packaged explosive products loaded into 2.493" tubes compared to loading 2" dynamite through the kelly bar. Load factor decreased. Same production rate.

(3) Same as 2, except that load factor remains constant, but a minimum increase in production rate occurs.

(4) Same as 3, except that increase in production rate is the maximum indicated by statistical analysis.

(5) 2" packaged explosive products loaded into 2.493" tube casings with a 32% reduction in load factor compared to 3" diameter dynamite loaded through the kelly.

(6) 1 3/4" diameter dynamite loaded into 2 1/4" tube casings compared to 2" diameter dynamite loaded through the kelly bar.

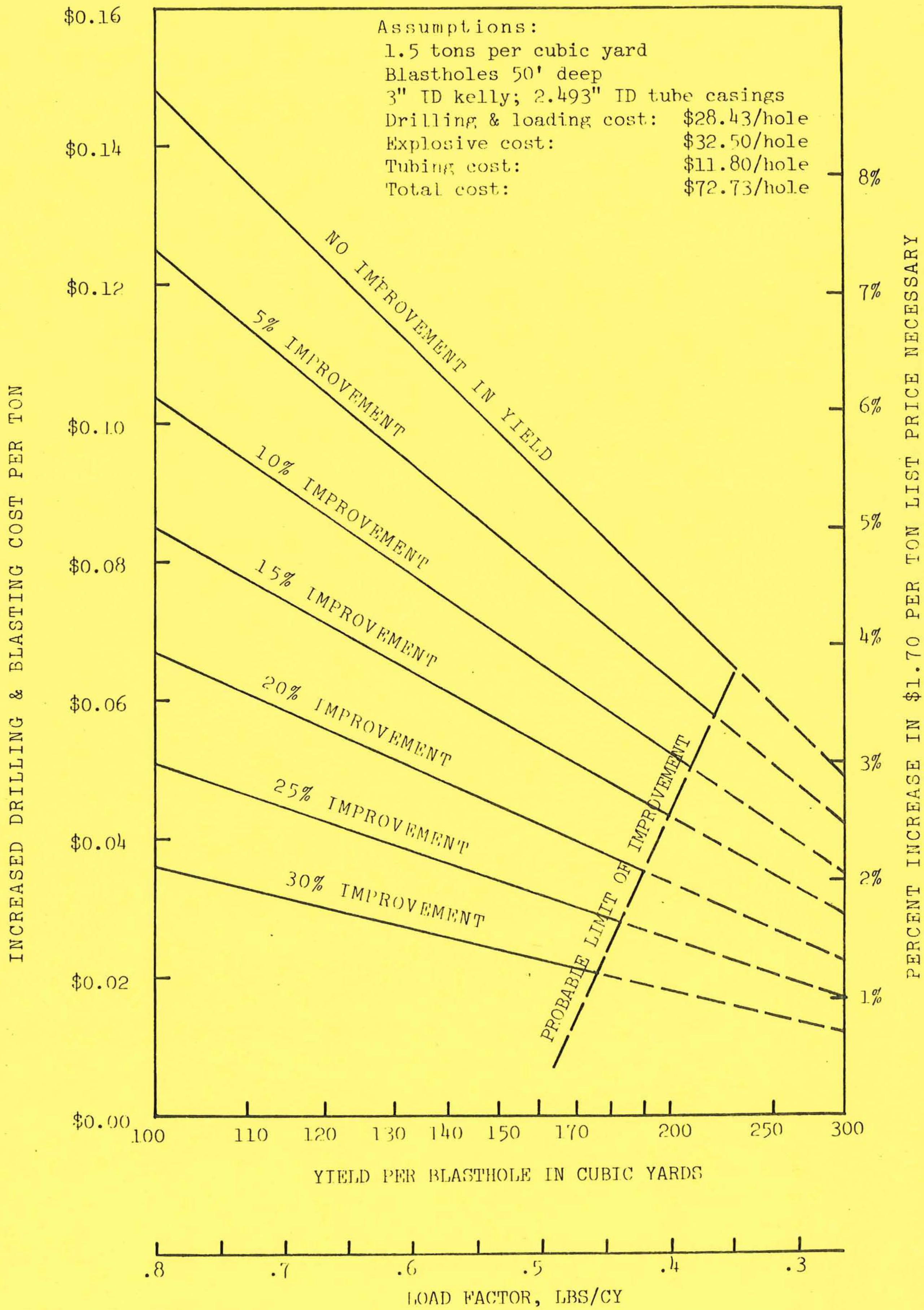
(7) Bulk slurry loaded into 4 1/2" diameter tube casings compared to 4" diameter dynamite loaded through the kelly bar.

These comparisons are site specific and are based on field trial experience and/or data from normal operations.

Graphs are presented which permit an operator to relate his present cost picture to that with the tube casings. These graphs are based on average field experience during the project and are not site specific. A method is also presented to determine the cost advantage of increases in production rate.

Except for increase in processing rate, these concepts are all incorporated in Figure 1. The operator can enter the base of the graph with his current yield in cubic yards per blasthole. With no optimization, his added cost is represented by the top line. For varying percentages of optimization, other values of added cost are given.

FIGURE 1



INCREASED DRILLING AND BLASTING COST FOR TUBE CASING METHOD WITH VARIOUS IMPROVEMENTS IN YIELD

CONCLUSIONS AND RECOMMENDATIONS

Expendable paperboard tubes will stabilize the blastholes in South Florida for long periods of time. Hot wax dipped tubes work well in large diameter operations (5" ID kelly bars) and phenoloc-resin treated tubes work well in the small diameter (2 1/2" and 3" ID kelly bars) operations.

Placing the tubes through the kelly and then loading them at some later time as a separate operation eliminates the hazards that are peculiar to loading explosives through the kelly.

Bulk slurry or packaged explosives can be loaded into the big tubes and with a moderate increase in yield a cost savings can result. Packaged products can be loaded into the small tubes but it is more difficult to effect a reduction of present cost. Bulk slurry in small diameter tubes does not appear to be operationally feasible.

Additional cost when using small diameter tubes will depend on the efficiency of the operator's present blasting practices and his effort to optimize his design when tubes are used.

For small diameter operations, converting the drill rig to accept a 3" ID kelly bar appears to be the best alternative because it provides more opportunity for increased yield. Conversion is best accomplished when the kelly bar needs to be replaced.

If conversion is not possible, or not desired, high density 1 3/4" diameter gelatin dynamite can be loaded into tube casings which can be placed through a standard 2 1/2" ID kelly bar.

Initiation only from the top of the explosive column yields consistently poor results. This does not apply to detonating cord downlines.

The tube casings permit the use of a higher degree of sophistication in blasting than previously possible in South Florida. This offers an opportunity for explosives manufacturers to utilize the sales potential of their expertise in advanced blasting methods.

Additional research is needed for a better understanding of the causes of desensitization due to precompression of slurry explosives in South Florida.

CHAPTER 2

INTRODUCTION AND ACKNOWLEDGEMENTS

INTRODUCTION

Open pit mining of Pleistocene limestone is conducted in South Florida (Dade, Broward, Collier, Monroe and Lee Counties) to obtain crushed stone aggregate, base course material, and cement plant feed. It is essential to the construction industry of the area, and a considerable amount of these products are exported to aggregate poor areas up the east coast. Other rock excavations involving blasting are performed in conjunction with real estate development, drainage canals, road building, trenching for sewer and water facilities, and harbor channel dredging. Inasmuch as the government share of funding for this project is from the U.S. Bureau of Mines, this report is concerned primarily with the loading and use of explosives in open pit mining applications. Nevertheless, a considerable part of the technology covered in the report is applicable to the other areas of endeavor.

Unusual methods are utilized in open pit mining in South Florida because of (1) the near surface water table, (2) the porous and permeable strata which preclude the possibility of economically pumping the water from the pit, and (3) the presence of unstable sand layers. In general, the surface muck and peat are stripped from above the rock and a pad of rock fill has to be emplaced to keep the operations above the free water surface. Blast holes are drilled to bottom with one continous length of hollow drill steel, and while the core-type bit is still on bottom, the explosives are loaded down the hole through the drill steel along with detonating cord. This is called the kelly bar method of loading. Multiple hole patterns are usually drilled and loaded. The rock is then blasted and recovered from the water filled pit with a dragline and placed on a surge pile parallel to the pit edge. Various loading, hauling and transporting systems are then used to process the rock into finished products.

There are a number of hazards specifically associated with the kelly bar method of loading and since 1964 seven South Florida mining accidents associated with this type of loading have been officially reported. These accidents involved seven fatalities, five injuries, and property loss.

Because of these accidents and the risk of future deaths and injuries resulting from continued use of this explosives loading method, a cost-sharing contract between the U.S. Bureau of Mines (80%) and the Engineering Contractors Association of South Florida, Inc. (20%) was negotiated in order to find safer, alternate methods.

This report covers the work accomplished and the results achieved under this contract, including a review of the present state-of-the-art.

ACKNOWLEDGEMENTS

Project guidance was provided by a Technical Committee composed of industry members and representatives of the Bureau of Mines and MESA. Mr. Richard W. Watson served as the Bureau of Mines representative and as Technical Project Officer. Mr. Joseph Fiedorek was the

MESA representative on the Committee. Mr. Jon Guerry Taylor of Capeletti Bros., Inc., was the initial chairman of the Technical Committee and served until changing employers and relocating to another area. He was succeeded by Mr. E.T. Foster, Jr. of Meekins, Inc. Other Committee members at various times included Messrs. Lawrence Capeletti (Capeletti Bros., Inc.), Steve Torcise (Florida Rock & Sand Company), and Ben L. Greene (Vulcan Materials Company). Committee alternates were Messrs. Arthur Larson (Sterling Crushed Stone Company), Ron Fish (Miramar Rock), Russell Cormican (L. W. Rozzo, Inc.), and William Vest (Vulcan Materials Company).

Many other quarry operators, equipment suppliers, explosives manufacturers, and personnel of Federal, State, county and municipal governments provided information used in this report. The following is a partial list of individuals or firms whose personnel provided valuable background for the state-of-the-art review or technical assistance in the design of field tests:

Vulcan Materials Company
Coral Aggregates Corporation
Rinker Southeastern Materials, Inc.
Pennsuco Cement & Aggregates, Inc.
Capeletti Bros., Inc.
Miami Crushed Rock
Rinker Portland Cement Corporation
Miramar Rock
L.W. Rozzo, Inc.
Redland Rock Company
Sterling Crushed Stone Co.
Troup Bros., Inc.
West Dade Rock & Fill Co.
Florida Rock & Sand Co.
Gator Rock
Warren Brothers Company
Harper Bros., Inc.
Florida Rock Products
Bergeron Land Development, Inc.
Meekins, Inc.
William May
Atlas Powder Company
Austin Powder Company
E.I. DuPont de Nemours & Company
Gulf Oil Chemicals Company
Hercules, Inc.
Ireco Chemicals
Trojan - U.S. Powder
Ensign-Bickford
Dynablast Corporation
Palmetto Drilling Company
Richardson Drilling & Blasting Company
Mowry Drilling Company
Ronnie's Welding & Machine Shop
Ingersoll-Rand
W.L. Sly Machinery Company
Pan American Equipment Co.
Division of Planning, Building & Zoning, Broward County

Department of Public Works, Metropolitan Dade County
U.S. Geological Survey
Alton Box Board Company
Sonoco Products Company

CHAPTER 3

STATE OF THE ART

STATE OF THE ART

Prior to developing and evaluating alternatives for the kelly bar method of loading, an in-depth investigation of current practice was made. Interviews with those persons in responsible charge of drilling and blasting sites were made. During the field visits additional information was obtained from drillers and their helpers.

Cycle time for various phases of the operation were obtained as time would permit. All of the quarry operators listed in the acknowledgements were included in this collection of background data. In addition, interviews and field visits were made with those contract drillers whose business is primarily in mining operations. Some drilling equipment suppliers, technical representatives of explosive manufacturers, and distributors of explosives were queried. Visits were made to the Dade and Broward county offices involved in blasting and quarry regulation, and to those federal offices where information on geology, hydrology and climate might be obtained. The Florida Department of Transportation's laboratory was visited to get specification information with particular reference to the limits for deleterious material in aggregates.

Development of the kelly bar method was necessitated by the need for a high yield drilling and blasting technique which would work under the hydrologic and geologic conditions found in South Florida. Because these conditions are controlling, it is essential to have a good working knowledge of them to understand the events leading up to the present state-of-the-art, and to evaluate the feasibility of possible alternatives.

GEOLOGY

The rock quarried in South Florida for aggregate, base course, and cement plant feed is from limestone of Pleistocene age. These limestones were deposited during the periods between the glacial advances called interglacials, when sea level was higher because of the melted ice. Several investigators (Sanford, 1909; Sellards, 1919; Cooke and others, 1943; Parker and Cooke, 1944; Richards, 1945; DuBar, 1958; Hoffmeister and Multer 1964; Brooks, 1968; Conklin, 1968) have addressed themselves to interpreting the stratigraphy of all or part of these sediments. The names Miami oolite, Key Largo limestone, Anastasia formation and Fort Thompson formation are well established in the literature and their surface distribution is shown on the Geologic Map of Florida, complied by R.O. Vernon and H.S. Puri, in 1964. Because of the flat topography, low elevation and the high water table, the above authors, of necessity, worked primarily with surface outcrops and particularly with a section exposed along the banks of the Caloosahatchee River, which flows westward from near Lake Okeechobee to the Gulf of Mexico.

Perkins studied 52 cores and the Caloosahatchee River outcrops for Shell Development Company in 1969. 41 of these cores were provided by the U.S. Army Corps of Engineers and 11 were taken by Shell.

The results were originally published as "Depositional Framework of Pleistocene Rocks in South Florida" in a confidential Shell report. This report has recently been released by Shell and is the basis of a publication of the Geological Society of America available to the general public (Geological Society of America, Memoir 147, Part II, 1977).

Perkins abandoned the older usage and established five limestone time-stratigraphic units separated by discontinuity surfaces. The limestone units were deposited during high sea level stands of interglacial periods. The discontinuities developed during periods of low sea level stands when the limestone was exposed subaerially. The units are identified as Q1, Q2, Q3, Q4 and Q5 in ascending order (Q for Quaternary). Perkins' correlation of these units with previous stratigraphic terminology is shown in Figure 2.

Perkins criteria for recognizing discontinuity surfaces included:

a) vadose sediment

b) land-plant root structures

c) laminated caliche-like crusts

d) corrosion and alteration zones (diagenetic soilstones)

e) soils and soil breccias

f) animal borings

g) solution surfaces

h) fresh water limestones

Usually a combination of two or more of these features is present.

Unit isopachs and unit lithofacies maps from Perkins' report are reproduced with permission from Geological Society of America, Dr. Ronald D. Perkins and Shell Development Company. These show the relationship of the test sites in South Florida to these units. (Figures 3,4,5,6,7 and 8)

The lithofacies maps utilize Dunham's (1962) classification of carbonate rocks. This system is based on depositional texture and is helpful in the interpretation of depositional environment. Carbonates that retain their depositional texture are subdivided into mudstone, wackestone, packstone, grainstone, and boundstone.

Mudstone is composed of lithified carbonate mud containing less than 10% grains of sand size or larger which are free-floating in the mud matrix (mud supported).

Wackestone contains more than 10% grains but the grains are still mud supported.

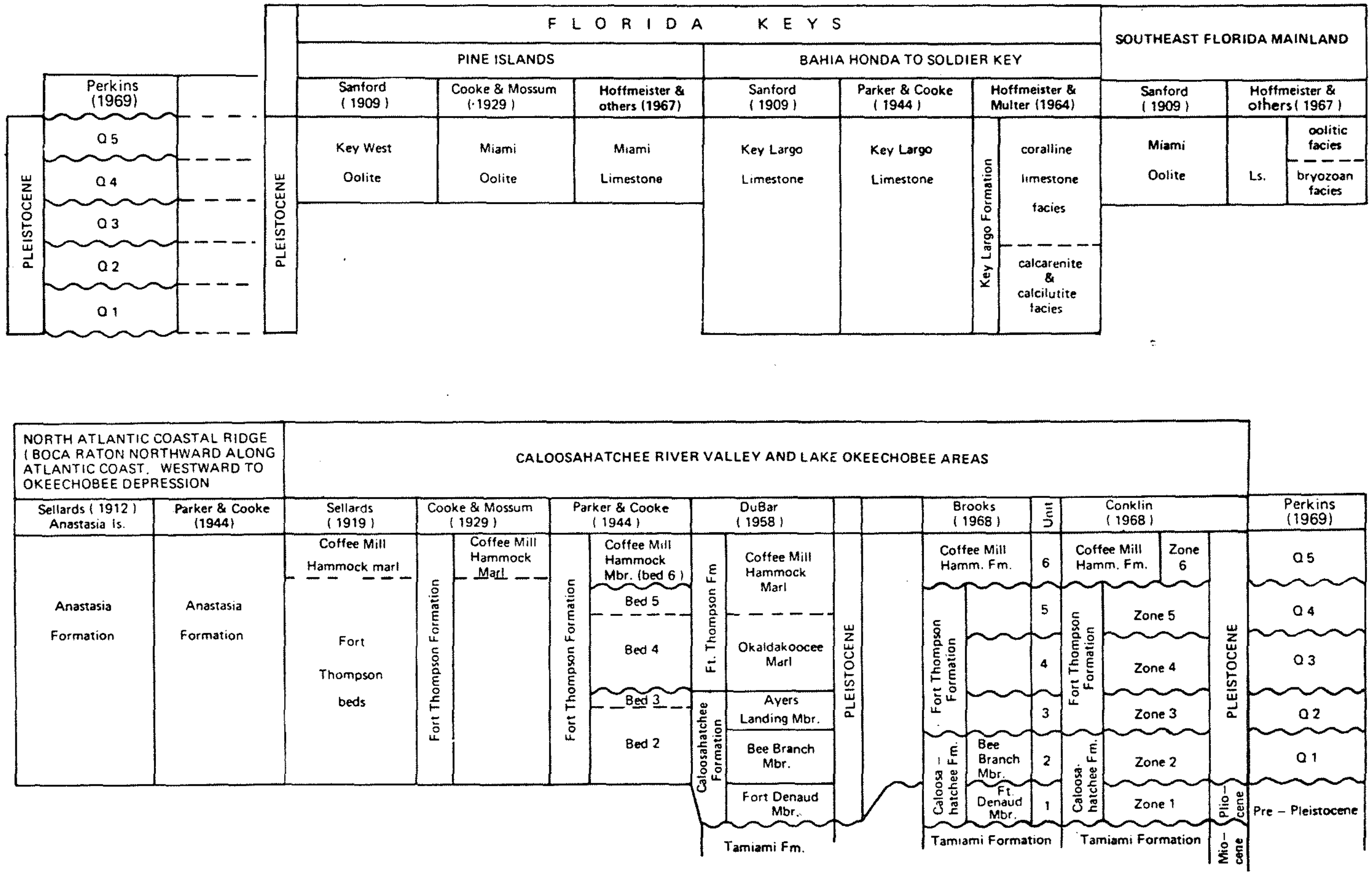
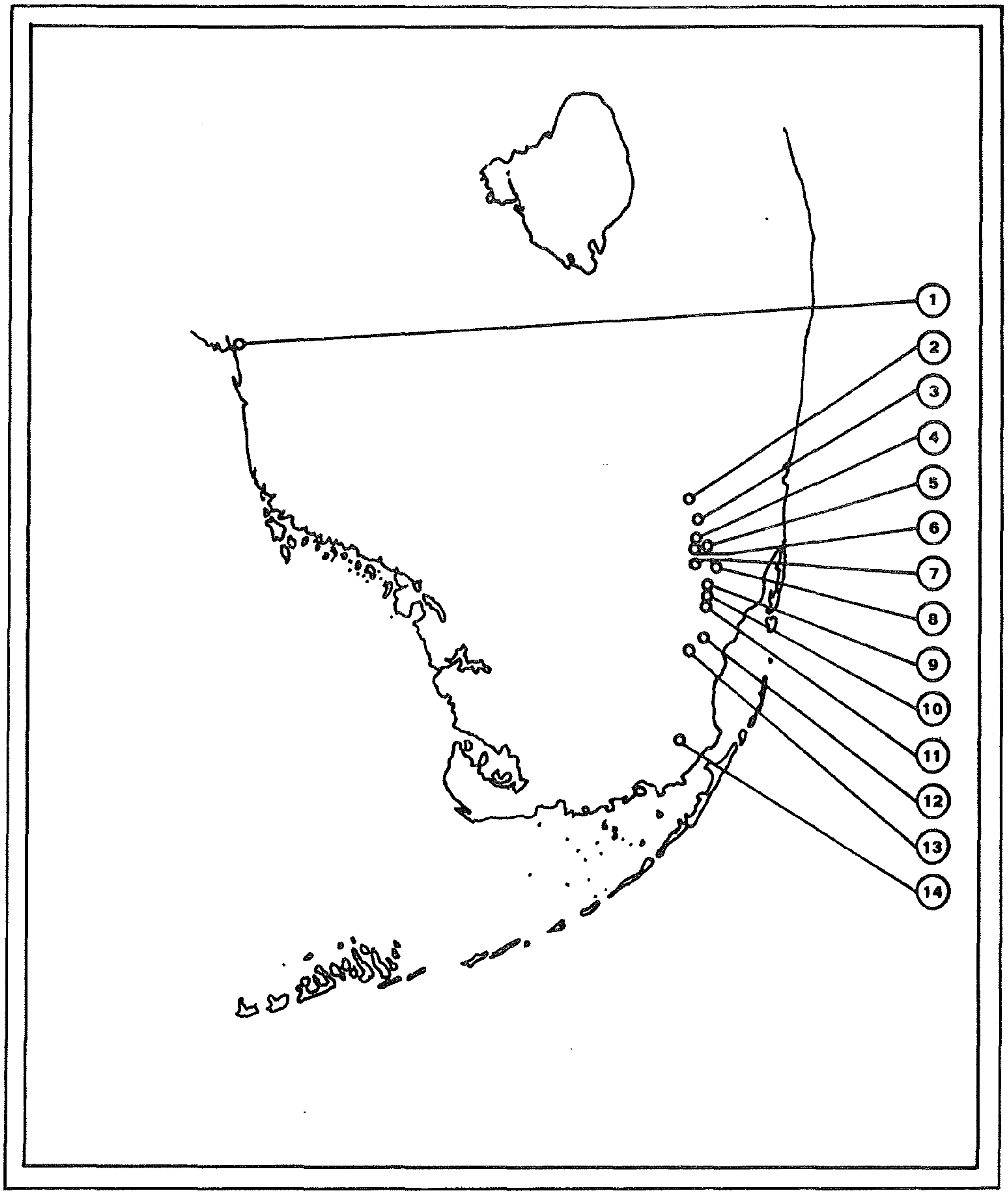


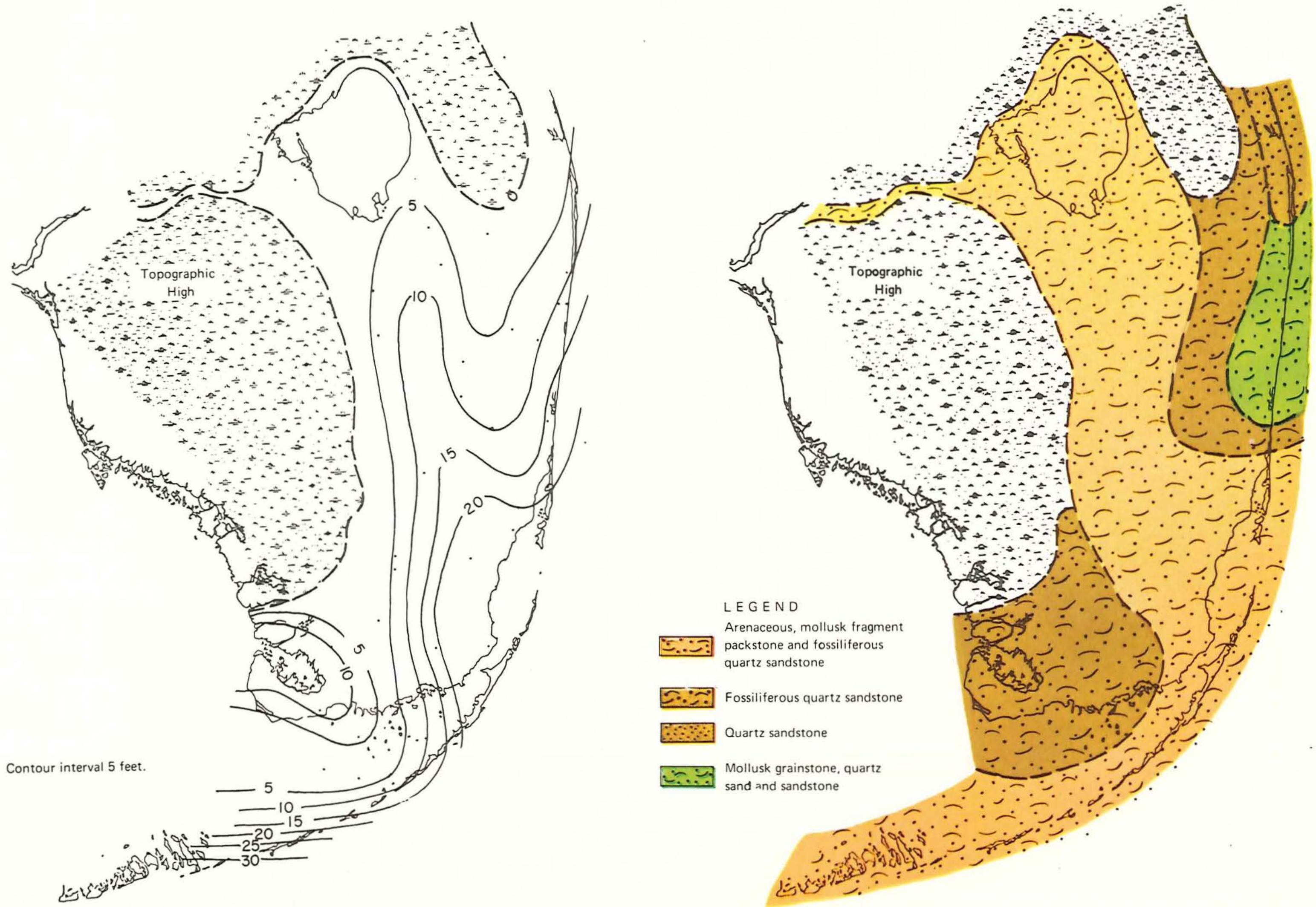
Figure 2. Correlation of previous stratigraphic terminology with Q-unit designations

FIGURE 3



INDEX MAP TO
TEST SITES

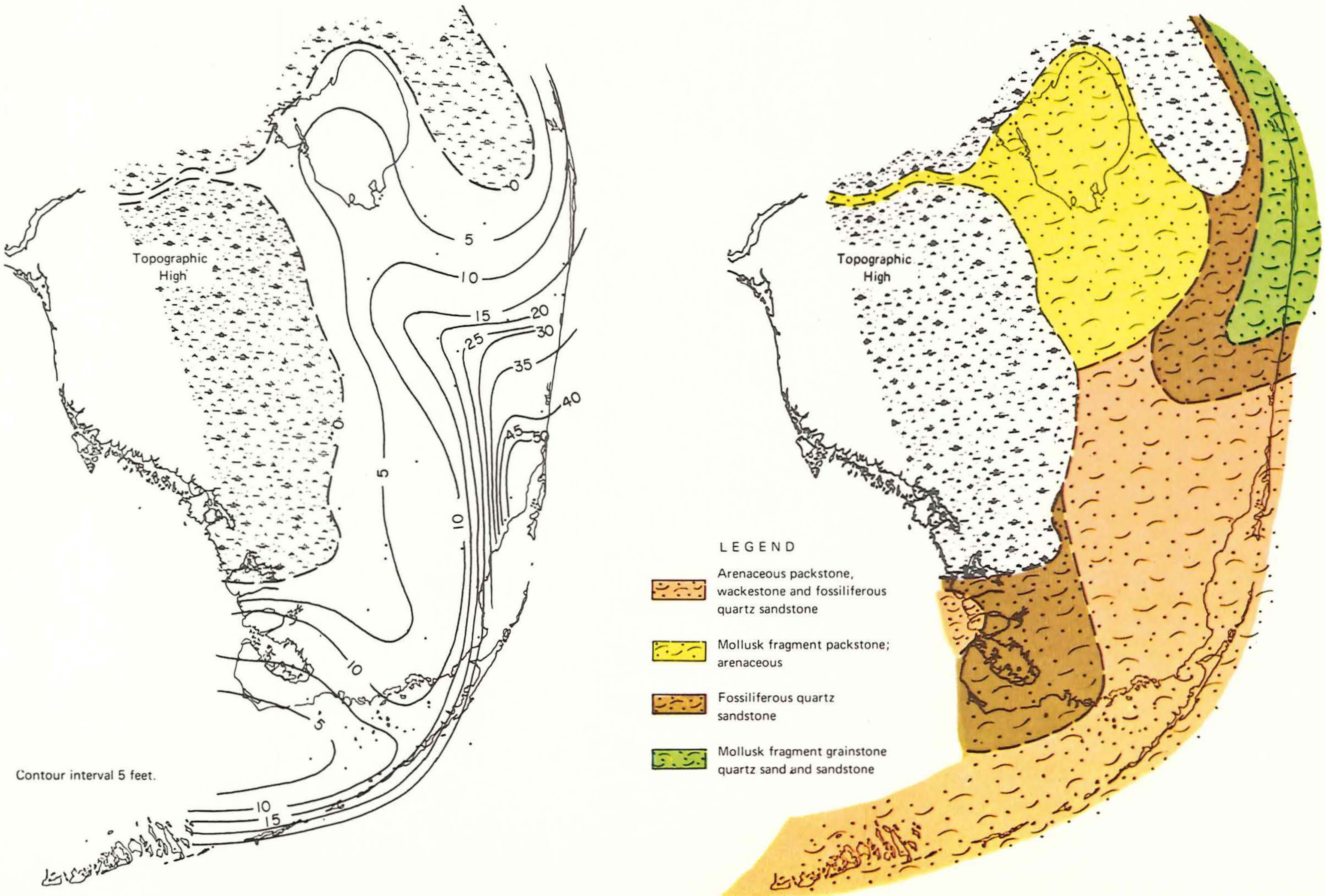
1. Harper Bros.,Inc.
2. Gator Rock
3. Bergeron Land Dev. Co.
4. Miramar Rock
5. L.W.Rozzo, Inc.
6. Hardrives, Inc.
7. Capeletti Bros. Pit #13
8. L.Dunn Indian Lakes Pit
9. Sterling Crushed Stone
10. Coral Aggregates Corp.
11. Miami Crushed Rock, Inc.
12. Capeletti Bros., Pit #12
13. Redland Rock Co.
14. Florida Rock & Sand Co.,Inc.



Thickness

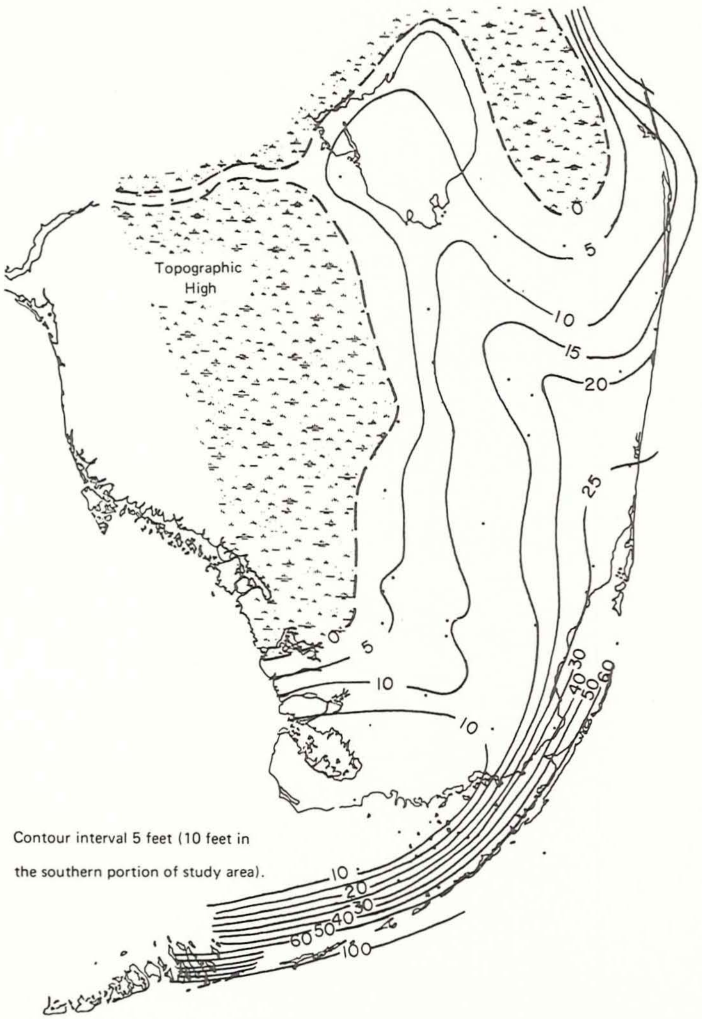
Rock Type

FIGURE 4. UNIT Q–1

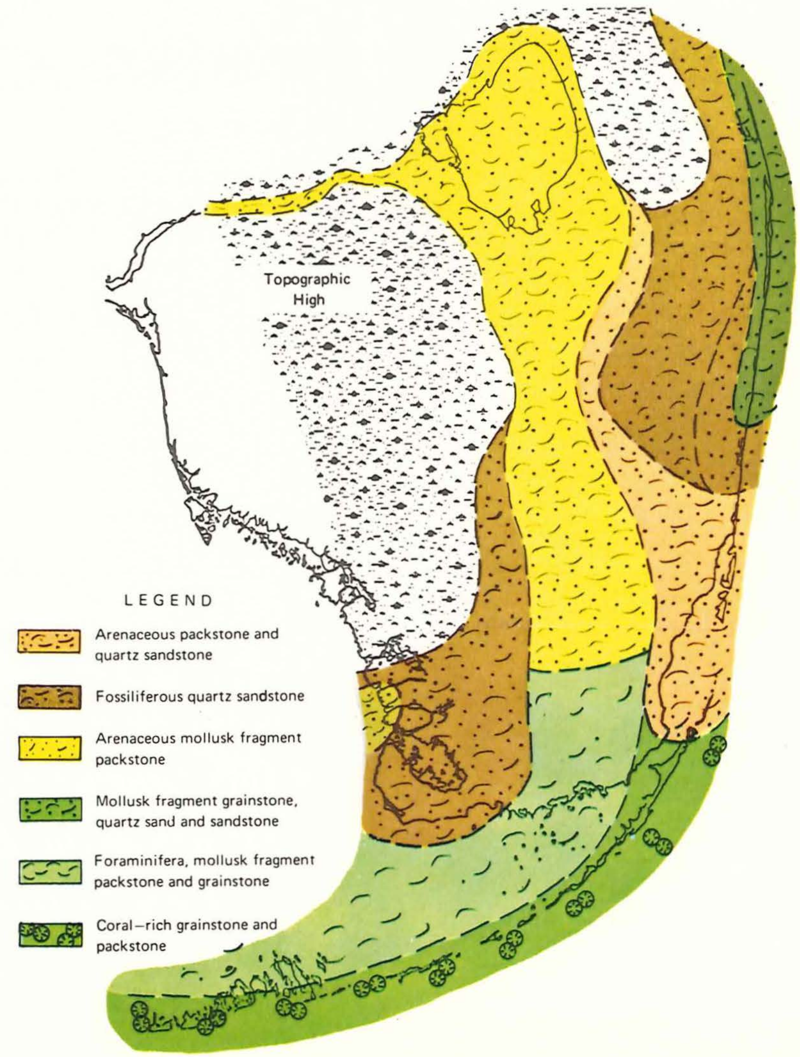


Thickness   Rock Type

FIGURE 5. UNIT Q–2

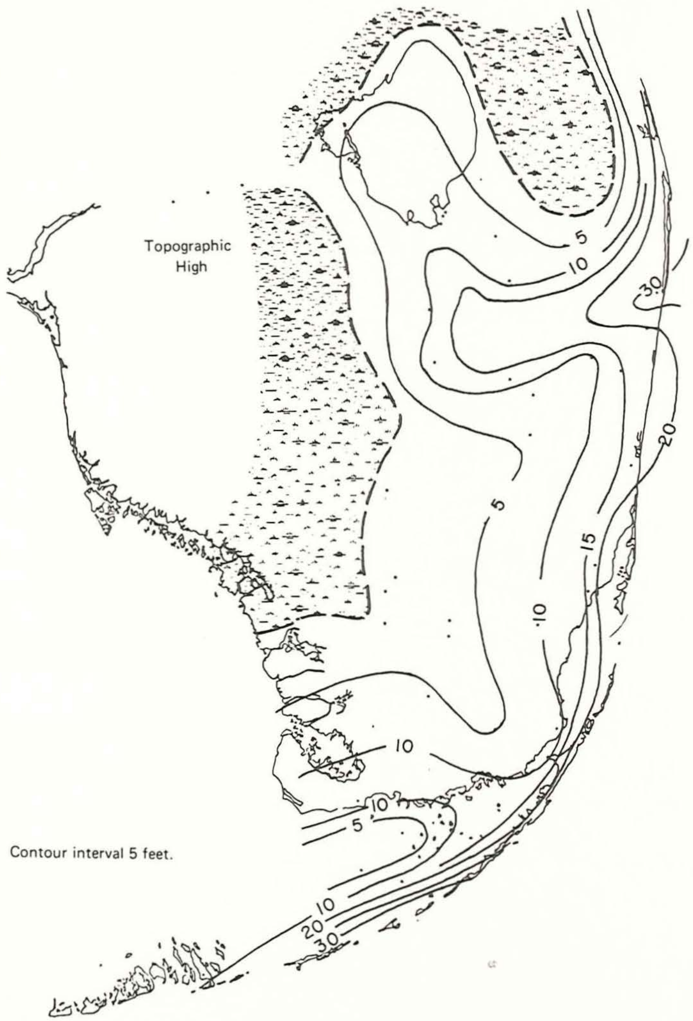


Thickness

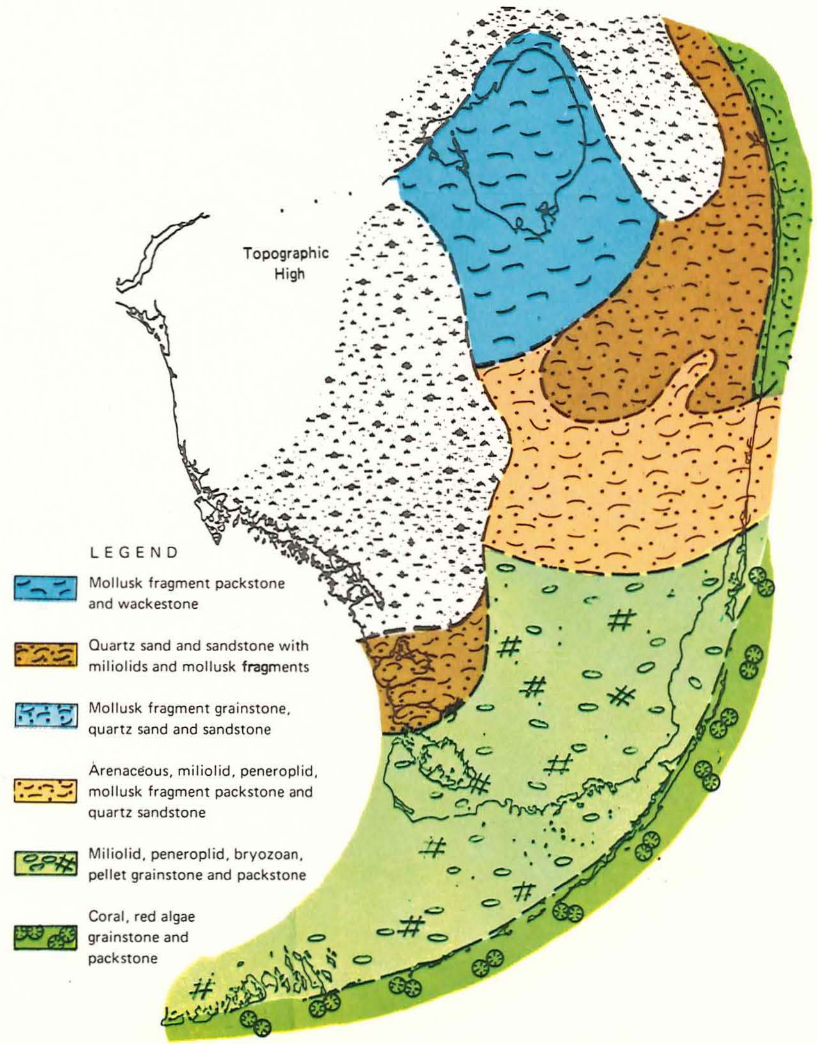


Rock Type

FIGURE 6. UNIT Q–3

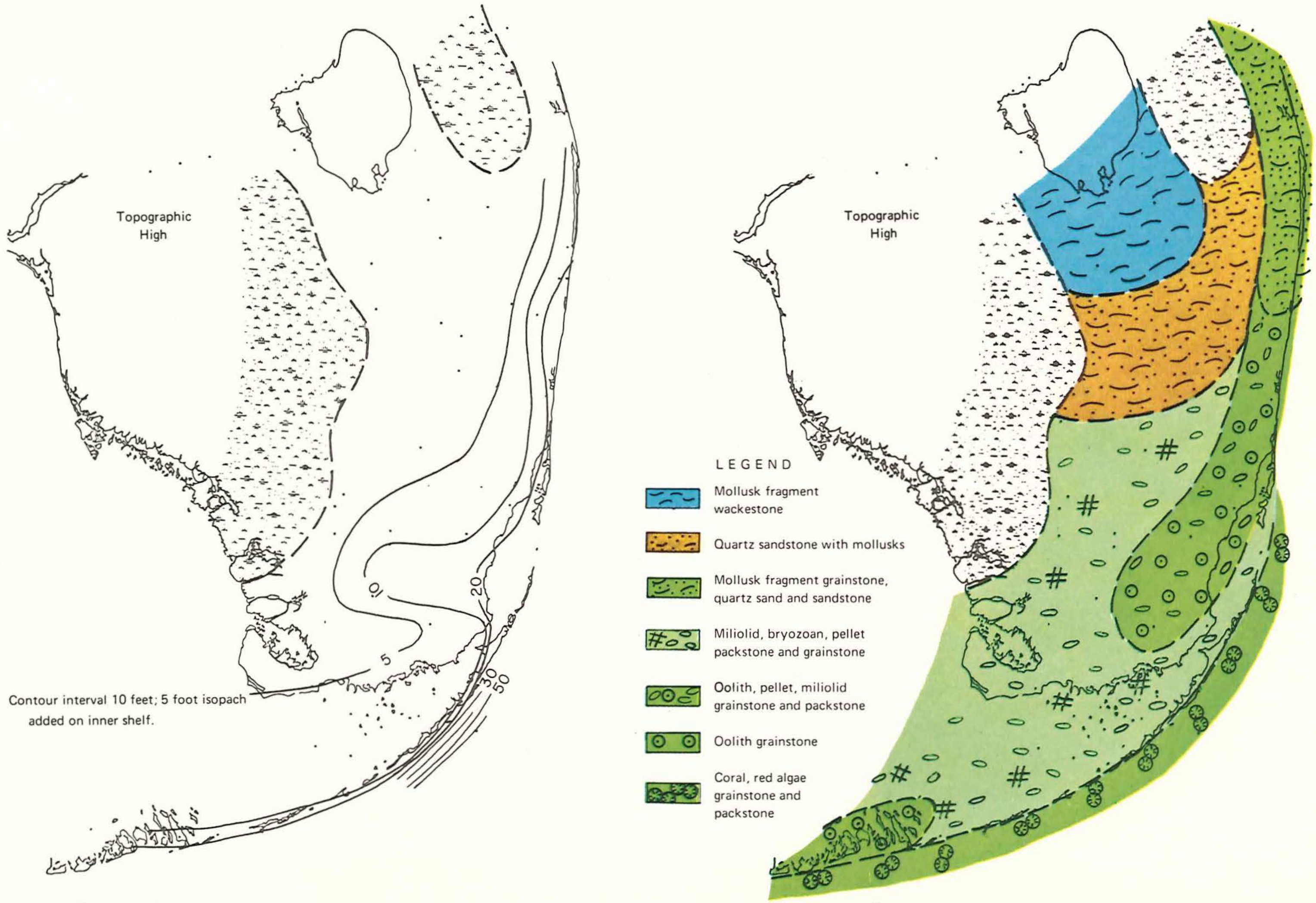


Thickness



Rock Type

FIGURE 7. UNIT Q–4



Thickness

Rock Type

FIGURE 8. UNIT Q–5

Packstone is a grain supported carbonate rock with carbonate mud in the grain interstices.

Grainstone is a grain supported carbonate rock which is mud free.

Boundstone is a carbonate rock which shows signs of being bound during deposition, such as intergrowths of colonial corals, or the laminations of stromatolites.

With the addition of quartz sandstone and some descriptive adjectives (particularly, arenaceous, which means that the rock contains sand, which in this suite of maps is mainly quartz sand imbedded in carbonate mud), this classification serves well in South Florida where the Pleistocene sediments are essentially undisturbed by tectonic movement.

Under ideal conditions, each of Perkins' units should contain basal sediments and fauna reflecting shallow water followed by a more open marine phase and terminating with a shallow water phase at the top but not necessarily containing representatives of the total range of sediments and fauna. This is because pre-existing topography apparently controlled the distribution of facies and in the total view, beach complexes are more or less stacked on previous beach complexes, and marine embayments on previous marine embayments.

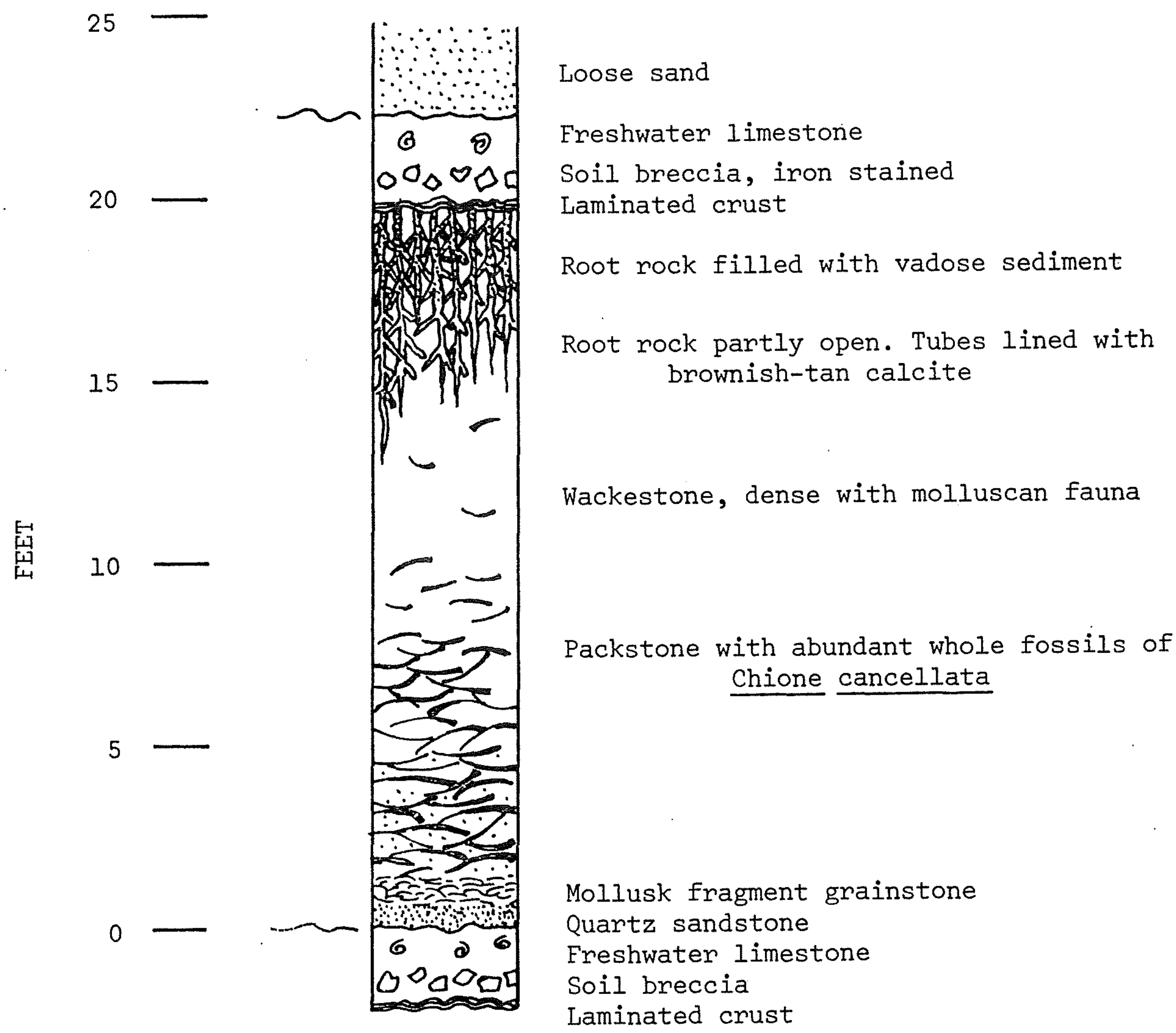
Consequently, most of the stone quarry operations of Dade and Broward County are in locations where the Pleistocene sediments would be characterized as open marine platform, open marine embayment, restricted marine shoal, or restricted marine bay. Unconsolidated fine-grain quartz sand which is commonly interbedded with limestone in the quarries of Broward County, and the more eastern operations in Dade County, is most probably the result of winnowing and redistribution in the restricted marine shoal and bay environments of sand originally deposited in the beach-dune-lagoon complex. This complex generally was situated along the present coastline in Palm Beach and Broward counties and is represented primarily by the rock-type group of mollusk fragment grainstone, quartz sand and sandstone. As the area where these rock types are dominant is approached, one can expect an increase in the amount of fine quartz sand.

Figure 9 is an idealized unit sequence as it might exist in the vicinity of the Florida Turnpike in central Dade County. This shows a relationship frequently found among the rock types of this area and provides a rational basis for interpretation of the rock types observed.

Of importance in evaluating different methods of blasting is the typical development of alternating competent strata with very open textured root rock or unconsolidated sand. Zones of blasthole enlargement probably occur opposite the open root rock because it is less competent than the other lithified strata. Hole enlargement certainly takes place opposite the layers of loose, unconsolidated sand.

There is no feasible way to determine to what extent cartridged explosives might tend to concentrate in enlarged sections when loaded

FIGURE 9



IDEALIZED UNIT SEQUENCE IN
CENTRAL DADE COUNTY
NEAR FLORIDA TURNPIKE

without some form of casing. It undoubtedly is related to normal hole diameter, cartridge diameter, cartridge length, amount of hole diameter enlargement, and vertical thickness of enlarged zone. Nevertheless, it is reasonable to assume that concentration of explosive charge does occur opposite those strata requiring little or no explosive fragmentation. The adverse effect of this concentration is compounded by the relatively high absorption of explosive energy by these more incompetent strata which further reduces the energy available for breaking the more competent beds. Because of these effects, the oversize material placed to one side for secondary breakage consists predominantly at most sites of blocks of dense, hard wackestone and the dense, hard discontinuity zone of freshwater limestone, soil breccia, laminated crust, and <u>filled</u> root rock. If open root rock is present and it comprises a significant part of the rock being quarried, it may also be prominent in the oversize because it tends to absorb the explosive energy in the water filled openings.

Using the criteria developed by Perkins and with judicious application to specific problems, each operator can develop or have developed, a model of the geologic conditions in his pit with a minimum of coring. Such a model would be useful in modifying blasting round design for different conditions in the pit, and in maintaining specification grade material. In connection with this effort, carefully compiled drilling time data in seconds per foot can be useful in making a proper interpretation.

This section would not be complete without some comment on the "Tamiami strata", a name commonly applied by drillers to hard drilling strata in western Dade and Broward counties. The name probably results from confusion with the much older Tamiami formation of Miocene age which crops out in the eastern parts of Monroe and Collier counties, and underlies the Pleistocene strata in part. This outcrop area was the emergent topographic high southwest of Lake Okeechobee shown on the rock type maps for each unit.

From examination of various specimens, the driller's "Tamiami strata" consists of one or more of the Pleistocene discontinuity surfaces. As each of the five Pleistocene units thins to the west and northwest, the separate discontinuity surfaces become less separated vertically and tend to converge into one another.

These areas may have been sufficiently removed from normal saline marine water that mangroves could not grow; these plants providing the most likely cause for the development of the root tube openings in the root rock. Such an area would best be characterized as a fresh water marsh which would be a logical site for development of the rock types which comprise the so-called "Tamiami strata". Without open root rock and loose sand, the rocks associated with converging Pleistocene discontinuity surfaces would provide a complex series of hard, dense strata which would require a modification in blasting procedure.

Most quarry operations lie well to the east and southeast of the most troublesome areas. Nevertheless, there are 5'-6' zones in parts of some pits where drilling and blasting incur greater than normal problems. These probably represent a local convergence of two or more discontinuity surfaces. When present, they are frequently referred to as the "Tamiami strata".

As the discontinuity surfaces converge, the departure from the idealized unit section in Figure 9 becomes greater because of missing portions. For example, a slight rise in sea level could cause mangrove development to encroach on fresh water marsh. This could be reflected in the geologic section by root rock occurring directly above fresh water limestone, unless destroyed subsequently by an open marine phase.

HISTORICAL DEVELOPMENT OF KELLY BAR METHOD OF LOADING EXPLOSIVES

The industry's impetus for successive steps in the development of the kelly bar method of loading appears to have been twofold; first, an effort to improve on the safety of the method as evidenced by changes widely adopted by industry following accidents, and second, an economic desire to achieve more efficient use of drilling and blasting effort. Consequently, there is some parallelism between historical developments and the accident record.

Placing exact dates on the developments is difficult because the changes were phased in at different times at different places, and the present sources of information sometimes have conflicting memories of certain events.

Because of the unstable hole conditions commonly encountered in South Florida and the need to develop aggregate sources for the local construction industry, a method of drilling and loading shallow (15 to 25 feet in depth) holes was developed in the 1930's. Holes were drilled with a track mounted rotary using a 2" bit. Centers were six feet apart or less. After drilling, the drill string and bit were pulled from the hole and a thin-walled steel tube called a "blow tube" was inserted in the drilled hole. Compressed air was blown through the tube which blew cuttings and sloughed material out of the hole, allowing the tube to go to bottom. When the tube reached bottom, the air hose was removed and cartridged explosives (1.5" x 8") were loaded thru the tube. The tube was then removed from the hole manually.

Because of the thin wall construction of the tube, the tube bent easily and when this happened, explosives would not pass through it. Another problem was the limited depth. Nevertheless, small drills continued to use this method until the mid 1950's.

In the 1950's, larger rotary drills were employed in order to use larger diameter explosives and increase the spacing and burden. Six-inch tricone bits were used and drilling depth was increased to about 30 feet. A standard square or round-fluted kelly bar was used with a relatively thick wall. Explosives were not loaded through this kelly because the inner diameter was not large enough to pass the 3" and 4" cartridges that operators wanted to use, and, of course, the explosives could not pass the tricone bit. After the hole was drilled, the drill rig moved on to the next hole and a steel pipe somewhat smaller than the 6" hole was positioned over the hole with a crane. This pipe was called a "loading tube". At the top of the tube, a weight could slide up and down between two flanges which provided a hammering action to drive the tube into the hole. Inside the loading tube was a 2 1/2" tube. Water was pumped down this tube with high pressure causing excess drill cuttings and sloughed material to be forced up the annulus between the two tubes, cleaning out the cased hole. The cartridged explosives and an electric blasting cap were then loaded into the loading tube. After loading, the tube was removed by reversing the hammering action to work upwards on the top flange.

About 1957 or 1958, the dyna-kelly was introduced; so called because dynamite was loaded through the kelly. This was a thin-walled (5/8 inch) section of tubular chrome alloy steel with three flutes, or round grooves, each 3/8" deep and with a 13/16" convex. The inner diameter could be made large enough to accomodate 4" cartridge explosives if so desired. Tubular sections were welded together to make an overall length sufficient to reach bottom without having to make any connections. This kelly was used with a drag-type core bit having approximately the same diameter. To eliminate the problem of rock cores going up into the kelly and causing explosives to get hung up in the kelly, a weight on a line was dropped down the kelly after drilling. The kelly was then raised a short distance and if the line went slack, the driller knew the kelly was not clear. By raising and dropping the weight, the kelly could sometimes be cleared. Other times might require the reapplication of water or air pressure along with hammering. If this didn't clear the jam, the drill string could be pulled out of the hole and the rock core removed by prying it out through the bit end of the string. The hole was not loaded until the drill string was clear.

An improvement on this method was to utilize a core-breaker bit. Initially, these were of a retrievable type which could be pushed up through the outer core bit before starting the drilling of a hole. The core breaker bit had two spring loaded dogs which would lock into two slots in a special sub-adapter and hold it in place. After drilling the hole, an overshot retriever was lowered down through the kelly which locked onto a 7/8 inch retriever pin at the top of the core breaker bit. The pin was tapered to release the dogs as it moved upward within the bit. When it reached the stop position it lifted the bit. After the core breaker bit was removed, explosives could be loaded down through the kelly. When the kelly was retracted, the explosives passed by the outer core bit and remained in the hole.

The retrievable core breaker presented a common problem when sand would get into the dogs and lock them in place. There was also the possibility that the drill crew might fail to remove the core breaker bit and load explosives on top of it. According to official reports, this is believed to have happened in an explosives accident at Pennsuco Quarry of Maule Industries, Inc. in January, 1964, in which three men were killed. It was assumed that after loading the explosives, the kelly was pulled from the hole and an effort was made to retrieve the core breaker by removing the sub-adapter with a wrench. The retriever pin could have punctured the explosives cartridges and loose explosive material could have accumulated about the thread areas of the sub-adapter and bit. Efforts to remove these parts could have set off the explosives in the kelly bar. To overcome just such a problem, a core-breaker bit removal system had been devised at about the same time as this accident. The system involved Allen wrench-type screws in the subadapter which released two rods which could move vertically in the sub and trip the locking dogs. The core breaker bit could then be removed through the outer bit. It is not known whether this type of safety device was being used at the time, or whether the men were familiar with its operation or whether the movement of the set screws and rods might not have been sufficient to cause the explosion.

Another method of using the dyna-kelly was to connect a four-foot subadapter to the base and use a standard rock bit. After drilling the hole, the drill string was pulled from the hole and the four-foot sub and bit were replaced with a four-foot long section of drill tubing. The drill tubing and the dyna-kelly were then run back into the hole using the circulation system for clean-out. Explosives were loaded through the dyna-kelly and drill tubing when the string reached bottom. The kelly and drill tubing were then pulled from the hole. In April, 1971, there was a fatal blasting accident at the Dade County plant of General Portland Cement Company, where this system was being used according to USBM Health and Safety reports. One man was killed instantly and another seriously injured. The inspector could not determine the actual cause of the accident but an eye witness said that about 15 feet of the 40 foot long dyna-kelly had been pulled out of the hole. The drive engine on the drill was idling and the drill steel was not turning or being raised. It was assumed that the explosive charge was stuck and was being lifted out in the tube.

Because of these accidents, industry sought safer ways to perform the necessary drilling and blasting, and this effort focused on eliminating the retrievable core breaker.

The present method of using the dyna-kelly in larger diameter holes, is to employ an inner bar and core breaker bit inside the kelly bar. Steel centralizers are placed about every 20 feet on the inner bar to provide rotational stability. Where loading is done through the kelly, the core breaker bar and bit are hoisted up in the mast by a solitary steel line where they hang suspended while the explosives are loaded into the kelly. Explosives cannot be loaded in the kelly until this is done. Detonating cord is tied to the bottom stick and is played out from a spool. A drag, consisting of a brass rod bent to form a V with a small loop at the point of the V, is used to keep the detonating cord from falling down in the hole. The cord is run through the small loop and a loosely knotted clump of cord is placed on the uphole side. The wings of the V can be adjusted to provide friction against the inner wall of the kelly. After the explosives are loaded, the inner bar and bit are placed over the outer bar and lowered slightly. The outer bar is then raised. This method is used where 3" or 4" diameter cartridged explosives are being loaded, and the dyna-kellys are referred to as 3" or 4" kellys. The actual dimensions are 4.75" OD X 3.5" ID and 6.25" OD X 5" ID, respectively.

The same system was used with the so-called 2" kelly bars (3.75" OD X 2.5" ID) in the early application of the system. Then on a job where the wrong size core breaker bits had been delivered, it was discovered that holes could be drilled without the core breaker with relatively infrequent rock jams. These few rock jams could generally be detected by the behavior of the air or water pressure used in circulating cuttings from the hole. With this discovery, further refinements were made in bit design by adding a 0.125" thick piece of tungsten carbide to the inner wall of the bit. With normal bit wobble, this addition tended to undercut any developing rock core and permitted the bit to break it up. A similar effect could be created by shortening the length of the teeth (generally called fingers in South Florida) on one side of the bit which increased the wobble but the former is the more popular method. Today, all small diameter kelly bars in South Florida are used without any type of inner core breaker bit.

There have been two nonfatal accidents involving jams in the small diameter kelly bars. Both were at the L.W. Rozzo plant in Broward County. The first occurred on March 1, 1971 and the second was on November 14, 1974. According to official reports, in the first accident the mast had been lowered in order to remove the jammed explosives from the kelly. All but two cartridges were removed without difficulty. It was reported that a metal rod was used to probe the blocked area. An explosion occurred in the drill steel which blew fragments of the steel into an explosives truck 75 feet away and detonated about 300 pounds of explosives on the truck.

The two workmen suffered minor cuts and bruises. The drill was extensively damaged and the explosives truck was destroyed.

In the second accident, the MESA report indicates that the driller and his helper noticed a section of 25 grain detonating cord hanging out of the drill bit when it was pulled out of the hole after drilling and loading. They were able to determine that part of the explosive charge was lodged in the drill stem. The mast was not lowered but the bit was raised until it cleared the rotary table. The two men used an aluminum pipe to break up and remove two cartridges of dynamite. While attempting to remove a third cartridge the detonation occurred. The driller received minor injuries to his head, shoulder and hands. The helper suffered a partial and possible temporary loss of hearing. The driller said that the detonating cord appeared to be twisted around the lodged cartridges.

A third accident has occurred with the small diameter kelly but unlike the other two, it happened while the drill steel was partially in the hole. At the Broward County quarry of Seminole Rock Products, Incorporated (This pit is now owned and operated by Vulcan Materials Company) three men were killed when explosives in the drill steel detonated. The direct cause of the accident is not clear but the USBM report states, ".... all but about 16 feet of the drill steel had been removed from the ground before the explosion, and about 4 feet of explosives had been in the drill steel above the ground. In the absence of evidence to the contrary, it was assumed that the explosion originated in the drill steel and flying metal from the bursting steel detonated about 1200 pounds of explosives remaining on the explosives truck which was located about 20 feet away from the drill rig. The six or seven fully loaded holes drilled earlier in the pattern had also detonated."

GENERAL QUARRY PRACTICE

Quarry operations in South Florida differ from those in most other parts of the United States in that the floor of the pit remains under 35 to 70 feet of water. One operator attempted to pump the water out after first digging a trench about 30 feet deep around the area to be excavated. The trench was backfilled with relatively impermeable material. The water level in the pit was then pumped down about eight feet before equilibrium was reached and the pumps couldn't bring about any further lowering. Reportedly, the pump rate was about 8 million gallons per day and the area of the pit was about 15 acres.

The first step in quarry development after the necessary zoning permits are obtained, is to strip the two to three feet of surface muck and soil. This is a black, peat-like material with numerous gastropod shells. Stripping consists of bull dozing the material into a mound along the long sides of the planned rectangular pit. Alternatively, the material may be stripped and piled with a dragline. Much of this material is sold as top soil for real estate developments after it is processed through a one-inch screen.

After stripping, the stripped area is covered with about one foot of water during dry periods. Because of the heavy rains which are common in the area, and the pumping activities of the South Florida Flood Control District, the depth of the water may increase by another foot or two in the period of a day. Consequently, the stripped area has to be built up with a rock pad in order to have a working surface. The initial rock material must be obtained elsewhere, but after enough pad area is available for the drill and a dragline to operate, the rock fill can be generated on-site.

An initial key cut is usually made. This is about 75 to 80 feet wide and is as long as the final pit will be, usually on the order of 1500 to 2000 feet.

Successive multi-row blasts are made across the width of the key cut with the delay pattern designed to heave the rock from each blast toward the shot rock of the previous blast. The shot rock is generally levelled by bulldozer which uses some of the material to advance the rock pad. After the drilling and blasting have advanced a safe distance, a dragline can start digging the shot rock and placing it in a surge pile on one side of the pit. This is usually done with a six to eight cubic yard bucket.

Because the angle of repose of the shot rock in the water is about 45°, the dragline cannot dig the pit to its full depth on this initial cut if this is more than 40 feet and the key cut width is about 80 feet. This is because the two side slopes will meet in the middle of the cut at a depth one-half the width.

In order for the dragline operator to know the depth at which his bucket is digging, one or more white indicator bands are painted on the line from the boom to the bucket. Generally, in digging the initial cut the operator will swing 45° to 60° from digging point to dump point. Equipment used in this digging is usually a Bucyrus-Erie 88-B, Lima 1860-SC, or equivalent.

Water carrying fines flows from the surge pile towards the lake and towards the back edge of the rock pad. This makes vehicular traffic on the far side of the pile difficult. For this reason, digging is usually started at the end of the pit closest to future haulage direction.

While the dragline is digging the first cut, drilling and blasting will commence on the side of the lake opposite the surge pile when the dragline has advanced a safe distance. Depending on the size of the drill, about 20 to 40 holes will be drilled and loaded through the kelly each working day. Local ordinances prohibit blasting after 5pm or leaving the loaded holes overnight. Consequently, each days drilling effort results in a blast. At some operations, where ground vibrations are a problem, 8 to 10 holes may be exploded as soon as they are drilled and loaded. This results in four or five blasts per day.

Local ordinances also prohibit having electric blasting caps at the site while the drill rig is still operating. Thus, the rig has to complete the pattern, move at least 300 feet from the site, and then the blaster goes to the cap magazine. Depending on location, this may take from 10 to 20 minutes, and if it is repeated five times a day it can reduce productive time by as much as one hour and forty minutes.

When the drilling and blasting have proceeded a safe distance on the new cut, the dragline can start digging at the near haulage end of the pit. Some operators will place the dragline on the shot rock with the crawlers parallel to the direction of the cut. Others commonly will position the dragline at an angle of about 45° to the direction of the cut. At least one operator positions the dragline on unshot rock with crawlers oriented at 90° to the cut. Drilling, blasting and digging will be continued in this fashion, first on one side of the pit and then the other, until the pit is developed to its full limits as defined in the development plan that was submitted for the county permit. Most owners contemplate real estate development around the edge of the final pit.

Some operators pick up the shot rock from the surge pile at the side of the cut with a front-end loader (usually 6 to 7 cubic yard capacity) and load it on to haul units which transport the material to the primary crusher for processing. Most of these permanently installed primaries are impact crushers, but there are some jaw crushers. More commonly, a portable crusher is moved to the end of the pile and the primary crushing is done at the site. A hopper and feeder before the primary crusher may be fed by a front-end loader or it may be fed by a small four cubic yard dragline operating from a position at the top of the shot-rock surge pile.

The portable primary crushers are usually Universal or Cedarapids impact crushers and they require daily buildup of breaker bars and impeller bars because of the abrasive action of the quartz sand contained in the rock. Buildup time varies with the quartz content. It is not uncommon for the buildup to require eight hours per crushing day.

The portable primary discharges on to a radial stacking conveyor which is supported and controlled by cables running to a gantry frame on the primary crusher unit. Limerock base course material is discharged from the conveyor if the rock meets the required chemical specification that it contains at least 70% calcium and magnesium carbonate. At many operations, this is the extent of the processing. Customer's trucks come to the pile of discharged material and are loaded with a front-end loader. They then proceed to the scale house.

At other operations, the crusher-run base material may be loaded on to customer's trucks without further processing and part, or all of it may be loaded on to haul units which transport it to the secondary plant. Here it is first screened to make sized aggregate. The oversize is crushed again in hammer mills or roll crushers and recirculated for screening. Various grades of sand are made in addition to the coarse aggregate products. The finished material is stockpiled by size for later sale to customers. A considerable quantity of material in some plants is loaded on to railroad cars for shipment up the east coast of Florida where aggregate materials are not available. Consequently, the importance of the quarry industry in Dade and Broward counties extends well beyond the boundaries of these counties.

DRILL RIGS

Drill rigs used in South Florida are generally grouped by the diameter of the explosive cartridges loaded through the dyna-kelly. Three sizes are presently used; 2", 3", and 4". Occasionally, 4 1/2" cartridges are loaded through the larger kelly. The corresponding inner diameters of the dyna-kellys are 2 1/2", 3 1/2", and 5" and these will be used in this report to avoid the confusion which accompanies categorization of drill rig size by explosive diameter.

It is difficult to get an accurate count of dyna-kelly rigs and to definitely ascertain ownership because of the frequent moving in and out of the region, the conversion to a water well rig or vice versa, cannabilization of old rigs, and the numerous sales which have taken place in the last few years. The tabulation in Table No. 1 has been compiled from many sources and field visits and is believed to be complete as to operable equipment. Several of the listed pieces of equipment have not been used for several years and probably are not operable without considerable repair. Many operating rigs are old and obsolete.

Most drilling equipment in the area is mounted on crawler tracks. This provides better traction and more stability in moving from hole to hole and is essential for those operations where the unit is operating directly on the surface muck. In quarry operations where there is a well-constructed, level pad, a truck mounted rig can be used but is certainly more prone to overturning during moves if a soft spot (such as a bridged backbreak fracture) is encountered because of the smaller bearing surface. Truck mounted rigs are indicated in the tabulation. All others are on crawlers.

The rotary table opening is shown where known. This is an essential consideration in contemplating any change in the size of the dyna-kelly. There must be enough opening to accomodate the larger kelly outer diameter plus the drive bushing. The drive bushing requires at least 1.25 inches more than the outer diameter of the kelly. If not large enough, some rotary tables could be custom-fitted with a drive welded to the top of the rotary table.

Table 1 includes a total of 89 rigs used in mining, real estate development, canal work, and general construction. The list shows sixty-four 2.5" rigs (72%), three 3.5" rigs (3%), and twenty-two 5" rigs (25%).

If only mining where blasting is done is considered, Table 2 reveals the situation. It indicates the quarry blasting operations south of Lake Okeechobee and if the operators conduct their own drilling operations, the rig size and flush system is indicated. If they use outside contractors for their drilling, that is indicated. In some cases, both rig ownership and contracting are shown which indicates the operator either supplements his own drilling from time to time, or prefers to use a contractor instead of his own equipment.

Nearly all contract drilling is small diameter and the breakdown by size as to the number of South Florida quarry operations is 68% 2 1/2" ID rigs, 6% 3 1/2", and 26% 5" ID. However, about 76% of the products are produced where the large diameter kelly bars are used.

The heavy dependence on contract drilling is apparent. Nevertheless, the cyclical economic swings that the South Florida quarry industry has experienced in recent years alternately creates a shortage of contract rigs during boom periods, and virtually no demand during the intervening slack periods. This is because the bigger producers have their own rigs and most of these are large diameter operations. In general, the large diameter operations are more efficient (less drilling time per cubic yard of stone) and because they are mainly used in conjunction with larger plants there are additional downline cost savings resulting from economies of scale. Such producers are capable of meeting most, if not all, of the product demand during economically slack times. In these situations, the more marginal producers (which usually rely on contract drilling) decrease, or cease, their activity and the demand for contract drilling dries up.

Most of the contract drilling firms consist of one owner or a partnership of two or three individuals, most of whom actively participate in the business. In general, their financial resources are considerably less than those of a quarry operator and the adverse

TABLE 1
Dyna-Kelly Equipped Drill Rigs

OWNER	MAKE & MODEL		KELLY I.D.	NORMAL DRILLING DEPTH	FLUSH	ROTARY TABLE OPENING
Aggregates, Inc.	Midway	1000	5"		Air	7½"
Alonzo Cothren, Inc.	Damco		2½"	35'	A	7½"
Alonzo Cothren, Inc.	Damco		2½"	35'	A	5 "
Anderson Contracting	GD	1000	5"			10 "
BSI	GD	1000	2½"	35'	A	7½"
BSI	GD	RDC-11	2½"	35'	A	5¼"
BSI	Davey		2½"	35'	A	GD 7½"
BSI	Davey		2½"	Not Operable	A	GD 7½"
BSI	Davey		2½"	Not Operable	A	GD 7½"
BSI	GD	RDC-11	2½"	Not Operable	A	5¼"
Capeletti Bros.	Davey	M5C	2½"	43'	A	
Capeletti Bros.	Mayhew	RDC-10	2½"		A	5¼"
Carroll Construction	GD	500	2½"*	30'	A	
Childers, J.	Mayhew	1000	2½"	40'	A	7½"
Childers, J.	Failing		2½"	24'	A	
Crucie Bros.,Inc.	Mayhew	RDC-15	2½"	30'	Water	5¼"
Curcie Bros.,Inc.	Mayhew	1000	2½"	50'	W	7½"
Devcon	GD	1000	2½"	45'	A	
Devcon	Davey	M5C	2½"	45'	A	
Devcon	Failing	RB-25	2½"	45'	A	
Dixie Lime & Stone	GD	1000(T)	5 "*			10 "
Dynablast,Inc.	Davey	M5C	2½"	50'	A	7½"
Dynablast,Inc.	Davey	M5C	2½"	50'	A	7½"
Dynablast,Inc.	Damco	2400	2½"	20'	A	5 "
Eden,Bob	Damco	2400	2½"	18'	A	7 "
Florida Machinery	GD	1000	2½"	50'	W	5¼"
Florida Machinery	Failing	1500	2½"	50'	W	
Florida Mining Materials	Failing		5 "*	80'		
Florida Rock & Sand	Damco	1850	2½"	50'	A	
Florida Rock Industries	GD	1000	5 "	30'	A or W	7½"
Florida Rock Industries	GD	1000	5 "	30'	A or W	7½"
Florida Rock Industries	Mayhew	1000(T)	3½"	30'	W	7½"
Florida Rock Industries	Damco	1850	5 "	30'	W	
Gulf & Western Sugar Co.	Damco	5000	2½"			
Harper Bros.,Inc.	Mayhew	RDC-10	2½"	20'	A	5¼"

OWNER	MAKE & MODEL		KELLY I.D.	NORMAL DRILLING DEPTH	FLUSH	ROTARY TABLE OPENING
Kelly, Jonathan	Damco	2400	2½"			
Klein, Ronnie	GD	1000	5 "	44'	A	7½"
Lacanto Rock Div.	GD	1000	5 "		A or W	10 "
Lebel, Ernest	GD	RDC-16	2½"	50'	W	5¼"
L.W.Rozzo,Inc.	GD	1000	2½"	54'	A	7½"
Meekins,Inc.	Damco	1200(T)	2½"	Not Operable	A or W	
Merrill	Damco	2400	2½"	36'	A	7½"
Miramar Rock	Damco	1250	3½"	50'	A or W	7 "
Mowry, Fred	Mayhew	1000	5 "	44'	W	7½"
Mowry, Fred	Mayhew	1000	2½"	43'	A	7½"
N.D.Allen & Sons	Damco	2400	2½"	30'	A	
Ocean Reef Dev. Co.	Davey		2½"	15'	A	
Oren, Paul	Damco	3000	2½"			
Oren, Paul	Damco	3000	2½"			
Oren, Paul	Damco	2400	2½"			
Palmetto Drilling	Damco	D-6000	2½"	62'	W	7 "
Palmetto Drilling	Damco	D-6000	2½"	62'	W	7 "
Palmetto Drilling	Damco	D-5000	2½"	47'	A	7 "
Palmetto Drilling	Damco	2400	2½"	25'	A	
Pennsuco Aggregates	GD	1000	5 "	57'	W	7½"
Pennsuco Aggregates	GD	1000	5 "	57'	W	10 "
Redlands Rock	GD	RDC-15	2½"	45'	A	5¼"
Redlands Rock	Mayhew	1000(T)	2½"	40'	A or W	4 3/4"
Rinker Lehigh Cement	GD	1000	5½"	65'	W	7½"
Ronnie's Welding & Machine Shop	GD	RDC-11	2½"		W	
Ronnie's Welding & Machine Shop	Mayhew	1000(T)	2½"		W	converted to water well drilling
Ronnie's Welding & Machine Shop	Mayhew	1000(T)	2½"		W	" " "
Southeast Materials,Inc.	GD	1000	5 "	50'	A	7½"
Southeast Materials,Inc.	GD	1000	5 "	50'	W	10 "
South Florida Drilling	GD	1000	5 "	50'	A or W	7½"
Sterling Crushed Stone	Mayhew(T)		3½"	50'	W	7½"
Sterling Crushed Stone	Mayhew		5 "	48'	W	7½"
Tirrell-Bruni,Inc.	Damco		2½"	25'	A	
Toppinno Bros.,Inc.	GD	1000(T)	2½"	50'	A	7½"
Toppinno Bros.,Inc.	Davey	(T)	2½"			
Toppinno Bros.,Inc.	Small truck mounted		2½"			

OWNER	MAKE & MODEL		KELLY I.D.	NORMAL DRILLING DEPTH	FLUSH	ROTARY TABLE OPENING
Troup Bros.,Inc.	Damco	2400	2½"	Varied	A	
Troup Bros.,Inc.	Damco	2400	2½"	Varied	A	
Troup Bros.,Inc.	Davey		2½"	Varied	A	
Troup Bros.,Inc.	Davey		2½"	Varied	A	
Troup Bros.,Inc.	Davey		2½"	Varied	A	
Troup Bros.,Inc.	Davey		2½"	Varied	A	
Upper Keys Marine	Damco	1200	2½"*	30'	A	
Upper Keys Marine	Damco	2400	2½"*	30'	A	
U.S. Sugar Co.	Damco	3000	2½"*			
U.S. Sugar Co.	Damco	3000	2½"*			
U.S. Sugar Co.	Damco	3000	2½"*			
Valious, Charles	GD	1000(T)	2½"		A	
Vulcan Materials Co.	GD	1000	5 "	47'	W	10 "
Vulcan Materials Co.	GD	1000	5 "	55'	W	7½"
Warren Bros.,Inc.	Mayhew	1000	2½"	18'	A or W	7½"
White Construction Co.	GD	1000	5 "			7½"
White Construction Co.	GD	1000(T)	5 "			7½"
White Construction Co.	Mayhew	1000(T)	5 "			7½"

(T) indicates truck mounted rig. Others are crawler mounted.

* indicates that open hole loading is normally done.

TABLE 2

QUARRY OPERATIONS SOUTH OF LAKE OKEECHOBEE

WHERE KELLY BAR LOADING HAS BEEN DONE

County	Operator	Drill Rigs	Flush
Broward	Bergeron Land Dev. Co.	Contract	
"	Curcie Bros.	Two 2 1/2"	Water
"	Gator Rock	Contract	
"	Griffin Bros., Inc.	Contract	
"	Hardrives Company	Contract	
"	L.W. Rozzo, Inc.	One 2 1/2"and contract	Air
"	Meekins, Inc.	Contract	
"	Miramar Rock	One 3 1/2"	A or W
"	Vulcan Materials	One 5"	Water
Collier	Ralph Crapse	Contract	
"	Florida Rock Ind., Century Quarry	One 3 1/2"(T)	Water
"	Highway Pavers, Inc.	Contract	
"	Jack Queen Heavy Industry	Contract	
"	Meekins, Inc.	Contract	
"	McCormick	Contract	
"	Ochopee Rock	(Closed?)	
"	Toppinno & Sons, Inc.	See Morgan Co.	
"	Warren Bros.	One 2 1/2"	A or W
Dade	A.J. Capeletti, Inc. Pits 7,9,10,12 & 13	Two 2 1/2"	Air
"	Coral Aggregates Corp.	Contract	
"	L.W. Dunn Co. Airport Pit	Contract	
"	" " " Indian Lakes Pit	Contract	
"	Florida Rock & Sand Co.	One 2 1/2"	Air
"	General Portland Cement Co.	Contract	
"	Marks Bros., Inc.	Contract	
"	Maule Industries (Pennsuco)	Two 5"	Water
"	Miami Crushed Rock	Contract	
"	Redlands Rock Limerock Pit	One 2 1/2"	Air
"	" " Eureka Pit	One 2 1/2"	A or W
"	Rinker Lehigh Portland Cement Co.	One 5"	Water
"	Southeastern Materials, Inc. (FEC)	One 5"	Water
"	" " " (Rinker Lake)	One 5"	Air
"	Sterling Crushed Stone, Golden Prince	One 3 1/2"	Water
"	Sterling Crushed Stone, SW 144 St.	One 5"	Water
"	Vulcan Materials	One 5"	Water
Lee	Florida Rock Industries, Inc.	Three 5"	Air or Water
"	Harper Bros., Inc.	One 2 1/2"	Air
Morgan	Alonzo Cothren	Two 2 1/2"	Air
"	Toppinno & Sons, Inc.	Three 2 1/2"	Air
"	Upper keys Marine	Two 2 1/2"	Air

effects of business cycles upon them are much more severe. This has created some very marginal situations and in some cases, funds may not be available for new capital investment.

The height of the mast varies considerably. For shallow drilling in canal work, the mast height may be approximately 30 feet measured from the drill platform. In quarry work where the usual depth of blastholes is 50 feet, the mast must accomodate a kelly about 57' long. In such cases, the mast height must be extended. The maximum in current use in South Florida is a 76' mast. At this operation, a 71' kelly is used and the blasthole depth is 65'. This mast is mounted on a Gardner Denver Model 1000 rig.

Water or air may be used to flush the drill cuttings from the hole. Water pumps are generally 5" X 6" duplex reciprocating pumps or larger. When air is used the normal operating pressure is about 30 to 40 psi. Some rigs are equipped to use either method. Industry concensus is that water is superior and that jammed kellys are less likely with water. This is because the very porous nature of the rock being drilled allows the air to push the water out of the rock for a considerable distance beyond the hole. While drilling, air can be seen escaping from adjacent holes and from fissures on the ground surface. At times this action is quite vigorous. When drilling ceases, the displaced water moves back toward the hole and in some cases the hole will continue to bubble for up to ten minutes after drilling has ceased. This water carries particles of sand and rock with it and these can cause particles to lodge between the explosive cartridges and the inner wall of the kelly. This prevents the explosives from dropping to the bottom when the kelly is raised. Air drilling and the re-entrance of water and air into the borehole after drilling also keeps the water and sand agitated which increases the density of the water inside the kelly. This makes it more difficult to get low specific gravity explosives to sink to the bottom. Cartridges have been known to come floating out of the hole after the rig has moved to the next position, and before the hole was stemmed.

When air is used with a core breaker, there is a tendency for the air to become trapped in the upper part of the annulus between the corebreaker bar and the kelly. This can cause a sudden release of air pressure when the swivel is opened and an inrush of sand and rock. To avoid this, one operator has drilled a small hole through the corebreaker near its upper end. This allows the air pressure to bleed back into the rotary hose and compressor before the swivel is opened.

KELLY BAR MANUFACTURE AND OPERATION

At the present time there is only one firm manufacturing dyna-kellys in South Florida. This is Ronnie's Welding and Machine Shop. Cold drawn 41-42 chrome alloy steel tubing with a 5/8" wall thickness is used in making the dyna-kellys. Cold drawn steel is used because the inner diameter cross section is more truly circular. The steel is delivered in approximately 30' sections. To make kellys longer than 30', two sections are welded together by placing a carbon cylinder inside the tubing at the butt joint to maintain alignment. After welding, three grooves, or flutes, are cut longitudinally at 120° to each other. These flutes are between .375 and .39 inches deep with a 13/16" convex. This leaves about 0.25" of metal at the base of the flute to maintain the strength of the kelly. Efforts to increase the inner diameter by reducing this thickness have resulted in torque failure of the kelly. Any reduction in the flute depth would require smaller drive pins and would significantly reduce the useful life of the kelly. The useful life of a kelly comes to an end when the drive pins will no longer remain in the flutes. Because the lower end of the kelly is in the hole more, it receives more wear. The useful life can be increased by about 50% by reversing the ends. Total life with constant use depends on location but the average is about six months.

After welding and fluting, the ends are threaded and a head adaptor is screwed onto the upper end and welded in place. The kelly is then ready for field installation. In use, a sub adaptor with female threads at the top and male threads at the base is screwed to the lower end of the kelly. The subadaptor may vary in length from 6" to 48" and is fitted with wearbars which help to maintain full hole gauge and reduce wear on the exterior surface of the kelly. Bits are screwed onto the lower end of the subadaptor.

The bits have the same inner diameter as the dyna-kelly and subadaptor but even with the best steel, the circular cross sections may be misaligned by as much as 1/16" when made up. The smaller size bits may have four or five fingers faced with tungsten carbide; the larger bits, six to eight. It is important to get the proper grade of tungsten carbide for bit facing because if it is too brittle it will chip off and bit life will be reduced. If it is too soft, it will abrade and bit life will be reduced.

Kelly bar cost at the present time is $57.50/ft. for 3 3/4"OD X 2 1/2"ID, $67.50/ft. for 4 3/4"OD X 3 1/2"ID, and $76.50/ft. for 6 1/4"OD X 5"ID. Head adaptors vary in price from $85 to $130 and subadaptors from $79.50 to $189. Bits vary from $76.50 to $159 each.

When installed, the head adaptor at the top of the kelly is screwed onto the male threads at the base of the swivel. The inner part of the swivel rotates with the kelly when it is driven by the rotary table and drive bushing. The inner part of the swivel is supported by the outer part and there are several courses of bearings which permit the swivel to rotate while the outer part does not. The outer part fits into a yoke which supports the swivel and the drill string. The yoke can be moved up by pulleys and a cable running to the drawworks. It can be pulled down by a single or double chain system. Most drill rigs used in quarries have double chains. The lower end of the kelly passes through a drive bushing which has flutes in the upper part of its inner diameter with depth and orientation similar to the flutes on the kelly. Drive pins are dropped into place when the two fluted surfaces are matched up and these transfer the rotary motion of the drive bushing to the kelly but allow the kelly to move up and down independent of the bushing. The outer surface of the drive bushing has a configuration which permits it to be locked in place when it is lowered into the rotary table. The rotary table is turned by a ring gear and pinion connected to the motor by a drive train consisting of universal joints, drive shaft, transmission, transfer case, chain coupling and a clutch.

SWIVEL CONSTRUCTION

Before describing in some detail the method of loading explosives through the kelly bar, it is important to have a general understanding of the differences in swivel construction.

On all of the large diameter kelly bar and core breaker assembly operations observed by the author, a patented Gardner Denver 1107 swivel was being used. This swivel has a 5" inner opening and the top part of the rotating inner assembly can be separated from the lower part. The general arrangement is shown in the sketch in Figure 10. The two are held in place during drilling by pins which protrude from the lower part and lock into bayonet-type slots on the upper part. On the top of the upper casting there is a fitting with an axial hole which will accept a smaller water swivel which is referred to by its trade name as a "Chiksan". This water swivel permits the entire inner part, top and bottom, to rotate with the kelly while it is supported by the outer part in the yoke. The bottom part of the Chiksan also rotates but the upper part which consists essentially of a 90° elbow, does not. There are two bearings courses between the upper and lower part of the Chiksan along with seals and packing. The rotary hose connects to the upper part of the Chiksan and air or water can then flow through the Chiksan down through the swivel and into the kelly. The upper part of the swivel casting is also designed to accept the core breaker bar on the bottom side, opposite the opening to the Chiksan. Consequently, the air or water flows down through the core breaker bar and enters the hole through an opening in the center of the corebreaker bit.

There are other large diameter swivels of different design.

Small diameter kellys have a different construction and there are several manufacturers. In general, these differ from the large swivel in that the upper part does not rotate and is fastened to the yoke by various arrangements. The upper part has the inlet for the hose without the need for a separate Chiksan water swivel. (See Figure 11) On the upper side of this inlet, axially aligned with the center of rotation of the swivel, is a vertical extension which receives the explosive cartridges. This whole arrangement on the upper part of the swivel, the inlet for the hose and the loading pipe extension, is called the loading tee. If air is used for flushing, as it usually is on the small diameter drills, a quick opening loading cap is used at the open end of the loading pipe. This loading cap has a self locking hinge assembly with sufficient resistance to remain closed when air is used. If water is used, then a screw-in plug with a hexagonal configuration at the top is normally used. Damco swivels with

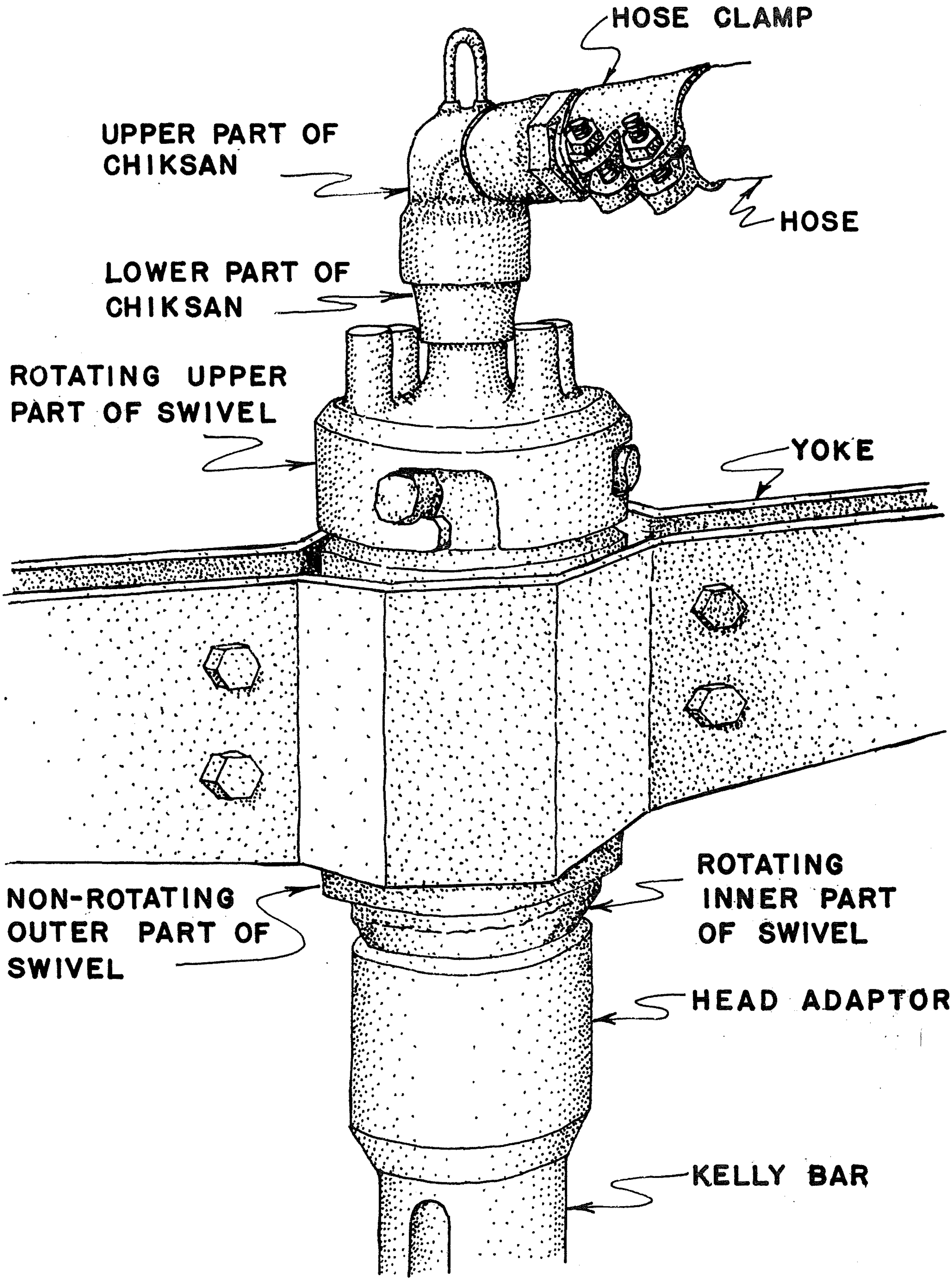


FIGURE 10

SKETCH OF THE GENERAL ARRANGEMENT OF A TYPICAL LARGE DIAMETER SWIVEL, CHIKSAN, YOKE AND HEAD ADAPTOR

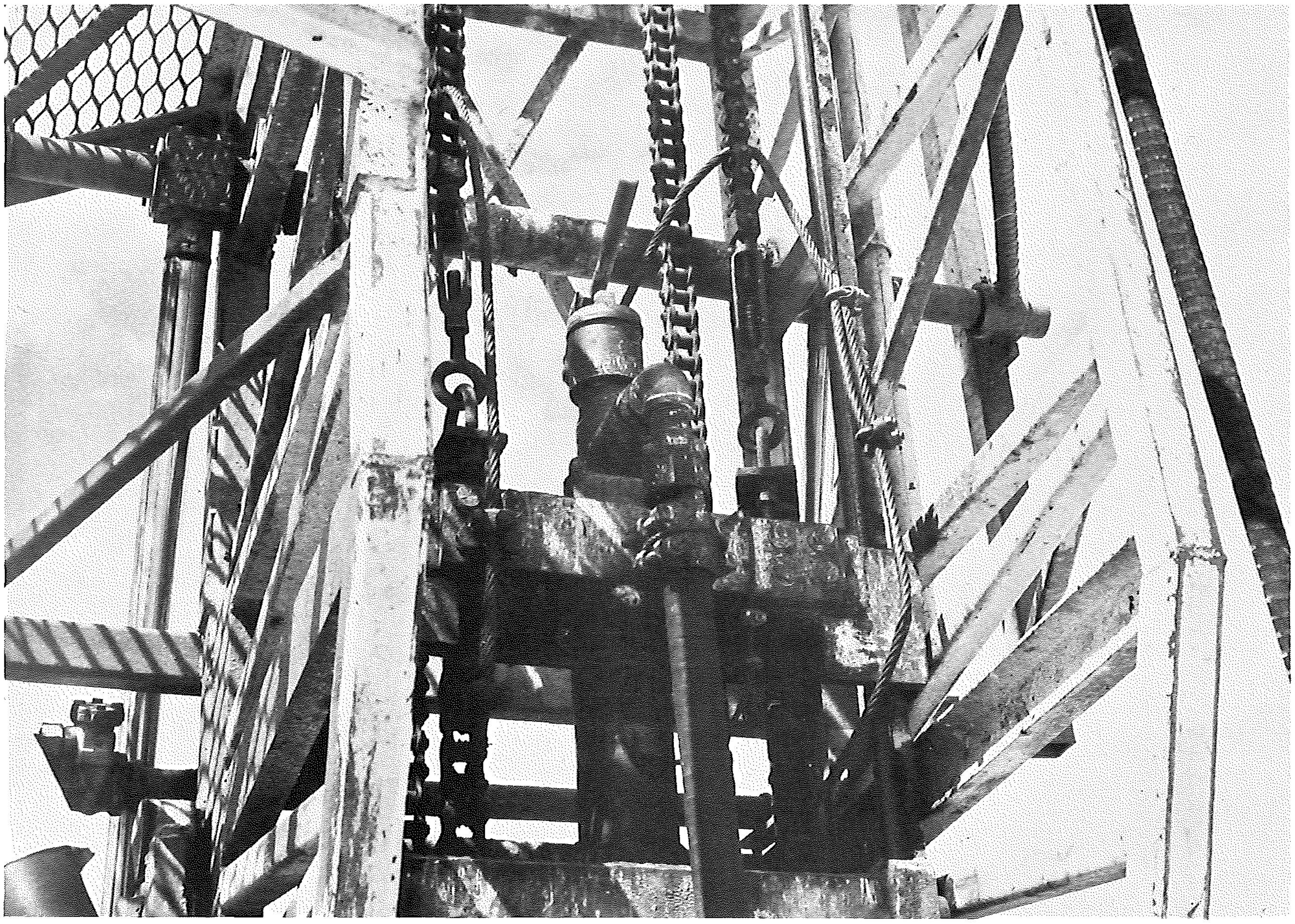


FIGURE 11

Typical small diameter swivel
and quick opening loading tee.

a 3 1/2" opening are available, and there is a King swivel made for Davey rigs with a 3 3/4" opening. The Gardner Denver 807 swivel, which is commonly used on the small diameter rigs instead of the larger 1107 swivel, has a 2 1/2" opening and the thickness of the metal is such that it can not be enlarged to accept 3" diameter material. The swivels with 3 1/2" to 3 3/4" openings can be adapted with proper design and manufacture of the head adaptor, to accomodate a 3" inner diameter kelly. This can be done in local machine shops.

LOADING EXPLOSIVES THROUGH THE KELLY BAR

When explosives are to be loaded through the kelly bar, the hole is first drilled to the desired depth. In doing this, the swivel moves down through the mast until the head adaptor comes to rest on the stabilizer which in turn rests on the drive bushing and rotary table. In this position, the swivel can be reached by the driller and his helper from the drill platform. If a large diameter kelly is used, the upper part of the swivel can be unlocked and hoisted with the core breaker bar until the core breaker bit is several feet above the lower part of the swivel. The core breaker assembly is moved to one side of the opening in the swivel, and then an explosive cartridge with detonating cord threaded through a hole in the cartridge and tied at the top is lowered down through the swivel opening into the kelly below. The detonating cord usually plays out from a spool. Additional cartridges are then dropped in the opening until the total charge weight is placed. The detonating cord is then attached to the drag as described in the section on historical development and the cord is cut. The corebreaker assembly is then guided by the helper to go into the swivel opening and is lowered slightly. The kelly is then raised up and around the core breaker assembly. If the line holding the corebreaker should fail at this time, the corebreaker would fall to the top of the explosive column and possibly detonate it. If the explosives do not pass through the bit, they will be raised with the kelly and be pressed against the corebreaker bit. This may push the cartridges past the bit and clear the jam, or it may detonate the explosives, or it may start lifting the corebreaker. If the latter happens, the driller may see the core breaker line go slack and stop hoisting the kelly. At this point the driller is faced with a real problem. If he lowers the kelly in an effort to free the inner bar he will be pressing the kelly and bit against any explosives which may have cleared the bit and dropped into the hole. If he continues to hoist the kelly, the jammed explosives may detonate at any time and the core breaker can only move upwards a few feet before the swivel hits the crown pulley at the top of the mast. If the driller should decide to take a chance and lower the kelly, he may find that the kelly will not go back down the hole because of rock sloughing in the hole. Fortunately this series of events doesn't happen often because there are only two alternatives at this point because it cannot be abandoned. If most of the corebreaker is out of the kelly, there might be enough flexibility in the core breaker bar to permit a crane to hook onto the top and work it forward and around the top of the mast. If this is done, the kelly could then be raised in the mast, the mast lowered and the jam washed out with water. If this cannot be done, the top of the mast must be removed in order to raise the core breaker. The only other alternative is to saw or cut the core breaker bar a few feet above the top of the kelly and pick it up with a crane and remove it. Then lay down the upper part of the corebreaker bar, hoist the kelly up in the mast, lay the mast down, remove the lower part of the corebreaker, and wash out the kelly. Such situations are unusual but the potential risk is there. Normally, the kelly moves upward around the core breaker without incident until the lower part of the swivel contacts the upper part. Rotation of the kelly then causes the two parts to lock together for subsequent drilling.

When a small diameter kelly is being used, a quick-opening loading cap is normally used for loading rather than to open the swivel. This loading tee was described in the previous section.

After the hole is drilled, the kelly is raised a foot or two and the air or water pressure is checked for indications of a jam. If not, the kelly is lowered, the air or water pressure is allowed to bleed off and then the lever on the cap is raised. Detonating cord is tied to the first stick of explosives and is dropped down the Chiksan. 25 or 30 grain cord is normally used, but operations using water gel slurries employ a 50-grain seismic cord which is similar to standard 50-grain cord except that it does not have a waxed fabric cover over the plastic. The cord plays out from a spool as with the large diameter operations.

After the explosives are loaded through the opening, the cord is cut, the drag fastened to the cord, and the quick action cap is locked in place. The kelly is then raised and the explosives remain in the hole.

The driller's helper watches as the bit comes out of the hole to retrieve the end of the detonating cord. If the detonating cord does not appear, the cord has either come loose from the drag and dropped in the hole, or the drag has failed to work and cord and drag are in the hole, or the explosive did not come out of the kelly. The next step is to examine the opening in the bit and see if there is a rock jam which would retain the explosives in the kelly, or possibly retention of the cartridges by mushrooming of the packaging, looping of the detonating cord, or cuttings carried up past the column by the returning air and water action. Safety procedure at this point requires that the mast be lowered before clean out attempts are made. Some operators use the levelling jacks to raise the rear of the platform so that the top end of the kelly is pointing slightly downward. The swivel can be opened and water can be pumped into the bit end of the bar. In most cases, the upper end of the detonating cord can be recovered and the explosives removed with a gentle pull. If not, additional water flushing can be continued. One operator has mounted a water tank, pump and hose on a pickup truck for the purpose of cleaning out explosive jams. Relatively high pressure is developed which can be directed into the packaging, washing the dynamite out.

Such jams are relatively infrequent. Some operators report having only one jam in the past ten years. Others indicate about one per year. Some measure the interval between jams in months. Such differences are probably related to the difficulty in removing the jam, which would have a bearing on whether if was reported or not, and to the geologic strata, the type of equipment being used, and/or the experience of the driller. A reasonable estimate is that most operations probably experience about two jams per year.

A more frequent occurrence is for the loaded kelly to get hung up in the hole. Minor hang ups are a daily matter and it is difficult to spend much time around an operation without seeing the driller rotate the kelly to free it. Rotation is kept well within the 5 second limit with the 30 second intervals between rotational effort.

The drawworks on most rigs is capable of pulling as much as, or more than, most cranes. Thus, if the kelly cannot be freed by pulling and rotation, it probably is not going to be pulled free by a crane. There have been instances where the kelly was sawed off at ground level. In another case, where there were no explosives in the kelly, the kelly was apparently broken off at ground level because the rig was rocked back and forth on the tracks in an effort to free it after pulling and rotation had failed.

In January, 1976, some safety rules for drilling and loading explosives with kelly bars were developed through a series of meetings between MESA and members of the Engineering Contractors Association. These rules have been distributed and are generally adhered to by both members and non-members.

BLASTING ROUND DESIGN

In Dade and Broward Counties, design appears to be generally informal and results primarily from discussion between the operator, his pit personnel, and the explosive manufacturer's technical representative.

In most cases, the initial design appears to be based primarily on a rule-of-thumb load factor of 0.56 pounds of explosive per cubic yard of rock. This may be later modified by the operator depending on the results obtained. Most of the operations have been in existence for many years so the present average load factor gives a fair idea of industry experience when explosives are loaded through the kelly bar. This average load factor is 0.53 pounds per cubic yard, with a standard deviation of 0.10.

A stemming interval is then selected, usually 7 or 8 feet where 2" explosives are being used, and 15 to 20 feet with 4" powder. On the average, the stemming interval is 0.76 times the burden. The operator may use drill cuttings and carefully hoe these back into the loaded hole. Others simply move to the next hole and depend on the usual sloughing of the upper part of the hole and the everpresent water column to do the job. The average for those using drill cuttings is a smaller multiplier (0.61 times the burden) than for those who make no attempt to stem the hole (0.86 times the burden). No one was found using a coarse sized (1 to 1 1/2 inch) aggregate as stemming although several use a loader to bring fines to the loading area when there are insufficient drill cuttings.

When the stemming interval has been selected, and where cartridged explosives are being used, the charge weight is usually determined by the remaining length of the hole. The length of the hole is determined by the depth to clay, or the depth to layers with less than 70% carbonate, or the length of the kelly bar and the capability of existing drilling equipment, or the capability of the dragline, or zoning restrictions, or a combination of these factors. The average depth of the blastholes in Dade and Broward Counties is 47.9 feet with a range from 31 to 65 feet. The standard deviation is 8.9 feet. With the charge weight and the hole length known, a burden and spacing can be determined which will give a load factor of approximately 0.56. Multi-row square, or nearly square patterns are the most popular. Multi-row staggered patterns are used in only 15% of the quarry operations in these two counties. All operators in these two counties shoot a rectangular shot off the corner as they proceed down the length of the pit.

Where water gel slurries on continuous plastic links are loaded, some operations can load the entire length of the kelly and thus increase the charge weight because the soft packaged explosives will slump in the much larger hole to a considerable degree.

70% of the operators maintain a "buffer" of shot rock between the first row of holes and the open water. The width of this buffer is usually equal to the burden. Consequently, the distance from the first row of holes to the water is twice the burden. This buffer is maintained to contain the shot and keep the rock from going out in the water beyond the reach of the dragline. Some operators feel that it is also effective in creating a resistance against which the rock can be crushed by the blast.

Delay patterns are so varied that it is impossible to make generalizations as to industry practice in South Florida. Apart from the ground vibration and air concussion problems, many operators make an effort to "hold" the first row until late in the delay pattern. This reinforces the effect of the buffer; if no buffer is present, it provides a substitute. About 30% of the operators use a standard echelon pattern. Others shoot a row, or a half row, per interval and this may be delayed to heave toward the lake, or toward the shotrock of the last shot. Where there are vibration problems, a separate delay will be used on each hole, generally starting from the lake side of the shot. Where air concussion is not a big problem, the detonating cord downlines from the holes in a row will be tied together with a cord trunkline and initiated by some operators with electric blasting caps at the end of the lines. Others will use millisecond delay connectors between rows and utilize a single electric blasting cap for initiation.

In Collier and Lee counties, blasting is only necessary to break up a relatively thin cap rock. The thickness of this cap rock varies from 10 to 30 feet. Where it is only ten feet, operators will generally drill the holes and then load the open holes later rather than to load through the kelly. Elsewhere, the explosives are loaded through the kelly in much the same manner as in Dade and Broward counties. Where there are vibration problems, design is determined by local government restrictions on the size of the blast. If the restriction is for a total charge of forty sticks of 2" explosives and there are ten holes in the shot, then there is a charge of four sticks loaded in each hole. If there are eight holes, five sticks are loaded. A separate delay is used for each hole.

Prior to the experimentation performed under this contract, no sequential blasting timers had been used in South Florida.

DRILLING RATES AND EFFICIENCY

Drilling records for all of 1976 pertaining to the quarry industry in Dade County were analyzed. This comprises 1,240,002 feet of drilling history at 14 locations. Daily rates can be affected by many factors and these are not indicated in the records. Weather may cause a shutdown, or the equipment can break down sometime during the day, or personnel may not be present for the whole workday, or the length of the workday may vary. These factors will influence the results but they should tend to be present to roughly an equal degree within each of the groups averaged. A ten-hour working day for the drilling and blasting crew is fairly standard. Weather on any given day has a fairly similar effect on all operations.

1976 was a year of low activity for the industry but the effect of this was to reduce the number of days worked. The rate per day should remain fairly consistent regardless of number of days worked.

Average daily drilling rates were determined for each rig employed in the quarry industry and then these were grouped according to the kelly bar ID to determine a group average. These figures are:

Kelly ID	Average number of feet drilled daily
5"	942 feet
3 1/2"	1194 feet
2 1/2"	1542 feet

To determine the relative productivity of the different size groups, the drilling rate for each rig was multiplied by the number of cubic yards per foot of drilled depth that could be attributed to that drill considering the spacing and burden at the location being drilled. This yielded the following productivity rates:

Kelly ID	Average cubic yards per day
5"	10,147
3 1/2"	5,900
2 1/2"	5,001

Admittedly, these figures do not tell the whole story because there are significant differences in blasthole depth, the nature of the strata being drilled, and the efficient use of powder factors at the various locations. Nevertheless, some of these effects are offset in the averaging and the results clearly show that the larger diameter operations are more efficient.

This raises the question as to why all operators don't switch to large diameter operations. The answer is that the larger charge weight in the large diameter holes creates too much ground vibration for the adjacent neighborhoods. There is no way to deck the explosives in a hole loaded through the kelly bar without loading an electric blasting cap through the kelly. Consequently, if the column length is to be maintained, the only way to reduce the charge weight is to decrease the size of the explosive diameter.

PREVIOUS EXPERIMENTATION

There has been considerable industrial and governmental effort to develop alternatives to the kelly bar method of loading explosives prior to this project.

Hercules Incorporated, with funding by the U.S. Bureau of Mines, conducted a project to find an alternate approach. This work consisted of a series of tests at five quarries, and one construction site (Maule Industries, Inc., Sterling Crushed Stone Co., Miramar Rock, Coral Aggregates Corp., Southeastern Materials Inc., and Troup Bros.) performed during the latter part of 1975 and the first part of 1976. Primary effort was directed toward use of a Hercules product called Corflo™. This is a flexible plastic duct that can be shipped and stored in a continuous roll. The concept was to load this material through both large and small diameter kelly bars and then load the Corflo™ cased hole later with explosives after the drill crew had moved on. Hot-waxed treated paperboard tubes and phenolic resin treated tubes (Kemax tubes) were also tested after some problems developed with the Corflo™ concept. In general, Hercules had difficulty with placement of small diameter Corflo™ tubing; loading explosives into the small diameter tubes; loss of drill productivity because of reduced loading of holes; softening of the heavy walled spiral paperboard tubing with attendant loading difficulties; and product contamination with the non-degradable plastic.

Gulf Oil Chemicals Company and Vulcan Materials Company by early spring of 1975 developed a system of pumping aluminized NCN slurry into 4 1/2" ID hot-wax dipped, spiral paperboard tubes which had been placed in the holes through the kelly. By the end of 1975, 102,000 pounds of the pumped slurry had been tested and the system was operational at Vulcan's quarries in Dade and Broward Counties, Florida after some difficulty was experienced in getting the proper strength tube for the more unstable geologic situation at the Broward County operation. By October, 1976, Vulcan claimed that the new system had resulted in a cost reduction for drilling and blasting because of expansion of the borehole pattern, smaller drill crew, and lower explosive cost. In the development of this system, 4" cartridged products were loaded in the tube casings with limited success before the pumpable slurry was developed. The pumped slurry was preferred because it provided higher borehole loading density and could pass by any egg-shaped cross-sectional areas of the tube.

Atlas Powder Company conducted a series of pumped slurry experiments with hot wax spiral tubes and Kemax tubes at Sterling Crushed Stone Company's quarry in early 1975. Details of the results were not made public.

In addition to participating in the Hercules program, Troup Brothers, Inc., also experimented with a retrievable aluminum casing which could be placed through the kelly and loaded after the drill had moved off. After loading, the casing could be recovered. The additional time required for this operation made it uneconomic and did not sufficiently eliminate the safety problems.

Redlands Construction Company placed some cardboard tubes of unknown specification and manufacture, and loaded the tubes with some site-mixed-slurry. They concluded that the tubes hung up too frequently for the method to be operationally feasible.

Trojan Powder Company and L.W. Rozzo, Inc. experimented with plastic, screw-together, explosive cartridges without successful result.

Florida Rock Industries, Inc. used the hot wax paperboard tubes for two shots but because of the limited depth (30') of their blasting they felt that operational use of the tubes was not justified because they could not increase their borehole pattern to absorb the cost.

There has been other experimentation in the drilling and blasting associated with real estate development. Pumping slurry into open holes was reportedly attempted and the slurry went out into the formation and extended the area of the shot well beyond the design.

Several operators have attempted to load cartridged products in open, uncased holes after the drill has moved off. In some places, this might be operationally feasible as in Key West. At some other locations, a fair percentage of holes (say 50%) might be loaded in this manner. This does not provide an answer for the other 50%, or for those locations where open hole loading is impossible.

After Gulf Oil Chemicals Company's success with pumping slurry into large diameter tube casings at the Vulcan Materials quarry, they turned their attention to developing a similar system for the small diameter operations. Four series of tests were conducted at Capeletti Bros., Pit #13. These utilized a 2.243" ID Kemax tube with internal couplings which was placed in the hole through a standard 2 1/2" ID kelly. Gulf GX-19 slurry, an aluminized NCN product, was delivered to the site in 300 lb. drums. These were dumped into a hopper and the slurry was pumped into the tube casing by running the hose to the bottom of the hole and letting it pump itself out. The first test was in September, 1975, followed by a two shot series in October, 1975. A two shot series was conducted in October, 1976, and another two shot series in December, 1976. ECA/USBM Committee members were invited to witness the last two series and all data were made available. The second shot of the October, 1976 series gave excellent surface results with superior heave and swell. The other shots in these two series presented very marginal results. Nevertheless, it appeared that Gulf was making significant progress and that an operational method might soon be forthcoming.

Unfortunately, when two of the latter series of tests were finally dug with the dragline in the summer of 1977, there was a lot of oversize material and hard digging below a depth of 35 feet.

In conjunction with one series of these tests, Gulf loaded 2" X 24" cartridged dynamite with a 25 grain detonating cord downline into a 2.243" tube casing (0.045" wall thickness and with internal couplings). It would not go down the casing by gravity but had to be poled down with a tamping rod.

TRAINING OF PERSONNEL

The level of training for drillers and blasters in South Florida is an uneven matter as it is in many places. Some drillers have many years experience in operating different pieces of equipment. Some have many years experience operating the same piece of equipment. And some have only a few months experience as driller's helpers before they take over the responsibility of drilling and blasting. Training is primarily a one on one on-the-job situation, with instruction received from a more experienced driller. Once the basic operation of the equipment is mastered along with the loading and hookup procedure, there is virtually no more instruction.

As to safety instruction, the E.C.A. rules have been given fairly wide circulation and most drillers are familiar with them. Few have attended a formal course in the safe handling of explosives.

The job is one of the least desirable in the quarries because the operation is frequently conducted in several inches of water and there is the risk that is always persent in handling explosives. Most accept the danger as a necessary risk. The pay scale is generally better than many other classifications in the quarry but this is largely offset in the minds of many by the hazards involved. Basic pay for a driller is in the neighborhood of $6.00 per hour with the drillers' helper getting about $4.00 per hour. The latter is comparable to the pay for a quarry laborer. Most crews work a ten hour day and a four day week is common, particularly during slack times.

Most of the operations consist of two men, but three-man crews are common.

In a small diameter operation, the crew is responsible for the efficient application of about $1500 in operating and owning expense each day. ($850 in explosives and $650 in drilling cost.) Large diameter crews largely control the application of about $4100 per day ($3400 in explosives and $700 in drilling cost). Considering a 200-day work year (2000 hours divided by 10 hours) these individual crews are responsible for the proper application of from $300,000 to $800,000 of operating and owning expense per year. Apart from the safety aspects, this would seem to justify activities that would make sure that driller/blasters thoroughly understand and practice good hole layout procedures, proper blasthole pattern design, effective delay systems, and preventive maintenance, to mention a few items.

Blasters are required to have a license from the State of Florida, Department of Public Safety. Applicants are screened and fingerprinted, and if the review board feels it is necessary, a test may be required. The cost of a State license is $25 per year.

Dade County also requires a license. There is no specified period of experience but letters are required which indicate the past experience and character of the applicant. The cost of a blaster's license in Dade County is $25 per year. If the individual doesn't have local experience known to the examining officials, he may be required to take a written examination.

In Broward County, no letters of reference are required but all applicants must take a written examination. Cost of a blaster's license in Broward County is $250 per year. Handlers (driller's helpers) must also have a license and this costs $50 per year.

Collier, Lee and Monroe Counties all have blasting ordinances and these require licensing of blasting personnel.

As with most licensing, these procedures seek to insure that the licensee has at least a minimum level of competence and knows the regulations.

HAZARDS IN THE KELLY BAR LOADING METHOD

There are a number of hazards that are created by loading explosives through the kelly bar. Chief among these is the fact that two high risk operational functions are being performed as though they were one, i.e., drilling and loading. Drilling is a relatively complicated task and requires the driller's attention to many parts of his machinery and to the drilling action. This activity is done in the midst of holes that have been loaded with high explosives (semi-gelatin dynamite, 60% extra dynamite, or sensitized water gel slurry) and detonating cord downlines. All of these materials can be detonated by impact and the porous, water saturated strata create conditions which are conducive to propagation. Problems which arise in the drilling function (equipment breakdown, rig overturning, injuries, hangups, fires, etc.) are worsened by the fact that they occur within 8 to 20 feet of a charged hole. There can be no argument with the conclusion that nearby charged holes present an additional measure of risk to drilling over and above that which pertains to drilling where no explosives are near the site.

All of the other hazards are intensified by this proximity.

The safety rules developed by E.C.A. and MESA sought ways to minimize or eliminate these hazards, but people can be unaware, or forgetful of the rules and bring about disaster. In defense of the kelly bar, it is commonly said in South Florida that the kelly bar never killed anybody, that it was the ignorance of the people trying to use it improperly. This, of course, is precisely the point. If the potential is there for someone to use it unsafely, albeit improperly against the rules and common sense, there is the potential for an accident.

In evaluating possible alternatives, it is essential that they eliminate the hazards associated with kelly bar loading. Therefore, a list of the principal hazards in addition to the proximity problem is given below, but not necessarily in order of importance.

Hoist cable breaking and dropping inner bar on explosive column.

Brake or clutch failure dropping inner bar on explosive column.

Brake or clutch failure dropping kelly down hole on top of explosive column.

Rotating drill steel to free it from hang ups when coming out of the hole. This can generate heat which can detonate column.

Using air or water pressure to help free explosives hung in drill steel. Rapid expulsion could result in detonation.

Drilling into a charged hole by not keeping the holes vertical.

Explosives retained in the kelly because of:

(1) a rock jam in the bit

(2) looping of detonating cord around cartridges during rotation.

(3) a cartridge mushrooming at one end causing it to stick in the kelly.

(4) misalignment of inner diameter of kelly, subadaptor and bit.

(5) a rock particle between a cartridge and the inner wall of the kelly.

Looping of detonating cord around bit teeth during rotation causing it to be abraded, impacted and heated against the rock wall of the hole. Looping of the detonating cord around bit teeth during rotation, causing bottom cartridge to be pulled up through column.

Moving hung drill steel down the hole onto explosive column in attempting to free it.

Hammering on pipe to free jammed explosives in kelly.

Lightning striking the drill rig mast after explosives have been loaded.

Breakdown when coming out of hole requiring repair while loaded kelly is partly in the hole. This is usually the chain.

Detonating cord dropped down hole so that hole can't be tied in.

Probing jammed explosives with a rod, ferrous or nonferrous.

Spark and fire potential during loading.

Moving a crawler track over a charged hole or a downline.

Sawing a charged, hung kelly off at ground level.

Cartridges falling into sidewall cavities and failing to detonate posing a potential risk to the dragline operator.

Cleaning out a jammed kelly.

No certainty that stemmed interval is adequate which can cause excessive flyrock.

OTHER PROBLEMS

There are some other problems with the kelly bar method of loading which relate to the managerial control of the operation and the efficient use of explosives.

If something cannot be controlled, it cannot be managed. This is pretty much the case with the kelly bar method of loading. The size of the shot is determined by the number of holes that can be drilled in one day. This depends on the weather, the condition of the equipment, whether employees show up for work on time, and a multitude of factors beyond the control of management. Management may want to shoot 30 holes at a time because they seem to get better movement and breakage with a shot of this magnitude, or because this reduces the number of times a dozer or loader has to travel to the blasting area to level up the shot. It cannot be done consistently with the kelly bar method. If only 20 holes can be drilled and loaded by 4:30pm, then the shot is going to be 20 holes because it must be shot by 5:00pm inasmuch as local ordinances do not permit blasting after 5pm and charged holes cannot be left overnight without a guard at the site. If only 10 holes can be drilled, then the shot will be 10 holes. In other words, management cannot decide the size of the shot. The size of the shot is decided by the method. By denying this option to management, it eliminates a major area of possible cost reduction.

Associated with this managerial limitation is the fact that the method dictates the level of capital expenditures in drilling equipment, or production levels. If only a certain number of holes can be drilled and shot by 5:00 pm and sales indicate that more production is desirable, the operator must either contract for another drill, buy another drill, or pass up the opportunity for additional sales. Some might add a fourth alternative, perhaps with justification, get a new driller. As mentioned previously, when sales improve the services of contract drillers quickly reach a point where demand exceeds the supply. Thus, the method dictates that another rig must be purchased or leased (long term commitment), or pass up the increased sales (probably short term opportunity). Other methods might offer a more acceptable third alternative; use the drill rig for another shift but load and blast only during the permissible daylight hours (adjustable to long or short term needs without commitment of new capital).

In addition to these limitations on management control, there is the more serious limitation on optimizing the blast design. In the hole loaded by the kelly bar method, utilizing simple concepts of blasting design such as explosive column length and stemming interval is not really possible. No one knows where the cartridges are except that they are somewhere down in the ground.

This is because the hole is drilled with a bit that has an outer diameter considerably greater than the diameter of the explosive. Where 2" diameter explosives are loaded, the hole is drilled with a 4 3/4" OD bit. The hole is probably slightly larger than this because of bit wobble. If it is 1/2" larger, say 5 1/4", it will accomodate four 2" diameter cartridges side by side. If this happened throughout the column, a planned build of 40 feet with 8 feet of stemming might actually be 10 feet of build and 38 feet of stemming. Needless to say, this might produce good breakage at the bottom of the hole but the rock in the upper part probably would require a lot of secondary breakage. It is rather pointless to try to optimize a 30-hole shot pattern when any six of the holes might have a 42' explosives column, six might have a 30' column, six might have a 24' column, six might have a 16' column and six might have a 10' column, or whatever, but not to exceed 42 feet. Considering that each hole represents about $40 in drilling and explosives cost, it seems that it would be worthwhile economically to devise a method in which each hole is loaded in the same consistent manner.

Many believe that as the cartridges drop out of the kelly and pass through the bit into the hole, sand and rock fill the annulus between the 2" diameter cartridge and the 4 3/4+" diameter hole. This is possible, and probably does happen at times. No one knows how often.

There is the question of the partial vacuum at the top of the kelly when it is pulled out of the hole. Does it tend to retain the water and the cartridges in the kelly in a manner similar to water being held in a soda straw when one holds a finger over one end and draws the other from a glass of water? Some drillers report that they have seen cartridges and water drop out of the end of the drill string and into the hole after the bit has come out of the hole. To what extent does this effect destroy the effectiveness of the $40 per hole expenditure?

In an effort to get some measure of this effect, a 1/16 scale model of the 2 1/2" ID kelly loading situation was made. (2" X 24" cartridges in a 2 1/2" ID kelly in a 5" hole.)

Twenty-two plastic "cartridges" measuring 3.175mm X 38.1mm (.125" X 1.5") were "loaded" into a 4mm (.158") ID glass tube representing the kelly. This tube was inside an 8mm (0.315") ID glass tube representing the hole. Both tubes were filled with isopropyl alcohol. The first "cartridges" sank to the bottom in about 60 seconds which is approximately the same time as it takes for a 2" cartridge of dynamite to drop to the bottom of a 50' tube-cased hole.

The experiment was repeated about 50 times but one could obtain about any desired effect depending on the rate of withdrawal and the extent of partial vacuum at the top of the "kelly" and the moment of release. If the column became discontinuous, it failed in nearly every case. A "perfect" column was 33" long. Collapse of the column resulted in a length which was not less than 15".

The experimental results were not treated statistically because it was too easy to bias the results unintentionally, and surface tension effects probably masked any direct relationship to the kelly bar loading. Nevertheless, the experiment pointed out how little is probably known of the distribution of explosives in the blastholes of South Florida, and that a substantial amount of money and energy is probably being wasted.

Efforts to measure the actual top of the explosive column in real blastholes were inconclusive. Most times, the tape would go only about 3 or 4 feet below the surface; much less than the minimum stemming interval. In other cases, it was impossible to tell whether the weight was resting on explosive or rock. Usually it appeared to be rock.

There was insufficient time to conduct another experiment which would probably give a statistical probability of column collapse. This would involve tying a 50' length of twine to the top cartridge and having the twine play out through the loop in a V drag, such as that used to retain the end of the detonating cord in the kelly. The length of twine on the ground could be measured and subtracted from 50' to determine the depth to the top stick. This would not necessarily yield the depth to the top of the column, but it would indicate the likelihood that the column had collapsed.

The principal advantage of the kelly bar method of loading is its speed. Drilling and loading in South Florida is judged primarily by the number of holes drilled per day. As long as the surface results are acceptable, and the apparent gradation of the shot rock surge pile doesn't change, the daily drilling rate usually will be the only parameter of drilling and blasting performance measured by operators. Consequently, because of its speed the kelly bar is considered efficient without regard to the powder factor it requires or the effect on downstream crushing rates.

CHAPTER 4

UNIT TESTS OF ALTERNATIVES

SOLICITATION OF PROPOSALS

After assembling some state-of-the-art data, the first step taken in this project was to seek out proposals from industry for possible alternatives to the kelly bar method of loading.

A letter was prepared which outlined the problems and hazards and the desire to develop a safe and economical method. The letter invited the recipients to an informational meeting in Miami at which more information would be given.

A mailing list was compiled from the buyer's guides published by Engineering & Mining Journal, Rock Products, and Construction Equipment magazines. Manufactures of explosives, detonators, drill rigs, drill accessories, excavating equipment, rubber, glass, aluminum, ceramic and paper products were included. All members of the Engineering Contractors Association were included. The list consisted of 255 addresses. The letter was mailed to all of these parties.

The meeting was held on November 15, 1976, and was attended by 105 persons. A one hour slide presentation was made. Requests for Proposals were given or mailed to interested parties. Proposals were to be received by December 17, 1976.

Proposals were received from Gulf Oil Chemicals Company, Atlas Powder Company, Ireco Chemicals, and Sonoco Products. All of these proposals offered to perform tests using pumpable slurry in expendable cardboard tubes, except for Sonoco's proposal which was concerned with development of a storage rack to facilitate the loading of the tubes down the kelly. The Gulf and Atlas proposals visualized tube casings which could be placed through a standard 2 1/2" ID kelly bar. With Gulf, the bulk slurry would be delivered to the site in a truck with a large hopper and pump from their facility in Brooksville, Florida. A total of three tests were planned.

Atlas planned to ship their slurry in a truck with a tank and a pump unit from Tamaqua, Pennsylvania. Alternative facilities would permit bagged slurry to be dumped into a small portable hopper and then pumped into the tube casings. Up to twelve 10-hole tests would be performed.

The Ireco proposal was for a 30 to 60 day series of tests with delivery from a local staging plant. Slurry would be pumped into both large and small tube casings.

The ECA/USBM Technical Committee decided to proceed with the Gulf and Atlas offers. The Ireco proposal was so large that it would have left little money for testing other possibilities.

It was decided that the Atlas tests should be performed at Coral Aggregates Corporation pit because this would test the feasibility of using pumpable slurry in small diameter tubes at a depth of 60 feet.

The Gulf tests would be performed at Capeletti Bros., Pit #13 to provide continuity with the other tests that Gulf had conducted on their own account and a good basis for comparing results.

Sonoco's proposal was deferred to a later stage in the program because it represented a refinement beyond that sought in the preliminary tests.

Because no proposals were received which addressed the problem of loading packaged products into expendable tube casings, the Committee authorized a series of tests to be designed and arranged by the Project Manager. Discussions were held with the General Manager of Miramar Rock and their quarry was selected for these tests because it offered a test area with moderately adverse geological conditions and where there was no buffer to contaminate the results and the shot rock would be dug at an early date. There was also an inactive contract rig at the location which could accept tubes up to 3" ID.

TESTS AT MIRAMAR ROCK

It was decided to try two new types of tubes at Miramar. The first was a Kemax* tube with 2.493" (63.32mm) ID and a 0.060" (1.524 mm) wall thickness with external wall couplings. The thickness of the material used for the couplings was only 0.045" (1.143mm). The external coupling consisted of separate tube material 10" long, which was glued to the outside of the main tube. This helped maintain an outer diameter of the assembled package of 2.703" (68.66mm). With the couplings on the outside, the tube was internally flush and it was felt this offered less resistance to the downward movement of cartridges.

The second type was a hot-wax dipped paperboard, spiral wound tube with 2.730" (69.342mm) ID and 0.100" (2.54mm) wall thickness with swaged-end couplings. These couplings are formed by compressing the end of the normal tube so that the normal outer diameter is reduced to form a snug fit in the inner diameter of another similar tube. The fit is designed so that as the coupling is put together at the top of the kelly, the female part of the coupling is on the bottom, and the male part is on the top. When fitted together, the inner diameter at the swaged coupling is approximately 2.5", which is sufficiently large to permit 2" cartridges to pass.

At Miramar, 3" packaged products are normally loaded into the holes through the kelly bar. Details of hole depth, spacing, burden, stemming, and type of explosive normally used at this quarry is summarized along with similar details for the test series in Table 3.

*Kemax is a trade name used by Sonoco Products Company for a fibre material impregnated with phenolic resin by vacuum-pressure.

TABLE 3

SUMMARY OF TESTS AT MIRAMAR ROCK

	Normal Quarry Use	Test No.1	Test No.2	Test No.3	Test No.4
Type of tube	None	Kemax	Hot Wax	Kemax	Kemax
Burden & Spacing	12' X 12'	8.5' X 8.5'	8.5' X 8.5'	9.5' X 9.5'	11.5' X 11.5'
Stemming	7' - 8'	6'	6'	7'	7'
Hole Depth	50'	50'	50'	50'	50'
No. Holes	21	16	16	16	16
Explosive Name	Power Gel D	Powel Gel D	Power Gel D	Tovex 255	Tovex 255
Explosive Type	Semi-Gel Dynamite	Semi-Gel Dynamite	Semi-Gel Dynamite	Water Gel	Water Gel
Cartridge Size	3" X 24"	2" X 24"	2" X 24"	2" X 16"	2" X 16"
Cartridge Type	Rigid	Rigid	Rigid	Linked plastic	Linked plastic*
Effective diameter in hole	3"	2"	2"	2.1"	2.5"
Detonation Velocity	16,500'/sec	16,500'/sec	16,500'/sec	16,000'/sec	16,000'/sec
Pounds/Hole	175	83.6	83.6	78	110
Number of sticks	21	22	22	38	53
Cubic Yards/Hole	267	134	134	167	245
Load Factor lbs./CY	0.65	0.63	0.63	0.47	0.45
Weight Energy x 10^6/CY	0.65 ft.lbs.	0.63	0.63	0.65	0.63
Initiation	EB cap with detonating cord downline	EB cap with detonating cord downline	EB cap in top stick	EB cap with detonating cord downline	EB cap with detonating cord downline
Booster	None	None	None	None	None
Loading time/hole	100 sec.	100 sec.	100 sec.	163 sec.	450 sec.
Tube placement time/hole	---	125 sec.	125 sec.	125 sec.	125 sec.
Density of explosives	1.4	1.4	1.4	1.25	1.25

*Plastic links were slit and the explosive removed before dropping in hole. This permitted the explosive to fill the entire 2.5" ID of the tube.

For control, the first two shots at Miramar utilized the same semi-gelatin dynamite, Hercules, Power Gel D, used by the quarry operator but the spacing and burden were reduced to maintain an explosive weight energy per cubic yard of stone approximately equal to that normally used (0.65×10^6 ft-lbs per cubic yard) with the larger diameter explosive. In the first test, a 25 grain detonating cord downline was used with an EB cap at the top of the hole. In the second test, there was no downline and only an EB cap in the top stick was used.

In the first test, the Kemax tubes were used. The pattern consisted of 16 holes and no tubing failures were experienced. The holes were drilled with an inner bar, as is normally done with a 3 1/2" ID kelly so there were no rock cores retained by the bit which would have prevented the tubes from remaining in the hole when the kelly was retracted.

The Kemax tubes were shipped in cartons containing 613 feet of 7.3' sections. This consisted of 84 sections or enough for twelve 51' holes. In each carton there were 12 bottom sections with a nylon screen at the bottom end. The screen was held in place by a short internal sleeve section of Kemax tube which was glued in place. There were also 12 top sections in each carton. These had no coupling sleeve.

The screens grew out of the pumped slurry experiments and were used with the packaged explosives because it was thought that they would help prevent rocks and sand from coming up into the tube.

A cylindrical 10 pound weight was used to cause the assembled tube column to sink in the kelly. This weight had a shoulder about 1/3 of the length from the top. This shoulder when in place, rested on the top rim of the tube column and the lower part of the weight extended down into the tube opening. There were no buoyancy problems experienced with the Kemax tubes. Specific gravity of the Kemax material is 0.843.

Hot wax tubes were used in the second test. These were shipped in open bundles of three tubes consisting of one 13' bottom section with nylon screen at the bottom and two 19' sections. These tubes were assembled as they were placed in the kelly and a circular pressed wood disc with a shoulder was then placed in the top opening. The tubes could then be pushed down the kelly with a wooden pole, or by the inner core breaker bar. In these tests, a line was painted on the inner bar which was lowered until the reference line reached the top of the kelly. At that point, the tube column should be on bottom. The kelly was then raised and the tubes remained in place. A weight could not be used as with the Kemax tubes because it tends to split the sides of the hot wax tube and wedge itself down in the tube. The hot wax tubes are more buoyant and more difficult to place. (Specific gravity is 0.678). One or two feet of hole is frequently lost at the bottom because of the tendency to rise. On the other hand, the hot wax tubes cost only about two-thirds as much as the Kemax tubes.

One of the holes with the hot wax tube casings failed. This was at a depth of 12 feet and the hole had to be redrilled and cased with new tubing. Because of the longer handling time, the buoyancy difficulties, and the loss of bottom, the hot wax tubes with the swaged couplings were abandoned in the remaining tests at Miramar and only the Kemax tubes were used.

The third shot at Miramar utilized a packaged water gel (Tovex 255) in flexible plastic links, similar to a string of sausage links. 50-grain detonating cord was tied to the connection between the bottom two links and the string was dropped down the preplaced Kemax tubes. Because of the slight slump of this material in the hole the effective diameter of the explosive column was increased from 2" to 2.1" (50.8mm to 53.34mm). This was determined by calculation and later confirmed by the actual build of the explosives in the hole. This increase, and the greater weight energy per pound of this explosive, permitted the spacing and burden to be increased from 8.5' X 8.5' to 9.5' X 9.5' (2.6m X 2.6m to 2.9m X 2.9m). In this test, the Kemax tubes were left overnight in the holes before loading. All had maintained full depth when measured the next morning. Electric blasting caps were used at the top of each hole to detonate the cord which in turn detonated the water gel.

The fourth test consisted of placing the Kemax tubes as before, leaving them open for 60 hours (over the weekend), and then loading them with the same packaged water gel explosive. All tubes again maintained full depth. In this test the detonating cord was tied to two links and dropped to bottom. Other links were slit, the explosive removed from the plastic wrapper, and then dropped down the tube. This permitted it to slump to the full 2.493" ID of the tube so that a greater amount of explosive could be loaded into the hole. Consequently, the pattern was increased to 11.5' X 11.5' (3.5m X 3.5m) to maintain a constant weight energy per cubic yard with the other tests.

The material was dug between April 4, and April 8, 1977. Daily interviews with the dragline operators revealed that the digging was as good as, or perhaps slightly better than normal. Amount of oversize was less than normal.

Primary crushing of the test material started on April 25 and continued thru April 28, 1977.

Number of loader buckets per hour are recorded by the crusher operator. These were analyzed for the period from April 19 - 28 inclusive. This provided background data for the crushing rate under normal drilling and blasting conditions at this quarry.

The analytical method used was to determine the range of difference between the means at various confidence levels, using the following formula:

$$\text{Range of difference in means} = (\bar{x}_h - \bar{x}_l) \pm \left[\left(\frac{(n_h - 1)\, Sx_h^2 + (n_l - 1)\, Sx_l^2}{(n_h + n_l - 2)} \right) \left(\frac{1}{n_h} + \frac{1}{n_l} \right) \right]^{1/2} t$$

where $\bar{x}_h$ = mean of the data set with the highest standard deviation

$\bar{x}_l$ = mean of the data set with the lowest standard deviation

Sx_h = standard deviation of h data

Sx_l = standard deviation of l data

Sx_h^2 = variance of h data

Sx_l^2 = variance of l data

n_h = number of h samples

n_l = number of l samples

t = t score for level of certainty

This method is described on page 4-9 of "Calculating Better Decisions", a publication of Texas Instruments, Inc. Learning Center.

From these data, the following conclusions can be made:

(1) By placing the same explosive with the same load factor in the tube, and initiating the shot in the same way, (Test #1), one can be 90% certain to obtain an increase in primary crushing rate at this plant between 7.5 CY per hour and 90.0 CY per hour above the normal, or kelly bar method of loading. In terms of percent, this would be an increase between 2.8% and 33.5%. (Test #U-4)

(2) By initiating a tube cased column of the same explosive with the same load factor at the top only with an electric blasting cap (Test U-5), one can be 95% certain that the primary crushing rate at this plant will be decreased between 7.7 CY per hour and 130.7 CY/hr. below normal. In terms of percent this would be 2.8% to 48.2%.

(3) By designing the shot to take advantage of differences in weight energy per pound, one cannot be 80% certain that there will be any effect on primary crushing rate at this plant by using water gel explosives (in plastic links) in the tube cased holes at the same weight energy per cubic yard as the "normal" explosive in holes loaded by the kelly bar method. Stated in a different way, the same data indicate that by using only 71.4% as many pounds of water gel explosives as the "normal" explosive in kelly bar loaded holes, one cannot be 80% certain that it causes any change in

primary crushing rate.(Test #U-6)

(4) By removing the water gel explosive from its individual plastic packages and loading the material in a tube-cased hole, keeping the weight energy per cubic yard equal to semigelatin dynamite loaded through the kelly bar, one can be 80% certain that the primary production rate at this plant will <u>increase</u> between 7.1 CY per hour and 69.2 CY per hour, or an increase between 2.6% and 25.5%, above normal drilling and blasting procedures (Test #U-7).

These conclusions will be utilized later in the section on economic analysis.

GULF OIL CHEMICALS COMPANY TESTS AT CAPELETTI PIT #13

In the Gulf Oil Chemicals Company's test at Capeletti Bros., Inc. Pit #13, the following variations were introduced which made the test series different from the previous tests conducted by Gulf on their own account (See section on Experimentation, in Chapter 3).

1. The bottom section of tubing had a double weight wall to eliminate the tendency observed in Gulf's other tests for the tubing to blow out near the bottom of the hole.

2. A device using air pressure which was lowered into the tubing just before loading in order to blow the hole free of water.

3. A new type of hose to facilitate the loading of the bulk slurry.

4. A decrease in the depth of the holes from 43 feet to 40 feet because the pit operator thought there was a possibility that the shots might be blowing out at the bottom through less competent strata.

5. A new slurry truck with large hopper and pump to eliminate the handling of the 300# drums.

The first test (Test U-1) consisted of 35 holes in seven rows of five holes each. Burden was 9 feet (2.7m), and spacing was 9.5 feet (2.9m), in a staggered pattern.

Kemax tubes with internal couplings were placed in the holes through the kelly bar after the hole was drilled and while the bit was still on bottom. Eight foot sections were joined together above the opening at the top of the kelly and a ten pound weight was placed in the top of the assembled tubing to make it sink to bottom.

The inner diameter of the tubing was 2.243" (56.97mm) and the wall thickness was .045" (1.143mm) except for the bottom section which had the double thickness. Tubing was joined together by internal sleeves of .045" wall thickness. Tubing clearance was .018" (.457mm) when the bit was retracted because of a 0.149" (3.785mm) carbide insert on the inner diameter of the bit which is used to reduce the coring problem. This reduces the ID of the bit to 2.351".

Considerable difficulty was encountered in placing the tubing. On one hole, the entire 40' of tubing hung up in the kelly and the hole had to be redrilled. On other holes, the weight used to sink the tubing split the top of the tubing and the hole had to be redrilled. On two holes, the tubing was two feet short of being on bottom. Some tubes split along the spiral when a twisting motion was used to make up the joints. It was concluded that a wall thickness of .045" was inadequate.

The attempts to blow the water out of the holes with an air jet just prior to loading was a failure. The air pressure ruptured the bottom tube sections in the two holes where the effort was made and these holes had to be redrilled. Water re-entered the tubing within seconds after the water was blown out.

The new type of hose (Gates 421 W, 1.375" X 1.88", 38 pounds per 50') was tried because it had a greater wall strength and it was thought that the previous hose might create pressure bulges which would retard its upward movement. The new hose did not produce the desired result, probably because the weight was too great for the buoyant forces to be effective. After nine holes, the hose was replaced with the previous black hose (Gates 75 W, 1.375" X 1.75", 26 pounds per 50 feet).

After the tubing was placed and the drill rig had moved off a distance of several rows, the slurry truck backed into the area where the tubes had been placed. With an articulated boom and an air operated reel at the end, the hose was placed over the hole. The hose was then lowered into the hole, pumping commenced, and the hose moved out of the hole by natural buoyancy (after replacement with the old hose).

Slurry was pumped to a depth of 5 feet from the surface. Two half-pound primers, taped top to top and with a delay EB cap inserted, were pushed down with a wood tipped aluminum pole to a depth of 12 feet (7 feet below the top of the slurry). The delay pattern is shown in Figure 12.

The reason for using the two 8L primers is that Gulf's laboratory has performed tests which indicate that these primers are not only directional with more explosive force at the ends of the cylinder, but that the bottom end yields more force than the top.

The shot produced good forward heave of material and fragmentation at the surface appeared good but the swell was not as great and the cavity at the rear of the shot (where the No.7 delays were placed) was not well-developed.

Velocity measurement at the top of Hole No. 1, in the south corner of the shot indicated a detonation velocity of 14,410 feet per second.

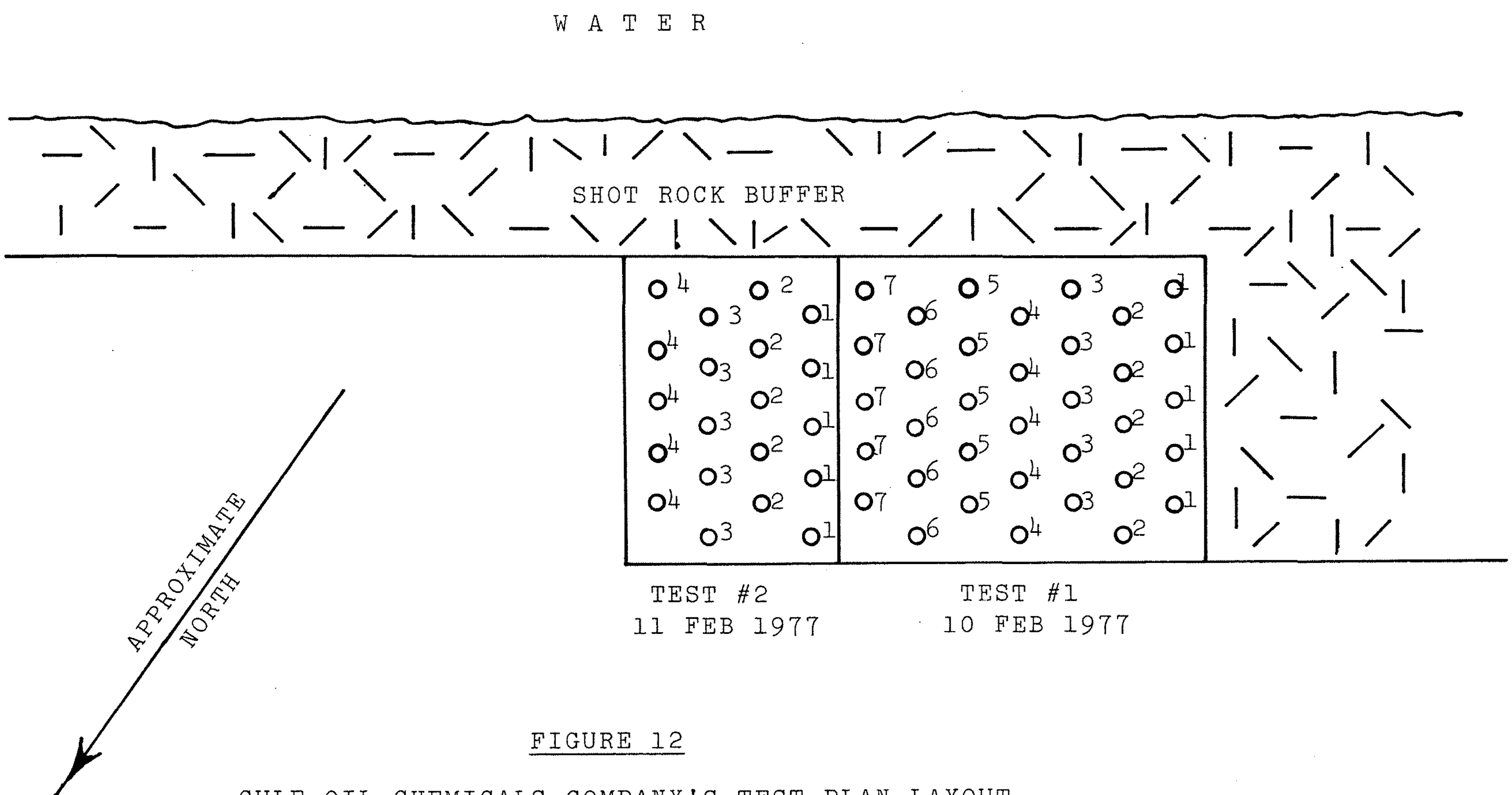


FIGURE 12

GULF OIL CHEMICALS COMPANY'S TEST PLAN LAYOUT

NUMBERS REFER TO DELAY PERIOD OF DUPONT EB CAPS

SCALE: 1" : 30'

The second test (Test U-2) in the series consisted of only 20 holes. Burden was reduced to 8.5 feet (2.6m) and spacing remained at 9.5 feet (2.9m). This was also a staggered pattern. The shot was limited to 20 holes because the bearings in the drill rig's pull-down system failed and drilling could not be continued for several days.

Drilling, tube placement, and loading were accomplished in the same manner as the first shot. Hole No. 20, at the north corner and in the row with the longest EB cap delay period, was instrumented for velocity determination.

On this shot, movement was fair and there was a minor amount of swell. Based on observation of previous shots using tube casings and GX-19, the result was disappointing. Hole No. 20 failed to detonate. On these two shots, the slurry was double pumped for the first time. Once into the hopper and the second time into the hole. This may have had a very pronounced effect on the results.

An attempt was made to dig these two shots in March, 1977, but the dragline could not break out the material below 35 feet and the effort was abandoned. Consequently, these two tests had to be considered failures, and no analysis was made of results.

ATLAS POWDER COMPANY TESTS AT CORAL AGGREGATES CORP'S QUARRY

This site was selected because it provided an opportunity to test pumpable NCN emulsion-type slurries to a depth of 60 feet which is the maximum drilling depth for any quarry in South Florida where a small diameter kelly bar is used. This operation also has little or no buffer and normal production digging will provide data for complete evaluation at an early date.

Normal practice at this quarry is to drill a ten hole pattern with a 9' (2.7m) burden and an 8' (2.4m) spacing. Pattern is rectangular with 2 rows of five holes. Holes are normally charged with 90 pounds of Austin AL-4 dynamite with 8 feet of water stemming. A separate delay is used for each hole.

Atlas performed 10 tests using their P-DYN-2 and P-DYN-4 NCN emulsion slurries. Locations of the tests and the delay patterns used are indicated on Figure 13. Details of each test are shown in Table 4.

The P-DYN-4 slurry was brought to the site in a 6000 pound capacity hopper mounted in a truck. From the hopper, the material was pumped by a Haskell Engineering and Supply Co., air-driven reciprocating pump also mounted in the truck. A water tank off to the side supplied water by air pressure applied to the tank. This water entered the system at the pump discharge and formed a circumferential film of water between the hose and the slurry. This permitted rapid pumping rates and the holes were normally loaded in less than one minute after pumping was commenced.

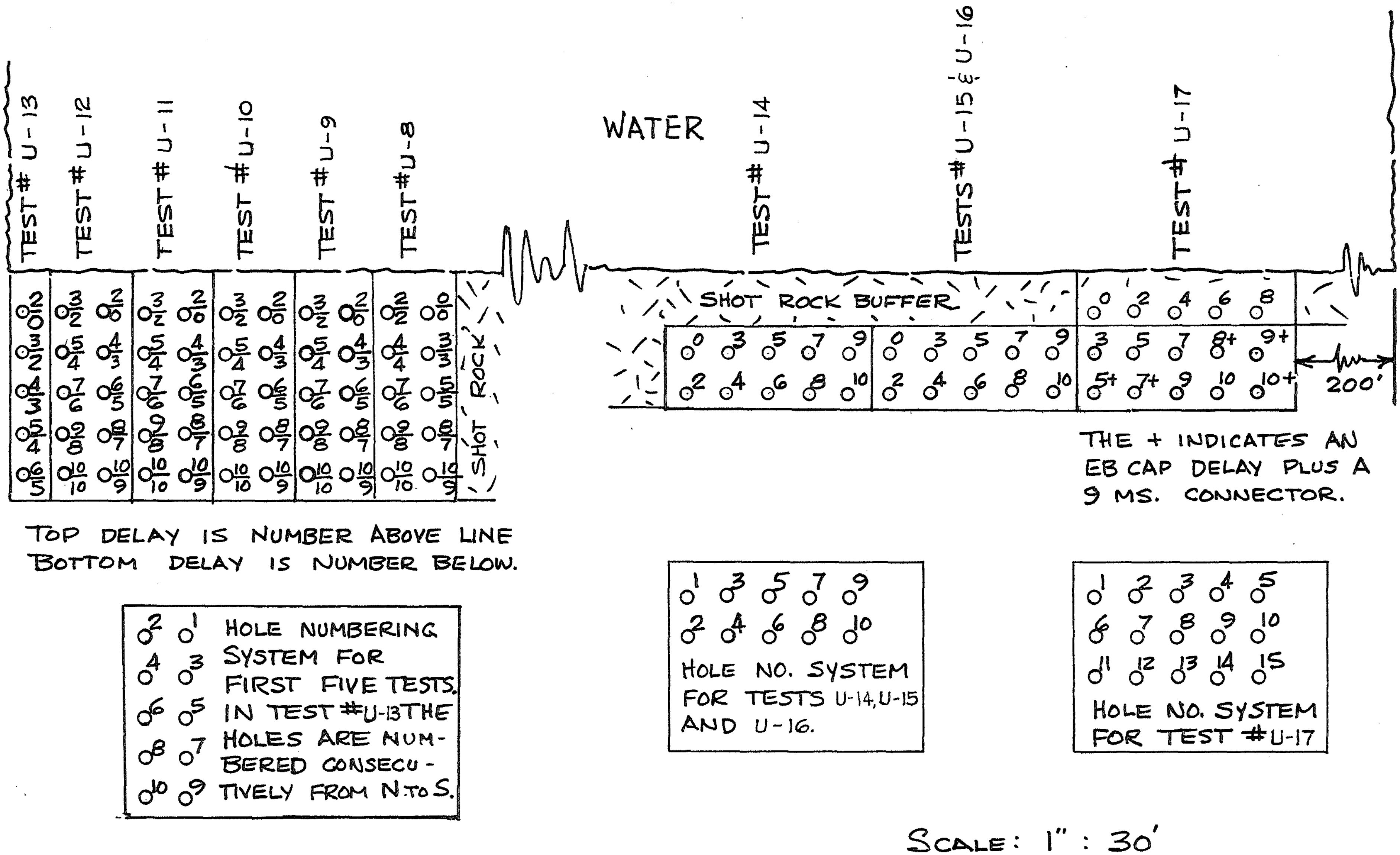


FIGURE 13

ATLAS POWDER COMPANY TEST PLAN LAYOUT AND DELAY PATTERNS

TABLE 4

SUMMARY OF ATLAS POWDER CO. TESTS

CORAL AGGREGATES CORP.

	No. Holes	Slurry	Pumped From	Pounds/ Hole	Loading Time	Bottom Primer Depth	Top Primer Depth	Stemming	Results by Surface Evaluation
Test U-8	10	P-DYN-4	Truck	113	56 min.	56'	10'	6'	Very poor
Test U-9	10	P-DYN-4	Truck	90	86 min.	56'	10'	6'	Very poor
Test U-10	10	P-DYN-2	Portable	114	47 min.	56'	10'	6'	Fair Heave
Test U-11	10	P-DYN-2	Portable	114	127 min.	56'	10'	6'	Good Heave, Swell & Scarp
Test U-12	10	P-DYN-4	Portable	111	41 min.	56'	10'	6'	Excellent, heave, swell & perimeter scarp
Test U-13	5	P-DYN-4	Truck	112	27 min.	56'	14'	6'	Very poor
Test U-14	10	P-DYN-4	Portable	112	48 min.	--	8'	6'	1st five delays good, 2nd five delays poor
Test U-15	10	P-DYN-2	Portable	114	102 min.	56'	--	6'	Very poor
Test U-16	10	P-DYN-2	Portable	114	---	--	8'	6'	No Swell, No Movement. Some hole enlargement.
Test U-17	15	P-DYN-2	Portable	114	59 min.	--	8'	6'	No Swell, No Movement. Some hole enlargement.

The P-DYN-2 was in 30 pound polyethylene bags. These were hand loaded to a rack of hooks over a small portable hopper. This hopper fed another Haskell pump mounted on the same frame which pumped the slurry into the hose and loaded the hole.

The holes were drilled and the Kemax tubing(2.200" ID with 0.060" wall thickness; 55.88mm with 1.524mm wall) was placed in the same manner as in the Gulf tests. Much less breakage and placement trouble was encountered with this thicker walled tubing than the 0.045" thickness used by Gulf.

Several holes were lost in the beginning until the drill crew became accustomed to handling the tubes. Only two holes had to be redrilled because of failures attributable to the tubing. This is a 2% failure rate.

The good results obtained by Tests U-10 and U-11 after the poor results of Tests U-8 and U-9 suggested that P-DYN-2 was better adapted to the test conditions, but there was the possibility that the double pumping required in the use of the truck hopper (pump into hopper, then pump into hole) might be desensitizing the slurry. Test U-12, therefore, called for loading the P-DYN-4 in the same manner as the P-DYN-2, i.e. 30 pound polyethylene bags and the portable pump. The success of this test indicated the problem was not with the explosive but more probably with the double pumping. Test U-13 was a negative test of this hypothesis and the poor results tend to confirm it. Consideration of these results and the fact that Gulf also had greater difficulty with their new bulk hopper than they had experienced with the 300 pound drums suggest that double pumping may cause a serious problem.

Test U-14 tested the feasibility of using only a top primer detonated by 50-grain cord with an electric blasting cap at the surface.

Test U-15 was a Primadet simulation. Because of production lead times, Primadets with different delay periods were not available at the time of the tests. A simulation was made by placing a delay cap and primer at the bottom of the hole. A 4 grain detonating cord was placed from top to bottom of the hole and opposite from the leads to the electric blasting cap. A zero delay at the top of the hole detonated the 4 grain cord. Electric blasting caps at the bottom of the hole had delay periods ranging from 0 to 250 milliseconds. This shot was a failure. There was no apparent movement or enlargement of the holes. The detonating cord apparently desensitized the slurry.

A length of 4 grain cord inside a five-foot length of .060" wall thickness tubing was detonated in the lake water to see what the effect on the tubing would be. In this unconfined situation, the tubing was blown into fragments with the greatest dimension being about 8 inches.

Test U-16 consisted of redrilling and reshooting Test U-15 with detonation similar to Test U-14.

Test U-17 was similar to Test U-14 except that five holes were drilled in the buffer to make sure that the shot was not contained by solid rock, and the delay pattern was changed to a standard diagonal en echelon arrangement.

Tests U-14, U-15, U-16, and U-17 were dug and processed first. The dragline operator reported that Test U-14 was excellent digging but U-15 and U-16 were very difficult. He said Test U-17 dug as if it had never been shot and that he was ready to give up.

Tests U-8, U-9, U-10, U-11, U-12, and U-13 weren't dug until May, 1977. The dragline operator stated that digging was moderately rough and that a considerable amount of oversize was encountered.

Three sets of data were collected from observations of the primary crushing rate. Nine hours of observation provided nine hourly rates for material which was drilled and blasted in the manner normally used at this quarry. For Tests U-8 thru U-13, twenty-eight hourly rates were determined, and for Tests U-14 thru U-17, twenty-nine hourly rates were obtained. The individual test shots were not separated because of their small size. These were treated in the same manner as the production data from the Miramar tests. Statistical parameters were:

	$\bar{x}$	Sx	Sx^2	n
Normal production	593 CY/hr	59.35	3522.8	9
Shots 1,2,3,4,5, & 6	431 "	89.87	8075.9	28
Shots 7,8,9, & 10	488 "	66.6	4441.3	29

By determining the ranges of difference of the means at the 95% level of confidence, one can conclude:

(1) Rock from Test Areas U-8 thru U-13 caused a decrease in primary crusher production rate of at least 97 CY/hr and not more than 228 CY/hr. Translated into percentage, this is a decrease ranging from 16.4% to 38.4% less than the normal production rate.

(2) Rock from Test Areas U-14 thru U-17 caused a decrease ranging from 55 CY/hr to 155 CY/hr, or a percentage decrease of 9.2% to 26.2%.

Because of these disappointing results and the poor results from Gulf's pumped slurry tests in the small diameter tube casings, in contrast to the encouraging tests at Miramar, it was decided that the future course of the program should be directed to more fully evaluate the use of packaged products in the 2.493" ID tube casings with external couplings. Contributing to this decision was the fact that all of the small-diameter pumped slurry tests required relatively expensive and repair-prone equipment at the loading site, and that even with many technical people on hand, there were repeated delays in performing a fairly simple operation. Hoses would become clogged, pumps wouldn't pump, hoses wouldn't pump themselves out of the hole, and pressure would blow out the wall of the tube and the slurry wouldn't be contained.

Nevertheless, one more unit test using pumped slurry was subcontracted to Gulf (Test U-3) in which the tube casings would be placed through a standard 2 1/2" ID kelly bar. This was a 30-hole test at Capeletti Bros., Pit #13 conducted on April 13, 1977.

In this test, the thicker-walled (0.060") Kemax tubes which had been successful in the Miramar tests were used and Gulf incorporated a longer delay interval between successive rows of holes. The longer delay was introduced because of their experience with much improved results in using a 50 millisecond interval rather than a 25 millisecond interval when initiating the pumped slurry in the large, 4 1/2" ID tubes where the pumped slurry system was operational. The results of this change in the large diameter blasting were inspected by the Project Manager and one of the blasts was witnessed which verified that post-blast appearance in the pit was indeed much improved by the use of the longer delay interval.

In the 30-hole test, GX-19, a pumpable slurry was pumped into the tubes as it was in previous tests. Spacing was 9.5 and burden was 9'. There was 6 1/2 to 7 feet of stemming. Two Trojan 8L primers were placed at 12-14 feet below the surface (6-7 feet below the top of the slurry) with an electric blasting cap. The first row of six holes contained No. 3 period DuPont caps. The second row, No.5 period delays, the third row, No. 7 period, the fourth row No.9, and the fifth and last row No. 10. There was a 23 foot buffer of shot rock between the front row and the open water.

Surface evaluation of the results of the shot indicated that it was unsuccessful. It appeared that the first row, or front of the shot had remained pretty much in place. There was some swell and forward heave of the material but it was mainly evident in the middle of the shot (3rd and 4th rows). Capeletti, Gulf and ECA representatives concurred that the shot would have to be redrilled and reshot.

Subsequent analysis of results by Gulf caused them to abandon efforts to initiate the slurry with a primer in the top only.

The better shots in the tests conducted by Gulf and by Atlas were obtained when top and bottom primers with electric caps were used. As with the dynamite tests at Miramar, initiation from the top only consistently gives poor results, either in the post-blast appearance of the shot, or in the production rate when processed, or both.

This adds two disadvantages to the pumped slurry systems in the small diameter holes. The first is a problem of safety in that the electric blasting cap at the bottom of the hole has to be in place while the slurry is being pumped and loaded. This means that the loading process with all of the equipment is moving about with charged holes nearby. This is one of the main disadvantages of loading explosives through the kelly bar. In addition, caps are in place for a relatively lengthy period of time in contrast to present methods, and this increases the lightning hazard. This is particularly important in South Florida where thunderstorms are almost a daily occurrence from May to the end of September. These thunderstorms appear within a matter of 15 to 30 minutes while it requires about two hours to complete pumping slurry into 30 holes.

The second disadvantage is a matter of economics. Operators currently use a 25, 30 or 50 grain detonating cord downline with an electric blasting cap at the top of the hole, or with a trunkline and an electric blasting cap. Such downlines cost about \$2.00 to \$2.50 per hole and the caps at the top, with short leg wires, cost about \$0.70 each, depending on length of leg wire and delay interval. This is a total cost of \$2.70 to \$3.20 per hole for detonators. Top and bottom priming requires two half pound primers at about \$1.10 each, one cap with 50' to 60' leg wires at roughly \$1.95 each, and one cap with short leg wires at \$0.70 each. This is a total of \$4.85 per hole, or an increase ranging from \$1.65 to \$2.15 per hole. To be economically competitive with present methods, the savings in explosive cost and loading time, along with any possible advantages in improved fragmentation, greater yield, and increased production rate, would have to offset this additional cost for detonators <u>and</u> the cost of the tube casings.

In both the Gulf and Atlas tests, there was no indication that any savings in loading time could be accomplished. Loading time for Gulf averaged 5.25 minutes per hole and for Atlas 6 minutes per hole. Assuming that the tube sections could be assembled and placed in the hole through the kelly in approximately the same time as it would take to load explosives through the kelly, the above loading times represent additional times. This means added cost instead of savings. Gulf attempted to increase the yield by increasing the burden and spacing but the rock couldn't be recovered with the dragline. Atlas didn't try to increase the yield and the fact that the production rate was lower when the same burden and spacing was used indicates that such efforts would have had negative results. From the pumped slurry tests conducted in small diameter tubing, it can be concluded that there is no decrease in loading time, no improvement in fragmentation, no increase in yield, and no increase in production rate. The cost of the bulk slurry is generally less than the cost of cartridged explosive products, but this advantage is not sufficient to offset these negative factors, much less the added cost of tube casings and detonators.

The advantages and disadvantages of pumping bulk NCN slurries into small diameter tube casings can be summarized as follows:

ADVANTAGES

1. Eliminates most of the safety hazards which are peculiar to the kelly bar method of loading explosives, e.g. no

possible hangup of explosives in the kelly bar, no explosives involved if drill steel is hung up in the hole, and drilling can be physically separated from loading.

2. Safer to handle than dynamite.

3. Easier explosive inventory control.

4. Less likelihood of explosive theft.

5. Generally lower explosive cost per pound.

6. Full diameter loading.

7. Little or no nitrous oxide cloud.

8. Can be shipped and stored as an oxidizer, eliminating the need for Class A magazines.

DISADVANTAGES

1. Sensitizing air bubbles are too easily collapsed by depth of water and/or detonating cord downline as small as 4 grains per foot, and/or double pumping. When this happens, the slurry doesn't explode.

2. Multiple priming with 1/2 pound primers and electric blasting caps is necessary to achieve consistently good results as judged by post-blast inspection of the surface.

3. Multiple priming requires a cap in the hole during loading and increases the lightning hazard.

4. The maintenance and operation of the equipment requires a level of skill significantly above that presently needed.

5. Accidental overloading of the holes can result in excessive flyrock. Removal of the overload would be difficult if not impossible.

6. Loading facilities require new capital investment for truck or trailer with hopper, pump, hose, and tanker for bulk storage.

7. If loading facilities are not purchased, operator must coordinate loading with distributor's schedule. This would be impractical for those operations which shoot 8 to 10 holes per shot, four or five times per day.

8. The method takes approximately 2 to 5 times more loading time than that required for loading packaged products in the tube cased holes.

9. Multiple priming increases the cost.

10. No offsetting advantages in improved fragmentation, greater yield, or increased plant thruput, to cover partly or all of the additional cost of multiple priming and tube casing. In fact, when the rock can be dug, production rate is substantially decreased.

11. Greater waste of explosives.

12. Probably more temperature sensitive.

13. Electric blasting cap at bottom of blasthole may be 20° to 25°F. cooler than the cap in sunshine at top of hole, possibly introducing additional error in the precision of average delay times.

CHAPTER 5

3" I.D. KELLY BAR PROGRAM

TUBE CASING SELECTION

As the unit tests were completed by various subcontractors, the data indicated not only the superiority of loading packaged explosive products in the small diameter tubes as compared to bulk slurry, but also the capability of the Kemax tube casings to stabilize the blastholes under a variety of conditions.

The severest test among these early efforts was at Miramar Rock where the tube casings performed excellently to a depth of 50 feet under moderately adverse geologic conditions. The casings used on one of the test blasts were left in the ground for 60 hours before loading without any failure. Others were left overnight, as they were in both the Gulf and Atlas tests. In addition to the casings used in the test blasts at Miramar, others were set at distances of 10, 20, 30, and 40 feet from one of the blasts. Only the tube casing at 10 feet failed. (Additional data on this subject is presented later in the report.) Ten holes were drilled to 50' at the north end of the Miramar pit and five of these were cased with Kemax tubes (2.493" ID with .060" wall thickness) and five were cased with spiral-wound hot wax tubes (2.730" ID with 0.100" wall thickness). These casings were placed on March 7,1977. Depth was measured daily by using a tape with a 2" cylindrical weight, until March 11 when one of the hot wax tubes failed at a depth of 35 feet. After that date the intervals for measuring were gradually spread out from weekly, to biweekly, and finally monthly. When the 3" ID kelly test plan was being formulated, the tubes had been in the ground for about one month with no additional failures, and it was clear that the Kemax tube gave superior performance. (Actually, all of the Kemax cased holes were open at full diameter to their original depths on December 12, 1977, nine months and five days after placement. The four surviving hot wax tubes remained open to full depth until sometime after September 27, 1977, for a minimum life of six months and twenty days.)

In developing a test plan for the remainder of the project, it was considered essential to expose some type of expendable casing to as many different sites as possible to determine its general applicability to South Florida mining conditions. Considerable thought was given to the different possibilities.

Inquiry to another major supplier of cardboard tubes revealed that they did not make a product similar to the Kemax tube or the hot wax tube. Other efforts to locate a competitive source revealed that the phenolic resin treated tube could be made and produced by others but not as readily as Sonoco because of their previous experience.

With respect to other materials, local suppliers of glass, aluminum and plastic products were questioned about the availability and price of their tubing. With Kemax tubes at 18.9¢ per foot delivered at that time, when ordered in trailer load quantities and delivered by piggyback rail, the plastic and glass suppliers, indicated they could not provide a competitive product. One plastic supplier indicated that a similar sized plastic product would run about 70¢ per foot. This might be further reduced to about 35¢

per foot if quantities on the order of a million feet were being ordered. He further indicated that the tubing would not shatter or disintegrate when the encased charge was exploded and would probably show up in the final stone product as it did with the Hercules Corflo™. The glass supplier felt that the price for 2 1/2" glass tubing would be more than 70¢ per foot and that couplings and handling would be problems. Aluminum producers were included in the general solicitation for ideas but no replies were received. Assuming a minimum wall thickness of 0.050" would be required, local suppliers indicate that the cost for 2 1/2" ID tubing would be about 41¢ per foot plus the cost of swaged couplings. (Approximately 90¢ per pound in quantities of 30,000 pounds per month.) Aluminum has the disadvantage that it might be a deleterious substance if the aggregate was used in concrete in that it might cause an excessive amount of gas formation and weaken the concrete. Aluminum powders are frequently added to concrete mixtures as air entraining agents but the percentage by weight is usually less than 0.02 percent. Although the tubes would constitute a smaller percentage if evenly distributed throughout the rock (on the order of 0.004 percent by weight), it might be possible for a local concentration to occur which could produce undesirable expansion in structural concrete. In addition, large pieces in the stemmed interval would probably cause processing difficulties. Aluminum might also produce unsafe electrical effects.

Undoubtedly there are other substances, such as a cellulose product, which might be used as expendable tubing in stabilizing blastholes in South Florida. Two people active in the industry mentioned that they have patentable ideas which they will pursue if kelly bar loading is outlawed. In selecting the Kemax material for testing the expendable tubing concept, considerable weight was given to the fact that it (1) was a product that was already on the market, (2) could be manufactured in quantity to size specifications as desired, (3) was available for delivery in three weeks time, and (4) had a previous record of success in Hercules, Gulf and Atlas testing.

It is assumed that if tube casings of other materials or other manufacture possess the same general strength and water resistant properties, they could be used in place of the Kemax tubes. If lower in cost, it would enhance the economic analyses used in this report with respect to the expendable tubing concept.

Details of the manufacture of the tubes is a trade secret of Sonoco Products Company. Their promotional literature indicates that the process varies and is custom-made to fit different applications. In general, it is made from kraft paper which is a tough, brownish paper made from high grade sulfate wood pulp. Some of the material in the Kemax tube may be recycled kraft.

The tube sections, called cores in the paper trade, are treated with phenolic resin to provide water-proofing and strength. This may or may not involve vacuum pressure treatment.

The material is brittle and will break rather than deform. If broken, it remains resistant to water on the broken edge surfaces.

It resists axial deformation when dry, but if placed on an uneven surface after it has been immersed in water, it may warp on drying.

The manufacturer apparently maintains tight control on tolerance levels because the friction couplings rarely give any difficulty in being either too tight or too loose. This control is essential when dealing with tubes in the standard 2 1/2" ID kelly because the clearance at the bit for a 2.320" OD X 2.200" ID (0.060" wall) tube is only 0.031" (2.5" less 0.149" for the carbide core breaker).

The hot wax tube on the other hand appears to be composed of a gray paper pulp which is probably largely recycled paper of many grades. The hot wax coating is applied by dipping the tube into a vat of hot wax.

The thickness of the wax coating varies so the tube does not maintain the same tolerance level as the Kemax tube. If the hot wax coating is ruptured so that water can get to the unprotected paperboard, the tube loses its strength rapidly and may fail completely.

Storage of the hot wax tube is also a problem. On hot days the wax melts and then resolidifies causing the tubes to stick together. Storage must be on a plane surface or the tubes will warp severely and cannot be placed in the kelly.

When drilled up, or when exploded, and the paper is exposed to water, it becomes a pulp-like mass which breaks up with any water action.

If a hot wax tube becomes jammed in the kelly it can be very difficult to remove.

Preliminary results and the experience of others indicated that neither type of tube would cause any deleterious final product contamination or processing difficulty.

Because of the probable importance of maintaining close tolerances in the small diameter kelly bars, and the need for couplings which would provide an internally flush tube, along with the other advantages such as water resistance and uniform quality, the Kemax material was selected as the most likely to give successful results.

The cost of the Kemax tube is roughly 35% more than the hot wax tube but this difference can be rapidly offset if the number of failed holes increases because of tube failure, or if the tubing cannot be recovered and reused if prevented from going into the hole by a rock jam in the bit opening. For example, in April, 1977, the cost of 51' of 2.493" ID Kemax casing in a 50' hole was $9.33. Comparative cost of 2.793" ID hot wax tubing was $7.04. The cost of drilling the hole was about $18. Total cost of a cased hole was:

	Hot Wax Tube	Kemax Tube
51' Tubing	$ 7.04	$ 9.33
50' Hole	18.00	18.00
	$25.04	$27.33

Total increase in cost for a hole cased with Kemax is $2.29 or 9.1 percent. If the hot wax tubes caused the loss of 10% more holes through tubing failure, inability to load past swaged couplings, or frictional retention in the kelly because of warping, the average cost per hole would exceed that of the Kemax tube. In addition, storage and handling loss would be greater, and in the event of a rock jam in the bit, the hot wax tubes couldn't be recovered as frequently as the Kemax tubes. Placement of the hot wax tubes also requires more time.

Considering all of these factors, it was decided that the Kemax tube was the most feasible expendable material to use for small diameter holes among the commercial products that were available.

Selection of the tube casing diameter was determined in the Miramar tests. The 2.493" ID was used there because in the conclusions of the Hercules report it was stated that the outside diameter of the cartridged explosive must be at least 1/4 inch less in diameter than the inside diameter of the tube. This was confirmed by Gulf in one of their tests in which 2" X 24" cartridges of semi-gelatin dynamite were loaded into a blasthole cased with 2.243" ID Kemax tubing with internal couplings of 0.045" wall thickness. Inner diameter at the coupling was 2.153". A 25-grain detonating cord downline was tied to the bottom stick by punching from side to side and tying at the top. The sticks would not pass the first internal coupling by gravity alone but had to be pushed by the addition of more sticks. When the column reached the third internal coupling, a tamping pole had to be used because sufficient downward force could not be exerted by simply pushing another stick in at the top.

Further project testing at a later date also confirmed that an electric blasting cap leg wire with a half hitch around a 2" rigid cartridge will not pass by a 2.153" ID internal sleeve by gravity in air. If the cap is placed in the bottom stick and no half-hitch is used, it will pass the sleeve by gravity in air but not in water. The same negative performance was obtained with Primadets and with Nonel trunkline connectors.

Similar tests were not conducted with water gel explosives in plastic links but their tendency to slump when an obstruction such as an internal coupling is encountered would certainly cause their passage to be more difficult and more unlikely than a rigid cartridge. It was established during later project test blasts that 2" diameter plastic links can't be poled down a casing with a inner diameter as large as 2.493". They either go down by gravity or they don't go down.

Consequently, an inner diameter was selected that permitted a 1/4" more diameter than the combined diameters of the 2" explosive and a 1/4" diameter detonating cord. This resulted in the 2.493" ID tube with 0.060" wall thickness.

The wall thickness was selected because Gulf experienced a high rate of end failure during coupling and it was felt this could be corrected with the 0.060" wall which is 33% thicker. Coupling material for the external sleeves was only 0.045" thick in order to reduce the uneveness of the outer surface and minimize the possibility of being snagged by the bit.

The external sleeves provided a flush inner surface which facilitated the passage of the packaged explosives and did not prevent its use with bulk slurries.

The final tube design was the result of the efforts of the Project Manager and Lynn Roper of Sonoco Products Company.

DEVELOPMENT OF TEST PLAN

After determining the proper diameter and the most likely material for the expendable tubing, and designing an external coupling which would permit the loading of 2" packaged products, a test plan was developed and approved by the USBM/ECA Techincal Committee. Approval of this plan was also obtained from the USBM Contracting Officer.

This plan emphasized finding an acceptable alternative for the small diameter operations because working alternatives were already available for the large diameter kelly bars. Also, the small diameter presented more constraints and complications because of the limited space available and the increased difficulty of reliably initiating small diameter explosives. Any newly developed technology could most likely be transferred from the small diameter to the large diameter but not vice versa. In addition, there are many more drill rigs with the small diameter kelly bars as indicated in Part II, State of the Art.

An obvious question arises. If acceptable alternatives have been found for the large diameter kelly bars and these alternatives are operational, why can't the other rigs be converted to use a 5" ID kelly and accept the same method.

The answer is threefold. First, many of the rigs do not have a rotary table which can accomodate the larger kelly, nor would the mast and the hoisting system be able to handle the additional weight. New pumps or compressors would be needed to flush the cuttings from the hole. Mainly, conversion to this large size would not be feasible.

Second, many contract rigs alternate between the mining industry and construction work. For many construction projects the use of a 4" diameter explosive column would be unwise and dangerous.

Third, most of the quarries that utilize the small diameter kelly bars do not do so through choice, but because they are operating near residential areas and have had to use 2" explosives to reduce the complaints. When loading through the kelly, this is about the only way to reduce the charge weight per delay. As the population of South Florida increases, many of the operations that now use large diameter explosives may find it necessary to convert to a smaller size unless it becomes possible to reduce the charge weight per delay in some other manner such as decking. The almost exclusive use of small diameter kelly bars in real estate development and construction activities is largely the result of an effort to avoid ground vibration problems.

For the large diameter operations, the plan proposed the utilization of data made available by operators who are using or experimenting with alternative methods.

For the small diameter kelly bars, the test plan was to replace the 2 1/2" bar with a 3" bar, and modify the drive bushing, swivel, and stabilizer to accept the slightly larger size. The additional weight of the 3" ID kelly is only 180 pounds, and the additional weight of a larger swivel is about 75 pounds, for a total of 255 pounds. This is not enough to create an overload situation for the mast and drawworks unless the rig is currently being operated in a marginally safe manner.

The 3" ID bar would be used without a core breaker inner bar, which would keep the increased weight down and which could be handled by rigs with only one reel on the drawworks. There was no way to predict whether rock jams in the bit would be a serious problem or not, or whether the external couplings might become snagged by the carbide insert on the bit as it was retracted.

The principal objectives were to determine:

(1) if the holes could be drilled and the tubing placed using the 3" ID kelly without any serious problems.

(2) if the tube casings could stabilize blastholes and be used under the wide variety of hole conditions found in South Florida.

(3) if blast design could be optimized with the use of the tube casings with a view to making their use economically attractive.

Answers for the first two items would be best obtained by testing as many different sites as possible with a wide geographic spread. Two or three shots at each site would be sufficient to prove or disprove the technical feasibility at that location.

The other item would best be answered by conducting a fairly extensive series of tests at a few locations where the results of different blasting designs could be compared statistically with others at the same location. Evaluation would further require that the shot rock be processed before the project was completed.

The plan adopted was a compromise to accomodate both needs to the greatest extent possible. Greater emphasis was placed on the broad geographic coverage because this was more directly related to the technical feasibility of developing a safe alternative. Optimization was more concerned with the economic aspects of the problem. Nearly every active 2 1/2" ID kelly bar operation in Dade and Broward Counties was included in the testing program along with one operation in Lee County which is on the west coast of Florida.

RIG CONVERSION FOR TEST PROGRAM

Because it could not be predicted whether the use of a 3" ID bit without a core-breaker was feasible, a contract drill rig had to be selected that required the minimum expense in conversion and which offered considerable flexibility in testing if preliminary results indicated that this part of the project should proceed.

The drill rig that was finally selected is normally used with a 3 1/2" ID kelly with a core-breaker inner bar. It is a Damco Model 1250, equipped with an air compressor for drilling with air, and with a 5" X 6" duplex reciprocating type water pump. The swivel was a Gardner Denver 1107 model which has a 5" ID opening. The rig was about 15 years old and in fair to good condition. It was not equipped with automatic pull-down equipment and its average drilling rate in 1976 was about 1200 feet per 10 hour day which is about the average rate for a 3 1/2"ID kelly bar operation. In other words, it was about an average drill rig without any special equipment that would improve its performance over other average mining equipment in South Florida. If anything, its age probably resulted in more than normal downtime which was included in the total drilling time if the drill rig was down for less than one day. This made the drilling time statistics compatible with drilling records of other rigs which are normally kept on a holes per day basis.

Because of the large swivel, and the normal use of a larger kelly, conversion consisted of removing the existing kelly and inner bar, putting a bushing in the stabilizer, modifying the drive bushing to accept a 4.25" OD X 3.0" kelly in place of a 4.75" OD X 3.5" ID kelly. The latter was accomplished by welding a smaller diameter drive bushing to the top of the existing bushing. (Figures 14, 15, and 16) This smaller diameter bushing was essentially a short section of tubular steel with an opening slightly larger than 4.25", with three flutes on the inner diameter positioned to match the flutes on the outer diameter of the kelly bar. The length was sufficient to accomodate the drive pins which were 4 1/4" long. There was a flange plate at the top to hold the pins in place and the base of the flutes was built up with weld to keep them from falling through. The bushing was fabricated in a local machine shop but could be installed or removed in the field.

The kelly bar was made from 41-42 chrome alloy steel, hot drawn. Hot drawn steel was used instead of cold drawn because it was available and could be shipped immediately to the local machine shop.

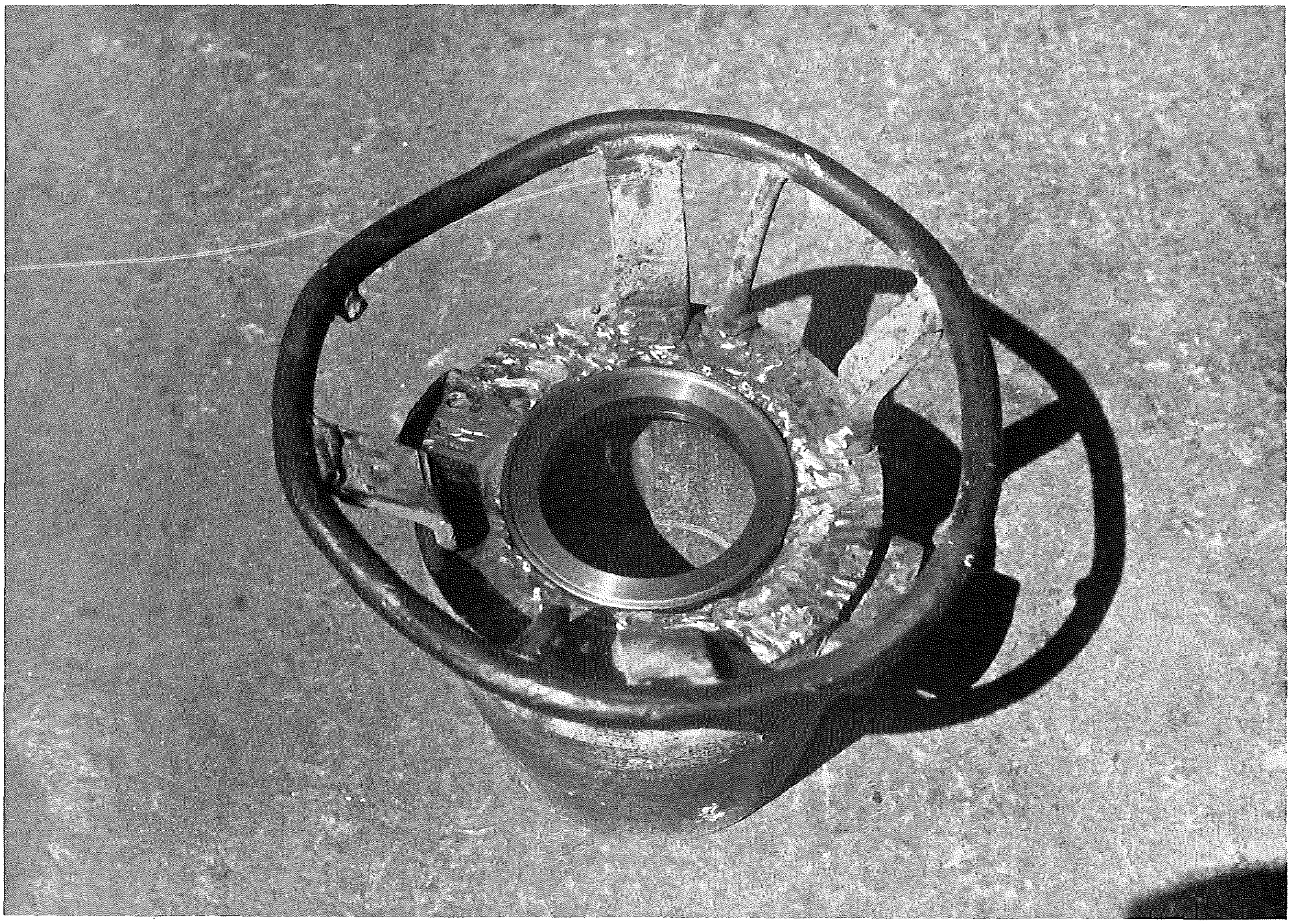


FIGURE 14

Regular drive bushing with seat prepared for drive modification on top of bushing.

Cold drawn steel would have required 90 to 180 days for shipment. In addition to these conversions to the drill rig, a new bit supply had to be obtained. For a start, six bits were ordered with the thought that the supply could be increased later if testing was continued.

The total cost of this conversion was:

56' kelly bar 4 1/4" X 3" @ $69.25/ft.	$3,878.00
18" X 4 1/4" X 3" Sub Adapter	112.00
Head Adapter	112.75
Fabricate drive bushing and rethread swivel	275.00
Six new six-point dynabits @ $77.50 each	465.00
Labor and start up cost	1,430.00
	$6,272.75

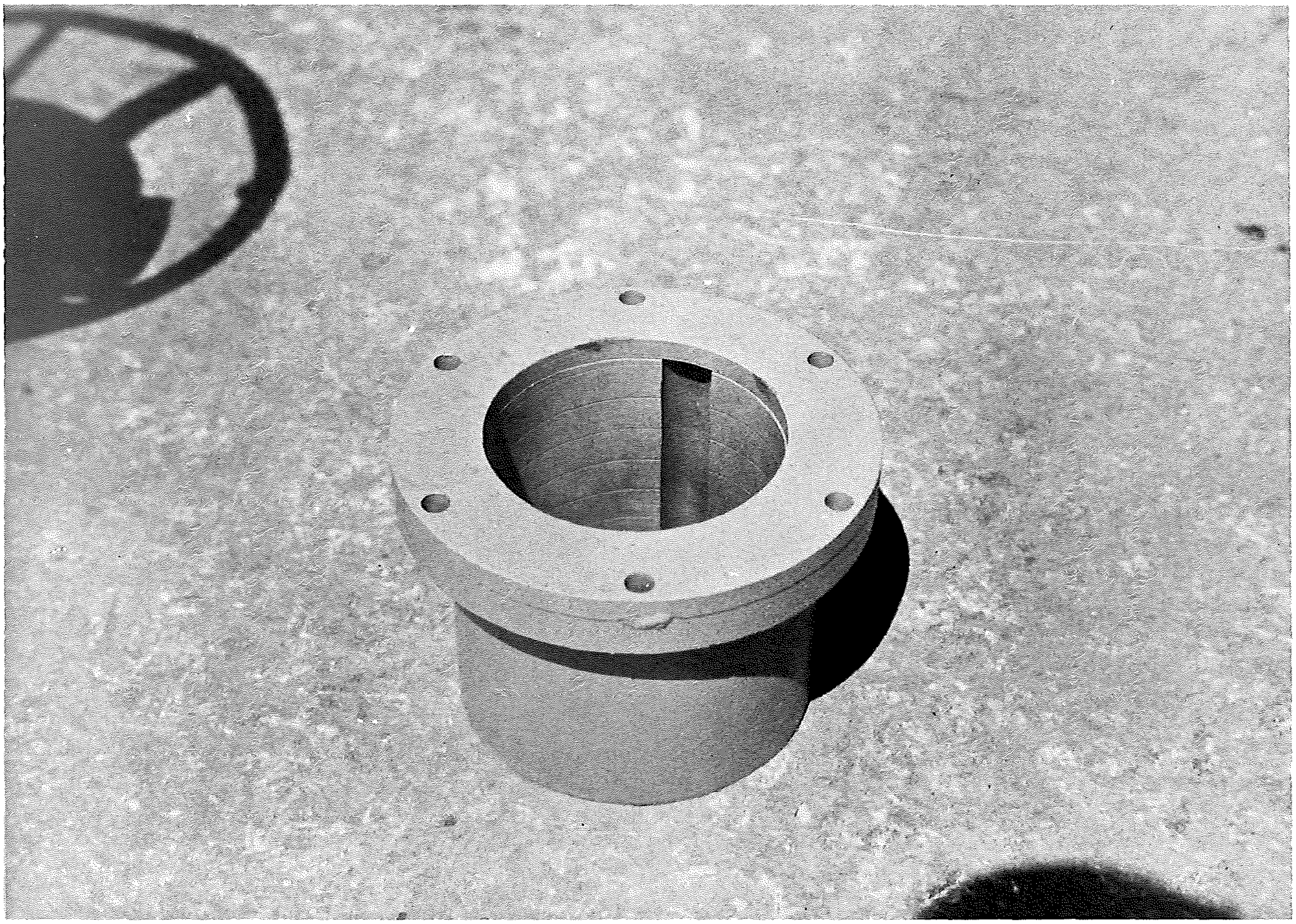


FIGURE 15

Drive modification assembly.

At a later date it was discovered that drilling without the inner bar permitted the upper part of the swivel too much freedom and caused excessive wear on the bearings in the Chiksan. This was corrected by putting a two foot stem on the swivel head in place of the inner bar. This stem extended down inside the kelly bar and stabilized the movement of the head and Chiksan.

There was continued difficulty with the swivel head coming free where the lower body pins fit into the J slots. Without the core breaker there was only water pressure on the swivel head to keep the pins in the locked position. Consequently, the head would work loose and spray water in excessive amounts. Correcting this situation caused a lot of short periods of downtime in the early stages of the program. It was finally corrected by welding a U-shaped metal loop to one of the pins, which held a hook attached to one end of a short section of rubber belt. After opening the swivel and placing the tubes down the kelly, the top of the swivel was replaced and the rubber belt was brought up over the top and

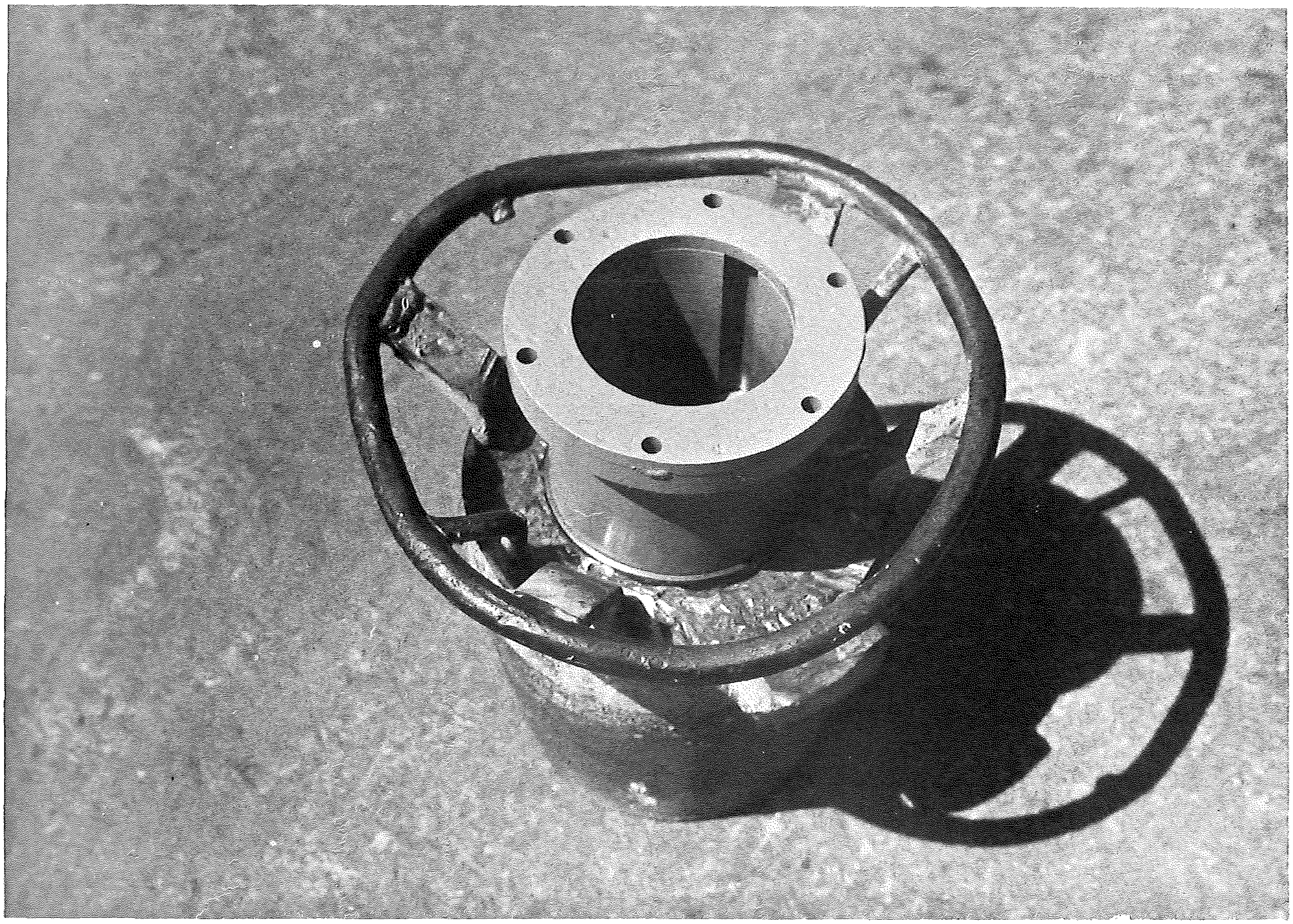


FIGURE 16

Drive modification assembly in place on top of regular drive bushing.

around the Chiksan, then placed over one of the vertical pins on the top of the swivel where it was held in place by a loop in the end of the belt. For a permanent conversion, one would most likely make modifications in the swivel to provide a quick opening cap which would eliminate this problem and decrease the time required to place the tubes.

Apart from these relatively minor problems, no difficulty was experienced with the conversion. The kelly was reversed about midway through the program because the drive pins started to ride out of the flutes but it was discovered later that this was probably caused by improper seating of the pins. Additional weld and a new set of pins were installed.

Because of the experimental nature of the program, it was necessary to contract for the drilling on a time basis rather than per foot. Prices were obtained from two contract drillers who had suitable equipment and the final selection was based on the lowest price which was $55.50 per hour, including liability and workmens compensation insurance, bit sharpening, fuel oil, and normal repair costs. This competitive figure is used for the owning and operating cost of a drill in the economic calculations later in the report.

TUBE PLACEMENT AND LOADING METHOD

With the 3" ID kelly bar, the tube casings were placed through the swivel opening in a manner similar to that used at Miramar Rock.

Seven individual 7.3' sections including one section with bottom screens were carried to the drill rig by the helper while the hole was being drilled. All but the bottom section had external couplings or sleeves. The seven sections were counted at the rig as a check.

When drilling of the hole was completed, the top part of the swivel was opened and the driller handed the tube sections to the helper for placement in the drill string. To insure a good friction fit at the sleeve connection, the tube end was dipped in water by the driller before passing it on. If the connection was still loose, fine sand could be rubbed onto the connection surface.

Starting with the screen section and checking to make certain that the screen end was down, the helper placed and connected the seven sections. The driller counted the sections as they were placed as a further check that the proper length of tube casing was being used. With one hand the helper could easily hold the column of tubes in the hole while reaching for the next section.

When the last section was connected and lowered, a machined weight with a shoulder was placed in the opening at the top of the tube column. This was sufficient to cause the column to drop to the bottom of the hole and remain while the drill string and bit were extracted. A central hole or side grooves allowed trapped air in the tubes to escape through the weight.

Upon retraction of the drill string and bit, the drill rig moved to the next blasthole location. The helper retrieved the weight and checked the depth of the hole. Then he used a hoe to drag the drill cuttings back into the hole around the tube casing.

When the entire pattern was drilled, the drill rig moved off and the explosives were brought to the site and laid out by the holes. The depth was checked again. Loading was then done by the driller and helper. Measurements were made to make certain that stemming was adequate.

The separate charges in the tube cased holes were then tied together by the driller and checked by the Project Manager. The area was then cleared and the driller checked any electric blasting cap circuits. Flagmen were positioned on roads leading to the blast site to stop traffic.

After ascertaining that the area was clear, the driller signalled for the blast signal to be given. He then proceeded to initiate the blast.

The steps in the method are illustrated in the photographs in Figures 17 to 27 inclusive.

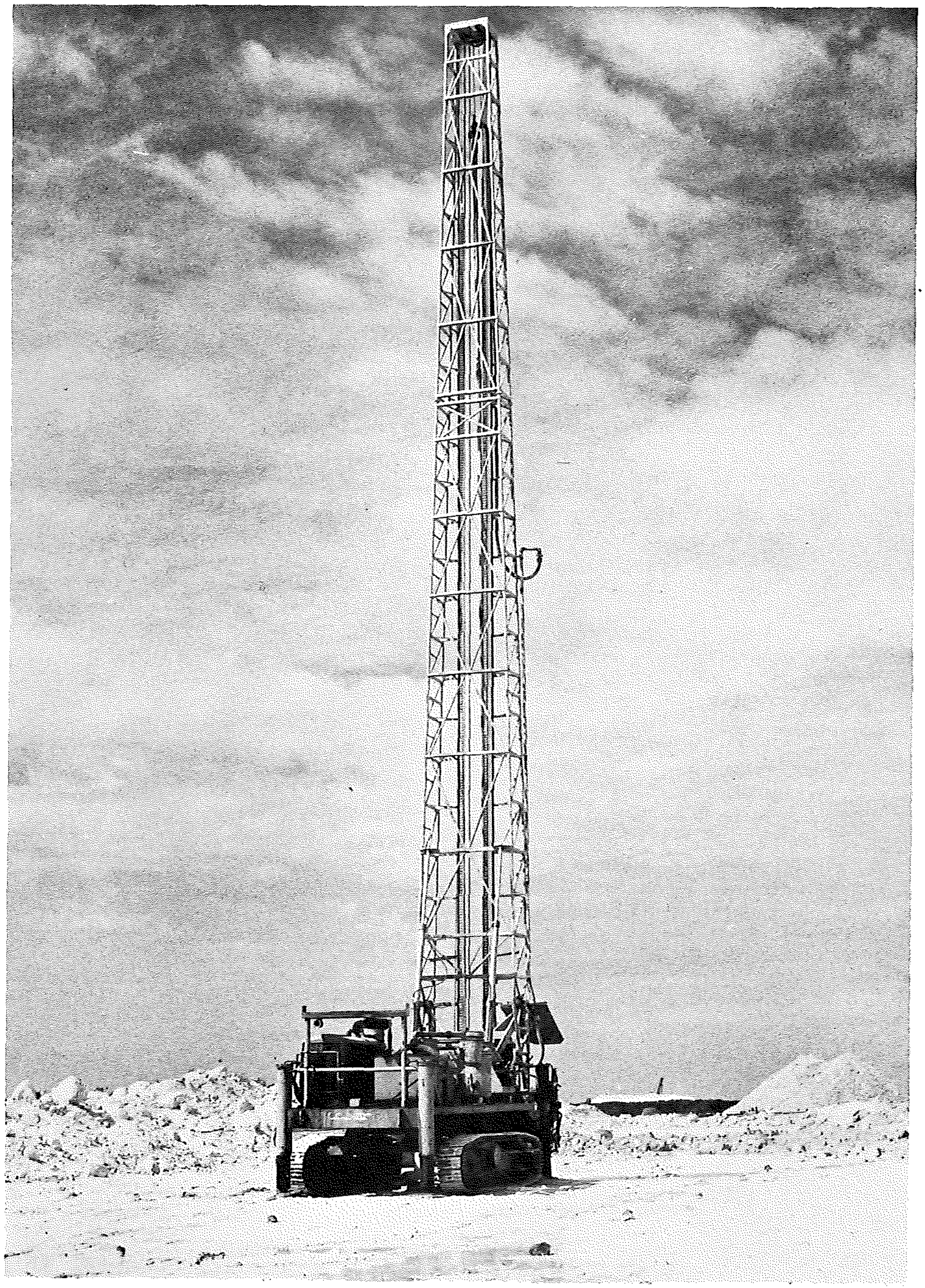


FIGURE 17

Typical drill rig used in South Florida. Because of the modifications made in order to load explosives through the kelly bar, it is called a "kelly bar rig".

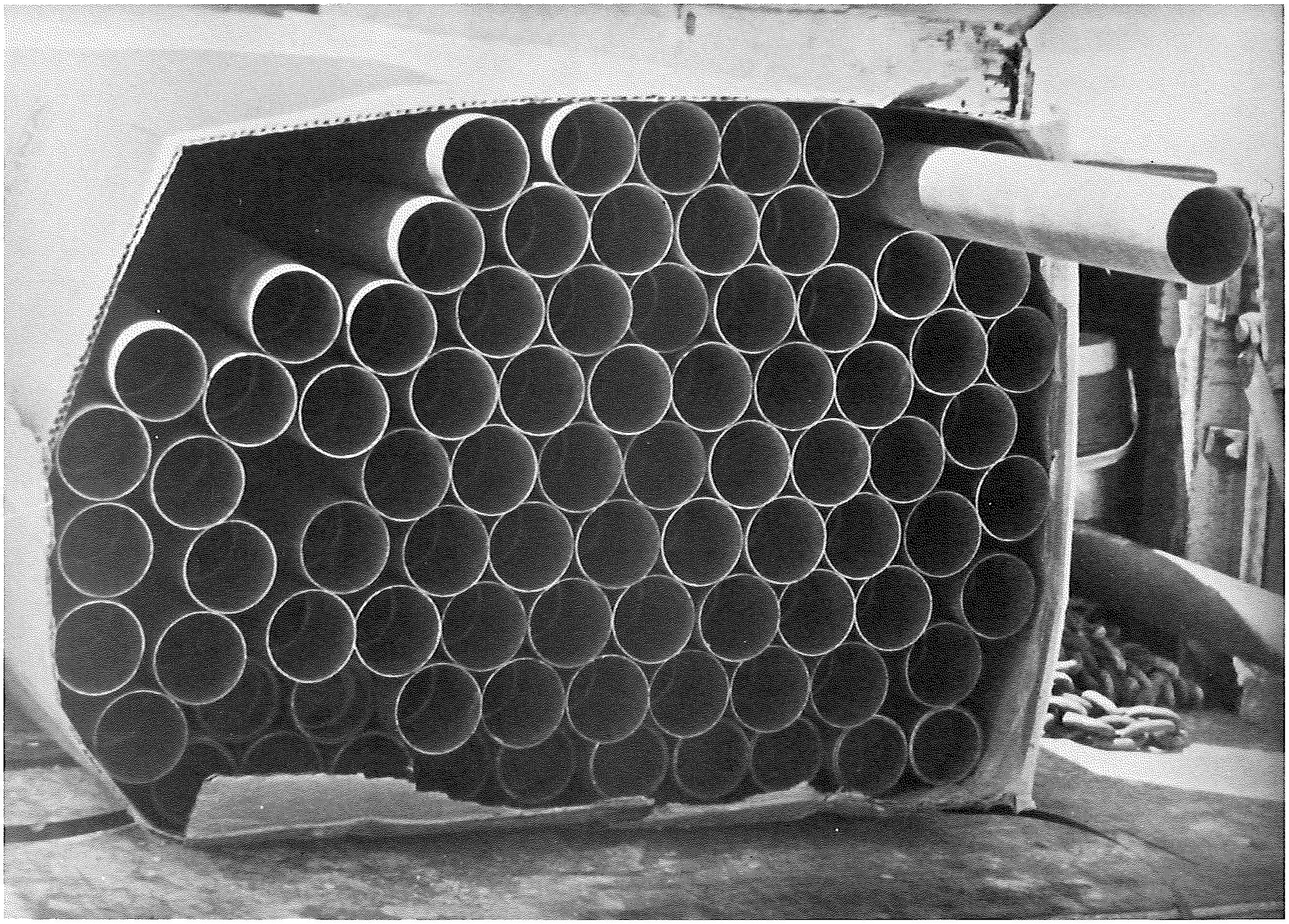


FIGURE 18

Open end of a box of 2.493" ID Kemax tubes.
Note external couplings and screen sections.

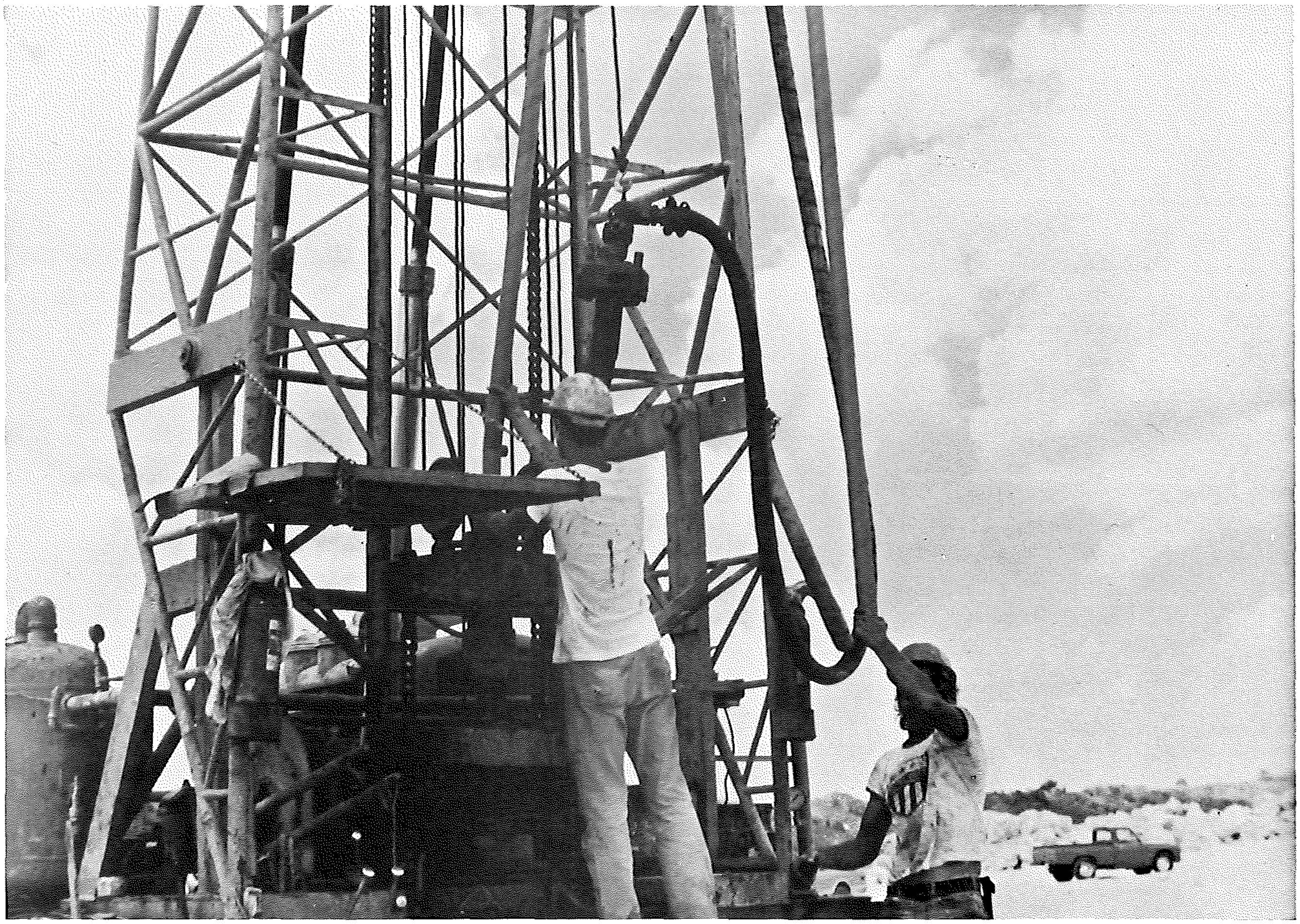


FIGURE 19

Driller handing a tube section to helper for connection to tubes in the kelly.

FIGURE 20

Closer view of tube placement

FIGURE 21

View of tubing in hole just after the kelly bar is extracted from the hole.

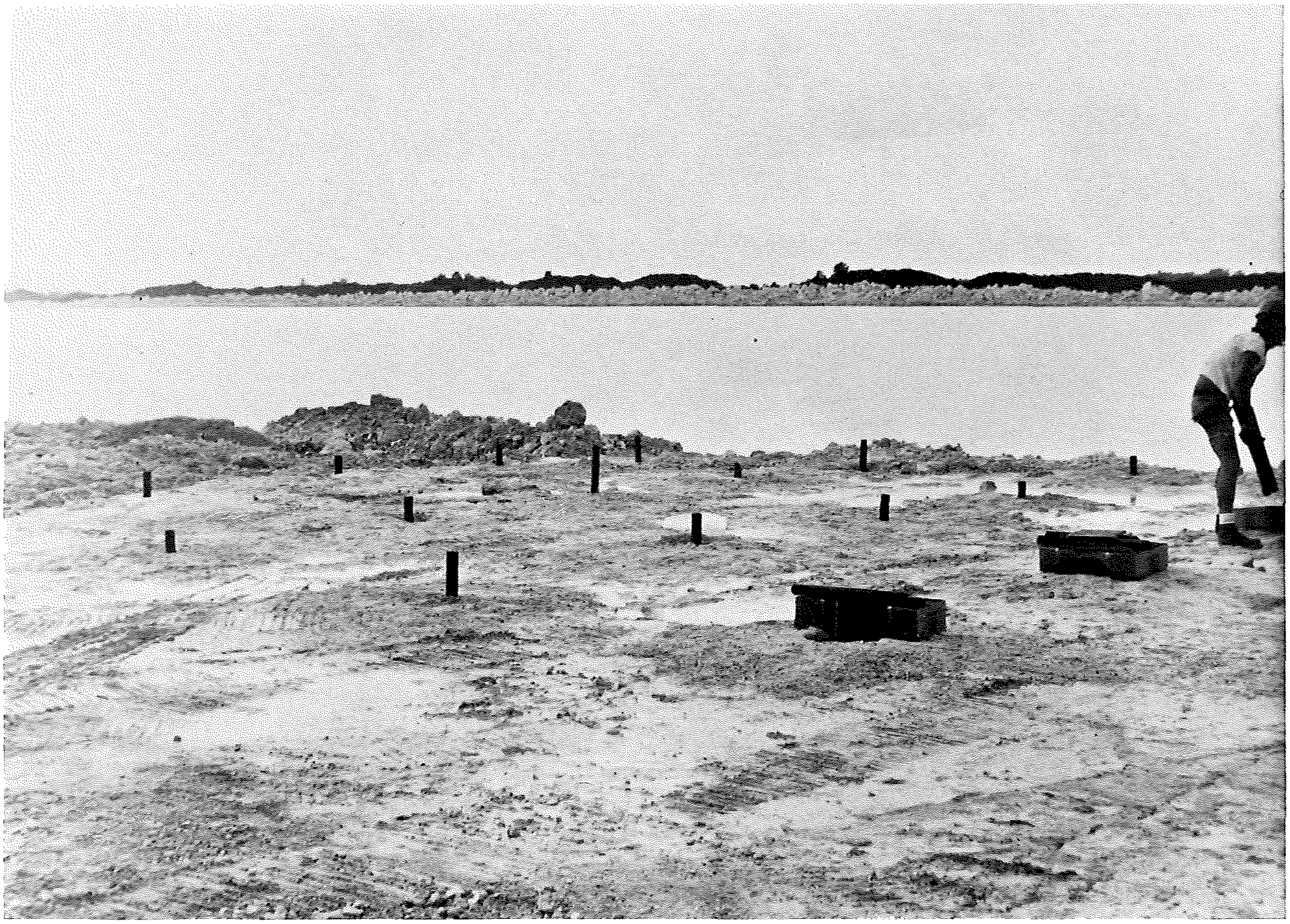


FIGURE 22

View of tube cased blastholes in a shot pattern prior to loading.

FIGURE 23

Loading continuous links of water gel explosives with detonating cord into tube cased blastholes.

FIGURE 24

Loading 2" X 24" dynamite cartridges into tube cased holes. Electric blasting cap leg wires can be seen extending to right of hole.

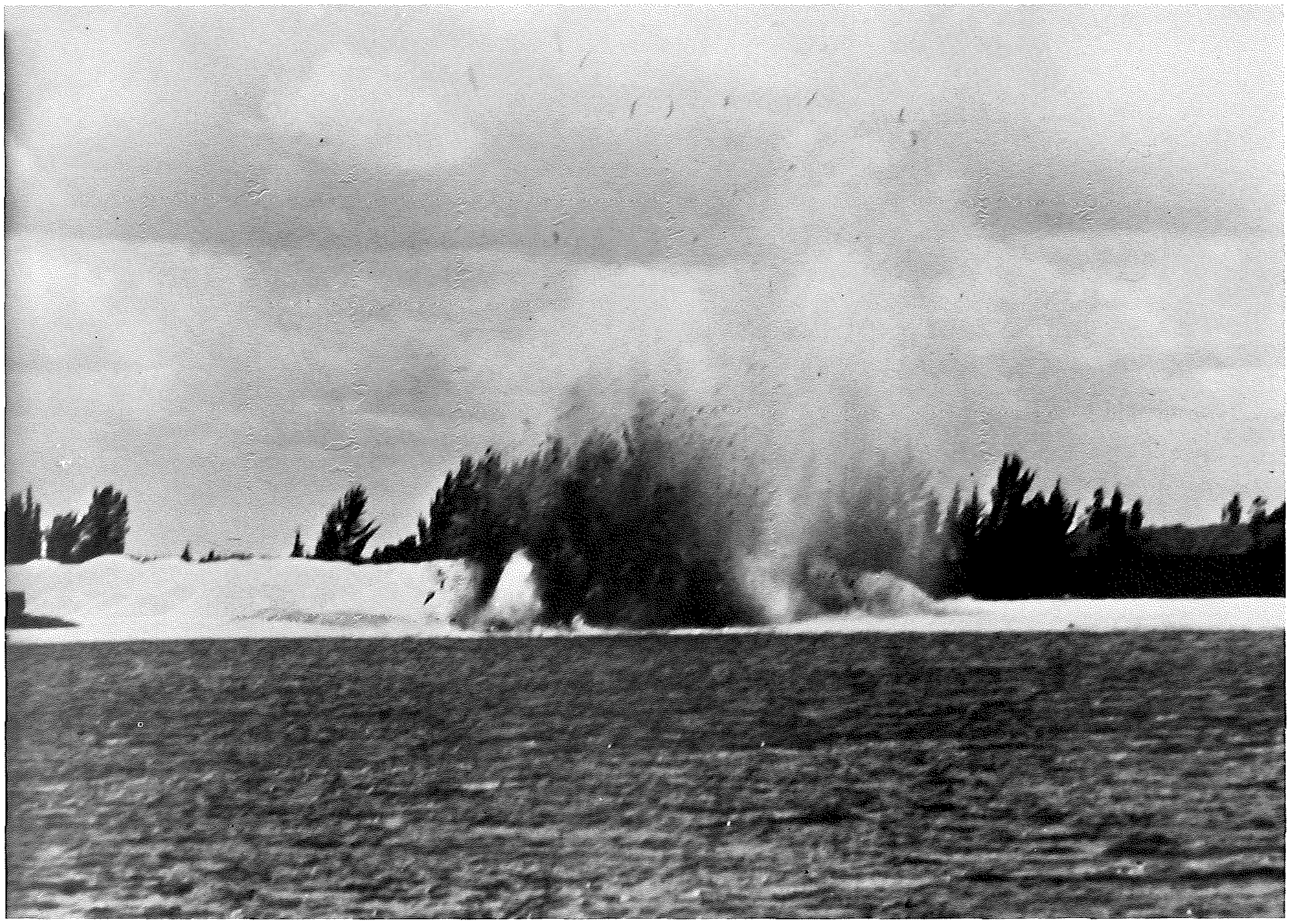


FIGURE 25

View of typical blast using
tube cased blastholes.

FIGURE 26

View of shot area after a blast. Delay pattern was a standard V toward the lake. Note well defined perimeter at rear of shot and the heave of the material in accord with the delay pattern. There was no buffer in front of shot.

FIGURE 27

View of typical dragline operation where dug material is placed in windrow parallel to water filled pit.

FIELD TEST RESULTS

In the 3" ID kelly bar part of the project, a total of 76 tests blasts were made at twelve different locations. Table 5 summarizes the amount of test effort at each location and provides totals for pertinent statistics such as number of holes (not including redrilled holes), pounds of explosive used, etc. All holes were 50 feet deep although the explosives were not always loaded to the full depth.

TUBE CASINGS:

Slightly more than 12 miles of Kemax tube casings were used in this part of the project. There is no doubt that this tubing can stabilize blastholes under the conditions found in South Florida. Of the 1206 holes that were stabilized, only one failed for reasons other than coring, proximity to blast, or human error. This was one of four tube-cased holes at Florida Rock and Sand Co. which were placed in a row 60 feet to the side of a planned test. All of the holes, including the 24 in the test blast, were left overnight. In the morning, the measuring tape would not go to the bottom of one of the four holes. There was no apparent explanation for the failure except that when the shot was made, two of the three remaining proximity test holes also failed at a depth of about forty feet, for a blast failure rate on this test of 67%. This is abnormally high and suggests that the tubes were placed in an unusual zone of weakness. For the whole project, 10 proximity test holes were placed at a distance of 60 feet to the side of a blast; only 3 of these failed of which two were in the high loss row at Florida Rock and Sand Co. When the proximity test data are examined, it can be seen that the loss at 60 feet is anomalously high because of this particular test.

Mention has already been made that Kemax tubes placed at Miramar Rock on March 7, 1977, were still open to full depth and full diameter of December 12, 1977. Normal production blasting was done within 165 feet of these tubes sometime around June 12, 1976 and there was no effect on the cased holes. In addition to this data, 69 tube casings were placed in very close proximity to blasts at various sites as the program progressed. Considering the water as the front side of the shot, 53 of these were placed to the side of the nearby blast, and 16 were placed behind the shot. The data for the 53 side proximity tests are listed in Table 6. It was surprising to discover that at a distance of twenty feet or more, there is at least a 70% probability that the tubes will remain open to full depth and full diameter, and can then be loaded as normal production blastholes. Had it not been for the failures at a distance of 60 feet at Florida Rock and Sand Co., this statement could have been modified to read, at least a 75% probability, and at 30 feet and beyond, an 89% probability that the tubes would be unaffected.

For the 16 tests placed behind the shot, the data are not as encouraging. It should be remembered, however, that these 16

TABLE 5: SUMMARY OF 3" ID KELLY BAR TEST EFFORT.

LOCATION	No. Holes	Drilling Time Hours	Loading Time	# Explosive	No. Holes Cored	% Holes Cored	Ft. Tubing Lost by Coring	Ft. Tubing Lost by Proximity	Total feet of Tubing used
Bergeron	82	52	6	6651	14	17.1%	259	51'	4492
Miami Crushed Rock	80	26.5	8.5	6416	7	8.8	0	104'	4184
Capeletti Pit #13	84	43	6.5	7250	7	8.3	204		4488
Sterling-Golden Prince	248	98	18.5	18530	31	12.5	729		13377
Miramar Rock	40	16	6	3120	7	17.5	23		2063
Rozzo	61	18.5	3	4575	3	4.9	7		3118
Florida Rock & Sand	247	98	27.25	20464	9	3.6	156	204'	12957
Redlands	100	45	7.5	5341	21	21.0	306		5406
Indian Lakes	63	24	10	4349	8	12.7	51		3264
Hardrives	96	43.3	7.25	8160	14	14.6	51	204'	5151
Gator Rock	52	22	3.5	1948	1	1.9	0		2652
Harper Bros.	53	14	5	710	2	3.8	0		2703
	1206	500.3	109	87514	124	10.3%	1786	563	63855

TABLE 6

SIDE BLAST

TUBE CASING PROXIMITY TESTS

LOCATION	DISTANCE FROM BLAST	NO DAMAGE	FAILED
Miramar Rock	10'		1
	20'	1	
	30'	1	
	40'	1	
Bergeron	18'		1
	27'	1	
Miami Crushed Rock	20'	6	2
Florida Rock & Sand	40'	8	1
	50'	8	1
	60'	7	2
Hardrives	9'		1
	18'	3	1
	27'	2	1
	36'	2	
	45'	1	
	60'		1

DISTRIBUTION BY DISTANCE

	NO DAMAGE	FAILURE	SURVIVAL PROBABILITY
9'		1	0.0
10'		1	0.0
18'		3	0.0
20'	7	2	0.77
27'	3	1	0.75
30'	1		1.00
36'	2		1.00
40'	9	1	0.90
45'	1		1.00
50'	8	1	0.89
60'	7	3	0.70

holes were all in connection with one blast. The results are shown in Table 7. Most of these failed at a depth of about 40 feet where a loose sand layer was present.

TABLE 7

PROXIMITY TEST DATA FOR TUBE CASINGS PLACED

OPPOSITE THE LAKE SIDE OF THE SHOT.

DISTANCE	NO DAMAGE	FAILED	SURVIVAL PROBABILITY
24'	0	8	0.0
32'	3	5	0.38

These proximity tests indicate how closely an operator could set additional tubes along the intended cut. For example, if one was limited to a shot of a certain size for other reasons, and if weather prohibited loading after drilling and placing the tubes, one could continue to drill additional holes in the next pattern. This would give better utilization of the equipment and the manpower.

This was done during the test program at Miami Crushed Rock where vibration complaints have caused the company to restrict their blasts to eight holes (two rows perpendicular to the water) at one time, each hole on a separate delay. There were two occasions when the eight holes were completed but special explosives or detonators had not yet arrived at the site for loading. Drilling was continued, placing tube casings 20 feet from the intended shot. Of the eight tubes so placed, two of them,or 25% failed. Although this may seem high, the drill rig was kept busy for 3 to 4 hours instead of being idle. This time is worth 3 X $55.50/hr.= $166.50 to someone. The loss of the two holes caused additional cost of:

102' Tubing @ $0.183	$18.67
45' minutes to redrill holes= $\frac{45}{60}$ x $55.50/hr.=	$41.63
Total	$60.30

Savings resulting from being able to continue drilling:

$166.50 - $60.30 = $106.20

If an operator wanted to do this routinely, such as drilling every other row on a second shift, the loss could be reduced by using a slightly thicker walled tube, increasing the spacing another foot with the same diameter cartridge, or use a 2 1/4" cartridge and increase the spacing by 28%. Explosives of this size, both in spiral wound cartridges, and in polyethylene bags, were loaded into the 2.493" ID tube casings with 1/4" downlines without difficulty, if the downline was held taut.

The tube presented no product contamination problem. In obtaining processing rate statistics, only three small (6"-8") sections were seen entering the primary crusher. Some operators reported seeing a very minor amount of the tube casing material when aggregate was screened, but not in any quantity that would present processing problem or a product quality problem.

Samples of rock were taken from material that had passed through the primary crusher. The samples were blended and taken to Wingerter Laboratories for examination. In a 1.5 cubic foot combined sample, they found three small particles, the biggest of which was .01" X .08" X .12" in size. This is substantially below the 1% limit on deleterious substances if, in fact, the material is deleterious.

In another test, 1/2"± fragments of Kemax tube were added to concrete in measured proportions. 6" X 12" cylinders were then poured and tested after 7 days for compressive strength. The results of these tests are given in Table 8. A graph of the test results is shown in Figure 28 which indicates that Kemax tube material can be present in quantities up to 1.3% by weight, or 3.5% by volume before any loss in concrete strength occurs. In lesser amounts it appears to improve the strength of the concrete.

In a 10' x 10' pattern, with blastholes 50' deep, the uniform distribution of a 2.493" ID tube would result in the presence of .002% Kemax material by weight in the coarse and fine aggregate, and even less in concrete made with such aggregate. The test cylinders indicate that concrete can tolerate 650 times this amount with no loss in strength.

Coring is a word that is used to describe an unintended rock jam in the bit which prevents the tube casing from remaining in the hole. When a hole is cored, the air or water pressure normally remains high when the bit is pulled off bottom. The driller can then take corrective action before the tubing is placed in the kelly. However, when drilling with water, the pressure difference is relatively small with respect to the gradations on the gage and the driller will commonly depend on other signs such as excess water continuing to spray out of the swivel when the bit is raised. Nevertheless, a rock core sometimes remains in the bit and this is evident when the bit is pulled out of the hole and the tubing cannot be seen.

When this happens, the safest procedure is to raise the bit above the drive bushing, secure the lower end of the kelly and lower the mast. The rock jam and tubing can then be removed. Pulling the tubing out while the kelly and mast are upright is unsafe because nothing is holding the kelly but a single line cable and the brake on the drawworks. This could be made safer by using two 3/8" steel rods which could be placed between the links of the pull-down chains just above the sprockets. This would provide two more

TABLE 8

RESULTS OF CONCRETE CYLINDER TESTS

Sample No.	1	2	3	4	5
Kemax Weight %	0%	0.5%	1.0%	1.5%	2.0%
Kemax Volume %	0%	1.4%	2.7%	4.1%	5.4%
Slump in inches	5.75"	5.75"	5.75"	5.75"	5.75"
Unit load in psi	5094	5518	5306	4988	4492

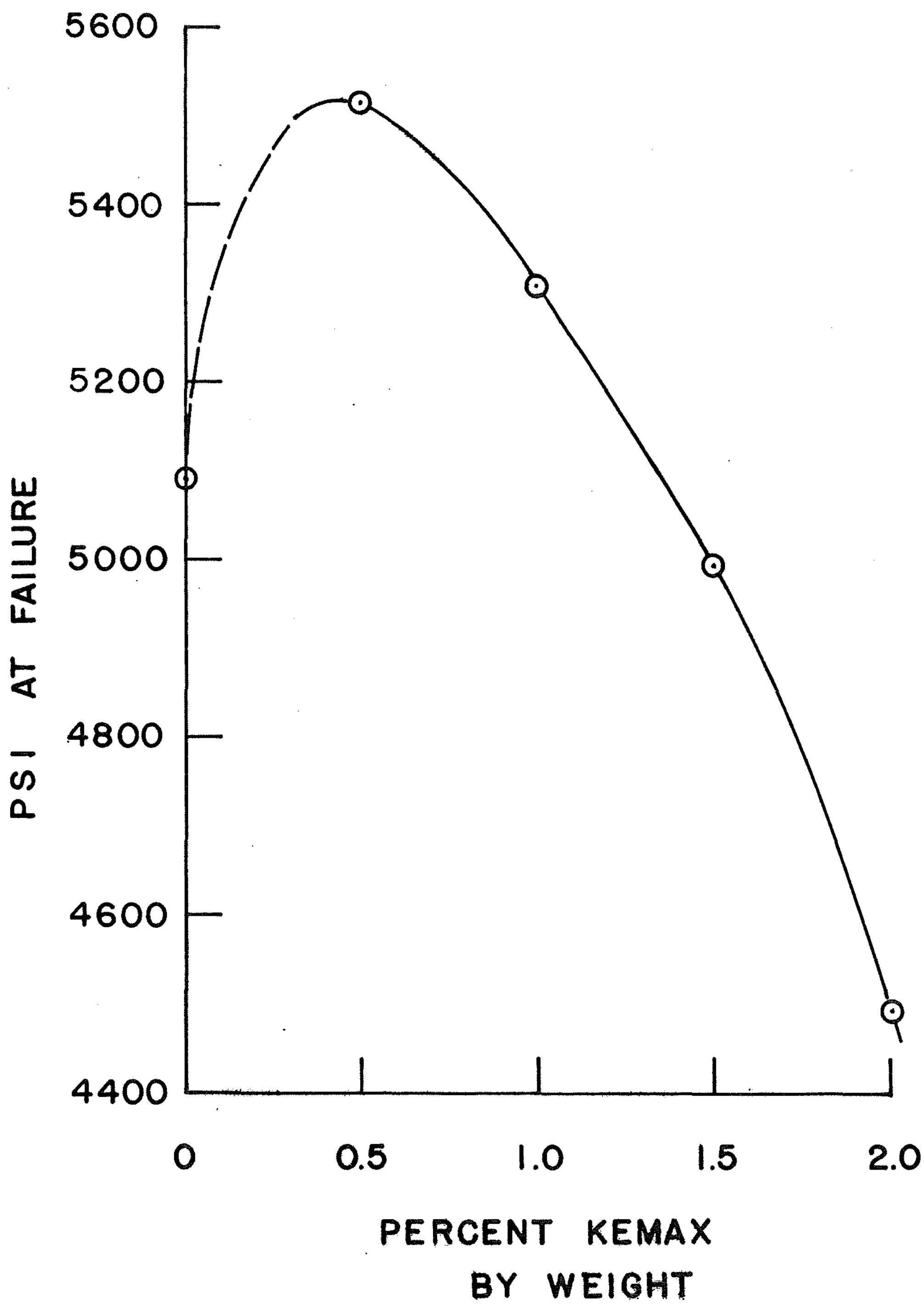


FIGURE 28

GRAPH OF CONCRETE CYLINDER COMPRESSIVE STRENGTH VERSUS PERCENT OF KEMAX TUBE MATERIAL BY WEIGHT IN THE CONCRETE independent supports for the kelly. In addition, a hinged split steel tube section could be clamped on the drill string just below the wear bars on the subadaptor. This clamp-on device would have legs or flanges which would extend out far enough to prevent the drill string from passing downward through the rotary table.

In most cases the tubing that is in the kelly can be recovered and reused. The hole has to be redrilled to clean out the sloughed materials. When the tubing is recovered, the main problem is the loss of time in redrilling the hole. All of this lost time during the test program is included in the drilling times shown.

Coring at Bergeron, Miramar, and Redlands appears excessive, but there were extenuating circumstances. Bergeron was the first location tested and the driller was just acquiring experience with the 3" ID kelly. Five of the 14 cores taken at this location were on the first test blast. At Miramar Rock, the regular drilling crew was not present. At Redlands Construction Co., 19 of the 21 cores were encountered in the first 57 holes. Only two cores were taken in the last 43 holes. This was because the driller learned to reduce his pulldown pressure for the last three or four feet of hole. Because of these three locations, the overall average of 10.3% cored is probably on the high side. For a driller who operates in the same pit for a considerable length of time, the percentage could be reduced considerably. Of the tubing involved in the cored holes, only 28% was lost. The remainder was recovered and reused. This represents a coring loss of only 2.8% of the total tubing used in the program.

The tubes can be stored in any place where the outside corrugated box does not get wet. This doesn't hurt the tubes but handling is difficult if the box loses its strength. One man can pull a box across a smooth floor and load it in a pickup. Two boxes will fit side by side in the standard pickup bed, and the tail gate can be closed, if the 7.3' sections are used.

Bottom screens were used in the program to provide the capability to use bulk slurry. They are not necessary with cartridged products and with air drilling, where the water is muddy, they increase the buoyancy to the point that placement is more difficult. Many of these screens were cut open to aid placement. Screens can be a problem in another way because they offer the opportunity for a screen section to be placed upside down. When this happens, the explosives won't go to bottom and the screen cannot be removed in the hole sufficiently to permit the passage of cartridges. In these circumstances the hole needs to be redrilled if the explosives can be removed with the cord downline.

Care must also be taken that the tube sections are counted as they are brought to the drill rig, and as they are placed in the hole. If less than the proper number is placed, the hole, the tubes, and the 10 lb. weight will be lost. The weight could damage the impact crusher if it is loaded into the feed hopper.

If too many tubes are placed, the bit will not clear the top of the column when it is pulled out of the hole. The column has to be broken or cut and the top section is generally lost, and there is a possibility that the weight might be lost in the hole beside the tube.

Obviously, it is good practice to have an extra weight on hand in case of accidental loss. Otherwise, loss of the weight will shut down the placement operation.

If the stemming interval is greater than the length of the top section, the top section can be recovered and reused. The two main problems with doing this are that downlines or legwires have to be pulled through the tube, and one might pull more than one section of tubing out of the hole. An expandable packer on the end of a tamping pole could be designed which would expand outward against the walls of the top sleeve, and permit the top section to be recovered. This would increase the possibility of the cord being lost in the hole and eliminates the possibility of handling a misfire.

Even when tubes fail at a distance of 10 feet from the blast, it is usually possible to reenter the top 10 to 20 feet of the hole. In case of a cut-off where a hole or holes don't explode, one can return to the area after a suitable time and prime the top of the holes with a single cartridge. When these are shot they will probably cause the whole column to explode because of a continuous length of detonating cord even in the broken tube area, or propagation because of the close proximity. This argues against putting rock cuttings in the tube as stemming material.

Stemming the top of the tube with cuttings did not appear to do anything more than the water which is always above the charge. If the holes were going to rifle (eject a geyser of water) because of poor explosive action, they did so in either event. Coarse sized stone of nominal 1" size was not commonly available and this might have made some difference. No difference was observed in amount of flyrock between water stemmed or rock stemmed holes.

Backfilling the cuttings in the annulus between the tubing and the rock bore is, on the other hand, important. This could not be quantified but backfilling produced better results. Backfilling can be done by the driller's helper while the next hole is being drilled. He generally has sufficient time to backfill the hole and deliver the tube sections to the drill rig for placement. Two men can easily handle the drilling and placement operation. Backfilling also appears to improve the chances for survival of unloaded tube casings near the blast.

Sand fill-up in the tubes after placement is not a problem when water is used in drilling. With air, the bubbling action at the top of nearby holes can be very active. If sand is present near the bottom of the hole, there may be some fill-up. Because the drill rig was equipped to drill with water or air, both were used in the test program although the great majority of the drilling was done with water. The most severe sand fill-up was at the Bergeron pit where most air drilled holes experienced a bottom loss of one to two feet. One hole lost four feet. If this was a common problem, one could use a longer sub and drill one or two extra feet.

Air drilling at Miami Crushed Rock, Sterling Crushed Stone's Golden Prince quarry, Florida Rock and Sand Co., L.W. Rozzo's pit, Miramar Rock, and Capeletti Pits #12 and #13, caused no significant problem in tube placement or in maintaining bottom. Nevertheless, drilling with air is more difficult regardless of whether tubes are being used or not.

EXPLOSIVES, DETONATORS & DELAY INTERVALS

The 76 tests conducted in the 3" ID kelly bar program are listed in Table 9. The abbreviations used for explosives represent the following:

AL-10	Austin AL-10 gelatin dynamite
APG-B-1:	Austin Apcogel B-1
FLO:	Atlas Florigel semi-gelatin dynamite
GPA-2:	Hercules Gel-Power A-2 aluminized water-gel
G-915W:	Gulf 915 W aluminized water-gel
Herc.40%E:	Hercules 40% Extra dynamite
Herc.60%E:	Hercules 60% Extra dynamite
PGD:	Hercules Power Gel D semi-gelatin dynamite
PM100:	Atlas Powermax 100 emulsion
PM200:	Atlas Powermax 200 aluminized emulsion
RXL330:	Atlas gelatin dynamite with 1.7 density
SLM100:	Austin Slurmex 100, aluminized water-gel
TOV:	DuPont Tovex 255 aluminized water-gel
TOV(4):	DuPont Tovex 255 in 4 mil. loose pack chubs
US-318:	Trojan US-318 semi-gelatin dynamite

At each site, the operator indicated his supplier, and purchases were made accordingly unless certain products were unavailable. Each distributor or technical representative in the Miami area was asked for assistance and input to the program.

TABLE 9: TESTS IN 3" ID KELLY PROGRAM

Test No.	Location	B X S	No. Holes	Drilling Time Hours	Loading Time Hours	Explosives	#/hole	CY/hole	LBS/CY	Weight energy in million ft-lbs. per/CY
1	Bergeron	8 x 8	18	14.0	1.0	PGD	83.6	119	0.70	0.70
2	"	8 x 10	16	11.0	2.0	PGD	83.6	148	0.56	0.56
3	"	9 x 10	16	9.0	1.5	TOV	79.0	167	0.47	0.66
4	"	10 x 10	16	12.0	1.0	TOV	82.0	185	0.44	0.62
5	"	10 x 10	16	6.0	0.5	TOV	77.0	185	0.42	0.58
6	Miami Crushed Rock	8 x 10	8	2.5	0.5	PGD	83.6	125	0.67	0.67
7	"	10 x 10	8	3.0	0.5	PGD	83.6	185	0.45	0.45
8	"	10 x 10	8	2.0	0.5	PGD	83.6	185	0.45	0.45
9	"	10 x 11	8	4.0	1.0	PGD	83.6	204	0.41	0.41
10	"	10 x 11	8	3.0	1.0	PGD	83.6	204	0.41	0.41
11	"	10 x 11	8	2.0	1.0	GPA-2	75.0	204	0.37	0.46
12	"	10 x 11	8	2.5	1.0	GPA-2	75.0	204	0.37	0.46
13	"	10 x 11	8	2.5	1.0	GPA-2	75.0	204	0.37	0.46
14	"	10 x 10	8	2.5	1.0	GPA-2	75.0	185	0.41	0.51
15	"	10 x 11	8	2.5	1.0	TOV	84.0	204	0.41	0.57
16	Capeletti Bros.	9 x 8	24	12.0	1.5	PGD	83.6	133	0.63	0.63
17	"	9 x 9	24	11.0	2.0	PGD	87.4	150	0.58	0.58
18	"	10 x 9	36	20.0	3.0	PGD	87.4	167	0.52	0.52
19	Sterling	8 x 9 3/4	18	7.0	1.0	FLO	77.2	144	0.53	
20	"	9 x 9	12	4.5	0.5	FLO	78.6	150	0.52	
21	"	9 x 9 3/4	8	4.0	1.0	PM100	69.5	163	0.43	0.48
22	"	8 x 9 3/4	18	7.0	1.0	PM100	69.5	144	0.48	0.54
23	"	10 x 9 3/4	18	8.0	1.5	FLO	78.5	180	0.43	
24	"	9 x 9 3/4	18	10.0	2.0	PM100	69.5	163	0.43	0.48
25	"	10 x 9 3/4	18	11.0	0.5	PM200	71.0	181	0.39	0.52
26	"	13 x 9 3/4	18	8.0	1.0	RXL330	100.0	234	0.43	
27	"		1	0.25		PM200	71.0			
28	"	9 x 10	7	2.25	0.25	PM200	71.0	166	0.43	0.48
29	"	8.3x 10.5	32	12.0	2.75	PM100	69.0	160	0.43	0.48
30	"	8.3x 10.5	32	8.5	3.5	PM200	68.0	161	0.42	0.56
31	"	9 x 11	24	8.0	2.0	FLO	75.0	183	0.41	
32	"	9 x 11	24	7.5	1.5	FLO	79.0	183	0.43	
33	Miramar Rock	10 x 10	20	8.0	3.0	GPA-2	75.0	185	0.41	0.51
34	"		20	8.0	3.0	GPA-2	81.0	185	0.43	0.55
35	L.W. Rozzo	8 x 9	20	5.5	1.0	US-318	75.0	133	0.56	
36	"	8 x 10	20	6.0	1.0	US-318	75.0	148	0.51	
37	"	8 x 10	21	7.0	1.0	US-318	75.0	148	0.51	

Test No.	Location	B x S	No. Holes	Drilling Time Hours	Loading Time Hours	Explosives	#/hole	CY/hole	LBS/CY	Weight energy in million ft-lbs per/CY
38	Fla.Rock & Sand	11 x 10½	15	9.0	1.0	TOV	78.0	214	0.37	0.51
39	"	11 x 10½	15	7.0	1.0	TOV	78.0	214	0.37	0.51
40	"	11 x 10½	18	9.0	1.0	AL-10	81.0	214	0.38	0.34
41	"	11 x 10½	18	9.0	1.0	AL-10	81.0	214	0.38	0.34
42	"	11 x 10½	18	6.0	1.0	TOV	78.0	214	0.37	0.51
43	"	11 x 12	15	6.5	4.0	TOV(4)	72.0	244	0.30	0.41
44	"	11 x 10½	20	6.5	1.0	TOV(4)	72.0	214	0.34	0.47
45	"	11 x 14	20	7.5	1.5	G-915W	104.0	285	0.37	0.59
46	"	11 x 13	20	6.5	7.0	G-915W	104.0	265	0.39	0.62
47	"	11 x 10½	20	6.5	1.0	G-915W	91.0	214	0.43	0.69
48	"	11 x 10½	20	6.75	1.25	TOV	78.0	214	0.37	0.51
49	"	11 x 10½	24	8.0	5.0	TOV	78.0	214	0.37	0.51
50	"	11 x 10½	24	9.75	1.5	TOV	78.0	214	0.37	0.51
51	Redlands	9 x 10	20	8.0	1.0	APG B-1	47.0	120	0.39	0.43
52	"	9 x 11	23	8.0	1.0	APG B-1	47.0	132	0.36	0.47
53	"	9 x 11	14	7.0	1.0	SLM 100	59.0	132	0.45	0.55
54	"	9 x 11	19	9.0	2.0	AL-10	58.0	132	0.44	0.40
55	"	9 x 11	18	10.0	1.5	AL-10	58.0	132	0.44	0.40
56	"	9 x 11	6	3.0	1.0	AL-10	58.0	132	0.44	0.40
57	Indian Lakes	9 x 11	9	4.0	1.0	PGD	83.6	183	0.46	0.46
58	"	9 x 11	9	5.0	1.5	PGD	83.6	183	0.46	0.46
59	"	9 x 11	9	3.5	2.0	PGD	68.0	183	0.37	0.37
60	"	9 x 11	9	3.0	1.5	PCD	65.0	183	0.36	0.36
61	"	9 x 11	18	5.5	3.0	FLO	61.0	183	0.33	
62	"	9 x 11	9	3.0	1.0	GPA-2	61.0	183	0.33	0.41
63	Hardrives	8 x 9	30	16.5	2.0	AL-10	85.0	133	0.64	0.58
64	"	8 x 9	15	7.5	1.0	AL-10	85.0	133	0.64	0.58
65	"	8 x 10	21	8.25	1.5	AL-10	85.0	148	0.57	0.51
66	"	8 x 10	20	6.75	2.0	AL-10	85.0	148	0.57	0.51
67	"	8 x 10	10	4.33	0.75	AL-10	85.0	148	0.57	0.51
68	Gator Rock	8 x 9	16	6.25	2.0	GPA-2	34.0	80	0.43	0.54
69	"	9 x 11	18	8.0	0.75	Herc.40%E	39.0	110	0.35	0.32
70	"	9 x 11	18	7.75	0.75	Herc.40%E	39.0	110	0.35	0.32
71	Harper Bros.	7 x 13	8	2.0	0.75	Herc.60%E	18.0	101	0.18	0.18
72	"	7 x 13	10	3.0	0.75	Herc.60%E	14.0	101	0.14	0.14
73	"	7 x 13	10	2.75	0.75	Herc.60%E	14.0	101	0.14	0.14
74	"	7 x 13	10	3.00	0.50	Herc.60%E	14.0	101	0.14	0.14
75	"	7 x 13	10	2.00	1.25	Herc.60%E	14.0	101	0.14	0.14
76	"	7 x 13	5	1.25	1.00	Herc.60%E	14.0	101	0.14	0.14

Cooperation was excellent and four companies, Atlas, DuPont, Gulf, and Austin sent technical specialists to Miami to participate in certain aspects of the test program.

The author has never been employed by an explosives company and is not privy to any special inside information. The author's experience with explosives was gained from their use in seismic and quarry operations. Consequently, the scope of the program and the evaluation of results is weighted primarily towards an operator's practical viewpoint rather than that of an explosives technical specialist.

The Engineering Contractors Association's contract with the U.S. Bureau of Mines called for the testing of different types of explosives in at least three different locations.

Most small diameter quarry operations in South Florida use semi-gelatin dynamite, or ammonia dynamite, and to a lesser extent, packaged water-gels. There is only one water gel explosive which can be obtained in South Florida without a special order and which is in everyday commercial use. This placed some limitations on the program because of the primary stress on geographic coverage. Some operators were reluctant to use slurry products citing bad experiences in the past. Some manufacturers did not want to make slurry products available. Consequently, the program moved from site-to-site demonstrating that the tube casings could stabilize the holes. At each site, a test program was utilized which was as broad as the operator and time would permit. Cooperation of operators and quarry personnel was excellent.

Although there are differences in the dynamites used, and even greater differences in the packaged water gels and emulsions, these explosives are grouped into two main groups for evaluation analysis; packaged slurry and dynamite. A third group, bulk slurry, is included but the data for these products comes from the unit tests and not the 3" ID kelly bar program. These tests are designated with a U for unit. Other unit tests are also included because they were performed with 2" products in 2.493" ID tubes. All packaged water gels and emulsions used were Class A high explosives. All bulk slurries were blasting agents.

Slurries are poorly understood in South Florida, and probably by the quarry industry in general. This becomes especially critical in South Florida where the small diameter application in water filled holes creates special problems.

Making slurries and water gels is considerably more complicated than making dynamite according to knowledgeable sources and there are many more trade secrets. Consequently, it is difficult for an operator to obtain a good concept of the differences between dry blasting agents, bulk slurry blasting agents, bulk slurry explosives, water gels and emulsions. USBM Information Circular 8560, "The Impact of Blasting Agents and Slurries on Explosives Technology" by Richard A. Dick provides a good summary. The classification of blasting agents and slurries given in that paper is given in Figure 29. In Dick's classification, water gel and emulsion explosives would be included within the group of slurry explosives. However, in this report the terms "water gel" and "emulsion" are frequently used to conform to the usage of the explosive manufacurers.

Slurries may be blasting agents which are sometimes called NCN slurries. These do not contain explosive ingredients and are not sensitive to a No. 8 blasting cap. They may be slurry explosives which do contain explosive sensitizers and are usually cap sensitive. Both must contain air and this is particularly important when used in a water filled hole.

Air bubbles will tend to collapse as the slurry is placed in increasing depth of water so artificial means of supporting the bubble structure must be employed. This is generally done with glass or plastic microballoons, or cork, and such support is generally considered necessary at depths below 30 feet. Not only are the bubbles affected by depth of water, but they may be affected by shock pulses from detonating cord in the same hole or by pulses from nearby holes which are exploded earlier in the pattern. If these pulses collapse the bubbles, the slurry will not explode. This phenomenon is referred to as precompression, or compression prior to detonation.

In some slurries the bubbles may recover after the compressive stress is removed. In others, the bubble support structure may be ruptured and the explosive is premanently damaged. Sometimes by rubbing the precompressed slurry between the fingers, the skin will be scratched by the broken microballoons, not sufficiently to bleed but to be visibly noticeable after the hands are washed and dried.

Successful blasting with slurries in South Florida obviously requires a good understanding of this precompression phenomenon and recognition of its effects. This project addressed the problem to some degree and some ideas were generated but the subject requires considerably more research and field testing, particularly with nearby instrumented holes to monitor the arrival of various pulses.

Post-shot analysis was performed on some test blast series, but for many operations the test blasts have not yet been dug and/or processed. The time schedule for the project required that some semi-quantitative method for evaluating the shot had to be developed to draw conclusions from different tests that would not be dug by project completion time.

A system of hole-by-hole appraisal was developed when the unit tests were being conducted in February, 1977. This focused primarily on the enlargement of the blast-holes and the development of radial cracks. It served well for marginal to sub-marginal shots but too easily resulted in a perfect score for anything that looked acceptable. This system was abandoned.

Another system was developed which incorporated most of the visual characteristics used by quarry personnel in South Florida in making on-the-spot evaluations of a blast. The system consists of

(From USBM IC 8560)

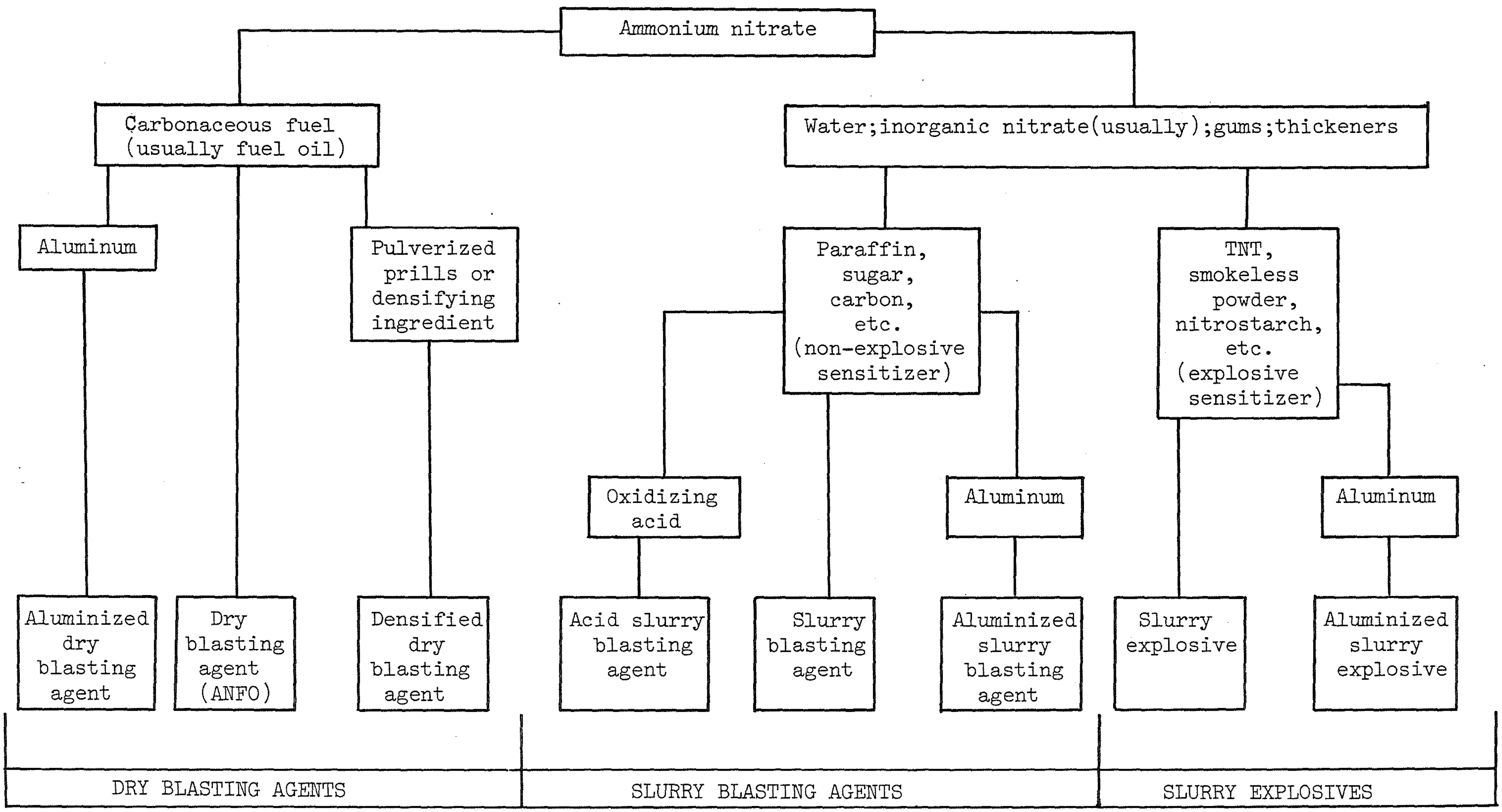


FIGURE 29
CLASSIFICATION OF BLASTING AGENTS AND SLURRIES

13 different points which are rated from 0 to 10. Equal weight is given to all points which is almost certainly not correct but any effort to weight them would probably be equally incorrect. The system is subjective but it provides values which agree fairly well with the impressions of trained observers. It applies only to South Florida where the evaluation has to depend on appearance during the shot and of the surface above water after the shot. A copy of the rating form is shown in Figure 30.

A brief explanation of the rated points gives an idea how they relate to the total shot performance.

Surface Fragmentation: Just what it means in other areas except that in South Florida it is affected strongly by the material used to construct the pad. If oversize is used in order to subject it to another breakage opportunity, the resulting after-shot appearance may be relatively coarse.

Heave with delay: This term is used to indicate evidence of horizontal movement in accord with the delay pattern.

Swell: This refers to evidence of vertical change and as used in these evaluations may actually be sink as well as swell. Although the material may not heave horizontally, if parts of the shot rock are two to three feet above the general pad level, or two or three feet below, something has happened.

Perimeter Cavity: This is related to heave if the delay pattern was designed to move the material from unshot rock. It gives a measure of how well the delay pattern was working at the tail end of the shot and whether the last few holes exploded as planned. Cavity may be 10 to 20 feet deep immediately after the shot.

Scarp: Refers to the sharpness of definition of the shot area by the scarp which separates shot rock from the pad.

Backbreak: In most areas, this is a negative sign and indicates energy wasted in unwanted breakage. In South Florida, most backbreak is probably limited largely to the pad and is caused by tensional relief of this unbound material. Some backbreak is nearly always present and if not, the shot is usually a failure. Therefore, it is considered a positive aspect in this grading system.

Absence of nitrogen oxide cloud: This develops with both dynamite and slurry explosives. It is included in the evaluation because the density of the cloud is probably inversely related to the effectiveness of the explosive.

Absence of Water Geysers: This refers to the visible sprays of water and rock stemming which rise vertically from the

FIGURE 30

SURFACE EVALUATION GRADING FORM

SHOT EVALUATION:	0	1	2	3	4	5	6	7	8	9	10
OVERALL: Total ÷ 13 =	-	-	-	-	-	-	-	-	-	-	-
SURFACE FRAGMENTATION:											
HEAVE WITH DELAY:											
SWELL:											
PERIMETER CAVITY:											
SCARP:											
BACKBREAK:											
ABSENCE OF NO_x CLOUD:											
ABSENCE OF WATER GEYSERS:											
ABSENCE OF STANDING TUBES:											
HOLE OBLITERATION:											
ABSENCE OF UNEXPLODED MAT.:											
WATER PUSH:											
ABSENCE OF FLYROCK:											

blasthole indicating that the explosive action is weak and the energy is largely being vented out of the blastholes rather than breaking rock. Sometimes called "rifling".

Absence of Standing Tubes: If the explosive action is normal, the tube casings in the stemmed interval will be well broken up and mixed with the rock. If subnormal, tubes may still be present sticking out of the hole. This is particularly true of those holes in which the explosives are apparently affected by precompression. Pulling the tube from the ground will sometimes reveal unexploded material still in cartridge form in the tube.

Hole Obliteration: Refers to the visual impact of looking at the shot area and not being able to identify the specific locations of the blastholes.

Absence of Unexploded Material: Applies to material either in standing tubes or lying about on the surface.

Water Push: This is the movement of the water out of the shot area which can leave dry-appearing perimeter cavities that may not fill up for 10 minutes or so.

Absence of Flyrock: Self explanatory.

Values for these 13 points were added and then divided by 13 to obtain an overall average surface evaluation grade ranging from 0 to 10.

The grades had to be applied retroactively from notes and slow motion movies to the unit tests and some of the early 3" ID kelly tests. Slow motion movies (32 frames per second) were made of most of the test blasts.

Surface evaluation grades for each of the 3" ID kelly tests, the unit tests, and two special tests are listed in Table 10, along with the initiation system, the delay interval, the load factor and the explosive used. These data are also segregated by Test Number in four matrices; Table 11, explosive type vs. load factor; Table 12, explosive type vs. initiation system; Table 13, explosive type vs. delay interval; and Table 14 for packaged slurry only, delay interval vs. load factor. These matrices are presented in Parts A and B: the first shows the individual test numbers; the second shows the average surface evaluation grades for each part of the matrix where data were obtained. Three test values were not included because these tests were conducted with the expectation of poor results. These were Test Nos. 29, 30, and U-13. The numbers are shown but are circled to indicate their exclusion from the set average.

The explosive type vs. load factor matrix reveals that a wide range of load factors can be used with dynamite with about the same results. This suggests that load factors of about 0.35 can be used with dynamite in the tube casings and achieve the same results as the industry now achieves with kelly bar loading of dynamite using an average load factor of 0.53. This is a reduction of 34% in explosives cost per cubic yard.

TABLE 10

SURFACE EVALUATION OF TESTS IN 3" ID KELLY BAR PROGRAM

Test No.	LOCATION	INITIATION SYSTEM	EXPLOSIVE	DELAY INTERVAL	LOAD FACTOR	SURFACE EVALUATION
1	Bergeron	25 gr. cord downline	PGD	25ms/100ms	0.70	9.0
2	"	EBC in top stick	PGD	50ms/350ms	0.56	9.0
3	"	50 grain cord downline	TOV	50ms/350ms	0.47	9.0
4	"	50 grain cord downline	TOV	50ms/350ms	0.44	8.5
5	"	Capped primer at top	TOV	50ms/350ms	0.42	7.5
6	Miami Crushed Rock	25 gr. cord DL	PGD	25ms/150ms	0.67	9.0
7	" " "	25 gr. cord DL	PGD	25ms/175ms	0.45	9.0
8	" " "	25 gr. cord DL	PGD	25ms/175ms	0.45	9.0
9	" " "	EBC in top stick	PGD	25ms/175ms	0.41	7.5
10	" " "	EBC in top stick	PGD	25ms/300ms	0.41	7.5
11	" " "	Capped primer in top	GPA-2	25ms/200ms	0.37	6.0
12	" " "	Capped primer in top	GPA-2	50ms/350ms	0.37	3.0
13	" " "	50 gr. cord DL	GPA-2	50ms/500ms	0.37	2.0
14	" " "	Primadet w/booster @ Bottom	GPA-2	50ms/300ms	0.41	2.0
15	" " "	50 gr. cord DL	TOV	50ms/450ms	0.41	6.0
16	Capeletti Pit #13	50 gr. cord DL	PGD	25ms/ 75ms	0.63	9.0
17	" "	50 gr. cord DL	PGD	50ms/350ms	0.58	9.0
18	" "	25 gr. cord DL	PGD	50ms/450ms	0.52	8.0
19	Sterling-Golden Prince	4 gr. cord DL EBC in top	FLO	25ms/100ms	0.53	9.0
20	" " "	EBC at bottom	FLO	25ms/100ms	0.52	7.5
21	" " "	Capped primer T & B	PM-100	25ms/100ms	0.43	5.0
22	" " "	Capped primer @ Bottom	PM-100	100ms/250ms	0.48	2.5
23	" " "	25 gr. cord DL	FLO	25ms/125ms	0.43	6.5
24	" " "	Capped primer T & B	PM-100	50ms/250ms	0.43	2.5
25	" " "	Capped primer T & B (Seq)	PM-200	10ms/ 50ms	0.39	9.6
26	" " "	25 gr. cord DL (Seq.Tim.)	RXL-330	10ms/ 50ms	0.43	9.7
28	" " "	capped primer @ Bottom (Seq)	PM-200	10ms/ 60ms	0.43	8.0
29	" " "	Nonel TL & Primer @ Bottom (Seq)	PM-100	10ms/ 70ms	0.43	2.0
30	" " "	Nonel TL & Primer @ Bottom (Seq)	PM-200	25ms/175ms	0.42	1.8
31	" " "	Nonel TL in Bottom stick	FLO	18ms/153ms	0.41	9.9
32	" " "	E-cord DL	FLO	18ms/153ms	0.43	9.8
33	Miramar Rock	Capped primers T & B	GPA-2	50ms/500ms	0.41	6.0
34	" "	Capped primer @ Bottom	GPA-2	50ms/250ms	0.43	1.5
35	L.W. Rozzo	25 gr. cord DL	US-318	17ms/ 51ms	0.56	8.6
36	" "	25 gr. cord DL	US-318	17ms/ 51ms	0.51	8.2
37	" "	25 gr. cord DL	US-318	17ms/ 45ms.	0.51	8.5

Test No.	LOCATION			INITIATION SYSTEM	EXPLOSIVE	DELAY INTERVAL	LOAD FACTOR	SURFACE EVALUATION
38	Florida Rock	&	Sand	50 gr. cord DL	TOV	25ms/ 75ms	0.37	8.5
39	"	"	"	25 gr. cord DL	TOV	25ms/ 75ms	0.37	8.6
40	"	"	"	25 gr. cord DL	AL-10	25ms/ 75ms	0.38	9.2
41	"	"	"	25 gr. cord DL	AL-10	9ms/ 63ms	0.38	9.5
42	"	"	"	25 gr. cord DL	TOV	9ms/ 63ms	0.37	9.2
43	"	"	"	25 gr. cord DL	TOV (4)	9ms/ 54ms	0.30	7.5
44	"	"	"	25 gr. cord DL	TOV (4)	9ms/ 63ms	0.34	8.3
45	"	"	"	50 gr. cord DL	G-915W	9ms/ 72ms	0.37	5.4
46	"	"	"	Nonel TL w/primer T & B	G-915W	25ms/175ms	0.39	7.0
47	"	"	"	Nonel TL w/primer @ Bottom	G-915W	9ms/ 72ms	0.43	8.3
48	"	"	"	Nonel TL w/primer @ Bottom	TOV	9ms/ 36ms	0.37	8.3
49	"	"	"	50 gr. cord DL (Seq.Timer)	TOV	10ms/105ms	0.37	9.3
50	"	"	"	50 gr. cord DL (Seq.Timer)	TOV	17ms/385ms	0.37	9.9
51	Redlands			30 gr. cord DL	APG B-1	9ms/ 81ms	0.39	8.9
52	"			30 gr. cord DL	APG B-1	9ms/ 90ms	0.36	9.0
53	"			Nonel DL w/primer @ Bottom	SLM 100	9ms/ 63ms	0.45	2.6
54	"			25 gr. cord DL	AL-10	9ms/ 90ms	0.44	8.2
55	"			30 gr. cord DL	AL-10	9ms/ 54ms	0.44	7.9
56	"			30 gr. cord DL	AL-10	Inst.	0.44	8.0
57	Indian Lakes			30 gr. cord DL	PGD	25ms/400ms	0.46	8.5
58	"	"		25 gr. cord DL	PGD	25ms/400ms	0.46	7.9
59	"	"		25 gr. cord DL	PGD	25ms/400ms	0.37	8.5
60	"	"		25 gr. cord DL	PGD	9ms/ 72ms	0.36	6.5
61	"	"		25 gr. cord DL (Seq.Timer)	FLO	17ms/385ms	0.33	8.1
62	"	"		Nonel TL w/primer at Bottom	GPA-2	9ms/109ms	0.33	8.1
63	Hardrives			30 gr. cord DL	AL-10	25ms/125ms	0.64	9.4
64	"			30 gr. cord DL w/conn.	AL-10	17ms/277ms	0.64	8.7
65	"			30 gr. cord DL	AL-10	17ms/119ms	0.57	7.2
66	"			30 gr. cord DL	AL-10	17ms/119ms	0.57	5.8
67	"			30 gr. cord DL	AL-10	17ms/102ms	0.57	6.6
68	Gator Rock			Nonel DL w/primer	GPA-2	9ms/ 27ms	0.43	5.7
69	"			25 gr. cord DL	Herc.40%E	25ms/240ms	0.35	9.1
70	"			25 gr. cord DL	Herc.40%E	25ms/150ms	0.35	8.2
71	Harper Bros.			25 gr. cord DL	Herc.60%E	25ms/150ms	0.18	6.9
72	"			25 gr. cord DL	Herc.60%E	25ms/184ms	0.14	8.8
73	"			25 gr. cord DL	Herc.60%E	25ms/175ms	0.14	8.4
74	"			25 gr. cord DL	Herc.60%E	25ms/175ms	0.14	8.4
75	"			25 gr. cord DL	Herc.60%E	25ms/184ms	0.14	8.4
76	"			25 gr. cord DL	Herc.60%E	25ms/184ms	0.14	8.5

SURFACE EVALUATION OF UNIT TESTS

TEST NO.	LOCATION	INITIATION SYSTEM	EXPLOSIVE	DELAY INTERVAL	LOAD FACTOR	SURFACE EVALUATION
U- 1	Capeletti Pit #13	Capped primer @ Top	GX-19	25ms/150ms	0.65	4.0
U- 2	" " "	Capped primer @ Top	GX-19	25ms/ 75ms	0.65	4.0
U- 3	" " "	Capped primer @ Top	GX-19	50ms/200ms	0.62	4.0
U- 4	Miramar Rock	25 gr. cord DL	PGD	25ms/125ms	0.63	9.0
U- 5	" "	EBC @ Top	PGD	25ms/150ms	0.63	8.5
U- 6	" "	50 gr. cord DL	TOV	25ms/125ms	0.47	8.5
U- 7	" "	50 gr. cord DL	TOV	25ms/150ms	0.45	8.0
U- 8	Coral Aggregates	Capped primers T & B	P-4	25ms/250ms	0.63	0.0
U- 9	" "	Capped primers T & B	P-4	25ms/250ms	0.50	2.0
U-10	" "	Capped primers T & B	P-2	25ms/250ms	0.63	6.0
U-11	" "	Capped primers T & B	P-2	25ms/250ms	0.63	8.0
U-12	" "	Capped primers T & B	P-4	25ms/250ms	0.62	8.5
U-13	" "	Capped primers T & B	P-4	25ms/100ms	0.62	2.0
U-14	" "	Cord w/primers Top	P-4	25ms/250ms	0.62	3.0
U-15	" "	Primadet/primer Bottom	P-2	25ms/250ms	0.63	0.0
U-16	" "	Cord & Primer @ Top	P-2	25ms/250ms	0.63	2.0
U-17	" "	Cord & Primer @ Top	P-2	25ms/259ms	0.63	2.0

SURFACE EVALUATION OF 1 3/4" DYNAMITE TESTS

E- 1	Capeletti Pit #12	4 & 25 gr. cord DL	RXL 330	25ms		9.0
E- 2	Capeletti Pit #12		RXL 330	25ms		9.0

TABLE 11

SMALL DIAMETER

EXPLOSIVE TYPE VS. LOAD FACTOR

TEST NUMBER MATRIX

LOAD FACTOR LBS./C.Y.	DYNAMITE	PACKAGED SLURRY	BULK SLURRY
0.3 to 0.35	61,69,70	43,44,62	
0.36 to 0.40	40,41,51,52,59,60	11,12,13,25,38,39,42 45,46,48,49,50,68	
0.41 to 0.45	23,26,31,32,54,55,56	4,5,7,8,9,10,14,15, 21,24,28,29,30,33,34, 47,53,U-7	
0.46 to 0.50	57,58	3,22,U-6	U-9
0.51 to 0.55	18,19,20,36,37		
0.56 to 0.60	2,17,35,65,66,67		
0.61 to 0.65	6,16,63,64,U-4,U-5		U-1,U-2,U-3,U-8, U-10,U-11,U-12, U-13,U-14,U-15, U-16,U-17
0.66 to 0.70			

TABLE 11 A

SMALL DIAMETER

EXPLOSIVE TYPE VS. LOAD FACTOR

AVERAGE SURFACE EVALUATION GRADE MATRIX

LOAD FACTOR LBS/C.Y.	DYNAMITE	PACKAGED SLURRY	BULK SLURRY
0.3 to 0.35	8.5	8.0	
0.36 to 0.40	8.6	7.1	
0.41 to 0.45	8.6	6.2	
0.46 to 0.50	8.2	6.7	2.0
0.51 to 0.55	8.2		
0.56 to 0.60	7.7		
0.61 to 0.65	8.9		3.8
0.66 to 0.70	9.0		
Weighted average	8.4	6.7	3.7

TABLE 12

SMALL DIAMETER

EXPLOSIVE TYPE VS. INITIATION SYSTEM

TEST NUMBER MATRIX

	DYNAMITE	PACKAGED SLURRY	BULK SLURRY
EB Cap at top	2,9,10,U-5		
EB cap at bottom	20,		
EB cap top & bottom			
Capped primer @ Top		5,11,12	U-1,U-2,U-3,U-14, U-16,U-17
Capped primer @ bottom		22,28,34	
Capped primer T & B		21,24,25,33	U-8,U-9,U-10,U-11,U-12 U-13,
Nonel TL to bottom	31		
Nonel TL & primer-bottom		29,30,47,48,53,62,68	
Nonel TL & primer T&B		46	
25 or 50 grain Detonating cord DL	1,6,7,8,16,17,18,23, 26,32,35,36,37,40,41, 51,52,54,55,56,57,58, 59,60,61,63,64,65,66, 67,69,70,71,72,73,74, 75,76,U-4	3,4,13,15,38,39,42,43, 44,45,49,50,U-6,U-7	
4 gr. det. cord DL	19,E-1,E-2		
Bottom primadet			
Bottom primadet w/bottom		14	U-15

TABLE 12A

SMALL DIAMETER

EXPLOSIVE TYPE VS. INITIATION SYSTEM

AVERAGE SURFACE EVALUATION GRADE MATRIX

	DYNAMITE	PACKAGED SLURRY	BULK SLURRY
EB cap @ top	8.1		
EB cap @ bottom	7.5		
EB cap top & Bottom			
Capped primer @ top		5.5	3.2
Capped primer @ bottom		4.0	
Capped primer T & B		5.8	4.9
Nonel TL to bottom	9.9		
Nonel TL & primer-bottom		6.6	
Nonel TL & primer-T&B		7.0	
25 or 50 gr. Detonating cord DL	8.4	7.8	
4 gr.detonating cord DL	9.0		
Bottom Primadet			
Bottom Primadet w/booster		2.0	0
Weighted Average	8.4	6.9	3.6

TABLE 13

SMALL DIAMETER

EXPLOSIVE TYPE VS. DELAY INTERVAL

TEST NUMBER MATRIX

9 milliseconds MS-connectors	41,51,52,54,55,60	42,43,44,45,47,48, 53,52,68	
10 milliseconds Sequential Blaster	26	25,28,29,49	
17 or 18 milliseconds MS-connectors	31,32,35,36,37,64, 65,66,67		
17 milliseconds Sequential blaster	61	50	
25 milliseconds EB cap	1,6,7,8,9,10,16,19, 20,23,40,57,58,59,63, 69,70,71,72,73,74,75, 76,U-4,U-5,E-1,E-2	11,21,30,38,39,46, U-6,U-7,U-8,U-9,U-10, U-11,U-12,U-13,U-14, U-15,U-16,U-17	U-1,U-2
50 milliseconds EB cap	2,17,18	3,4,5,12,13,14,15,24, 33,34	U-3
100 millisecond EB cap		22	

TABLE 13 A

SMALL DIAMETER

EXPLOSIVE TYPE VS. DELAY INTERVAL

AVERAGE SURFACE EVALUATION GRADE MATRIX

	DYNAMITE	PACKAGED SLURRY	BULK SLURRY
9 msec. MS-connector	8.3	7.0	
10 msec. Seq. Blaster	9.7	9.0	
17 or 18 msecs. MS-connector	8.1	---	
17 msec. Seq. Blaster	8.1	9.9	
25 msec. EB caps	8.5	5.2	4.0
50 msec. EB caps	8.7	4.8	4.0
100 msec. EB caps	---	2.5	
Weighted Average	8.5	5.8	4.0

TABLE 14

SMALL DIAMETER

PACKAGED SLURRY ONLY

DELAY INTERVAL VS. LOAD FACTOR

TEST NUMBER MATRIX

(Initiation at top only has been eliminated)

	LOAD FACTOR; LBS/C.Y.			
Delay Interval	0.3 to 0.35	0.36 to 0.40	0.41 to 0.45	0.46 to 0.50
9 msec. MS-connector	43,44,62	42,45,48	47,53,68	
10 msec. Seq.Timer		25,49	28,29	
17-18 msec. MS-connector				
17 msec. Seq. Timer		50		
25 msec. EB cap		38,39,46	21,30,U-7	U-6
50 msec. EB cap		13	4,14,15,24, 33,34	3
100 msec. EB cap				22

TABLE 14A

SMALL DIAMETER

PACKAGED SLURRY ONLY

DELAY INTERVAL VS. LOAD FACTOR

AVERAGE SURFACE EVALUATION GRADE MATRIX

	LOAD FACTOR: LBS/C.Y.				
	0.3 to 0.35	0.36 to 0.4	0.41 to 0.45	0.46 to 0.50	
9 msec. - MS connectors	8.0	7.6	5.5		7.0
10 msec. - Seq. Timer		9.5	8.0		9.0
17-18 msec. - MS connectors					
17 msec. - Seq. Timer		9.9			9.9
25 msec. EB cap		8.0	6.5	8.5	7.6
50 msec. EB cap		2.0	4.4	9.0	4.7
100 msec. EB cap				2.5	2.5
	8.0	7.8	5.3	6.7	

The matrix indicates generally lower values for packaged slurries than for dynamite. This is partly due to the need to experiment with each slurry type to determine the best method of initiation and the best delay interval, which is not so necessary with the dynamites. It may be very significant that the matrix indicates that better results are obtained with packaged slurries at low load factors. This may indicate that successful use of slurry in South Florida requires that the slurry be used within a very limited load factor range. If too little is used the rock is not broken but if too much is used, it may cause precompression in adjacent holes with longer delays. More research and testing of this possibility would be desirable.

Bulk slurries in small diameter tube casings appear generally unsatisfactory at all load factors tested.

The explosive type vs. initiation system matrix again reveals that successful-looking dynamite shots can be obtained with almost any system.

Slurries generally have lower grades and a lower weighted average. Performance appears to be better with top and bottom primers used either with electric blasting caps or with Nonel trunkline connectors, or with detonating cord. The latter is undoubtedly because Tovex 255 is designed to be used with 50-grain cord and many successful test shots were made in this manner or with 25 grain cord. The generally lower grades for slurries most likely are the result of the more experimental nature of some of the slurries used, and are not truly indicative of the results that can be obtained with products that are in normal commercial use in South Florida.

The bulk slurries again have generally unsatisfactory results in small diameter casings.

The next matrix compares explosive type to length of delay interval between adjacent holes.

Again, all dynamite values are fairly close to the weighted average indicating that dynamite is tolerant of almost any delay interval.

The packaged slurries on the other hand, indicate definite improvement with intervals of 17 milliseconds or less, and particularly good results when a sequential timer is used.

Bulk slurries were not tested with the shorter intervals and so it is not known if they might perform satisfactorily under such circumstances. At Florida Rock & Sand Co., an effort was made to use Gulf's GX-19 bulk slurry in the 2.493" ID tubes and at least one of these shots would have used a short delay interval. For some unknown reason, the slurry would not pump into the holes satisfactorily and the effort was abandoned after seven holes. The remaining 13 holes (Test No.46) were loaded with Gulf 915-W and the results are included with packaged slurries.

The final matrix shows the distribution of surface evaluation grades with respect to load factor and delay interval for packaged slurries only. This again shows the tendency for packaged slurry explosives to have improved performance with short delay intervals between adjacent holes and relatively low load factors.

The data in the last matrix were separated into two sets of grades, those with a delay interval of 18msec. or less, and those with intervals of 19 msec. or more, and the one lowest value in each set was eliminated. The two sets were then treated with the method used earlier to find the range of differences in the means at various levels of confidence.

The results indicate that at a 95% level of confidence, the shorter delay interval should increase the evaluation score by at least 0.3 points and not more than 3.94 points. At the 80% confidence level, the increase is at least 0.96 points and not more than 3.28 points.

Of course it should be remembered that the surface evaluation system is based on items which may not have equal merit, and the grades assigned to each item are subjective. Consequently, there may not be any real difference but if visual observation of the shot and its surface results have any significance at all, the above values probably reveal a real and important difference.

It is also important to remember that these data apply to 2" and 2 1/4" explosives in holes roughly 10 feet apart. Gulf's and Vulcan's experience with bulk slurry in large diameter tubes in holes approximately 20 feet apart indicates than an interval of 50 milliseconds produces better results. This suggests that some aspect of precompression phenomena in South Florida is a complex relationship between time and distance. The better small diameter slurry results obtained with relatively low load factor may be more dependent on the distance between holes than on the charge weight per hole. In other words, there may be a critical distance in South Florida below which slurry in adjacent holes cannot be effectively exploded if it has a longer delay period. Expanding on this hypothesis, slightly beyond this critical distance, a short delay of 9 or 10 milliseconds may be effective, increasing as the distance between holes is increased until a time approaching 25 milliseconds at some distance less than 20 feet is reached. At this time, a relaxation may occur in the controlling factor and holes beyond this distance may best be initiated at a delay interval greater than $25^{\pm}$ milliseconds. Precompression may permanently affect those holes between the critical distance and,say, $15^{\pm}$ feet so that they cannot be exploded even after the relaxation occurs. This conceptual scheme is illustrated in Figure 31.

Further testing is needed at one specific site with different spacing and burdens and different delay intervals, preferably controlled with a sequential timer,and with different packaged slurries to prove or disprove the above hypothesis. A rational explanation is also needed.

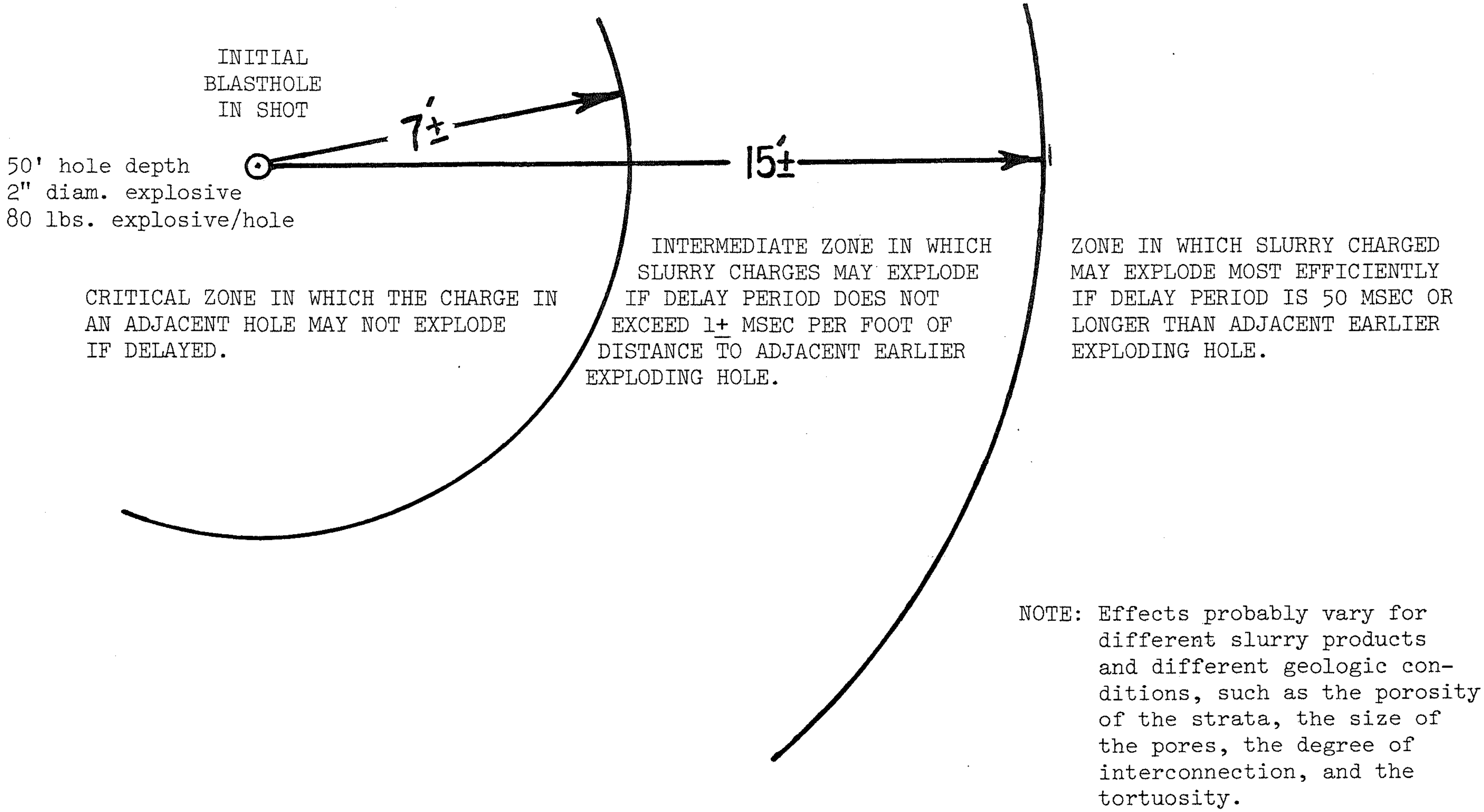


FIGURE 31

CONCEPTUAL SCHEME OF PRECOMPRESSION PHENOMENA IN SOUTH FLORIDA

Two additional tests were performed to develop data on using 1 3/4" diameter cartridged dynamite in Kemax tubes which could be placed through a 2 1/2" ID kelly bar.

The concept in this testing was to develop another alternative which would not require any conversion of the drill rig and would be applicable to those operations where little drilling is done, or where the depth of drilling might be only 10 to 20 feet, and the cost of rig conversion might not be justified. In either of these cases it might be better to incur the extra cost of tubing and pass the increase on to the customer. A typical example might be some of the operations on the west coast of Florida where only the cap rock is blasted. In these cases, most of the total stone is not blasted. Increased drilling and blasting unit cost is then less than those operations where full depth blasting is done. Many of these operations can load open hole.

Another example, but of a non-mining nature, is the drilling done in canal work. Holes are not precisely located and can be adjusted with conditions and the effectiveness of the explosives. Drilling is usually shallow. Most of the contractors doing this type of work have several 2 1/2" ID kelly bars of various lengths.

In these tests, Capeletti Bros., standard 2 1/2" ID kelly bar rig with air flush was used in their Pit #12. Inadvertently, two sets of Kemax tubing sizes were used: the first was 2.200" ID x .060" wall, and the second was 2.243" ID x .060" wall. Both had .060" thick internal couplings. Inner diameter at the coupling was 2.08" and 2.123", respectively. Difficulty should have been encountered with the second, or larger, size because there is insufficient clearance for it to pass the bit if a new carbide insert is present to reduce the normal 2.5" ID. The inserts must have been worn because the tubing did pass the bit and remained in the hole.

Although internal couplings were used in these tests at Pit #12, the same result could be obtained with an external coupling. The design would consist of an internally flush tube section with 2.08" inner diameter. Wall thickness of the main tube would be 0.060" and the external coupling material would have 0.045" wall thickness. Maximum outer diameter at the coupling would be 2.29". This would permit a clearance of 0.061" at the bit insert.

There is a lot of unconsolidated sand in Capeletti Pit #12 which provided a good test for the tube casings to remain open overnight.

The first ten included 18 holes as part of a total shot of 42 holes. The tube cased holes were the last 3 rows of 6 holes. The other 4 rows were loaded by the normal kelly bar method. Holes were 43' deep.

After placing the tube casings in the 18 holes, and completing the drilling, each of the holes was loaded with 28 sticks of 1 3/4" x 16" Atlas RXL 330 dynamite. Stemming was 6 feet.

This product has a density of 1.7 and was in paper wrappings which provided a full 1 3/4" diameter of explosive. This is nearly the same as the actual diameter of the explosives in a 2" spiral wound cartridge. (The tube casing eliminates the need for the spiral wound cartridge.) 61 pounds of explosive were placed in the column build of 37 feet, or 1.65 pounds per foot. This compares to 1.7 pounds per foot when 2" dynamite with a density of 1.25 is used, or 1.9 pounds per foot when the density is 1.4.; in other words, 86 to 97% of the loading density of the 2" dynamites commonly used in kelly bar loading. This is probably more than offset by the use of paper wrap instead of thicker spiral wound cardboard cartridges. The same fact would be true in the larger diameter tubes.

On three of the holes, Nonel trunkline connectors were used as downlines with the delay element placed in the bottom stick. On another three holes, the 4 grain tails of assembled Primadets were used as downlines without the delay elements. 25-grain detonating cord downlines were used on the other 12 holes. An electric blasting cap at the top of each hole provided the proper delay interval.

All 42 holes were detonated in the same shot. All had been drilled on the same pattern, 10' burden and 10' spacing. 18 sticks of Hercules Power Gel D in 2" X 24" spiral wound cartridges were used in the holes loaded through the kelly for a load factor of 0.43 versus a load factor of 0.38 in the holes containing the 1 3/4" dynamite.

Evaluation of the results indicated that the tube cased holes gave superior performance. The holes where 4 grain cord had been used were the best appearing part of the shot.

Of the 18 holes in which tube casings were placed,only one had to be redrilled and this was because the sections separated at the connecting sleeve. This may have resulted by mistakenly using the wrong size tubing (a 2.243" ID tube over a 2.200" ID tube) which would separate as soon as the tubes were released at the top of the kelly.

Drilling and placement of tubes was accomplished at the rate of four holes per hour. Loading the 18 holes required another 45 minutes. This gives a rate of 3.43 holes per hour for drilling, tube placement, and loading. This compares with the 1976 average rate for this drill rig of 5.19 holes per hour, drilled and loaded by the kelly bar method. Accordingly, drilling the holes, casing them with Kemax tubing, and then loading the explosive into the tubes is 66% as efficient as normal kelly bar loading. From the appearance of the shot, some of this loss in efficiency could be offset by increasing the burden and spacing.

A second shot of 51 holes with 18 of these utilizing tube casings was made. Loading was similar to the first test and so was the spacing and burden. The difference was that 12 of the tube cased holes were left in place for 24 hours before loading to determine if they would be squeezed and fail, or whether sand would fill in from the bottom.

The following measurements were made:

Hole No.	Depth when placed	Depth 24 hours later
1	42' 7"	38' 9"
2	42' 5"	40' 5"
3	41' 9"	41' 8"
4	43' 4"	43' 0"
5	43' 6"	43' 1"
6	43' 8"	43' 7"
7	42' 7"	42' 6"
8	43' 1"	43' 1"
9	43' 5"	43' 5"
10	43' 7"	43' 2"
11	43' 8"	43' 8"
12	43' 9"	43' 9"

SHOT ROCK BUFFERS:

Most operators maintain a buffer equal to one burden distance. This is to contain the shot and make certain that the rock is not blown into the lake beyond the reach of the dragline. There is considerable argument over the need for a buffer.

In the test shots, blasts without buffers were conducted at Bergeron, Miramar, Sessa, Gator Rock, Coral Aggregates, Capeletti Pit #12, Redlands, and Harper Bros. In none of these shots was there any indication of excessive throw into the lake, In the geometry of the blast situation in South Florida, it may be necessary to maintain a buffer when loading through the kelly because the charge probably tends to be concentrated at the bottom of the blasthole. This would direct most of the throw against the toe, and tend to cause the shot rock to be blasted farther into the lake.

From observation, it appears that a buffer is unnecessary and may even be undesirable when explosives are contained in tube casings. Elimination of buffers would avoid the tie-up of considerable working capital and the time value of the money involved could significantly, if not completely, offset the cost of tube casings.

More research is needed on the efficacy of buffers, with and without tube casings.

SEQUENTIAL BLASTING MACHINE

As mentioned earlier, a sequential timer was used for the first time in South Florida in conjunction with the test program. Both Atlas and DuPont provided these instruments at no charge.

There were too few shots with these timers to provide proof for a definitive statement, but the additional effort and cost to use a sequential blasting machine appears worthwhile because:

(1) the short delay interval which the matrices indicate is desirable for small diameter packaged slurry is better controlled with the sequential timer. Intervals are more precise and there can be more combinations.

(2) operations which are currently using only 8 or 10 blastholes per shot could increase the number of holes and shoot only once a day or less, if tubes are used.

(3) there seems to be an undefinable mass effect when packaged slurries are used. In other words, a well-designed shot with 40 or 50 holes seems to give better results than when the same design is used with 25 to 30 holes.

(4) whatever techniques improve slurry blasting, probably improve dynamite blasting as well but the improvement is not as apparent. This may be because dynamite appears to have a much wider tolerance of improper design.

(5) sequential timers were used on seven shots involving three different types of packaged slurries and two different types of dynamite. Except for two shots which were designed to prove that certain delay intervals would <u>not</u> work, all of the shots were successful and rated among the highest surface evaluation grades.

(6) there is no other satisfactory way to get a dependable series of short intervals for a large number of holes. For many blasts, 9 or 17 millisecond connectors were used as a substitute but these are very susceptible to water or humidity damage. When used from a freshly opened box the results were satisfactory, but when allowed to sit in the magazine for 60 to 90 days during the rainy months, they became very unreliable. In addition, they frequently have to be used when there is considerable surface water and this increases the risk of a misfire.

The main disadvantages of the sequential timer are that it is electrical, it costs several thousand dollars, and there is a long hook-up time. It also requires a lot of connecting wire.

Attempts were made in February, April, and July, 1977, through the local Hercules technical representative to try the Hercudet nonelectric system. No arrangements could be made.

GROUND VIBRATIONS

Although not required by the contract, nearly all blast vibrations were measured with a Sprengnether VS-1100 seismograph. All records obtained up to May 23, 1977 were lost in a briefcase which was stolen in the Washington National Airport on that date.

Peak longitudinal particle velocities for most subsequent blasts are plotted versus scaled distance on log-log graphs. Figure 32 is a plot of all blasts where packaged slurry was used. Figure 33 is a similar graph of dynamite shots. Equations and correlation coefficients are shown for each.

Although the equations suggest rather large differences between the two, the amount of difference within the practical range of observations is minor, and there appears to be no reason to consider slurry and dynamite separately insofar as ground vibrations are concerned.

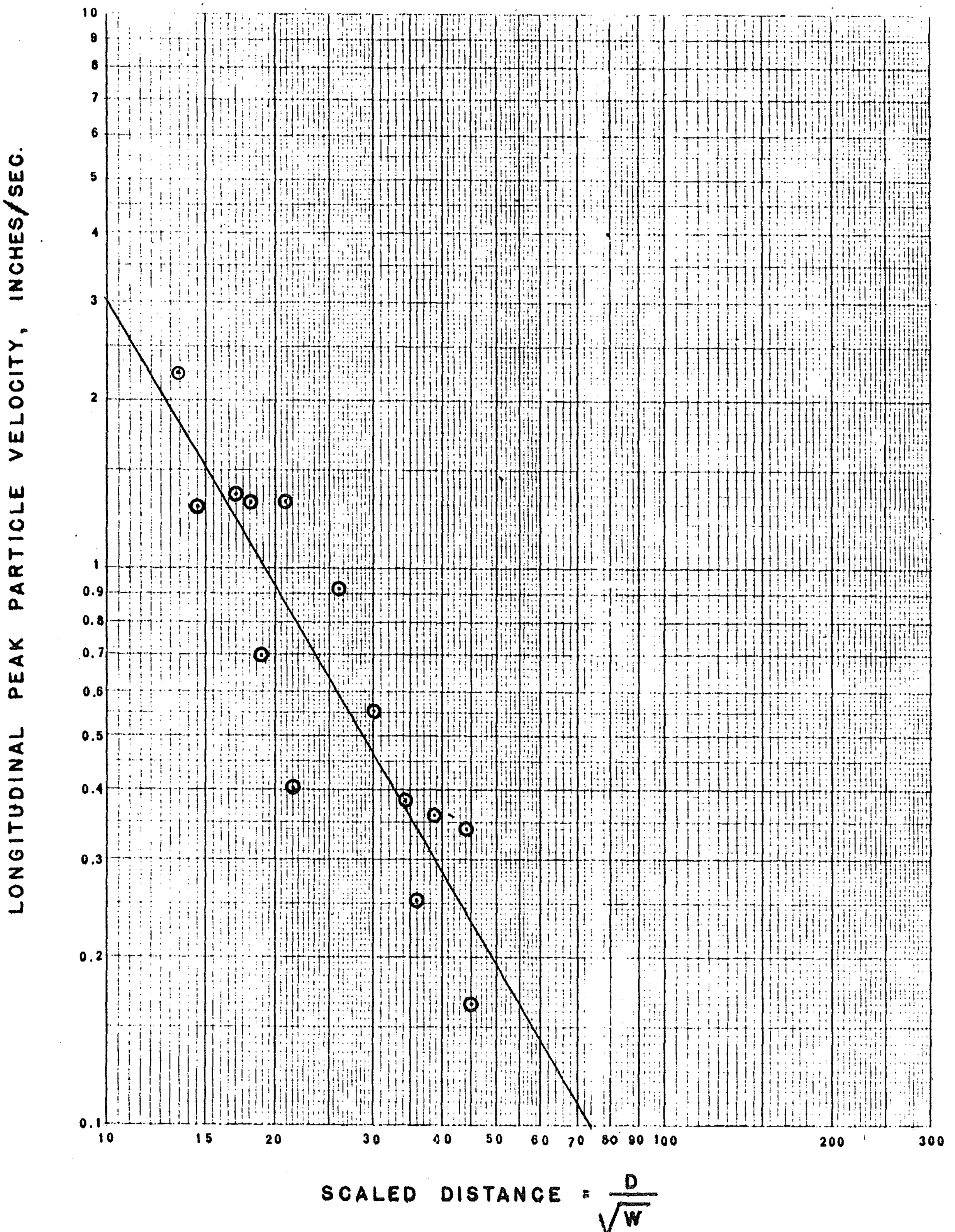
A combined plot of both sets of data (Figure 34) yields a similar curve which has value in determining the effectiveness of blasts. If a value plots significantly above and to the right of the least squares curve, it should be tested for the possibility that propagation may have occurred. This frequently happens with semi-gelatin dynamite in South Florida. In 17 dynamite shots there is definite evidence of propagation in six, and a good possibility of it occurring in one other. This is a rate of 35%.

Propagation is indicated when the plotted value is recalculated considering that all holes in the shot exploded as if on the same delay. In other words, the charge weight per delay is the total amount of explosives used in the blast. If the result plots close to the least squares line, propagation probably occurred.

Slurry explosives did not show any evidence of propagation, but blasts which gave evidence of considerable precompression plotted well below the least squares curve. In developing the curves, the obvious cases of propagation and precompression were eliminated from the data set.

When values plot consistently at some distance from the given curve, the ground conditions may be significantly different than normal. A series of six observations taken in a subdivision that had been built on an hydraulic fill indicated abnormally high vibration levels in relation to the scaled distance involved. This is compounded in South Florida by the fact that low frequency Rayleigh waves are abnormally strong and continue for several seconds. Their peak trace particle velocity is approximately equal to the peak trace particle velocity of the earlier arriving high frequency body waves, but they are much more perceptible to humans because of the frequency that may be as low as 2 to 3 hertz. They can be identified without instruments because they arrive only slightly before the air blast and continue well beyond. The first arrival of these Rayleigh waves is hard to define precisely but the velocity is approximately 1250 per feet per second and their appearance is restricted principally to the vertical and longitudinal traces. When values from these traces are plotted, the characteristic retrograde oscillation is present. It is these waves, undoubtedly, that create the frequent blasting complaints from neighbors.

FIGURE 32

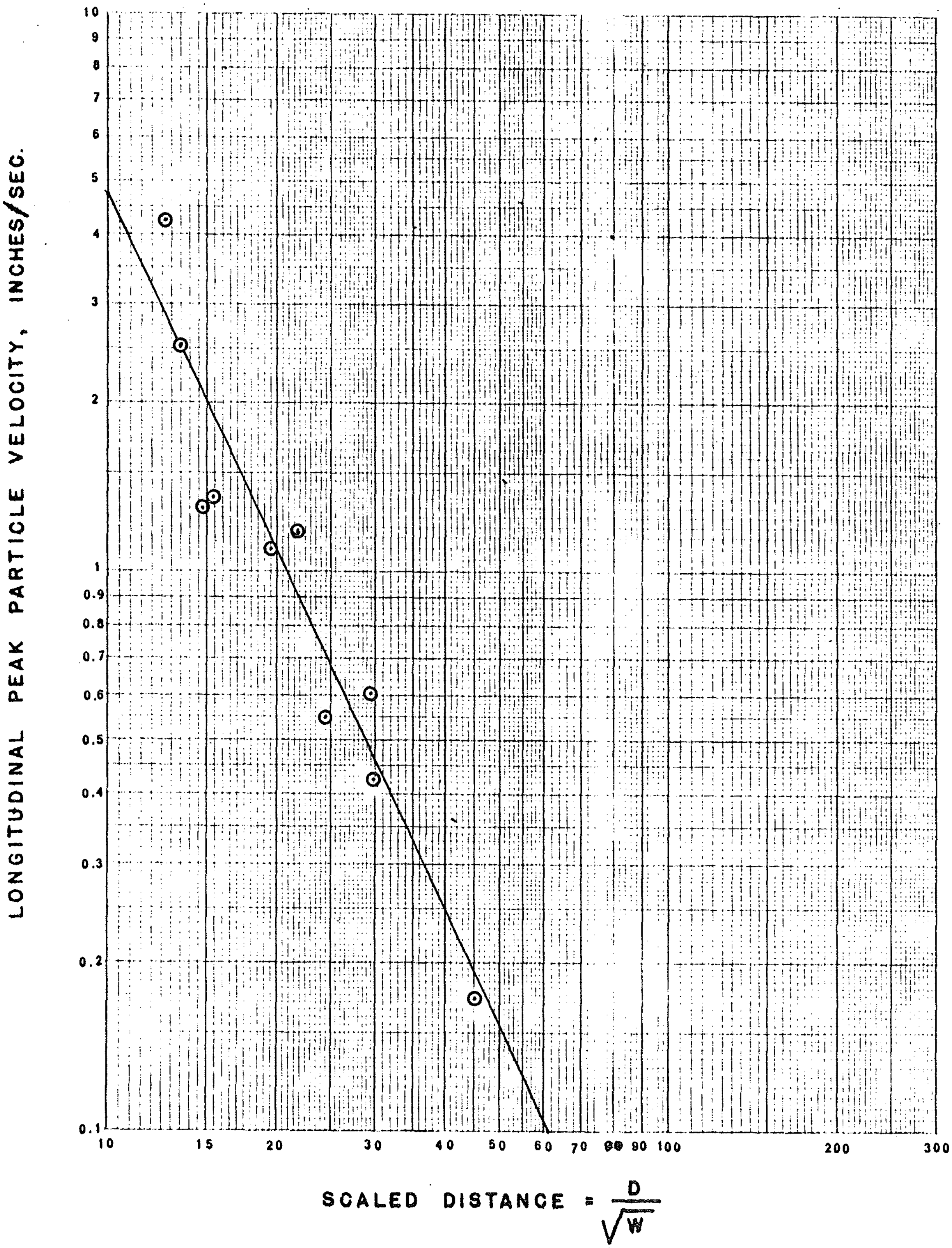


SLURRY ONLY

$$P.\ V. = 154\left(\frac{D}{\sqrt{W}}\right)^{-1.710}$$

Correlation coefficient: -0.914

FIGURE 33

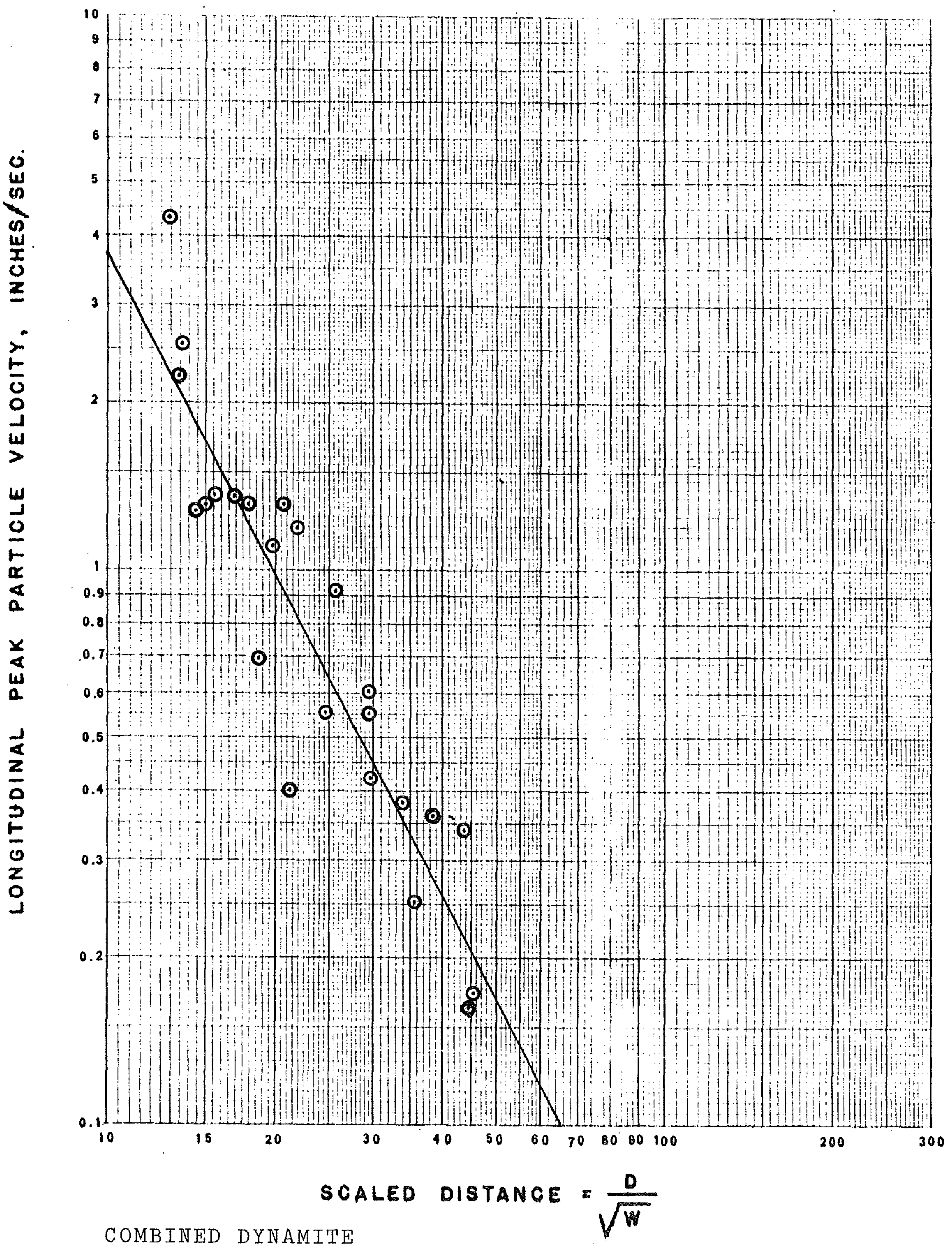


DYNAMITE ONLY

$$P.\ V. = 631\left(\frac{D}{\sqrt{W}}\right)^{-2.13}$$

Correlation coefficient: -0.95

FIGURE 34



COMBINED DYNAMITE
AND SLURRY BLASTS

$$P.\ V. = 289\left(\frac{D}{\sqrt{W}}\right)^{-1.894}$$

Correlation coefficient: -0.93

Ground vibrations are probably increased in shots loaded through the kelly bar because of the tendency of the charge to concentrate at the base of the hole. This can cause excessive containment in the upper part of the hole and inhibit the proper release of the explosive energy.

CHAPTER 6

POST-SHOT ANALYSIS

POST SHOT ANALYSIS

Twelve different locations were tested with the 3" ID kelly and two additional locations were tested in the other aspects of the project using the smaller 2.243" ID or 2.200" ID Kemax tube. Of these 14 sites, most of the test material has been dug by the dragline at 10 sites. At seven sites, a substantial part of the test material had also been processed.

Fairly extensive data suites were collected for digging and processing at six sites. Because of intermittent operation, machinery problems, weather, or lack of available observers, only four of these are adequate for statistical analysis.

A considerable amount of time was spent in collecting detailed dragline digging rates. This proved to be of very limited value because:

(1) most of the cycle time is involved with the swing, which is unrelated to diggability.

(2) total rate of advance is affected by pad building and downtime requiring constant observation for a minimal amount of information.

(3) operator variation

(4) variation in nighttime vs. daytime operation

(5) considerable variation in bucket-fill factor which is difficult to rate and record in a dynamic situation, and difficult to handle statistically.

Daily questioning of the operators as to the diggability provides about as much usable information as possible and this is only significant when the digging is tough.

Visual or photographic inspection of the surge pile is also hard to quantify and develop with sufficient precision to draw conclusions. Part of this problem is because the gradation of rock in the surge pile is not necessarily indicative of the quality of the blasting action. In South Florida, microfractures are developed in the blasting process and the extent of these determines in large part the rate at which the material can be processed in the customary impact crushers. The presence or absence of the microfractures is not apparent from the appearance of the rock in the pile. (Microfractures appear to affect the gradation of the final product in that some operators stated that small diameter bulk slurry shots seemed to yield a product which was deficient in fines.)

The most reliable information was obtained by observing the loading of the surge pile material into portable primary crushers. This loading is done either by a front end loader or a drag line which works from a position on top of the surge pile. Of the two, the front end loader provides the best data because it generally has more capacity (6 to 7 C.Y. bucket vs. 4 C.Y. for the dragline) and the bucket is filled more uniformly. Statistically significant variations in processing rates were obtained where loading was done with a front end loader. No significant variations occurred where a dragline was used. This may be because the dragline was always operating at full capacity and could not take advantage of the faster processing rate that might be obtained with better fractured rock.

Using a loader to load haul units which then took the shot rock to a permanently mounted primary crusher also provided good data but the observations were made at the crusher, not the surge pile. This provided an opportunity to observe the effect of oversize on the processing rate.

Results of these observations at Miramar Rock and at Coral Aggregates Corp. have already been presented in the section on unit tests.

Additional post-shot data were obtained for the tests at Florida Rock & Sand Co., and for the first 8 tests at Sterling Crushed Stone Co.; other material at Sterling has not yet been dug.

At Florida Rock & Sand Co. both the digging and processing operations were observed. The digging data revealed no significant change from normal digging except that considerable oversize was encountered at the location of tests 44 and 45.

The portable crushing unit was positioned a short distance from the end of the surge pile. The hopper was loaded by a 6 C.Y. capacity front end loader.

Surge pile positions of the dug material for the various tests were noted at the time of digging from baselines several hundred feet away but parallel to the orientation of the surge piles.

Buckets dumped into the hopper were counted for each 30-minute interval. Any rocks in the material over 2 feet in diameter and any downtime were recorded. Processing rates in cubic yards per hour were then determined for each 30-minute interval by removing the downtime (unless it was caused by oversize particles or other factors related to the feed material). Intervals where downtime exceeded 15 minutes were eliminated. Processing data were also obtained away from the test area for developing a "normal" production rate for rock which had been blasted with explosives which were loaded through the kelly bar.

For each test area, the 30-minute interval rates were summed, and the mean and standard deviation were determined.

At Sterling Crushed Stone Co., the same procedure was followed except that processing rates are expressed in cubic yards per 30-minute interval. At this location, a 4 cubic yard dragline is used to load the feed hopper of the portable crushing unit. The dragline appeared to be operating at full capacity at all times it was in operation. Therefore, it may be impossible to increase the production rate at the portable crusher unless a different piece of equipment is used to feed the hopper. In this case, the only change that might be noted between the test material and the non-test material could be in the value of the standard deviation. Examination of the results does indicate that the standard deviation for the test material is considerably less than the standard deviation for the composite non-test material, although the means are almost identical. This indicates that the test material provided a more uniform plant feed.

Data for Florida Rock & Sand Co., and Sterling Crushed Stone Co., are given in Table 15 and Table 16.

These data were treated in the same manner as the Miramar data in order to find the range of difference between the individual test means and the mean of the normal processing rates. At a 95% level of certainty, the following values were obtained:

Florida Rock & Sand Co. Primary Crushing: Range of difference in Means

Test Compared to Background	Production rate in CY/hr of test material is At Least	but not more than
38	28.8 CY/hr more	86.6 CY/hr more
39	42.6 " "	71.5 " "
40	33.6 " "	42.1 " "
41	20.9 " "	67.6 " "
42	11.2 " "	64.9 " "
43	26.3 " less	27.1 " "
44	43.7 " less	30.3 " "
45	23.4 " more	88.6 " "
46	33.3 " "	98.3 " "
47	23.3 " "	93.7 " "
48	29.6 " "	97.6 " "
49	38.2 " "	116.6 " "
50	Data not obtained	

Because all of the ranges of difference are positive with respect to the normal production rate, except for Tests 6 and 7, the increases are statistically significant at this level of certainty.

Sterling Crushed Stone Co. Primary Crushing: Range of Difference in means

Test Compared to Normal Rate	Production Rate in CY/hr of test material is At Least	but not more than
19	37.4 CY/hr less	34.6 CY/hr more
20 & 21	23.6 " "	46.8 " "
22	49.8 " "	21 " "
23	38 " "	22.8 " "
24	25.8 " "	36.6 " "
25	26.6 " "	45.4 " "
26	17.6 " "	48.4 " "
Composite of all above tests	11.6 " "	15.6 " "

Because the ranges for the Sterling tests straddle zero change, there is no certainty at the 95% level that the differences in the means are statistically significant.

Although the means do not always indicate a statistically significant increase in production rate, it is clear that the production rate was similarly not decreased even though many of the tests included explosives which were used for the first time in South Florida. Also, load factors used in the testing at Sterling and at Miramar were considerably lower for the most part than those normally used.

The means, standard deviations, and variances have been given for the tests at Coral Aggregate Corporation. For completeness, the data for Miramar Rock is given in Table 17.

TABLE 15

FLORIDA ROCK & SAND CO. - STATISTICAL PARAMETERS FOR PRIMARY CRUSHING OF TEST MATERIAL

	CU. YD./Hr Rate Sample Mean	Standard Deviation	Square of Standard Deviation	No. of Samples
TEST # 38	347.13	35.639	1270.11	24
TEST # 39	361.41	57.417	3296.75	29
TEST # 40	327.79	40.467	1637.60	29
TEST # 41	334.18	24.045	578.15	34
TEST # 42	328	24.518	601.13	24
TEST # 43	290.32	37.660	1418.28	37
TEST # 44	283.07	44.918	2017.64	15
TEST # 45	345.89	49.763	2476.33	27
TEST # 46	355.71	33.661	1133.09	17
TEST # 47	348.47	48.050	2308.82	19
TEST # 48	353.55	46.089	2124.16	20
TEST # 49	367.3	36.486	1331.22	11
BACKGROUND	289.94	55.808	3114.56	17

TABLE 16

STERLING CRUSHED STONE CO. - STATISTICAL PARAMETERS FOR PRIMARY CRUSHING OF TEST MATERIAL.

	CY/30 min. Rate Sample Mean	Standard Deviation	Square of Standard Deviation	No. of Samples
TEST # 19	176.6	28.30	800.8	8
TEST # 20 & 21	183.1	7.14	51.0	8
TEST # 22	170.1	12.30	151.36	9
TEST # 23	173.5	10.29	105.87	11
TEST # 24	180	16.81	282.44	10
TEST # 25	182	8.26	68.29	8
TEST # 26	185	18.20	331.11	9
Composite Non-Test	177.3	24.18	584.47	94
" Test	178.3	15.79	249.41	63

TABLE 17

MIRAMAR ROCK - STATISTICAL PARAMETERS FOR PRIMARY CRUSHING OF TEST MATERIAL.

Test No.	Mean	Standard Deviation	Square of Standard Deviation	Number of Samples
U-4	320.6 CY/hr	27.3	745.0	5
U-5	202.2 "	88.1	7771.4	9
U-6	291.7 "	33.9	1150	6
U-7	309.4 "	16.0	254.8	5
Normal	271.4 "	50.0	2500.5	13

CHAPTER 7

ECONOMIC ANALYSIS

ECONOMIC ANALYSIS OF RESULTS

From the test data and from information obtained in interviews or from public records, seven different situations were compared to investigate the effect of the tube casing method on the cost of drilling and blasting. These seven cases are:

(1) Bulk slurry loaded into 2 1/4" tube casings compared to loading 2" diameter dynamite through the kelly bar. No conversion of equipment.

(2) 2" packaged products loaded into 2.493" ID tube casings compared to loading 2" dynamite through the kelly bar. Load factor reduced about 15%. No change in processing rate. Conversion required.

(3) 2" packaged products loaded into 2.493" ID tube casings compared to loading 2" dynamite through the kelly bar. Load factor about the same. Minimum increase in processing rate indicated by post shot analysis. Conversion to 3" ID kelly required.

(4) Same as 3 except that maximum increase in processing rate indicated by statistical analysis of post shot data is used.

(5) 2" packaged products loaded into 2.493" ID tube casings with load factor reduced up to 32% compared to 3" dynamite loaded through the kelly bar. Conversion required.

(6) 1 3/4" dynamite with a density of 1.7 loaded into 2 1/4" tube casings as compared with 2" dynamite loaded through the kelly bar. No conversion of equipment.

(7) Bulk slurry in 4 1/2" ID tube casings. No conversion required for 5" ID kelly bars.

In making these analyses, the hours required to drill and load through the kelly bar are based on the average daily drilling rate for that site in 1976 reported on County blasting reports, or if these were not available, the rate was based on the average rate per hole obtained by observing at least a full day's drilling. The latter prejudices tha data in favor of the kelly bar method because it does not include any significant intra-day breakdown time. Observed rates for the kelly bar method were used in Case No. 2 at Sterling Crushed Stone when a contractor was using a 2 1/2" ID kelly bar in this pit. Drilling is normally done with a 3 1/2" ID kelly. Observed rate was also used at Miramar Rock because blasting reports were not required by Broward County in 1976.

The rates used for the 3" ID kelly bar system are the actual rates obtained during the testing at that particular site.

Tube casing cost is based on prices effective on December 1, 1977 which were higher than the cost of the tubing used in the testing program. Sales tax is added to the cost of the tubing and transportation by piggyback rail for trailer load quantities is included as well as local delivery. The amount of loss is the percentage experienced over the duration of the program.

Purchase of explosives during the test program involved relatively small quantities and consequently the cost per hundredweight was higher than that which would be paid by most operators. Costs varied for different products and different brands but this, and the quantity discounts introduce complications into the analysis which only cloud the picture. Consequently, one price was used for all products and all suppliers. That price is $37 per hundredweight or $0.37 per pound. One can adjust the analyses to their particular situation by multiplying the load factor in the analysis by the price they pay per pound and substituting the result for the Explosives cost per cubic yard used in the analysis. Other values should then be adjusted accordingly.

Cost for detonators assume that when detonating cord downlines are used, there will be an EB cap at the top of the hole with short leg wires. The following prices were used as required by the particular initiation system being analyzed:

EB caps with short leg wires	$ 0.70 each
EB caps with 60' leg wires	1.95 each
1/2 pound primers	1.10 each
25 grain boosters	0.40 each
25 grain detonating cord	0.04 per foot
50 grain detonating cord	0.05 per foot
4 1/2 grain detonating cord	0.03 per foot
Primadets with 50' tails	1.85 each

These costs are relatively straightforward and can be adjusted as prices change.

Drilling and loading cost was based on the contract rate for the test drilling of $55.50 per hour including fuel oil, insurance, normal repairs, and bit sharpening. By using this rate multiplied by the actual times, the costs for overhead (including depreciation), maintenance, and all normal operating costs are automatically taken care of at a competitive rate. Operators drilling with their own fully depreciated equipment would find their costs considerably lower. Again, the analysis introduces bias in favor of the kelly bar method. This is because the drill rig is not actually required for the loading of the tube casings. This can be done by a $10/hour + fringes loading crew and an explosives truck. The drill rig, of course, is required when loading through the kelly is done.

Amortization of conversion cost is based on an expenditure of $8000 depreciated over 10 years by straight line methods, and charged to the drilling operation on the basis of 1000 hours per year. The basis for the $8000 conversion cost is described in the section on Conversion of Existing Drill Rigs.

The dollar amount shown for the cost value of added production per cubic yard is more complicated to determine than any of the above. When improved blasting results in an increase in processing rate, it definitely improves the economics of the operation without any additional capital expenditure, if existing processing equipment is operating at capacity. The problem is to determine a reasonable value for this added production without making a cost analysis of the entire processing system. The method used in these analyses was to consider that most of the cases involved plants with capacities somewhere around 600,000 cubic yards per year or more. If the plant produced only limerock base course material, the maximum sales revenue would be about $1,530,000. (600,000 CY X 1.5 tons per yard X $1.70 per ton list price for limerock base course). Fixed costs were estimated to be about 20% at full capacity and the breakeven point was assumed to be reached at 67% of capacity, or 400,000 cubic yards. Figure 35 is a breakeven graph showing these relationships. By dividing the profit spread of $162,500 where annual production is 600,000/CY, by the difference between 600,000 cubic yards and the breakeven point (400,000 CY),a value of $0.8125 per cubic yard results. The assumption is made that the primary crusher is operating at capacity and that each additional cubic yard of production can be sold at the list price. With these assumptions, each additional cubic yard of rock that can be processed through the primary crusher makes a contribution of $0.8125 toward annual profit.

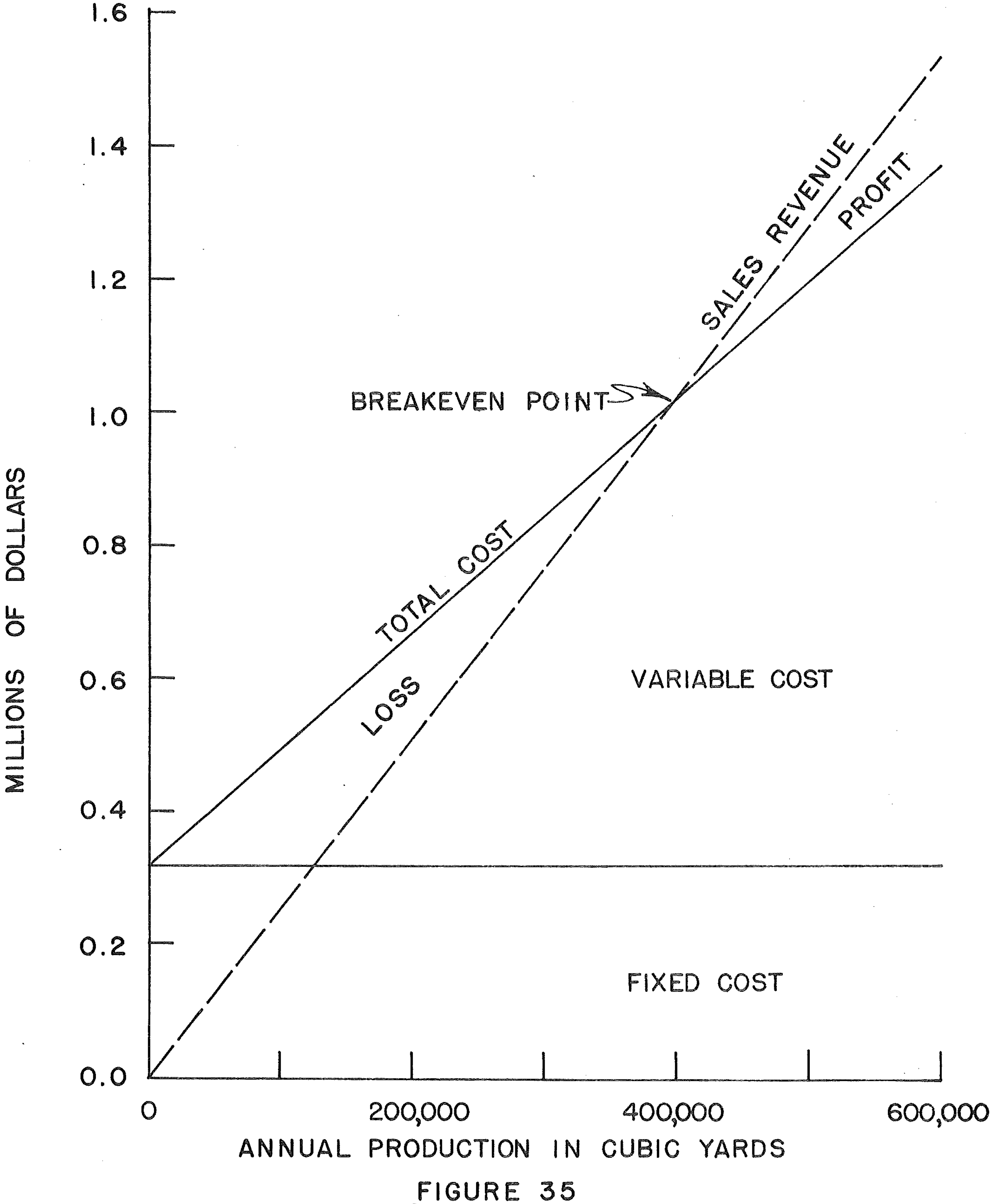
If the statistical treatment of the processing data indicates that improved blasting results in an increase of at least 20 CY/hr, and if the plant operates for 2000 hours per year, then the annual increase if 40,000 cubic yards. This has a dollar value of $32,500. When this amount is spread over the total annual production, its contribution to reducing costs is indicated. Under the conditions shown on the graph, the above amount of $32,500 would be divided by 640,000 cubic yards (600,000 CY capacity plus 40,000 CY additional). This yields a cost reduction of $0.051 per cubic yard.

Although the level of fixed costs and breakeven points may vary considerably from plant to plant, it is felt that the value obtained by this method reasonably reflects average experience.

In the case of a statistically significant decrease in production, the annual decrease in cubic yards is multiplied by $0.8125 and the product is divided by (600,000 C.Y. - Annual decrease) to give the additional cost per cubic yard of the decreased production.

Conversion used in converting an in-place cubic yard or rock to a ton of product is to multiply the in-place cubic yards by 1.5. In converting costs per cubic yard to cost per ton, the cubic yard amounts are divided by 1.5.

The list price of $1.70 per ton used for limerock base course material represents a common price in effect in Dade and Broward Counties in December, 1977. This price is considerably lower than the list price for sized aggregate and thus states the percentage increase as a higher value than if compared to average price or to the price for sized aggregate.



1.6
1.4
1.2
1.0
0.8
0.6
0.4
0.2
0.0
MILLIONS OF DOLLARS
SALES REVENUE
PROFIT
BREAKEVEN POINT
TOTAL COST
LOSS
VARIABLE COST
FIXED COST
0
200,000
400,000
600,000
ANNUAL PRODUCTION IN CUBIC YARDS

FIGURE 35

The analyses presented in the seven cases reveal the following economic facts about the use of tube casings in comparison to the kelly bar method:

Case 1. Bulk slurry in small diameter tube casings (2 1/4" ID) results in a direct drilling and blasting cost increase of 11.7 cents. The decrease in production rate increases total unit cost 25.2 cents per cubic yard. In the case presented, this would make the total increased cost 36.9 cents per cubic yard or 24.6 cents per ton. Compared with the other small diameter alternatives, this method is economically unattractive.

Case 2. With a 15% reduction in load factor, which can easily be accomplished at most sites with tube casings with no effect on production, the net increase in cost per ton is 4.5 to 4.9 cents per ton depending on whether packaged slurry or dynamite is used, respectively. This represents 2.6% to 2.9% of list price of $1.70.

Case 3. Where the load factor is already at a low value, and the primary crusher is operating at full capacity, the use of tube casings offers an increase in capacity by providing better feed. This increase in capacity reduces the additional cost in Case 3 where the minimum increase is assumed to 2.3 cents per ton with dynamite, and 1.4 cents per ton with packaged slurry.

Case 4. Where the maximum increase in capacity is assumed, the additional cost when dynamite is the explosive is 1.1 ¢ per ton. With packaged slurry, there is actually a cost decrease of 7.6 ¢ per ton.

Case 5. At this site, no attempt was made to reduce the load factor significantly when dynamite was used although, and the average value of added production was very small. The additional cost in this case is 12.9 ¢ per ton. With the packaged slurries, an effort was made to reduce the load factor and this did not affect the production rate. By shucking the plastic wrapper from the explosive, maximum loading density was obtained and the additional cost of using the tubes with this procedure was only 2.5 ¢ per ton. This could be further reduced because the loading time of 0.125 hours per hole is charged at the full rig cost of $55.50 per hour plus $0.80 conversion cost. This work could be performed by two men without a drill rig for a cost of about $12.50 per hour including fringe benefits. This would reduce the additional cost to 1.1 ¢ per ton. Although shucking increases the loading time, it is obviously worthwhile in minimizing cost.

Table 18

Economic Analysis: Case 1: Bulk Slurry in 2 1/4" Tubes
Coral Aggregates Corp: Holes 60' deep; 9' Burden X 9' Spacing

DRILLING & LOADING	2 1/2" Kelly Bar Loading 2" cartridges	2 1/4" Tube Casings Bulk
Hours/hole to drill & load thru kelly	.368	----
Hours/hole to drill & place tubes	----	.219
Hours/hole to load tubes	----	.136
Total Time	.368	.355
$55.50/hr. x total time	$20.419	$19.703
Drilling & Loading Cost/C.Y.	$ 0.113	$ 0.109
Tubing Cost		
Direct Use/hole @ $0.183 per foot	----	$11.163
Loss @ $2.8%	----	$ .313
Total Tubing Cost		$11.476
Tubing Cost per Cubic Yard		$ .064
Explosives & Detonators:		
Pounds per hole	90.	114.
Cubic yards per hole	180.	180.
Load Factor	0.50	0.63
Explosive Used	AL-4	P-DYN-4
Explosive Cost per cubic yard	$ 0.185	$ 0.233
Detonating cord cost per cubic yard	$ 0.014	----
EB cap cost per cubic yard	$ 0.004	$ 0.015
Primer Cost/per cubic yard	----	$ .012
Total Explosive & Detonator cost per C.Y.	$ 0.203	$ 0.260
Total Direct Cost per Cubic Yard	$ 0.316	$ 0.433
Increased Cost with tube casings per C.Y.	----	+$ 0.117
Cost Value of Added Production/C.Y.	----	+$ 0.252
Net Increase in cost/C.Y.	----	+$ 0.369
Net Increase in cost per ton		+$ 0.246
Increase as to % of $1.70 per ton list price		+ 14.5%

Table 19

Economic Analysis: Case 2: Packaged 2" Products in 2.493" ID Tubes
Slight decrease in Load Factor. Sterling Crushed Stone: Holes 50' Deep

Drilling & Loading	2 1/2" Kelly Bar Loading	3"ID Kelly Bar System	
	2" cartridges	2"Dynamite	2"Packaged Slurry
Hours/hole to drill & load thru kelly	0.303	----	
Hours/hole to drill & place tubes	----	0.395	0.395
Hours/hole to load tubes	----	0.075	0.075
Total Time	0.303	0.470	0.470
$55.50/hr. x total time	$16.817	$26.085	$26.085
Amoritization of conversion costs:$0.80 x total time		$ 0.376	$ 0.376
Total drilling & loading cost per hole	$16.817	$26.461	$26.461
Total drilling & loading cost per C.Y.	.108	$ 0.145	$ 0.145
Tubing Cost			
Direct Use/hole @ $0.225 per foot	----	$11.475	$11.475
Loss @ 2.8%	----	$ 0.321	$ 0.321
Total Tubing Cost	----	$11.796	$11.796
Tubing Cost per Cubic Yard	----	$ 0.064	$ 0.065
Explosives & Detonators			
Pounds per hole	75.	75.	71.
Cubic yards per hole	156.	183.	181.
Load factor	0.48	0.41	0.39
Explosive cost per cubic yard	$ 0.178	$ 0.152	$ 0.144
Detonating cord cost per cubic yard	$ 0.014	$ 0.012	----
EB cap cost per cubic yard	$ 0.004	$ 0.004	$ 0.011
Primer cost/C.Y.	----	----	.006
Total Explosive & Detonator Cost/C.Y.	$ 0.196	$ 0.168	$ 0.161
Total Direct Cost per C.Y.	$ 0.304	$ 0.377	$ 0.371
Increased Cost with tube casings per cubic yard	----	+$ 0.073	+$ 0.067
Cost value of added production per cubic yard	----	----	----
Net Increase in cost/C.Y.	----	+$ 0.073	+$ 0.067
Net Increase in cost/ton	----	+$ 0.049	+$ 0.045
Increase as Percent of $1.70 per ton list price	----	+ 2.9%	+ 2.6%

Table 20

Economic Analysis: Case 3: Packaged 2" Products in 2.493" ID Tube Casings
Minimum Increase in Processing Rate: Florida Rock & Sand Co. Holes 50' Deep

	2 1/2" Kelly Bar Loading	3" ID Kelly Bar System	
Drilling & Loading	2" Cartridges	2"Dynamite	2" Packaged Slurry
Hours/hole to drill & load thru kelly	$0.409	----	
Hours/hole to drill & place tubes	----	$0.397	$0.397
Hours/hole to load tubes	----	0.110	0.110
Total Time	$0.409	$0.507	$0.507
$55.50/hr. x total time	22.700	28.139	28.139
Amortization of conversion costs:$0.80 x total time	----	.406	.406
Total drilling & loading cost per hole	22.700	28.545	28.545
Total drilling & loading cost per C.Y.	0.086	0.133	0.133
Tubing Cost			
Direct Use/hole @ $0.225 per foot	----	11.475	11.475
Loss @ 2.8%	----	0.321	0.321
Total tubing cost		11.796	11.796
Tubing Csot per Cubic Yard	----	0.055	0.055
Explosives & Detonators			
Pounds per hole	90.	81.	78.
Cubic Yards per hole	265.	214.	214.
Load factor	0.34	0.38	0.37
Explosive cost per cubic yard	0.126	0.141	0.137
Detonating cord cost per cubic yard	0.010	0.010	0.010
EB cap cost per cubic yard	0.003	0.003	0.003
Total Explosive & Detonator Cost per C.Y.	0.139	0.154	0.150
Total Direct Cost per C.Y.	0.225	0.342	0.338
Increased Cost per cubic yard with tube casings	----	+0.117	+0.113
Cost Value of Added Production/C.Y.		-0.082	-0.092
Net Increase in cost/C.Y.		+0.035	+0.021
Net Increase in cost per ton		+0.023	+0.014
Increase as percent of $1.70 per ton list price		+1.4%	+0.8%

Table 21

Economic Analysis: Case 4: Packaged 2" Products in 2.493" ID Tube Casings
Maximum increase in processing rate: Florida Rock & Sand Co. Holes 50' Deep

	2 1/2" Kelly Bar Loading	3" ID Kelly Bar System	
	2" Cartridge	2" Dynamite	2" Packaged Slurry
Drilling & Loading			
Hours/hole to drill & load thru kelly			
Hours/hole to drill & place tubes			
Hours/hole to load tubes			
Total Time			
$55.50/hr. x total time			
Amortization of conversion costs: $0.80 x total time			
Total drilling & loading cost per hole			
Total drilling & loading cost per C.Y.			
Tubing Cost			
Direct Use/hole @ $0.225 per foot			
Loss @ 2.8%			
Total tubing cost			
Total tubing cost per C.Y.			
Explosives & Detonators			
Pounds of explosives per hole			
Cubic yards of rock, per hole			
Load factor in lbs/C.Y.			
Explosives cost per cubic yard			
Detonatign cord cost per cubic yard			
EB cap cost per cubic yard			
Primer cost per cubic yard			
Total Explosive & Detonator cost/C.Y.			
Total Cost per Cubic Yard	$ 0.255	$ 0.342	$ 0.338
Increased Cost/C.Y. with Tube Casings	----	+$ .117	+$ .113
Cost Value of Added Production per C.Y.	----	-$ .100	-$ .227
Net Increase in Cost/C.Y.	----	+$ .017	-$ .114
Net Increase in Cost/Ton	----	+$ .011	-$ .076
Increase as % of $1.70/ton list price	----	+ 0.7%	- 4.4%

Table 22

Economic Analysis: Case 5: 2" Packaged Products in 2.493" Tube Casings
Reduced Load Factor: Miramar Rock 50' holes

	3 1/2" Kelly Bar Loading		3" ID Kelly Bar System	
	3" Cartridges	2"Dynamite	2"Packaged	Shucked Packages
Drilling & Loading				
Hours/hole to drill & load thru kelly	0.273	----	----	----
Hours/hole to drill & place tubes	----	0.400	0.400	0.400
Hours/hole to load tubes	----	0.036	0.057	0.125
Total time	0.273	0.436	0.457	0.525
$55.50/hr. x total time	$15.152	$24.198	$25.364	$29.138
Amortization of conversion costs:$0.80 x total time	----	$ 0.349	$ 0.366	$ 0.420
Total Drilling & Loading Cost per hole	$15.152	$24.547	$25.730	$29.558
Total Drilling & Loading Cost per C.Y.	0.057	0.183	0.154	0.121
Tubing Cost				
Direct use/hole @ $0.225/foot	----	11.475	11.475	11.475
Loss @ 2.8%	----	0.321	0.321	0.321
Total tubing cost	----	11.796	11.796	11.796
Total tubing cost per C.Y.	----	0.088	0.071	0.048
Explosives & Detonators				
Pounds of explosive per hole	175	84	78	110
Cubic yards of rock per hole	267	134	167	245
Load factor in lbs./C.Y.	0.66	0.63	0.47	0.45
Explosives cost per cubic yard	$ 0.244	$ 0.233	$ 0.174	$ 0.167
Detonating cord cost per cubic yard	$ .008	$ 0.016	$ 0.016	$ 0.011
EB cap cost per cubic yard	$ .003	$ 0.005	$ 0.004	$ 0.003
Primer cost per cubic yard	----	----	----	----
Total explosive & detonator cost/C.Y.	$ .255	$ 0.254	$ 0.194	$ 0.181
Total cost per cubic yard	$ 0.312	$ 0.525	0.419	0.350
Increased Cost/C.Y. with Tube casings	----	+$ 0.213	+$ 0.107	+$ 0.038
Cost value of added production/C.Y.	----	-$ 0.020	----	----
Net increase in cost/C.Y.	----	+$ 0.193	+$ 0.107	+$ 0.038
Net increase in cost/ton	----	+$ 0.129	+$ 0.071	+$ 0.025
Increase as % of $1.70/ton list price	----	+ 7.6%	+ 4.2%	+ 1.5%

Table 23

Economic Analysis: Case 6: 1 3/4" Packaged Products in 2.200" ID Tube Casings
Capeletti Pit #12 43 holes

Drilling & Loading	2 1/2" Kelly Bar Loading 1.4 density dynamite 2" Cartridges	Tube Loading 1.7 density dynamite 1 3/4" Cartridges
Hours/hole to drill & Load thru kelly	0.193	
Hours/hole to drill & place tubes	----	0.250
Hours/hole to load tubes	----	0.042
Total time	0.193	0.292
$55.50/hr. x total time	$10.712	$16.206
Amortization of conversion costs: $0.80 x total time	----	----
Total drilling & loading cost per hole	$10.712	$16.206
Total drilling & loading cost per C.Y.	$ 0.067	0.102
Tubing Cost		
Direct use/hole @ $0.184/foot	----	$ 8.096
Loss @ 2.8%	----	$ 0.227
Total tubing cost	----	$ 8.323
Total tubing cost per C.Y.	----	$ 0.052
Explosives & Detonators		
Pounds of explosives per hole	68.	60.
Cubic yards of rock per hole	159.	159.
Load factor in lbs./C.Y.	0.43	0.38
Explosives cost per cubic yard	$ 0.159	$ 0.141
Detonating cord cost per cubic yard	$ 0.012	$ 0.009
EB cap cost per cubic yard	$ 0.004	$ 0.004
Primer cost per cubic yard	----	----
Total explosive & detonator cost/C.Y.	$ 0.175	$ 0.154
Total cost per cubic yard	$ 0.242	$ 0.308
Increased cost/C.Y. with tube casings	----	+$ 0.066
Cost value of added production/C.Y.	----	----
Net increase in cost/C.Y.	----	+$ 0.066
Net increase in cost/ton	----	+$ 0.044
Increase as % of $1.70/ton list price	----	+$ 2.6%

Table 24

Economic Analysis: Case 7:

	5" Kelly Bar Loading 4" Dynamite	4 1/2" Tube Casings Bulk Slurry
Drilling & Loading:		
Hours/hole to drill & load thru kelly	0.448	----
Hours/hole to drill & place tubes	----	0.466
Hours/hole to load tubes		0.139
Total time	0.448	0.605
$55.50/hr. x total time	$24.864	$33.578
Amortization of conversion cost $0.80 x total time	----	----
Total drilling & loading cost per hole	$24.864	$33.578
Total drilling & laoding cost per C.Y.	$ 0.060	$ 0.051
Tube Cost		
Direct use/hole @ $0.203 foot	----	$ 9.541
Loss @ 2.8%	----	$ .267
Total tubing cost	----	$ 9.808
Total tubing cost per C.Y.		$ 0.015
Explosives & Detonators		
Pounds of explosive per hole	210.	322.
Cubic yards of rock per hole	417.	662.
Load factor in lbs./C.Y.	0.50	0.49
Explosives cost per cubic yard	$ 0.185	$ .181
Detonating cord cost per cubic yard	$ 0.007	$ ----
EB cap cost per cubic yard	$ 0.002	$ 0.007
Primer cost per cubic yard	$ 0.018	$ 0.003
Total explosive & detonator cost/C.Y.	$ 0.212	$ 0.188
Total cost, per cubic yard	$ 0.272	$ 0.254
Increased cost/C.Y. with tube casings	----	- 0.018
Cost value of added production per C.Y.	----	----
Net increase in cost/C.Y.	----	- 0.018
Net increase in cost/ton	----	- 0.012
Increase as % of $1.70/ton list price	----	- 0.7%

Case 6. This case offers a very attractive alternative with no rig conversion needed, except for having the loading tee reamed out slightly. Additional cost is only 4.4 ¢ per ton.

Case 7. The analysis of this large diameter method indicates a cost <u>reduction</u> of 1¢ per ton and in this example the loading is charged at the full $55.50 per hour drill rig rate. This could be reduced in practice and a reduction of as much as 4¢ per ton is possible according to those who have operational experience with the method.

A frequent question is, "How much will it cost to use the tube casings?" Perhaps the foregoing analyses indicate there is no set answer. Cost compared to present methods,depends on how much an operator has already done to optimize his blasting, and how much effort he will expend in getting the maximum benefit from the use of tube casings.

To achieve the maximum benefit, it is essential for each operator who decides to use the tube casings, to determine through experimentation the broadest possible pattern. This is because all of the per hole costs, drilling, explosives, and tubing, are divided by the yield of that hole in cubic yards to determine the real cost per cubic yard. The effect of the yield on these costs is shown in a series of five graphs, all of which assume that blast holes are 50' deep and that 80 pounds of explosives are loaded into each hole, except for one case (Figure 37) in which a charge of only 72.5 pounds is assumed.

Figure 36 indicates the relationship of drilling and blasting cost to yield per hole in cubic yards when 2" diameter explosives are loaded through a 2 1/2" kelly. Equivalent load factors are indicated below the yield scale.

Figure 37, is a graph of drilling and blasting cost per cubic yard versus yield when 2 1/4" ID tubes are placed through the kelly and loaded afterwards with 1 3/4" dynamite of 1.7 density. Figure 38 shows similar relationships when 2.493" ID tubes are placed through the kelly and later loaded with 2" packaged explosives. Figure 39 is a comparison of the total drilling and blasting cost curves for all three of the preceding methods. On this graph, it can be seen that for a given yield, or load factor, the loading of 1 3/4" explosive results in a lower total cost than the 3" ID system. The problem is that the 1 3/4" diameter alternative does not offer as much opportunity to increase the yield per hole. Figure 40 shows the importance of increasing the yield in relation to the cost incurred through the use of tube casings.

In this Figure 40, the increased drilling and blasting cost per ton under the given conditions is shown in the top curve assuming no change is made in the drilling pattern in an effort to increase yield.

FIGURE 36

RELATIONSHIP OF DRILLING & BLASTING COST TO YIELD
WHEN EXPLOSIVES ARE LOADED THROUGH A 2 1/2" ID KELLY

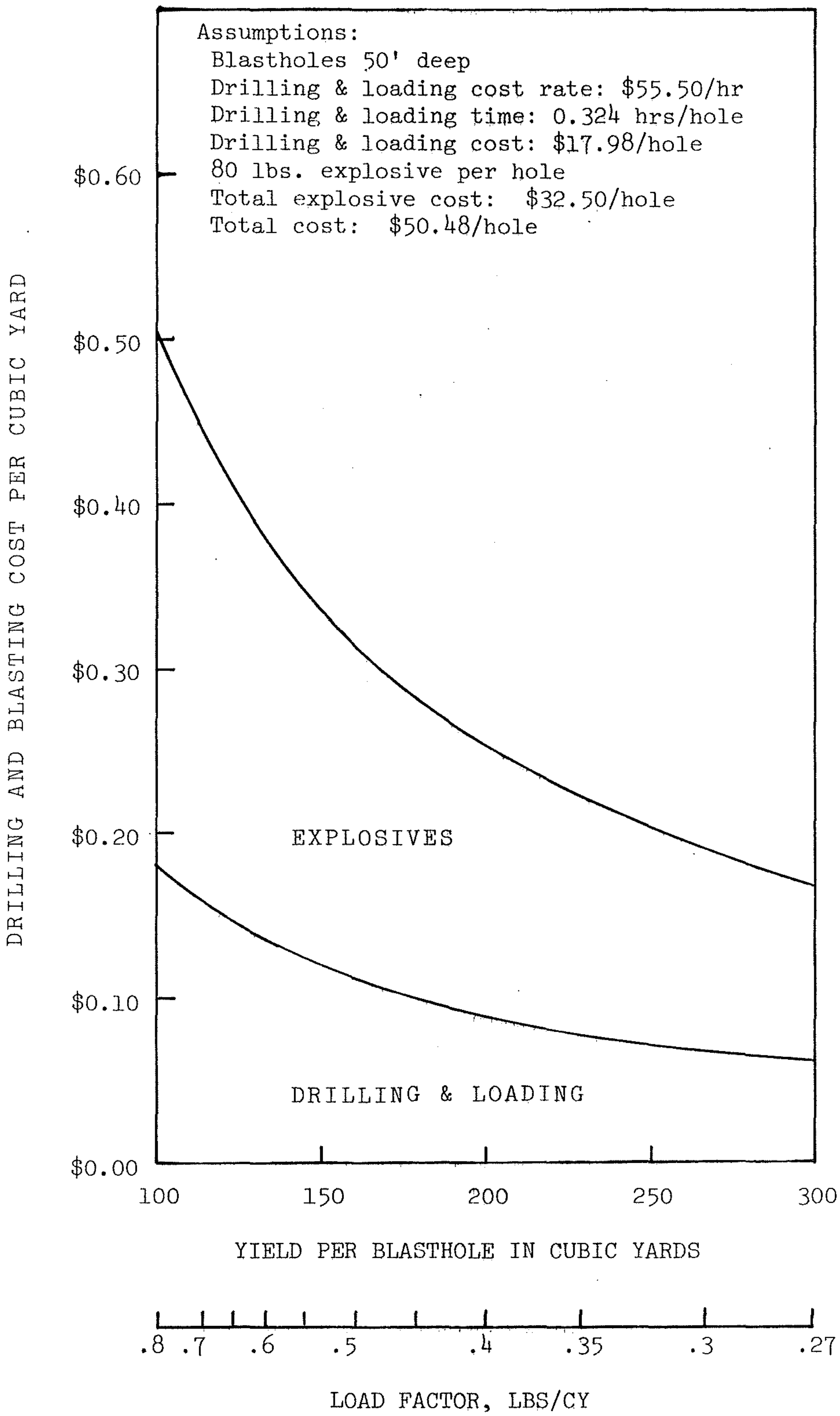


FIGURE 37

RELATIONSHIP OF DRILLING & BLASTING COST TO YIELD
2.200" ID TUBE CASINGS PLACED THROUGH A 2 1/2" KELLY
1 3/4" PAPER WRAPPED DYNAMITE: 1.7 DENSITY

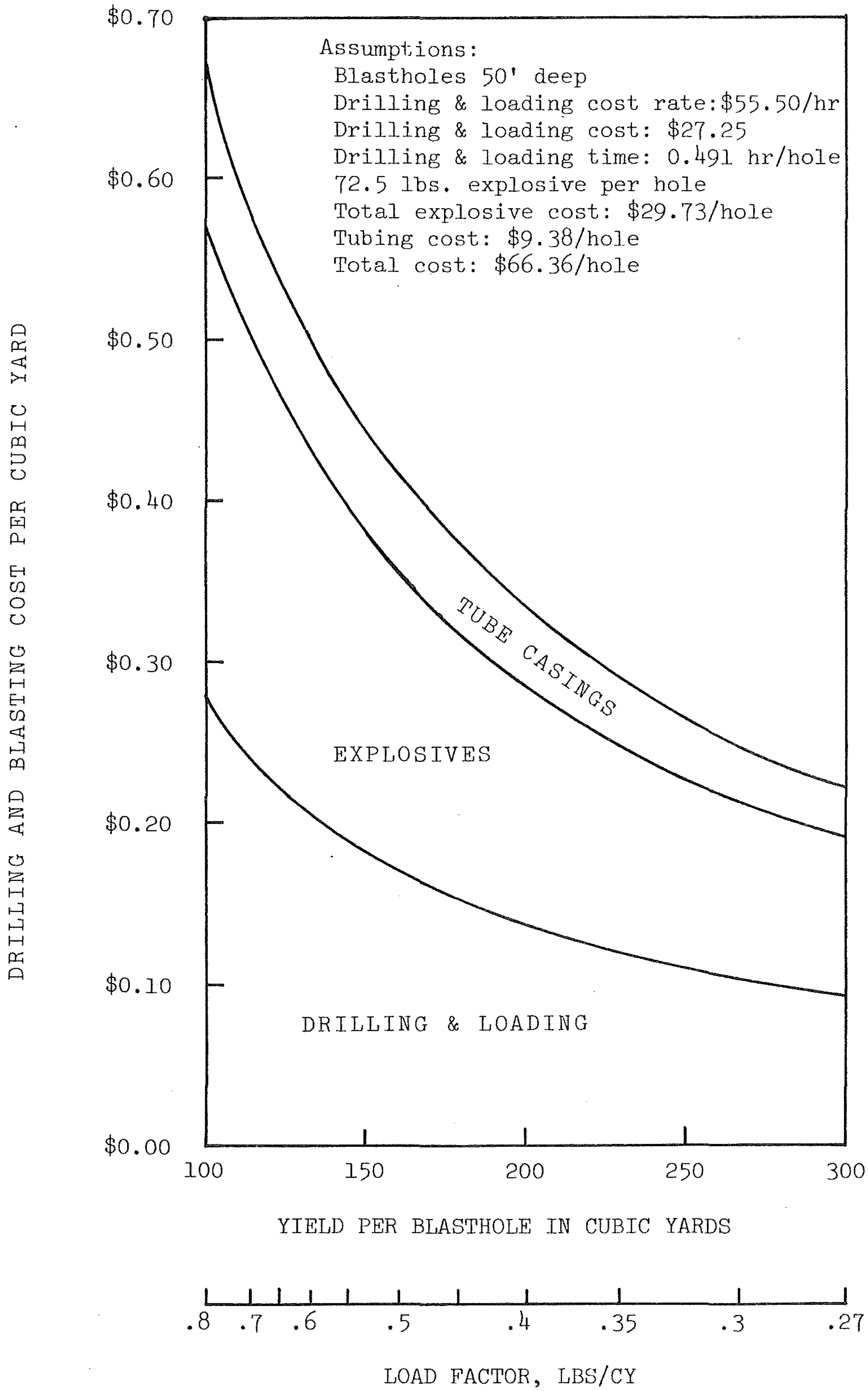


FIGURE 38

RELATIONSHIP OF DRILLING & BLASTING COST TO YIELD
2.493" ID TUBE CASINGS PLACED THROUGH A 3" ID KELLY
2" DIAM. EXPLOSIVES

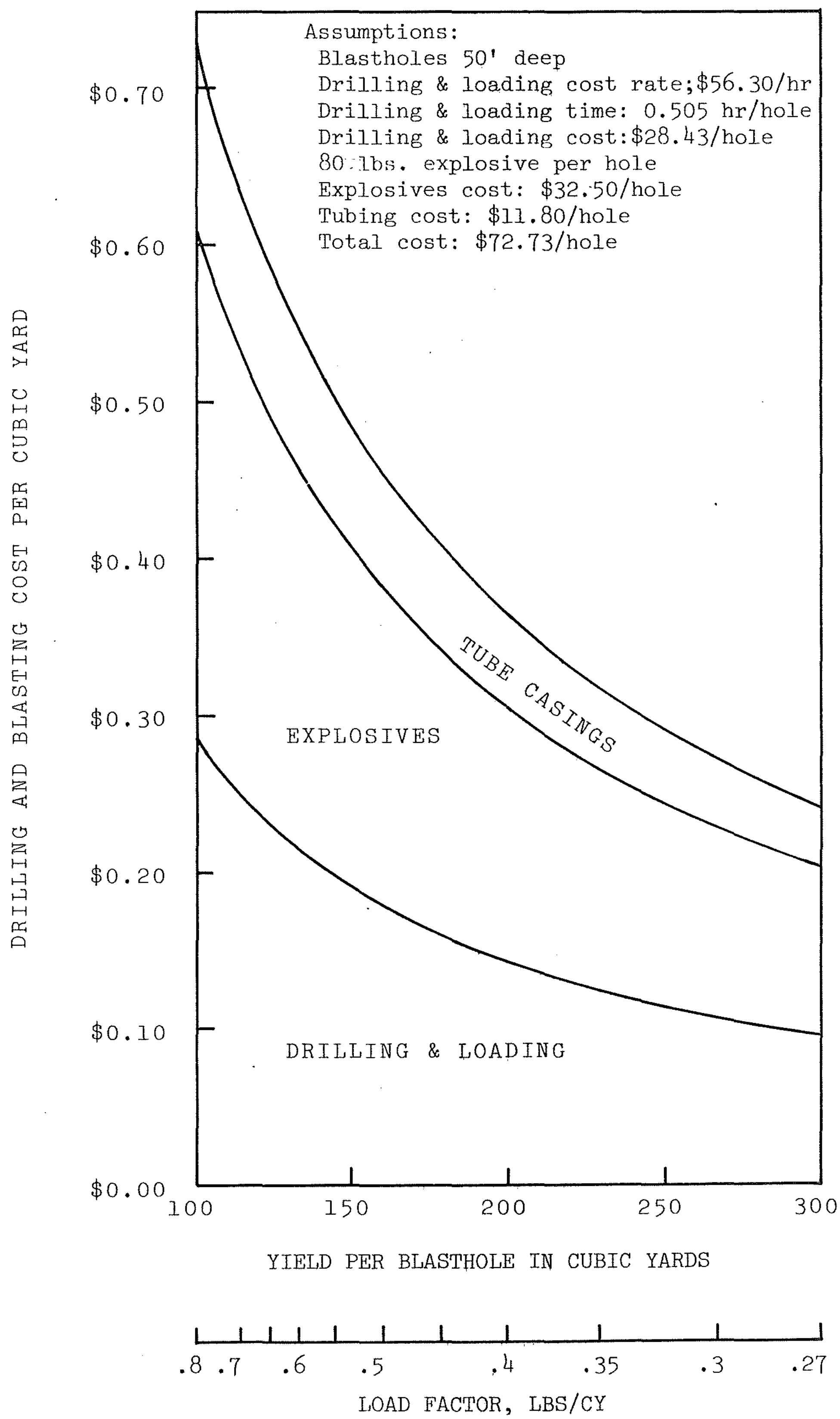


FIGURE 39

COMPARISON OF DRILLING AND BLASTING COSTS
FOR THREE DIFFERENT METHODS OF LOADING

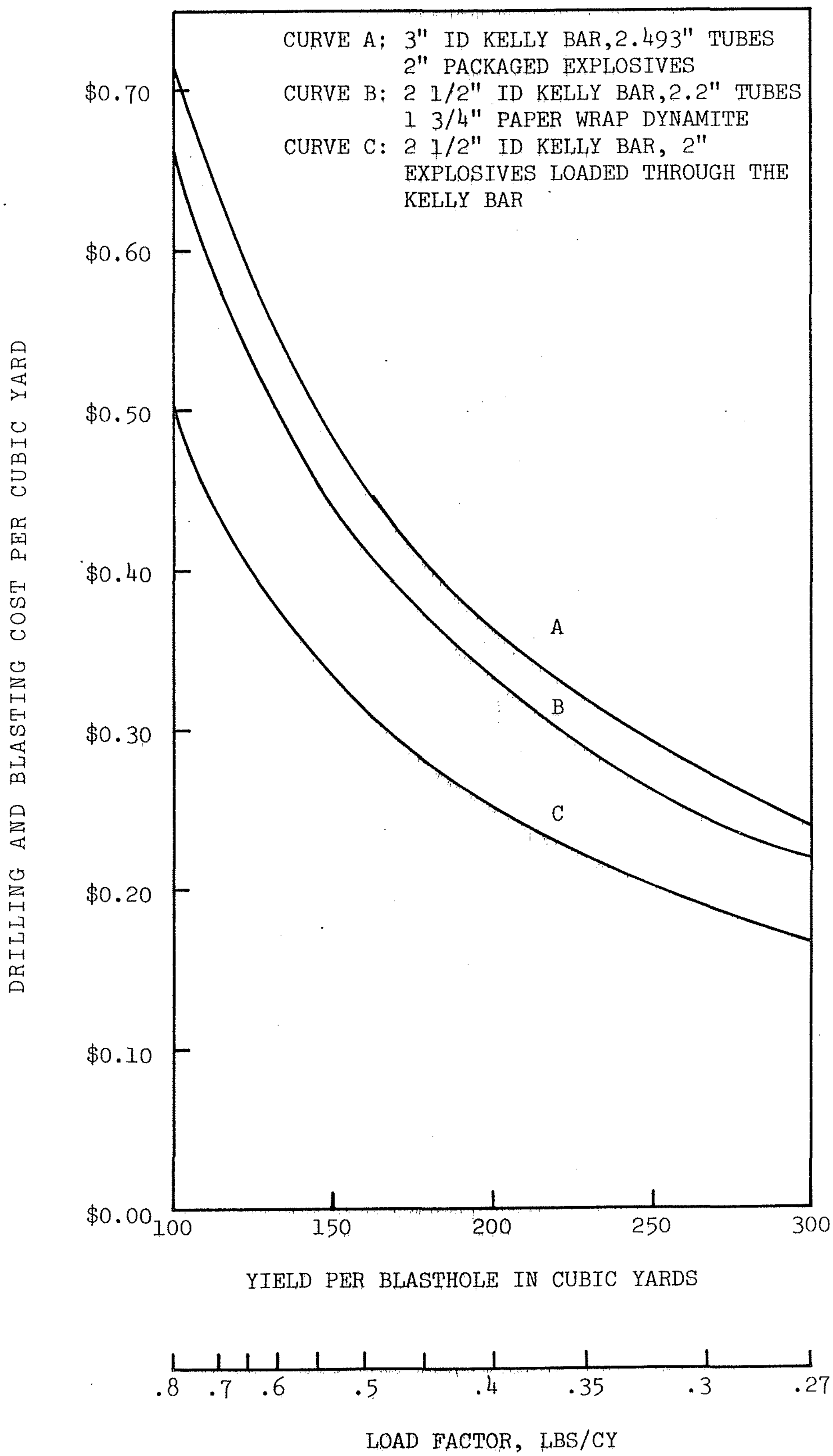
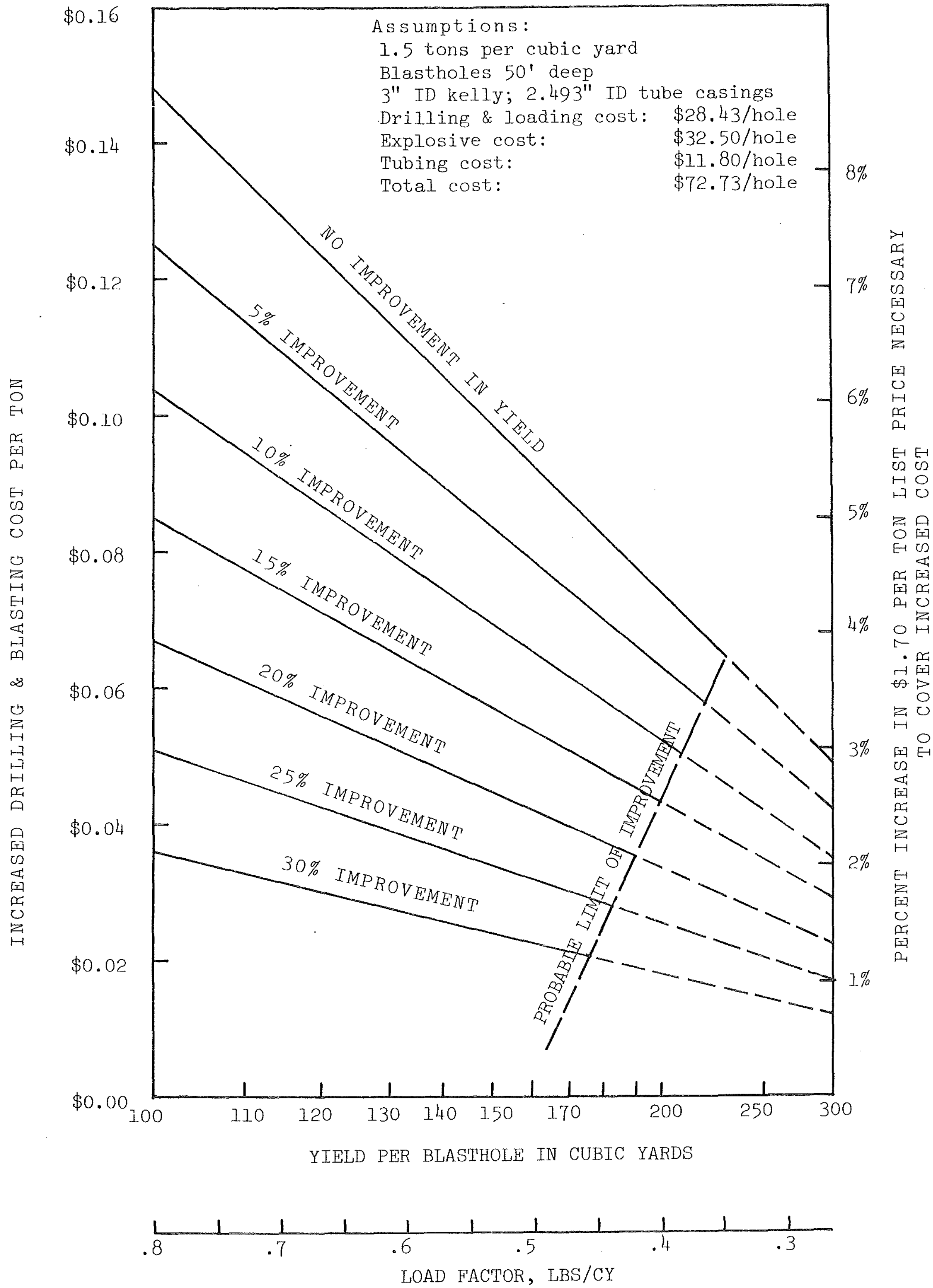


FIGURE 40



INCREASED DRILLING AND BLASTING COST FOR TUBE CASING METHOD WITH VARIOUS IMPROVEMENTS IN YIELD

Other curves are shown in this graph indicating the cost per ton if certain percentage increases in yield can be affected. This graph gives the best overall picture of the impact on cost when tube casings are used in South Florida. An additional scale on this graph indicates the percentage increase in a list price of $1.70 in order to cover the additional cost.

For example, if an operator is currently getting a yield of 160 cubic yards per hole with a load factor of 0.50 which is near the industry average, and no attempt is made to change any aspect of the blasting design, the additional cost will be about 9 cents per ton. On the other hand, if a 20% improvement in yield reduction can be made (and testing indicates this is probably feasible) the additional cost would be reduced to 3.2 cents per ton. This could be covered by an increase of 2% in the $1.70 list price of limerock base course material if that is the sole product, and less if sized aggregates are made.

It may seem that the foregoing indicates that the additional cost of the tubes is to be offset by a reduced cost for explosives. This would mean that the brunt of the additional cost would be carried primarily by the explosive manufacturers, distributors, and salesman, rather than the quarry operators. This need not be the case. If improved blasting results in an increased processing rate, then more material will have to be blasted to keep the plant operating at capacity. If the processing rate is increased 20%, and the load factor is reduced 20%, the same amount of explosives will be sold. This provides an opportunity and impetus for these concerns

(1) to utilize their technical expertise in helping their customers optimize their blasting with modern techniques, and

(2) to assist their customers in being able to continue full capacity operation through competitive drilling and blasting costs in those circumstances when the total market is inelastic.

CHAPTER 8

CONVERSION OF EXISTING DRILL RIGS

AND

TUBE CASING AVAILABILITY

CONVERSION OF EXISTING RIGS

Although 2.200" ID tubes can be placed through a standard 2 1/2" kelly bar and these can later be loaded with paper-wrapped 1 3/4" gelatin dynamite, the opportunity to optimize the yield is lessened. It is a good interim measure and it may be a satisfactory permanent solution in certain types of construction work. In high volume quarry work, however, the advantages of using explosives with diameters up to a full 2 1/4" (in polyethylene, or mylar or spiral wound cartridges) fully outweighs the cost of converting a drill rig to handle a 3" ID kelly bar.

Kelly bars used in the southern part of Dade County have a much longer drilling life than those used in Broward County. The former may have a drilling life of two or three years of fairly continous drilling; the latter may last only three or four months. Average life is about six months, or 150,000 feet of drilling.

If rig conversion is performed at a time when the kelly bar is to be replaced anyway, the cost of the conversion need not consider the cost of the bar because the difference in price between a 2 1/2" ID bar and a 3" ID bar is minor. The weight difference is only about 180 pounds.

Each kelly bar rig in South Florida is almost unique. Most basic rigs are of either Gradner-Denver, Davey or Damco manufacture (Mayhew was purchased by Gardner-Denver several years ago), but almost all have been modified in one way or another. A Damco rig may have a Gardner-Denver swivel, etc. Consequently, it is impossible to estimate what amount of conversion a rig will need without specifically inspecting that rig.

The essential aspects of converting a rig from a 2 1/2" to 3" ID kelly bar are:

(1) the swivel must have an opening large enough to accept a tube with an outer diameter of 2.733". In general swivels cannot be reamed out to increase the inner diameter.

(2) the rotary table must have an opening large enough to accept a drive bushing with a 4 1/4" opening, or be capable of having an adapter welded to the top of the rotary table which can accept such a drive bushing.

(3) The drive bushing must be reamed out or replaced.

(4) the opening in the stabilizer must be increased to accomodate the 4 1/4" OD kelly.

(5) a larger head adapter is needed for the upper end of the kelly.

(6) a larger subadapter is required.

(7) a new bit supply is necessary. This consists of about

30 bits to maintain the average quarry; more are needed in Broward County, less in southern Dade County.

(8) If a new swivel is needed in order to provide adequate inner diameter, then modifications must be made to the yoke which holds the swivel, or a new yoke must be purchased. In most cases, it can be modified.

(9) changing the swivel changes the axial alignment with the center of the rotary table. Consequently, the rotary table must be moved to maintain alignment. This requires extension of the drive train.

(10) If the present swivel has sufficient inner diameter, the loading tee may still require modification or replacement.

Some 2 1/2" ID kelly bar drill rigs already have swivels large enough to accept the 2.733" OD of the tubes (2.493" ID), and sufficient rotary table opening for the bigger drive bushing. In these cases, conversion is a relatively easy job. Some of the smaller drill rigs probably can't be converted but these are more likely to be employed in shallow construction work.

Keeping in mind all of these variations, one can consider a maximum cost of conversion knowing that the true cost is probably something less.

Swivel replacement and new bit supply are the main expenditures. Estimated cost of maximum conversion is as follows:

Swivel	\$3,321.02
New Yoke	361.11
Modify drive bushing	400.00
Align turntable	400.00
Modify stabilizer	200.00
30 bits	3,000.00
Subadaptor	112.00
Head adaptor	100.00
Labor to install swivel	100.00
	\$7,994.13

If this sum is depreciated over 10 years by straight line methods, it amounts to roughly \$800 per year. If this is distributed over 1000 hours per year it amounts to \$0.80 per hour. This is the conversion cost per hour used in the economic analyses. Most rigs would operate for 2000 hours which would cut this hourly cost in half.

Much of the conversion is a matter of custom cutting and fitting. Most operators and the larger contractors have in-house capability to do this. There are also several shops which can handle this type of work but they can only handle one or two rigs at a time.

Obtaining the proper steel for the kelly bars could take 60 to 90 days. These sections would then have to be welded, fluted and threaded. Bits would have to be made.

If a lot of operators wanted to convert, shops would have to schedule the work so that 3" ID kelly bars would be delivered first to those not requiring swivel replacements. This would allow time for new swivel delivery and alignment modifications to be made before delivery of the larger kelly bar.

Total time required for all 2" rigs to be converted is estimated by a kelly bar manufacturer to be one year. If a prohibition on loading explosives through the kelly bar was promulgated, many would have to rely on the smaller 2.200" ID tube and 1 3/4" explosives for up to one year.

TUBE CASING AVAILABILITY

Availability of the tube casings is not a problem. Sonoco Products Co. is one of the larger producers of the phenolic-resin treated paperboard tubes. Only their products were used in the tests because they were the source of the tubes for the unit tests and special mandrels were made to produce the 2.493" IK tube. A one time charge for this was avoided by purchasing the additional needs from them. Also, the least expensive way to purchase the tubes is in trailer load quantities and one trailer fulfilled the needs for about two-thirds of the program.

Nevertheless, there are other sources in the southeastern United States and the names of some of these companies are available from the Engineering Contractors Association.

Delivery of tubes by piggyback trailer from Sonoco Products requires about three to four weeks from date of order.

Sources in the paperboard tube industry state that if hole stabilization with tubes becomes a common practice in South Florida, numerous concerns will be fighting for the business. This should guarantee a competitive price.

CHAPTER 9

CONCLUSIONS AND RECOMMENDATIONS

CONCLUSIONS AND RECOMMENDATIONS

Expendable paperboard tubes will stabilize blastholes in South Florida. Tests at 14 sites with small diameter tubing establish that resin-treated tube casings with a wall thickness of 0.060" will remain open with full diameter for as long as nine months. In most cases full depth will be maintained but in very sandy areas where air drilling is used, there may be some sand fill-up at the bottom of the hole.

Paperboard tubes treated with hot wax are in operational use with large diameter (5" ID) kelly bars by the industry at two locations, and are being tested at two additional sites.

Wall thickness of the large diameter hot wax tubes depends on geologic conditions with 0.100" being satisfactory at a site in Dade County, and 0.125" being required for the sandier conditions in Broward County.

Placing the tubes through the kelly and then loading them at some later time as a separate operation eliminates the hazards that are peculiar to loading explosives through the kelly.

Tubes composed of other materials may be developed and the concepts of use developed in this report can be easily adapted.

Bulk slurry can be loaded into the big diameter tubes either from the top or by running the loading hose to the bottom and pumping out. A moderate increase in yield is generally possible because of the greater loading density per foot. This can produce a cost savings.

Packaged explosive products can be loaded into the small tubes but it is more difficult to bring about a reduction in present cost. The additional cost of using the tube casings can be largely offset by increases in yield and/or increases in processing rate. Additional cost when using tube casings will depend on the efficiency of the operator's present blasting practices, and his effort to optimize his design when tubes are used. Maximum increased cost would be about 13¢ per ton, but only 3¢ to 4¢ per ton for the average operation. The latter could be recovered by 2-3% increase in product price.

Bulk slurry in small diameter tubes does not appear to be operationally feasible.

For small diameter operations, converting the drill rig to accept a 3" ID kelly bar appears to be the best alternative because it provides more opportunity to increase the yield. Conversion performed at a time when the kelly bar needs replacement reduces the additional cost because expenditure for a new kelly would be required anyway. Conversion of all 2 1/2" kelly bar rigs to 3" ID would require up to one year, with a maximum cost of $8000 per unit.

If conversion is not possible because of rig design, or temporary unavailability of parts and services, or is not desired, high density 1 3/4" gelatin dynamite can be loaded into 2.080" ID X 0.060" thick tube casings with external couplings. These can be placed through the standard 2 1/2" kelly bar. Reaming of the loading tee may be required.

Initiating commercial explosives in blastholes in South Florida from the top of the column yields consistently poor results.

The tube casings permit the use of a higher degree of sophistication in blasting than previously possible in South Florida. Increased processing rates can offer significant reductions in overall plant cost. Improved breakage and more uniform plant feed may reduce maintenance cost on draglines, loaders, haul units, and perhaps most importantly, reduce the need for crusher buildup.

There is no convenient way to determine where the explosives are in a hole loaded by the kelly bar method. It undoubtedly varies from hole-to-hole making consistent blasting or moderately sophisticated techniques impossible. Because of the uneven distribution of explosives in the hole, relatively high load factors must be used to assure minimal breakage.

Dynamite is more popular than slurry in South Florida in the small diameter operations. Presumably this is because good design appears to be more difficult with small diameter holes and dynamite is more tolerant of poor design. Nevertheless, the inherent energy in dynamite is probably lessened by the same factors that affect slurry results but not to the same degree.

Blasting problems are probably not caused specifically by the hardness or denseness of the rock, but by the vertical distribution of rock types which vary considerably in their physical properties. Energy tends to be dissipated in the loose sand and very porous layers, leaving adjacent firm rock unbroken. This condition is aggravated by kelly bar loading because the explosive cartridges probably accumulate in areas of sand washouts and solution cavities.

Maximum yields will probably be obtained with slurry explosives because of the greater weight energy per pound.

In small diameter operations with blastholes on approximately 10' centers, results with slurry explosives are better when delay intervals of 9 to 17 milliseconds between adjacent holes in a row are used.

Additional research is needed for a better understanding of the causes of desensitization due to precompression of slurry explosives in South Florida.

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