



BLAST DESIGNS TO IMPROVE
DRAGLINE STRIPPING RATES

Final Report

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ABSTRACT

Field tests of blasting techniques to improve dragline productivity were undertaken at a midwestern strip coal mine. The program intent was to demonstrate the potential for increased dragline productivity using explosive casting.

Explosive casting is the displacement of overburden directly into a spoiled condition. This was accomplished during the field tests by the blasting of a shale-sandstone-limestone overburden into or across a mined-out cut. A total of approximately 16 percent of the overburden associated with any given unit of coal was explosively cast into its final spoils position during the field demonstration.

A 16 percent reduction in dragline materials handling was not the only cycle economy realized during the test. The blasting technique moved a large portion of the overburden towards, but not into, a final spoiled condition. This bulk movement shifted the center of mass of the dragline handleable materials towards the spoil area. This center of mass shift resulted in a reduction in dragline swing angle. The more acute angle led to approximately a 2 percent reduction in cycle time.

The above noted field observed efficiencies would be detrimentally affected by either poor overburden fragmentation, which could lead to decreased muckpile diggability and/or bucket filling, or an increased swell factor, which could lead to an increase in the volumetric requirement of the dragline for the the muck not cast into its final spoils location. Diggability, bucket fill factors and muckpile swell were monitored and adverse fragmentation or swell effects were not noted. Except for occasional large muck occurrences, usually traceable to loading procedure errors, diggability was assessed to improve slightly.

The test program realized stripping efficiencies greater than the previously noted 16 percent (cast) and 2 percent (dragline cycle improvement). An additional 7 percent productivity was attributable to changes introduced into the two-seam mining method employed at the test mine. Historically, the dragline worked 20-25 percent of its time in a "chop down" mode as opposed to more efficient below grade underhand excavation. The test program explosive casting of muck was characterized by a compensating vertical "highwall drop." The highwall drop void was filled so that a level working surface for the dragline was established (without tail room obstruction). The filling was done by dozers. The fill material used was the No. 6 Coal (upper seam) overburden normally taken in the dragline "chop down" activity. Elimination of the dragline "chop down" activity and replacement of this activity with the more efficient underhand excavation mode led to another 7 percent increase in dragline productivity.

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In summary, 25 percent more overburden was moved onto the spoils pile per operating hour due to the combined effects of the blast program implementation.

The increased explosives charge required for the casting was observed to displace coal. This destruction and/or spoiling of coal was reduced during the program by the addition of stemming at the bottom of the drill hole. Further experimentation is required to finalize the stemming requirement for coal-seam protection.

The maintenance effect on the dragline tub due to an uneven settling of the "highwall drop" fill was not, within the short duration of the test program, measurable. The fill settlement was substantially reduced by the dragline placement of shale (a 5- to 10-foot layer) onto the fill. Dragline operation off the horizontal as a result of uneven fill settlement is potentially a serious maintenance concern.

Discounted cash flow (DCF) analyses were undertaken for both the historical mining operation and the explosive casting mining method. The DCF rate of return for the property was increased by over 50 percent with the casting method. This assumed a full exploitation of the exposed coal; that is, a 25 percent increase in production.

A discounted cash flow rate of return analysis for the casting method at the historical production rate was also developed. This study incorporated all casting and cycle improvements observed during the test program. The study assumed the replacement of the existing BE2570-W stripping dragline with a smaller unit. This DCF rate of return was also found to be over 50 percent higher.

Both DCF analyses assumed the coal spoilage losses and tub maintenance costs to be identical to the historical.

The DCF analyses were keyed to observed test results and reflected dragline stripping productivity gains, averaged casting economies, and an increased utilization of drilling, coal loading and haulage, coal washing, and reclamation capacities of on-site equipment. A dozer was the only item of new equipment required to achieve this production increase.

The operating cost baseline was developed with new equipment pricing and 16 percent cost of capital, manufacturer's recommended capacities and consumables costs, and, historic equipment availabilities.

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FINAL REPORT

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EXECUTIVE SUMMARY

Previous Bureau of Mines studies have identified overburden stripping as a principle constraint to increased coal production at many existing operations. Increased stripping equipment productivity could therefore be expected to lead to corresponding coal production increases from given machinery combinations.

One approach which might increase stripping equipment productivity is to explosively condition overburden for easier diggability and better bucket filling. This hypothesis postulated that adequate overburden fragmentation, fracture opening, and bedding plane disruption are critical factors in bucket penetration and filling.

Blast designs were developed as a means of opening blast induced or stratigraphic fractures to promote displacement of the overburden. The lateral displacement component was later developed into a "casting" concept which, when extrapolated, indicated the potential for cost effective relocation of a portion of the overburden to a spoils area by explosives alone. The field test portion of the project dealt with experimental overburden casting, or lateral displacement, utilizing explosives.

The project was executed in two phases: (I) a review of existing literature and investigation of operations suitable for a field testing of the identified principles, and (II) field tests and subsequent reporting of results.

Phase I of the project identified a number of operating strip mines, most with dragline stripping equipment, where explosively enhanced diggability, bucket fill, and casting, could be implemented and analyzed. These mines were also characterized by coal production levels constrained by stripping equipment productivity.

Prior to any field effort, legal agreements between the Phase II contractor, the chosen mine(s), and the Department of Energy (who had taken over the project from the Bureau of Mines) were negotiated. The negotiations proved to be difficult. The mine operators, with coal delivery commitments in hand, were not inclined to allow experimentation which, in a worst case circumstance, threatened production continuity. Although interest in the concepts was high, particularly among the engineering staffs at the respective properties, management and corporate legal reviewers generally advised against project implementation because of production-risk concerns.

Agreement was finally reached with the Old Ben Coal Company. The Old Ben property chosen as a test site was the West Field at the No. 1 mine in

Oakland City, Indiana. The field is characterized by two coal seams, the near-surface No. 6 Seam and the deeper lying No. 5 Seam. Both seams are nearly horizontal with minor undulations. The No. 5 Seam is the principle production seam.

The project program was to evaluate the historical mining operation, to test and evaluate the "casting" technique, and then to compare the dollar cost and production levels of the two operations. Individual change elements, such as powder factors, drill hole spacings, stripping dragline productivity, and production cycle effects, were to be monitored and ranked, where possible, for cost effectiveness. The final reporting of test program results includes a methodology for the recognition of dragline underperformance due to blasting related factors, as well as an in-depth reporting on the results of field test experience.

The test program area had both the No. 6 and No. 5 Coal seam present. The No. 6 Seam overburden ranged between 3 and 25 feet in depth. The No. 6 - No. 5 Coal interburden ranged from 95-115 feet in thickness. The blast program was undertaken on the interburden, with the No. 6 Coal overburden being used as fill to re-establish the dragline working bench.

The existing mining practice employed the dragline in a "chop down" mode for the excavation of No. 6 Coal overburden (the dragline operated on a working bench whose elevation was set by the bottom of the No. 6 Coal seam). From the working bench, the dragline would turn 90° from the long axis of the strip to chop above grade materials. When the No. 6 Seam overburden had been cleaned, the dragline would proceed with below grade activities (working bench excavation) parallel to the long axis of the cut. The No. 6 Coal was extracted after the chop-down overburden removal. Drilling of the interburden bench could then be undertaken. The blasting took place back from the highwall, hence, a strip of previously shot muck existed between the shot and the pit. This is known as "buffer" blasting. This technique, while minimally disruptive of ongoing pit operations such as coal loading and hauling, is confining. Historical records indicate buffer blast fragmentation was good. Bedding plane disruption appeared inconsistent. The dragline excavation of this "buffer" blasted muck was observed to be characterized by bucket penetration resistance along geologic bedding planes. This is known as hard face. This geologic and blast design related bucket penetration problem was one of the primary stripping inefficiency targets to be evaluated during the test program.

The test program emphasis was on the blasthole patterning, loading, and shot sequencing. Historical procedure had been a 36-ft x 48-ft hole pattern varying in depth from 95 to 115 feet with 30 to 35 feet deep satellite holes between rows. The satellites were designed to break a limestone caprock which was 15-30 feet below the working bench level. Historical explosives loading had been 0.33 lbs ANFO/yd³. The recommended pattern for the test program initiation was 40 ft x 40 ft without

satellites. Hole loadings of .70 lbs AN/FO/yd³ were recommended as a starting point. With the doubled powder factor, air blast, flyrock and ground vibration were additional concerns going into the program.

A two-thousand-foot section of the mine was cleared of buffer, exposing unbroken interburden to the mined-out pit. The mined-out pit bottom width was fairly uniform at 100 feet. The highwall angle was about 65°. Historical observations had indicated 70° was common. The shallower highwall angle led to toe distances in excess of the predicted "throwing" power of regular AN/FO. An aluminized AN/FO toe load with an increased energy density was considered. It was decided to go with the standard AN/FO in use, observe the results, and, as required, switch to a more energetic mixture.

Individual blasts were 200-300 feet in length. In the first pass through the test area, seven individual test blasts were made. A second pass totalling eight (8) individual shots followed.

The blasthole configurations for the field test work varied between 27 ft x 50 ft and 36 ft x 36 ft. Hole diameter was 15 inches. AN/FO loading varied between 1.0 lb AN/FO/yd³ and 0.60 lbs AN/FO/yd³. A stronger aluminized AN/FO was not required during the test program. Bottom stemming in the holes varied between 2 and 5 feet. This consisted of drill cuttings pushed back into the hole to buffer the coal seam from blast effects. The thicker bottom stemming was utilized in the second pass of the test blast program, and, in view of continued coal destruction ("chilling") and subsequent spoiling, a third pass through the area might have increased bottom stemming to 7 feet. Collars, i.e., the distance from the working bench surface to the top of the AN/FO load, varied between 30 and 26 feet. The 26 foot collar (top stemming) was found to be effective in consistently shattering the limestone caprock.

Fragmentation of the overburden was observed to be good. Although some large limestone caprock muck was occasionally observed, the large muck was usually traceable to hole collaring greater than 30 feet. Overburden was cast into the pit with the muck pile toe running up onto the opposite spoils pile between 10 and 30 feet. No flyrock was observed. Air blast and seismic events were within Office of Surface Mining guidelines.

The dragline generally dug without difficulty in the cast muck but dragline performance during the test program would, in our judgment, have been better had large muck not periodically impeded progress. Bedding plane hard face was visually assessed to be nil in the test program muck.

The coal exposure rate per operating hour of the dragline was measured as 25 percent greater than the historical rate. The achieved economies were evaluated to be 16 percent due to casting, 2 percent due to improved cycle time, and 7 percent due to the elimination of "chop down" activities on the No. 6 Coal overburden. This latter economy was specific

to the test program area and is not expected to be so pronounced in either the remainder of the West Field or at single seam mining operations employing the casting technique.

In assessing the program's economics, two substantial costs were noted but were considered incalculable from the data generated by the testing. These costs were: (1) the cost of deleterious blast effects on the coal which cause losses to the spoil, and (2) the unsuitability of the highwall fill, especially in the wet season, to support the dragline.

Discounted cash flow (DCF) rate of return analyses were developed for both the historical and test program practice. Given assumptions of equal coal recovery and equal tub maintenance costs, the test program approach showed an improved capital rate of return. This DCF assumed a 25 percent coal production increase with the existing drilling, reclamation, coal loading and hauling, and processing equipment. On site machine and wash plant ratings indicate such a production increase can be supported. A second DCF for the casting program was run. In this case the historical mining rate was achieved by the use of a smaller stripping dragline. This DCF also showed an improved discounted cash flow rate of return.

The DCF's were not intended to reflect a mining program in the Old Ben No. 1 Mine West Field, rather they were comparisons of the historical practice versus the observed test bench operation.

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1.0 INTRODUCTION

1.1 Background

Demands on energy sources other than oil and gas have increased greatly in recent years. In the United States much of this increased demand has focused on coal.

A study of western U.S. coal mines by the Bureau of Mines indicates that 75 percent of the operations were inhibited from increasing coal production by their stripping capacity, whereas coal loading and hauling capacity exceeds demands. This indicates that coal production might be increased by improving stripping efficiencies.

Draglines were a principal stripping tool for the mines included in the Bureau of Mines study. Thus, if dragline excavation efficiency could be improved, coal production would be increased.

The Bureau of Mines awarded this contract (later transferred to the Department of Energy) to develop and demonstrate blast designs with the potential for increasing dragline productivity.

The program was to be accomplished in two phases:

- Phase I Literature review, field studies, improved blast design development, and specific field evaluation program design.
- Phase II Demonstrate potentially more effective blast procedures designed during Phase I at three surface coal mines; and report the findings, recommendations, and evaluations of the results in terms of potential impacts on surface mining systems, costs and productivity.

Phase I was completed in April 1979. Phase II will be completed upon the final drafting of this report, the draft final being submitted in October, 1982.

1.2 Phase II Objectives

The primary objective of Phase II was to design and test blast designs that might directly aid in increasing dragline stripping rates and thus coal production rates. The principal investigation method was field testing. The program was constrained by the requirement to utilize excavation equipment currently used at the test sites.

Phase II was designed to accomplish the following:

- (1) Select appropriate mine sites for the field tests.

- (2) Establish working agreements with those mine owners.
- (3) Monitor current blasting systems.
- (4) Monitor a series of test program blasts and record and evaluate the following:
 - (a) Cost and productivity by elements for the introduced changes.
 - (b) Effect on fragmentation and overburden stripping rates achieved.
 - (c) Effects of blasts on the underlying coal.
 - (d) Blast design effects on cost distribution between drilling, blasting, dozing and dragline operations.
 - (e) Maintenance impacts.
- (5) Analyze the results of the blasting test program, in the light of the entire mining cycle, and refine the program to maximize its cost effectiveness.

1.3 Negotiation Results

Blast programs were designed for three selected mine sites, but ultimately field tests were conducted at only one site. The three mines selected were the following:

- (1) Arch Minerals Company's Seminoe Mine near Hanna, Wyoming.
- (2) NERCO's Dave Johnson Mine near Glenrock, Wyoming.
- (3) Old Ben Coal Company's Old Ben No. 1 Mine near Oakland City, Indiana.

After several discussions with Arch Minerals and visits to the mine site, we were advised that they had decided not to participate in the blast design program.

Negotiations with NERCO personnel and visits to their Dave Johnson Mine near Glenrock, Wyoming, were prolonged, but an agreement could not be reached. Consequently, the Dave Johnson Mine was not used as a test site.

Efforts to negotiate working agreements with operating strip mines were time consuming. Excessive costs involved in negotiating agreements precluded a final selection of more than one mine site. The strip mine finally selected as a test site was the Old Ben Coal Company's No. 1 Mine in Oakland City, Indiana

1.4 Phase II: Old Ben Field Program Pictorial Representation

The following pictures and text summarize the field test effort.

The test program was a series of fifteen blasts. The blasts were designed to cast overburden into the mined-out strip. Each blast broke approximately 100,000 cubic yards.

The test bench area had a near-surface (No. 6 Coal) and deeper lying (No. 5 Coal) coal reserve. The blasting effort was limited to the interburden between seams.

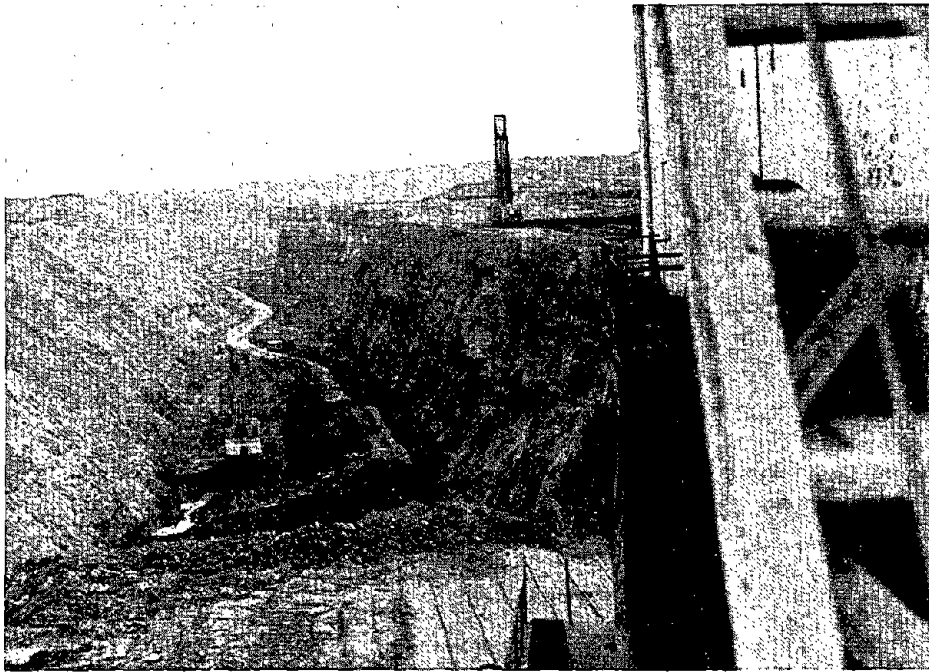


FIGURE 1 TEST BENCH OVERVIEW.

The test strip area as viewed from the principle stripping (BE2570) dragline. The BE61R drill rig is in the background; coal loading is in progress in the pit. The highwall angle in this area is 70°. The dragline working bench (tub level) was set by the bottom of the No. 6 Coal seam. Depth of interburden throughout the project test area was about 100 feet.

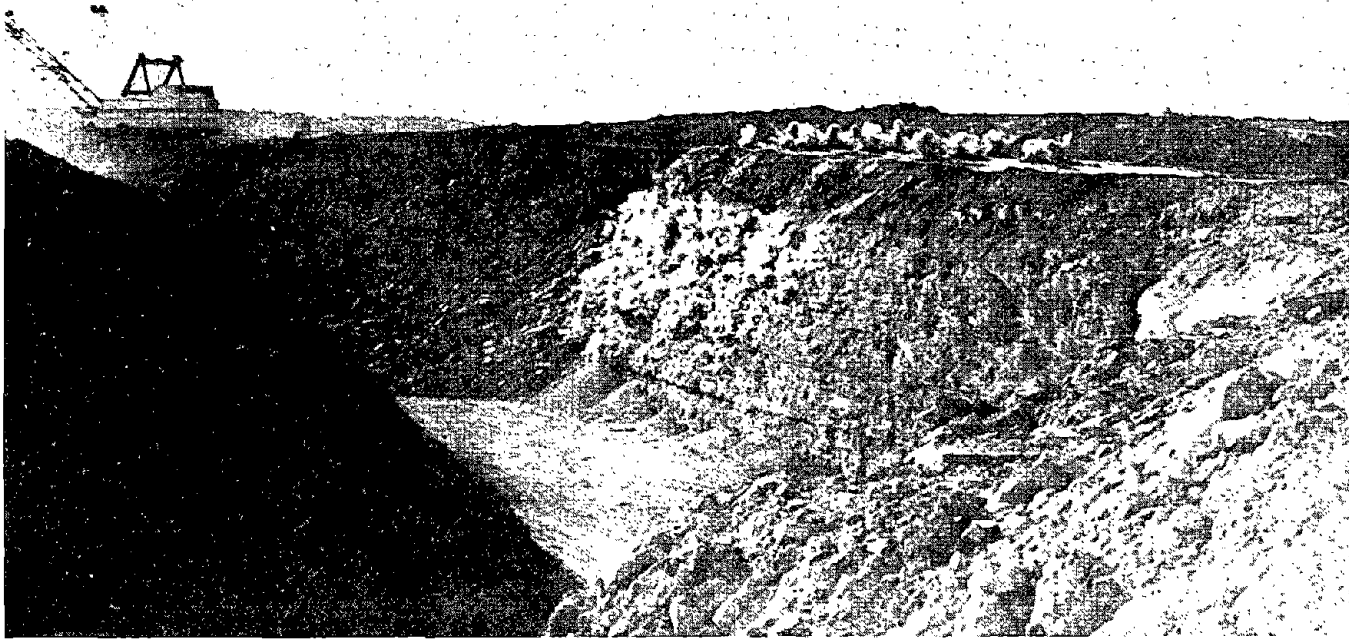


FIGURE 2 BLAST INITIATION

Barely discernible are targets strung down the highwall face on a 20-foot center. These targets were used in high speed filming analyses of each blast's face velocities. Blow-outs and flyrock were infrequent during the test program.

The blasts used double the power factor historically employed at the mine, and, through delay patterning, were designed to displace materials directly into the mined-out pit.

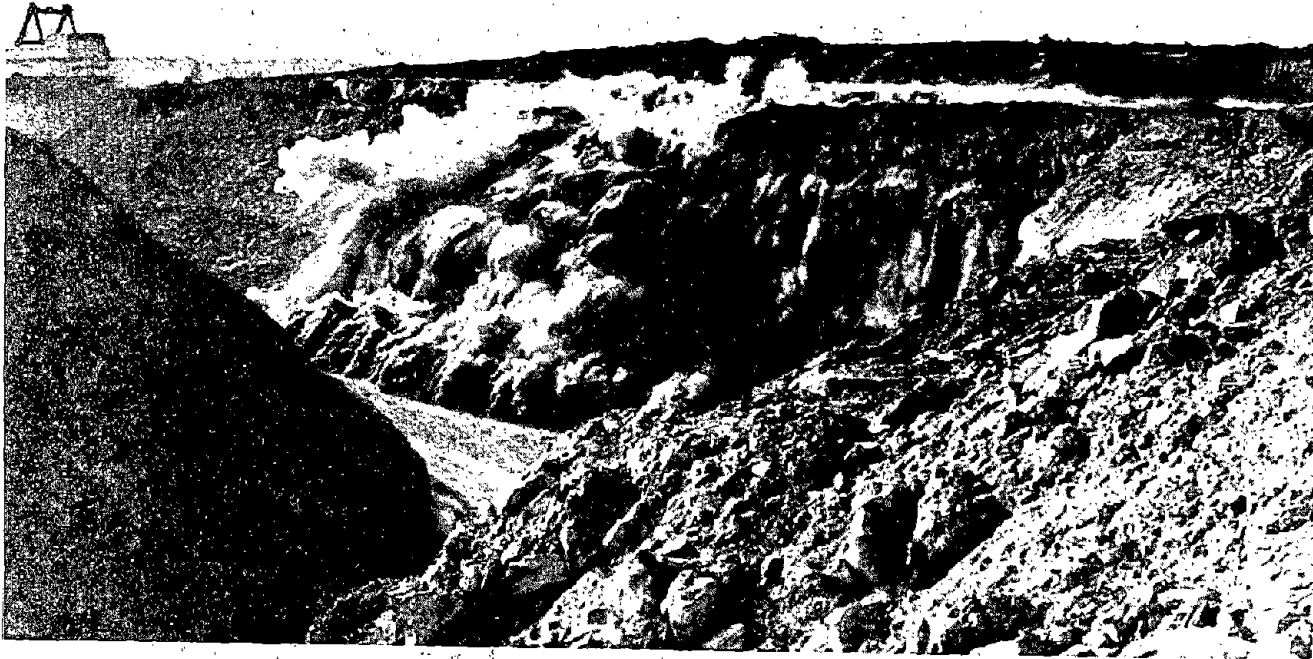


FIGURE 3 BLAST IN PROGRESS

The toe is kicking out into pit, and the highwall has dropped. Seismograph readings were taken at the dragline and at the edge of the property. No unacceptable ground motion or air blast values were recorded.



FIGURE 4 COMPLETED BLAST

Highwall drop and cast muck are apparent. Large fragments on the muckpile were generally cast into spoiled condition (as shown). Occasional large muck which did not reach a blast induced final location was handled without difficulty by the dragline.

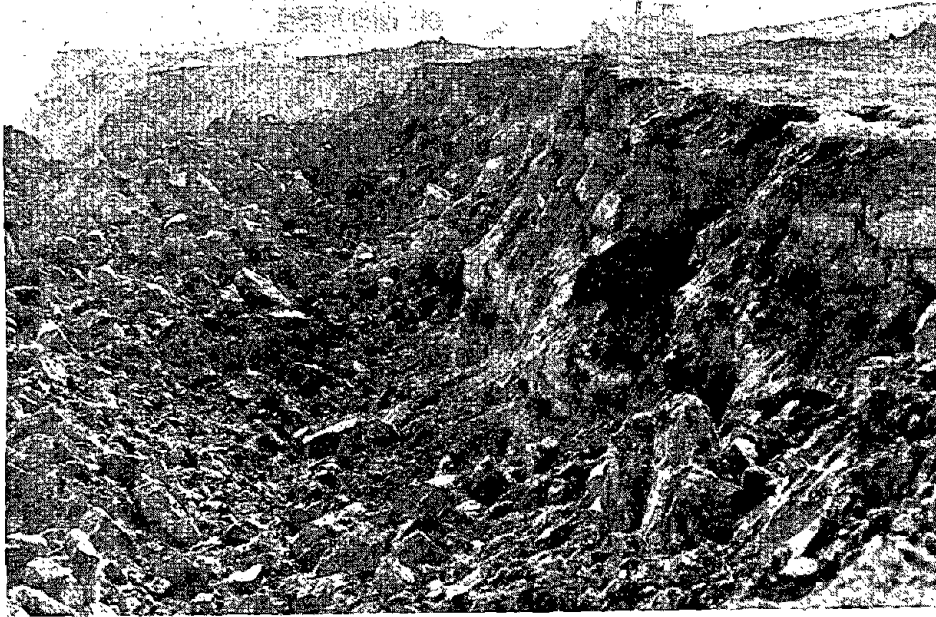


FIGURE 5 CLOSE-UP OF POSTBLAST HIGHWALL

Deepest part of the highwall drop, roughly coincident with the back row of holes, was 30 feet. This trough, on the pit side, rose to within 10 feet of the original bench level. Fragmentation was good (as shown).



FIGURE 6 WORKING BENCH RECONSTRUCTION

Working bench reconstruction was accomplished by the dozing of No. 6 Coal overburden onto the dropped highwall. This claylike fill deformed under the dragline during stripping operations. A 5 to 10-foot shale capping, dragline placed, was later found to alleviate most of the fill deformation problem.



FIGURE 7 WORKING BENCH RECONSTRUCTION

The clayfill requirement was often greater than the No. 6 overburden available within the preblast design strip of 125-foot width. These voids were filled with dragline rehandle.

2.0 HOW CAN MINES RECOGNIZE UNDERPERFORMANCE DUE TO POOR BLASTING?

A primary consideration is the actual dragline performance that is being achieved. Figure 8 is a plot of bank cubic yards per operating hour versus dragline bucket capacity. With increasing dragline size there is a corresponding increase in machine output.

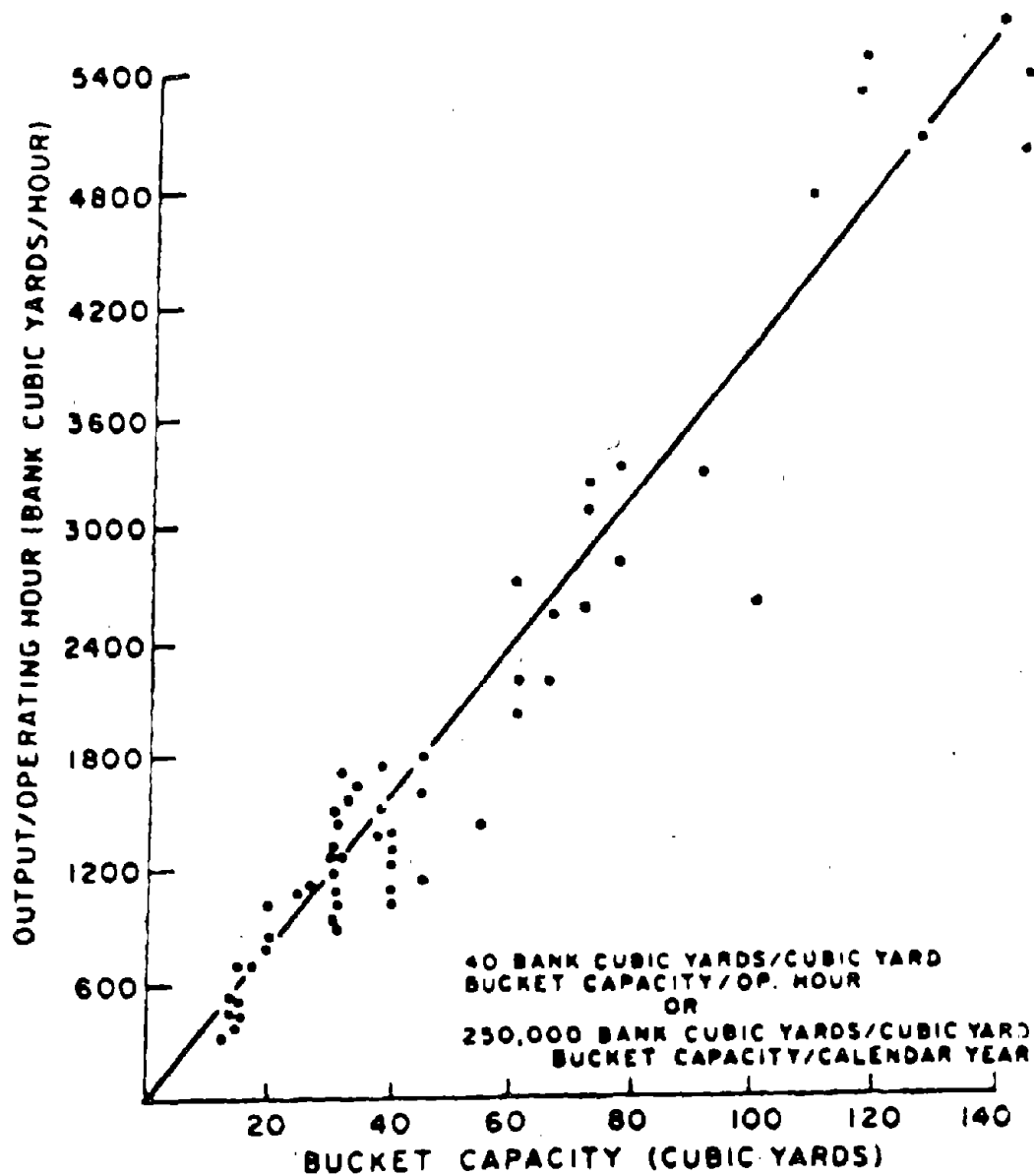
Typical machine outputs are reported as 250,000 bank cubic yards per cubic yard of bucket capacity per calendar year and 40 bank cubic yards per cubic yard of bucket capacity per operating hour. This method of expressing machine output helps enable comparison between draglines of different sizes.

A first check on machine output can be made against Figure 8. If a particular machine were performing drastically below the indicated levels it may be because of poor blasting performance. It must be stressed that Figure 8 does not consider the dragline operating method or the material swell factor characteristics. While Figure 8 is an interesting comparison of performance between machines, it does not definitively identify dragline underperformance. It may indicate situations which merit closer scrutiny.

One of the best methods of identifying dragline underperformance due to poor blasting is observation of the bucket loading operation. One major blasting influence on the dragline is the speed and efficiency of bucket filling. The bucket should be fully loaded within 2 bucket lengths. The bucket should not be required to break rock and should not bounce off "hard face" in the blast. Upon filling, the bucket should be fully loaded with a high bucket fill factor. The overall bucket loading time for normal passes should be on the order of 8 seconds. Following the loading operation, and as the bucket is swung to the dumping site, there should not be excessive fines production which can create poor operator visibility.

The next aspect to consider is the blast itself. Large oversize material should not be present. This means that none of the material that must be dug by the dragline should be too large to easily enter the bucket. It should be noted that large material can be produced by the blast without influencing dragline productivity if it is suitably placed by the blast in the mined-out area of the previous cut, thus requiring no further movement.

"Hard face" should not be present within the blast and undiggable areas should not occur. Even though the fragmentation may be good, the inherent fractures in the rock as well as those that have been produced by the blast should be opened up such that the dragline bucket can easily penetrate the bank.



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BANK YARDAGE MOVED PER
OPERATING HOUR FOR DRAGLINES
OF VARIOUS CAPACITY

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A final assessment can be made by considering the overall blast profile. As with the dragline performance, the dragline operating technique must also be considered in this assessment. If buffer blasting is in use, the blast profile is not significant. Massive surface disruption can lead to an excessive dozer leveling effort. If a conventional highwall dragline operating method is in use, the blast profile should be such that blast surface fill should be minimal. If an intermediate bench level is used with a blasted dragline tail room, the blast profile should approximate the level of the dragline tail room blast. This aspect will be covered in a later section.

Finally, if a pullback technique is in use with the dragline always operating from the spoil side, then blast throw should be maximized in order to reduce the blasted material to be moved to as small a quantity as possible. Again this topic will be described for both a single-seam and a multiple-seam operation in more detail in a later section.

2.1 Summary

To summarize, the factors that should be considered when trying to recognize underperformance of a dragline due to poor blasting are as follows:

- (1) Compare the dragline output on a cubic yards per cubic yard of bucket capacity basis with other machines. Make special note of the dragline operating method and material swell factor. The average productivity for draglines in the U.S. coal industry is 40 cubic yards per cubic yard of bucket capacity per operating hour.
- (2) Study the bucket loading operation. This should take place within 2 bucket lengths and in a time of approximately 8 seconds. High bucket fill factors should be maintained.
- (3) The blast fragmentation should be of a suitable size relative to the dragline bucket and should be opened up to allow easy bucket penetration.
- (4) The blast profile should conform to the selected dragline operating method. Throw should be maximized if the pullback technique is being used.

3.0 HOW CAN MINES IDENTIFY SOURCES OF POOR BLASTING PERFORMANCE?

In the original USBM report, "Blast Designs to Improve Dragline Stripping Rates," Phase I, April 1979, blast evaluation was covered in detail. Major factors which affect blasting results were discussed. These included the material type, burden to spacing ratio, direction of blasting relative to rock structure, explosive charge strength, delay

times, collar height, toe charges, primer types and sizes, and primer location. This section will present additional material that will help mines identify sources of poor blasting performance. Of prime importance is good blast record keeping.

3.1 Drill and Blast Record Keeping

The assessment of drilling and blasting results requires first, a good blast design containing the relevant parameters to be used, second, suitable documenting of the blast in the field, and finally, correlation of these first two records with the blast results.

The blast design parameters should be established for all areas of the mine. Typical design parameters that would be layed out would be:

- (1) Material type to be blasted.
- (2) Hole depth to be loaded relative to the top elevation of the coal seam. Also, the maximum loss of depth that will be tolerated before the hole must be redrilled.
- (3) Types of explosive to be used.
- (4) Priming practice to be followed.
- (5) Whether or not deck loading is to be used and, if so, the size and location of each deck.
- (6) The collar length to be used.
- (7) The drill hole size, spacing and burden.
- (8) The blast tie-up and delay pattern and location.
- (9) The highwall crest and toe location.
- (10) Charge weight of explosive per delay

Many open pit mines will lay out the blast on a 40-scale plan prior to its being staked in the field. Due to the uniformity of strip mines, this procedure is not usually warranted other than for specific problem areas. In general, dragline blasts can be satisfactorily layed out in the field by the survey department without the use of a blast design plan. The patterns should be accurately taped, rather than paced.

Information pertaining to the blast design can form the basis of a file for each blast fired in the field. Added to this file would be the actual field blast data and various assessment factors.

Accuracy in this type of reporting is important. Mistakes will occur and these should be faithfully recorded. In this way, if blasting problems do occur they can be explained by, say, poor loading, without management thinking a blasting practice change is necessary. In this respect, other benefits may result from the accurate records. If a mistake was made in loading and no problems resulted, an improvement in economy may be indicated with a suitable blasting practice change.

The field data to be recorded would be as follows:

- (1) A full survey of the hole locations and highwall crest and toe prior to the blast. It is not necessary to survey each hole if the drill pattern is regular, but holes off pattern by 2 feet or more should be surveyed.
- (2) Full records of hole depth (actual and loaded design), explosive loads and type, priming, collar height and decks if used.
- (3) Records of the bulk explosives quality control tests performed on the truck loads of explosives used in the blast.
- (4) Record of actual tie-in of the blast as fired showing the trunklines and relay periods and locations.
- (5) Record of redrills performed, if any.

All of these data would be recorded for each blast and added to the blast file. If blasting difficulties are occurring, it is also a good idea to take before and after Polaroid photographs of each shot as a further record.

Some mines prefer to record all explosive loads in tabular form. Table 1 illustrates an example of this type of form. One drawback is that when the data are used, they must be referred back to a master plan to obtain the relative hole locations. Tables of this form do allow convenient overall figures for the blast to be easily obtained. In particular, blast record tables allow the actual explosives load per foot to be conveniently calculated. This figure is important as it may indicate areas in the blast where excessive explosives loss predominates, thus indicating likely areas for future hole lining. It may highlight bad truck loads of explosive which contain low guar, resulting in a thin watery mix with little or no water protection, if slurries are in use. Accordingly, some mines use both a tabular form and plan form.

TABLE 1

TYPICAL EXPLOSIVES LOADING RECORD SHEET LAYOUT

Hole Number	Design Hole Depth (ft)	Actual Hole Depth (ft)	Design Explosives Load (lbs)	Design Explosives Type	Actual Explosives Load (lbs)	Actual Explosives Type	Design Collar (ft)	Actual Collar (ft)	Actual Explosives Load Per Foot of Hole (lbs)
1	61.5	61.5	2,150	Slurry F170	2,250	Slurry F170	16	16	49.5
2	62.5	62.0	2,250	Slurry F170	2,300	Slurry F170	16	16.5	50.5
3	62.0	61.5	2,200	Slurry F170	2,150	Slurry F170	16	16	47.3
-	-	-	-	-	-	-	-	-	-
-	-	-	-	-	-	-	-	-	-
-	-	-	-	-	-	-	-	-	-
-	-	-	-	-	-	-	-	-	-
25	64.5	64.0	2,500	Slurry F170	2,500	Slurry F170	16	15	51.0
Totals	1,630	1,598	58,750		60,500		400	385	49.9

The plan form is illustrated in Figure 9. This plan records the hole locations, the highwall crest and toe, the location of the last row of holes from the previous blast, any significant geological data, and the hole number and loading data. The design and actual hole depth, explosive load, and collar are written adjacent to each hole. If blasting problems are ongoing, it is a good idea to draw cross sections through the blast at representative locations. This should not be necessary on a continuing basis. Figure 9 locates two cross sections which appear in Figure 10. The cross section's primary use is to visually illustrate the critical front row toe area of the blast.

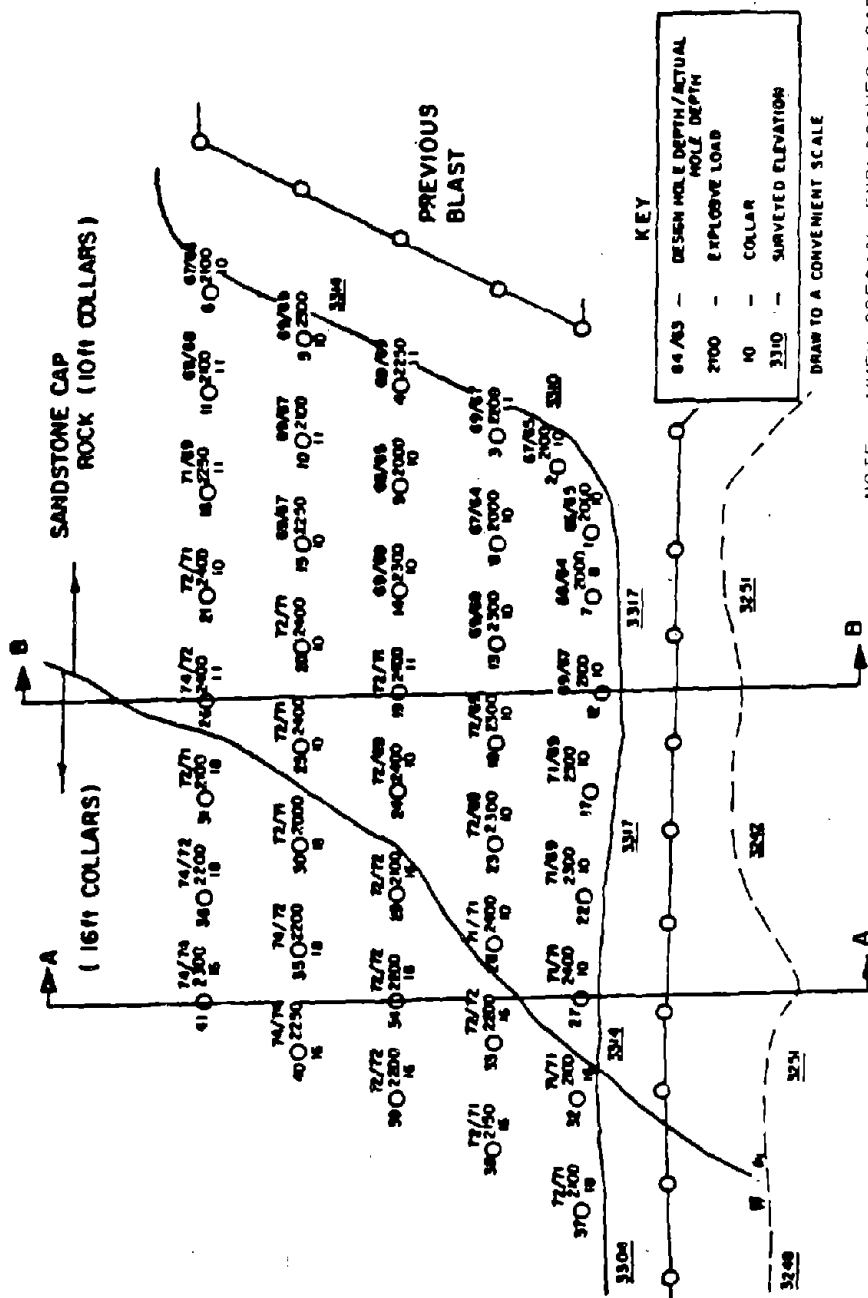
A copy of the blast record can be used to record the blast tie-in, as shown in Figure 11. This second plan would also be used to record after blast events such as the location of oversize material, hard toe and any misfires that are subsequently found.

The blast assessment record should include the following:

- (1) An estimate of the fragmentation achieved with location and percentage of oversize material.
- (2) Location and extent of hard toe and undiggable material (with photographs).
- (3) Record of muckpile profile with back throw, if any.
- (4) Estimate of flyrock produced.
- (5) Record of any blasting fumes produced.
- (6) Record of vibration and air blast, if a critical factor.
- (7) Production rates for equipment digging the blast.
- (8) Record of any secondary blasting required either for hard toe or oversize material.

As indicated in Figure 11, any misfires, hard toe and oversize should be plotted to highlight correlations that can be made between their occurrence and the blasting practice, geology, etc.

After-blast muck profiles superimposed on a Figure 10 type section are often useful to the long-term dragline technique and operating level planning.



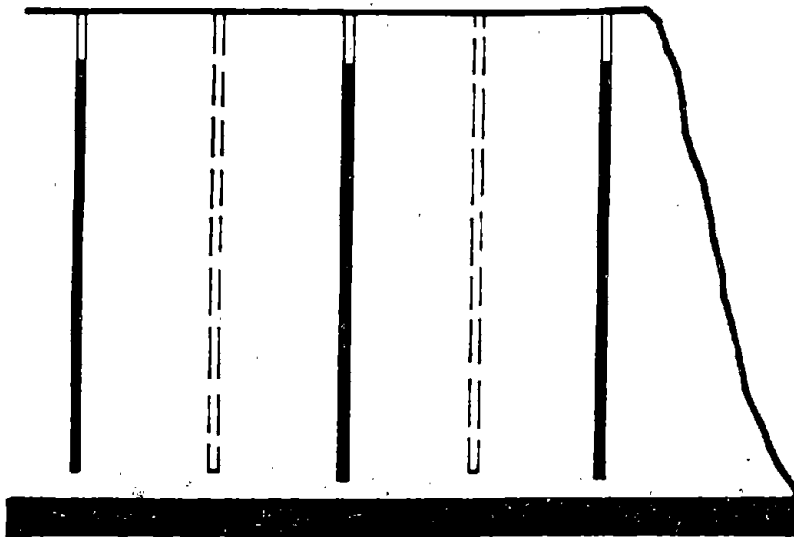
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RECOMMENDED BLAST PLAN
RECORD INCORPORATING HOLE
LOADING DATA AND CROSS
SECTION LOCATIONS IF ANY

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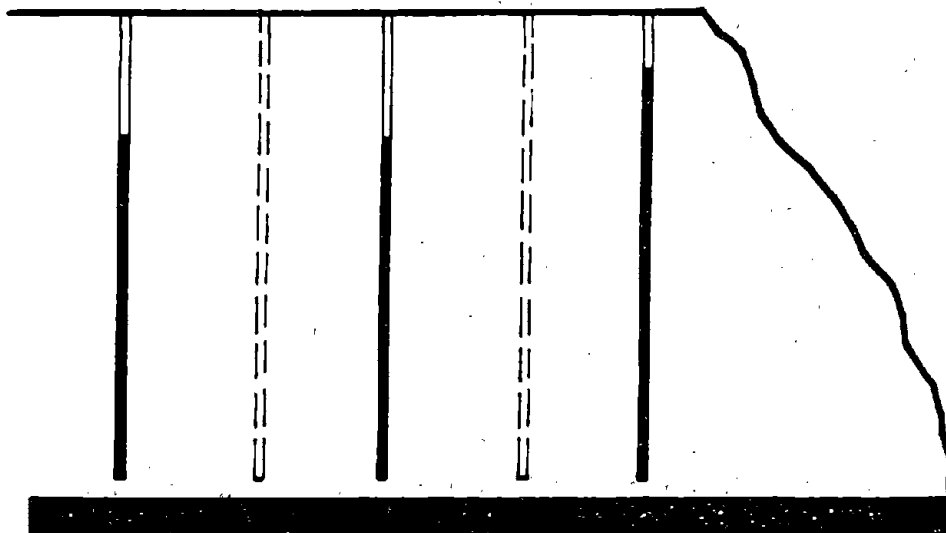
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GEOLOGY CAN ALSO
BE SUPERIMPOSED

SECTION B - B

(NOT TO SCALE)



SECTION A - A

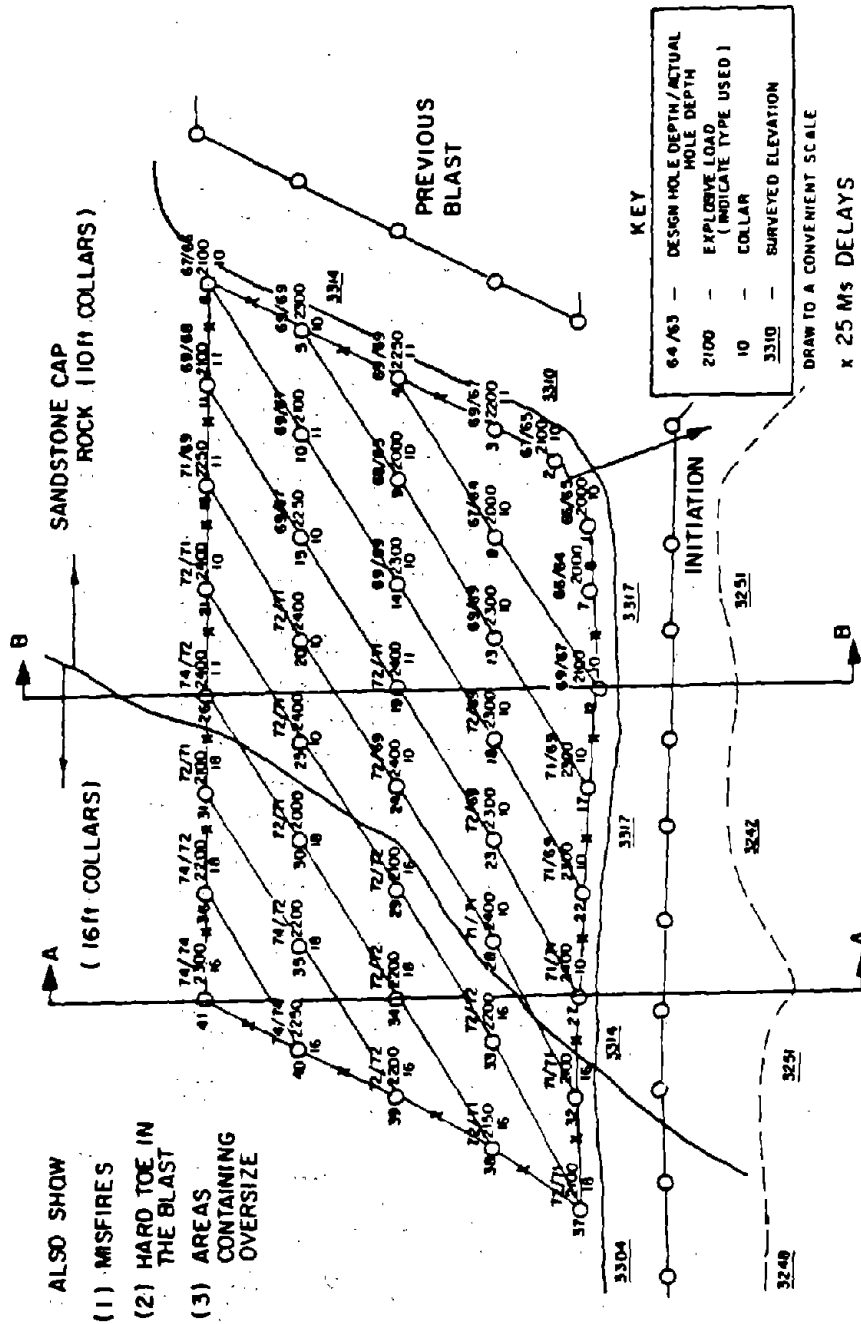
TYPICAL BLAST CROSS SECTIONS
TAKEN FROM THE BLAST PLAN
IN FIGURE 9

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RECOMMENDED USE OF THE BLAST PLAN TO RECORD THE TIE-IN AND SIGNIFICANT PROBLEMS EXPERIENCED WHILE DIGGING

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This section has outlined a set of drilling and blasting records that would allow good blast evaluation. As such, it represents an effort beyond that which is presently being done at most mines. However, if blasting difficulties are being experienced, this approach is worthwhile. As blasting improves, certain aspects of the record keeping can be reduced or eliminated.

When good fragmentation is consistently being produced, the next thrust should be to maintain that fragmentation but at a reduced cost. Good records are an important aid as to what changes have the best chance of being successful.

Supply companies are constantly improving their products, or claiming to have improved them. A good drill and blast record base is ideal for the evaluation of new products or blasting techniques.

A good drilling and blasting record guards against a problem that can occur at many mines; i.e., the loss of key personnel. A great many different blasting techniques are often tried in various parts of a mine in different geological conditions. The knowledge of what has been tried, with results obtained, could be lost to the company if key personnel leave, with the possible result of costly duplication of previous work or even retrograde steps in blasting practice.

3.2 The Use of High Speed Photography

High speed photography allows a series of still photographs to be taken in rapid succession during a short duration event such as a blast. By viewing these photographs at a slower rate, the event can be viewed in what is commonly referred to as "slow" motion.

The eye cannot resolve motion which occurs in less than 1/4 of a second. High-speed filming of a blast can allow motion to be slowed so it can be analyzed both qualitatively and quantitatively. Analytical procedures can be performed on the two-dimensional film record of the event which allow three dimensional data to be obtained.

For the purpose of filming blasts, the cameras used usually operate from 100 to 10,000 F.P.S. Speeds of 300 to 500 F.P.S. provide enough information for a reasonably accurate analysis. Speeds in excess of 1,000 F.P.S. are used for more accurate work.

High-speed motion pictures have two main uses in helping to identify poor blasting results. The principal use is the direct photography of the blast, this representing the bulk of the work. High-speed motion pictures can also be used to check out individual blast components, particularly actual delay element times for such accessories as delay connectors, down-the-hole delays and other delay systems.

The principle use of high-speed photography for blast evaluation are:

1. The firing sequence of the blast.
2. The actual firing times for the delay elements.
3. The velocity and direction of flyrock travel from the face and the top of the bench as well as the shape of the face movement.
4. Selection of the delay times for optimum rock movement and fragmentation.

3.3 Sources of Poor Blasting Performance

Using good drilling and blasting records and the technique of high-speed photography, the sources of poor blasting performance can be identified. The factors that should be checked include:

1. Overall blast energy input.
2. Distribution of energy input,
 - pattern size and shape
 - toe burdens
 - collar heights
3. Priming practice and initiation system in use.
4. Quality control of explosives.
5. Blast tie-in relative to rock structure.
6. Delay times in use.
7. Burden to spacing ratio.
8. Accuracy of delays.
9. Assessment of blast fragmentation and blast profile.

4.0 HOW CAN MINES DEVELOP THE NECESSARY CHANGES IN BLASTING PROCEDURE AND IMPROVE DRAGLINE PERFORMANCE?

There are two main approaches to improving dragline performance by developing blast design changes. The first assumes some form of conventional dragline operating technique and the improvement of dragline performance by improved blast fragmentation. The second approach is to modify the dragline technique to allow blast throw to reduce dragline handling requirements. This second approach has the potential of greater performance improvement because throwing the material also ensures that blast fracturing will be opened up. Dragline operating rate improvements may result.

4.1 Development of Blast Designs to Improve Fragmentation Alone

For improved fragmentation the two main factors to be assessed are the energy input into the blast and energy distribution. As many of these blasts are of the buffer type, the blast timing is usually of a lesser significance.

The first concern is the energy distribution. Collar heights should be checked using regular design procedures, especially if surface fragmentation in the blast is poor. If the collar heights are correct and fragmentation is still poor, then the design of stab holes or satellite holes or decking should be considered. This approach may improve breakage.

The blast toe should also be checked. Hole depth control is quite commonly a problem that results in caprock being left on top of the coal. Alternatively, if holes are too deep, coal damage can result. The correct hole loading standoff must be maintained.

The second concern is the overall energy input into the blast. Low energy input can occur for two reasons. The blast powder factor may be right but the quality control on the explosive may be poor. This can result in unexpectedly poor blasting performance for no apparent reason. Quality control testwork should be an ongoing part of a blasting operation, particularly where bulk AN/FO and/or slurries are used. Driliner checks should also be carried out if used for AN/FO water protection. Primers and detonating cord should also be periodically checked.

If the explosives products are known to be satisfactory, then low energy input must be changed by using more energetic explosives, by reducing the drillhole pattern size, or a combination of both. This problem is commonly first made evident by sections of poor fragmentation located in the body of the blast well above the coal horizon. To overcome random material type changes in the highwall, an explosive energy increase is best tested using a significant change of the order of 15-20 percent.

Provided the blast initiation system has an adequate delay between rows of holes, say 2 milliseconds per foot of burden, then blast timing does not usually contribute significantly to poor fragmentation.

4.2 Development of Blast Design to Both Throw Material and Improve Fragmentation

In this case a choice must be made between working the dragline on the highwall or on the spoil side of the pit.

If the dragline is to be used on the highwall side of the pit, blast throw should be designed so that the desired dragline operating level is achieved with the blast alone. An effort should be made to reduce dragline working bench filling to a minimum. A scaled burden (where scaled burden

is the burden in feet divided by the square root of the charge rate in lb/ft. of hole; see Phase 1 report for additional discussion of scaled burden blast design theory) on the front row holes of 3.5-4.0 ft/lbs^{1/2} will help control the throw. The tie-in angle with the face should be quite large, say of the order of 60°, to help throw the blast as much along the cut as across it. Required powder factors vary with material type but higher values are required, in similar overburdens, when compared to powder factors for blasts designed only for fragmentation. As the material type increases in strength the difference in powder factor between throw blasts and blasts designed only for fragmentation is reduced.

For highwalls containing caprock, an additional dragline tail room blast is required. This will be described in more detail in Section 5.

Throw is further enhanced by increased delay times between hole rows. Very long delays, however, are not necessary for controlled throw blasting.

If the dragline is to work from the spoil side, maximum throw from the blast is desired. The scaled burden on the front row holes should be reduced to a value on the order of 2.0 ft/lbs^{1/2}. The blast tie-in angle should be relatively shallow. A value of 30° in conjunction with an equilateral triangle pattern gives both good direction and burden to spacing ratio. With suitable front row burden, overall powder factors need not be significantly higher than for blasts designed for moderate throw. This type of blast will be described in more detail in Section 5.

The throw blasts described assume that the factors considered for fragmentation blasts have already been optimized.

Finally, the blast timing needs to be assessed. The delay time must be sufficient to allow time for horizontal movement of the burden. This minimizes the vertical movement which tends to reduce the casting effect. This factor can best be evaluated using high-speed photography. If the required delay time is very long then alternative blast initiation systems may be indicated.

5.0 ADVANCES MADE IN OTHER COUNTRIES IN BLAST DESIGN FOR DRAGLINES

As was outlined in Section 4.0, there are two main approaches to dragline operation in conjunction with throw blasting. The technique that maintains the dragline on the highwall requires a limited overburden throw. The alternate approach is to work the dragline from the spoils side. This procedure allows for maximization of the overburden throw. Both of these techniques have been developed overseas and will now be described individually.

5.1 Throw Blasting Using the Dragline on the Highwall Side

A number of coal mines throughout the world are experimenting with, or have already introduced, blasting practices designed to collapse the highwall. Access through the cut is not possible after the blasting operation in this method.

The most significant departure from common practice is the design allowance for good horizontal movement of the overburden toward the open cut. Such throw of the material will result in a portion of the overburden that does not have to be moved by the dragline. This adds to the productivity of the pit even if dragline production rates themselves do not improve. Figures 12 and 13 show the configuration of a blasted bank where good relief has been experienced. Figure 12 shows an initial blast creating a new dragline level. The short hole section of the blast for dragline tail room may or may not be blasted with the main shot. One possibility is firing the short holes separately and directly into the power trough created by the main blast. The power trough being the depression adjacent to the new highwall but lying behind the heaped, displaced, blasted muck. This introduces another operation into the mining sequence. Some mines are fortunate enough to be able to chop the dragline tail room either from completely virgin ground or following light crater blasting.

Due to the displacement, the dragline sits at an elevation somewhat lower than before, and also somewhat closer to the spoil, thus improving reach. Dumping height has to be checked to ensure that the machine can spoil the material adequately from the new location. Range diagrams should be drawn for the expected reach geometry.

Additional grade preparation for the machine is necessary as a result of the blast displacement. Ramps are required to facilitate movement of the machine onto the leveled bench. This work is normally done in advance by a dozer, so that dragline productivity is not inhibited by the need to make ramps or grade.

Cost is usually not a major factor. The use of a dozer for grade preparation is estimated to cost approximately 1 1/4 cents per cubic yard moved. Even though this additional cost is incurred, the productivity gains are such that the overall stripping cost can decrease or remain the same.

Figure 13 illustrates a cross-section of subsequent blasts following the initial move to this type of system. The highwall is now stepped requiring subsequent blasts to be drilled from two levels. There may be the requirement to shoot for the dragline tail room. The main difference between the blasts in Figure 12 and 13 is that after the blast in Figure 13, the dragline elevation is again slightly lower. Slightly less material is spoiled by the blast as the crest partly originates at a lower level. Subsequent cuts would produce similar profiles to that illustrated in Figure 13.

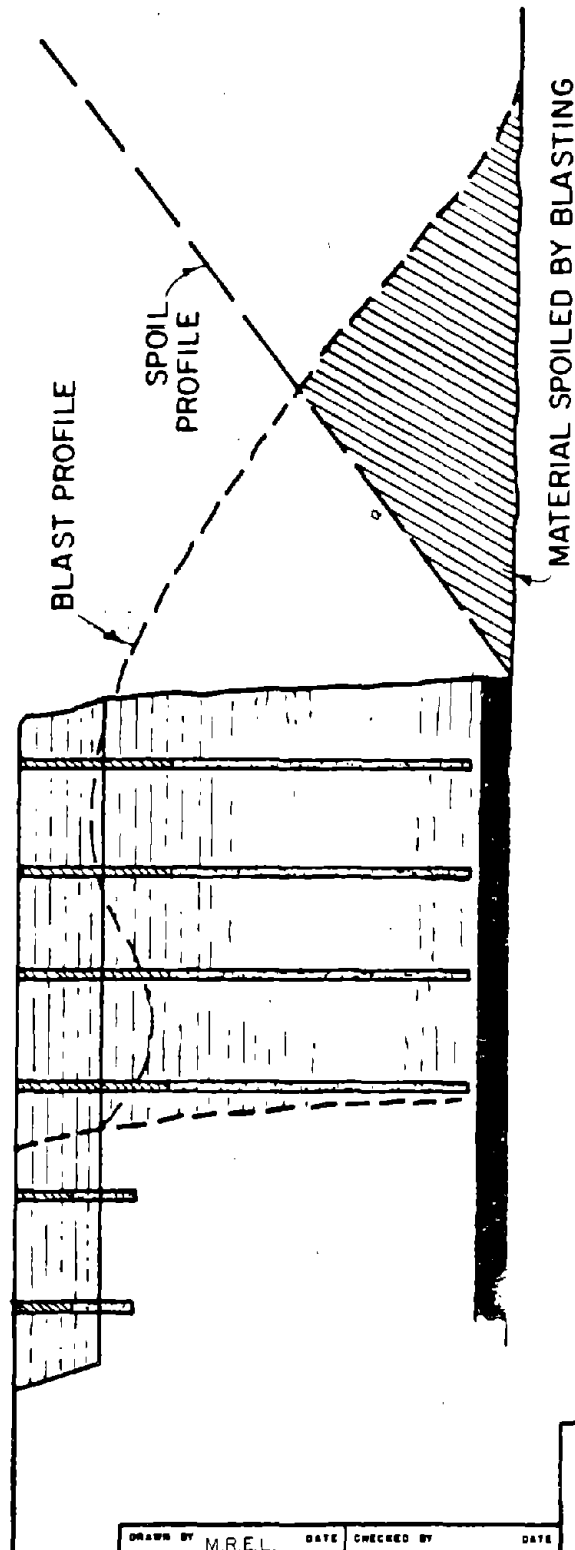


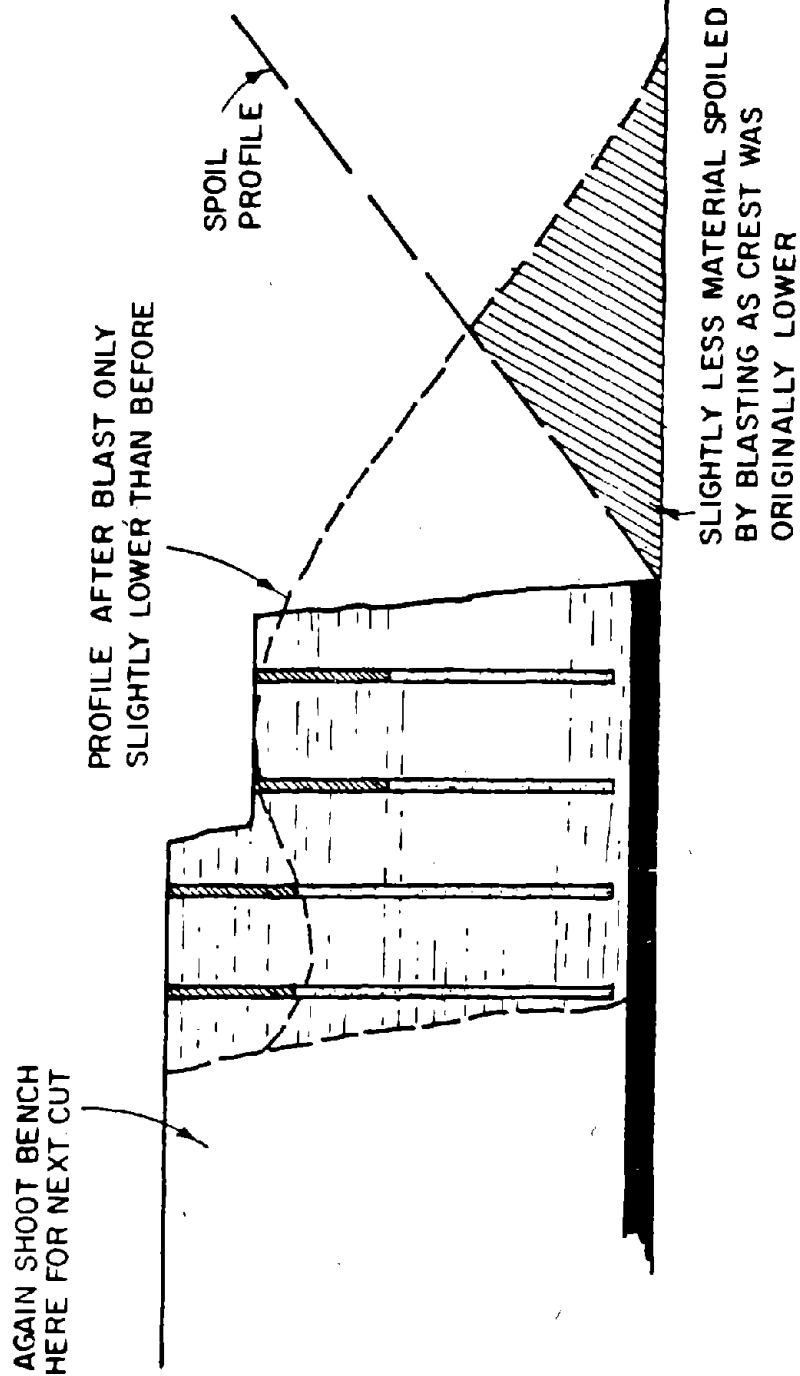
ILLUSTRATION OF THE DRAGLINE
TAIL ROOM BEING FORMED BY
SHOOTING A BENCH INTO THE
POWER TROUGH CREATED BY THE
MAIN HIGHWALL BLAST

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ILLUSTRATION OF THE NEXT
PRIMARY BLAST FIRED WITH
HOLES ON BOTH THE DRAGLINE
BENCH AND THE ORIGINAL
GROUND ELEVATION

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Figure 14 shows one mine's surface blast tie-in for an essentially square pattern using the V-1 angle. This produces good throw into the cut. For even more throw, the V2 angle is used. The V2 is formed by joining diagonal holes in a six-hole rectangle as opposed to the diagonal of the four-hole square for the V-1 angle.

Figure 15 shows the corresponding equilateral triangle V2 tie-ins pattern which is now generally favored because of its better explosives distribution in the blast as well as good spacing to burden ratio. Excellent throw is achieved with this design. Control of the throw is afforded by control of the burden on the front row holes.

Finally, Figure 16 shows the modification of the blast tie-in needed for the initial blast in a new cut using the equilateral triangle pattern. Blasting can proceed in either or both directions from this initial blast.

All of the rear row holes as well as the holes along the side are detonated individually. This helps to pull the blast away from the wall in a peeling action. It also limits the energy released into the wall at any instant.

5.2 Throw Blasting Using the Dragline on the Spoil Side

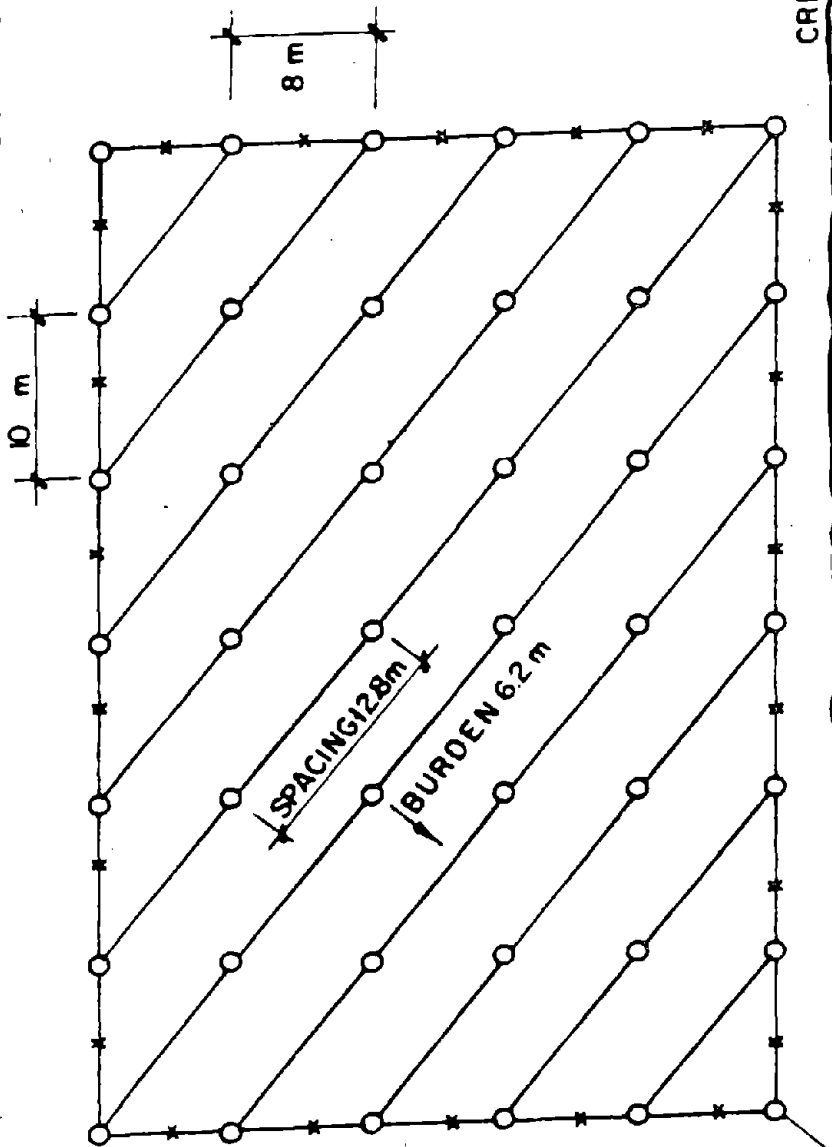
Mines using the dragline on the spoil side incorporate a presplit operation into the mining cycle. Presplitting is also used by a number of mines which have limited primary blasthole size. The presplitting operation will first be described followed by the dragline operation in a single-seam and a two-seam operation.

A number of dragline mine operators have introduced the use of pre-splitting in coal pits. Reasons for its introduction are in addition to the improved safety of the coal extraction within the cut itself.

For operations with water problems, presplitting dewateres the blast-holes which enables the easy use of dry mix explosives rather than the higher cost slurries. When presplitting a 150-ft wide pit for say a 1000-ft length, the entire presplit block moves up to 1.5 ft towards the mined out cut. This movement is most common when the overburden bedding plane-coal seam contact is weak. The main blastholes drilled after the presplit has been fired, are usually found to be completely dry.

A second major reason is the enhancement of the throw blasting technique which has now been introduced at many properties. A presplit face allows accurate control on the burden for the full length of the front row blastholes (see Figure 17). This enhances throw blasting control.

SPACING / BURDEN RATIO = 2.1 : 1

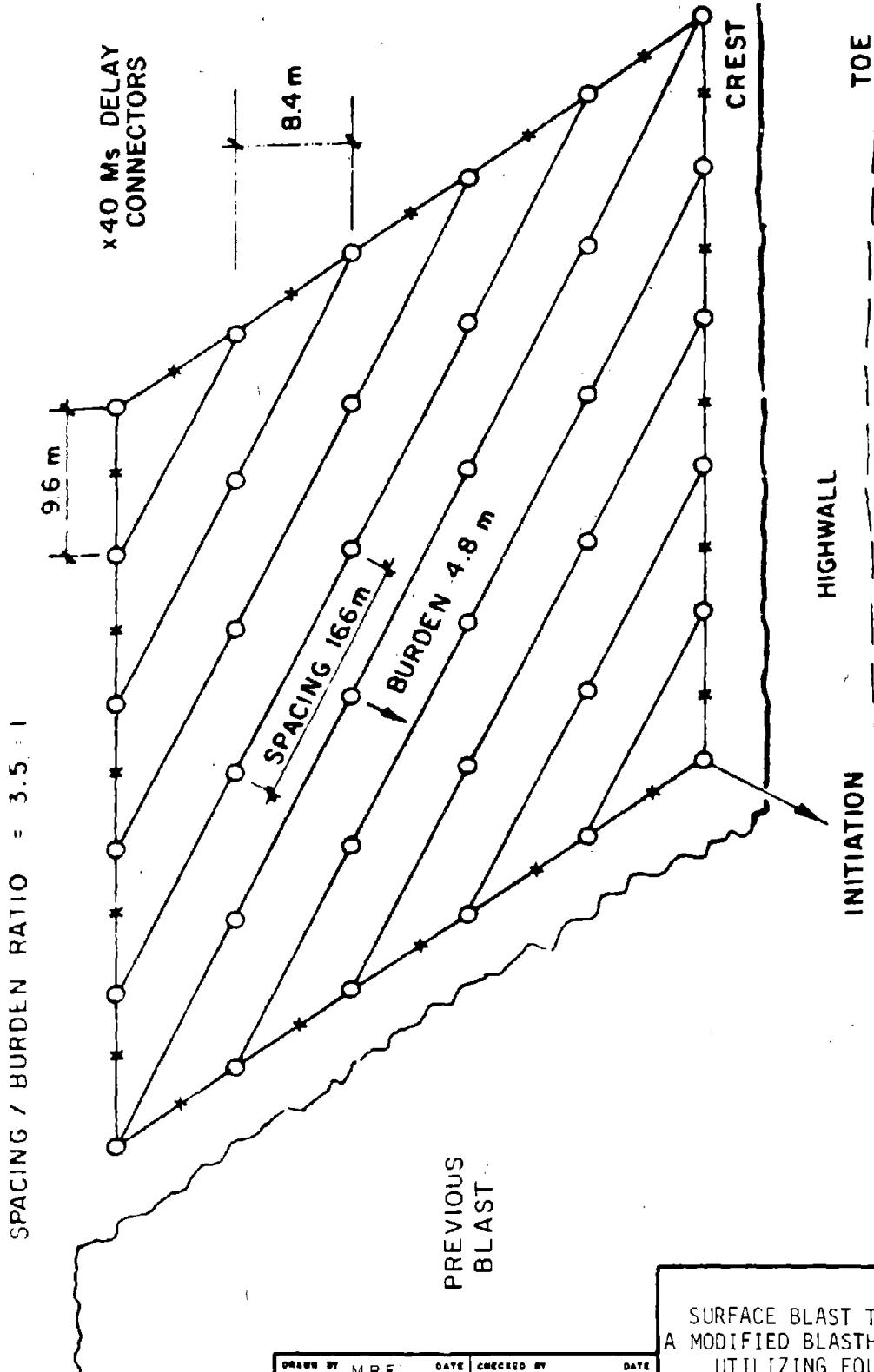
* 40 Ms DELAY
CONNECTORSPREVIOUS
BLAST

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SURFACE BLAST TIE-IN FOR AN
ESSENTIALLY SQUARE PATTERN
FIRED ON THE DIAGONALWOODWARD - CLYDE CONSULTANTS
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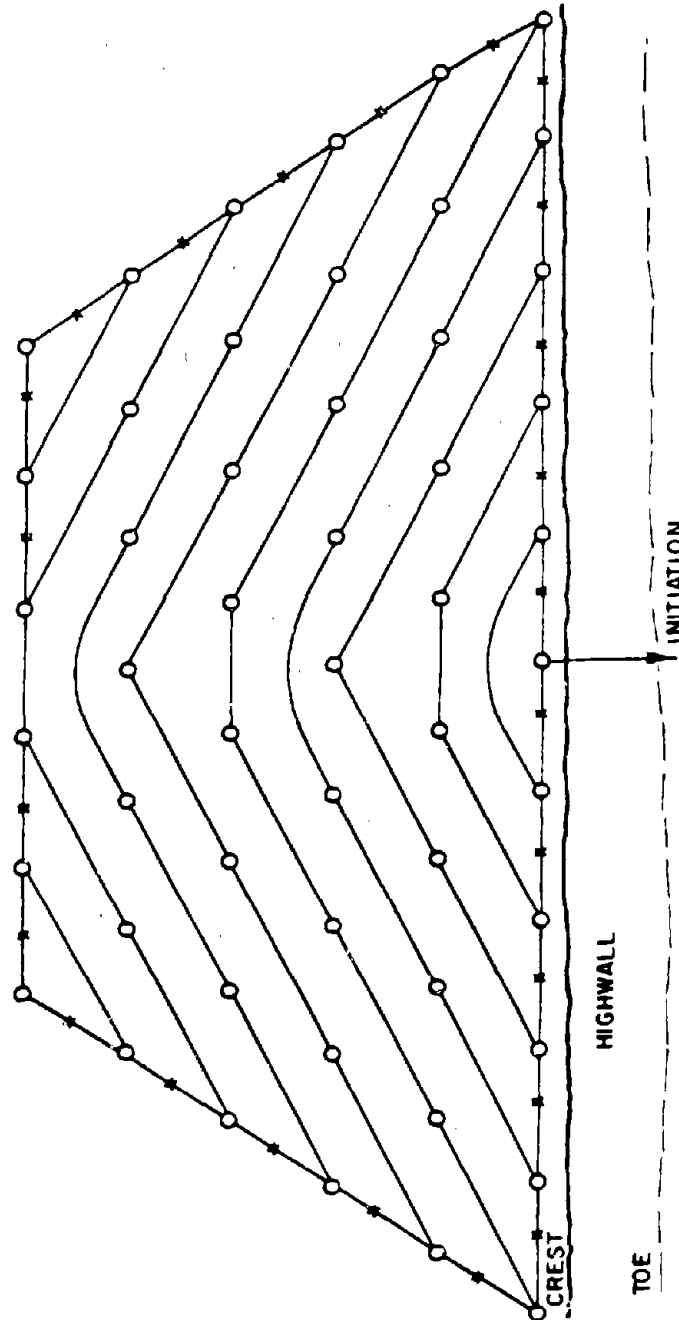
SURFACE BLAST TIE-IN FOR
A MODIFIED BLASTHOLE PATTERN
UTILIZING EQUILATERAL
TRIANGLES

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x 40 Ms DELAY CONNECTORS



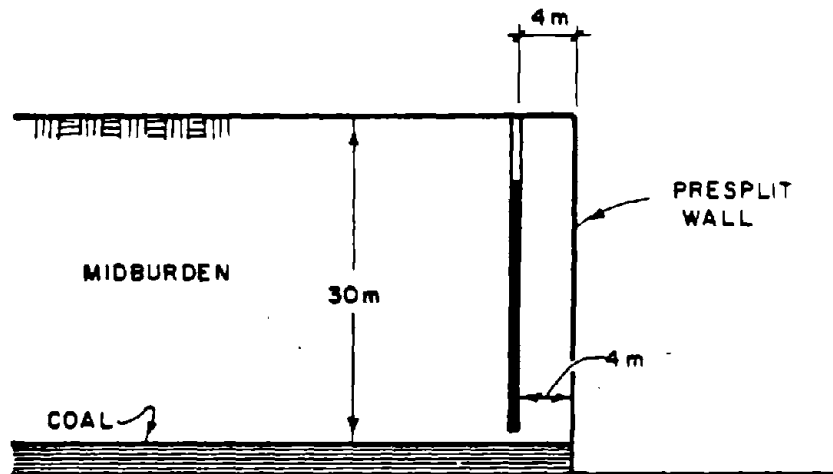
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SURFACE BLAST TIE-IN FOR THE
INITIAL BLAST OF A NEW CUT
FOR A MODIFIED BLASTHOLE
PATTERN UTILIZING
EQUILATERAL TRIANGLES

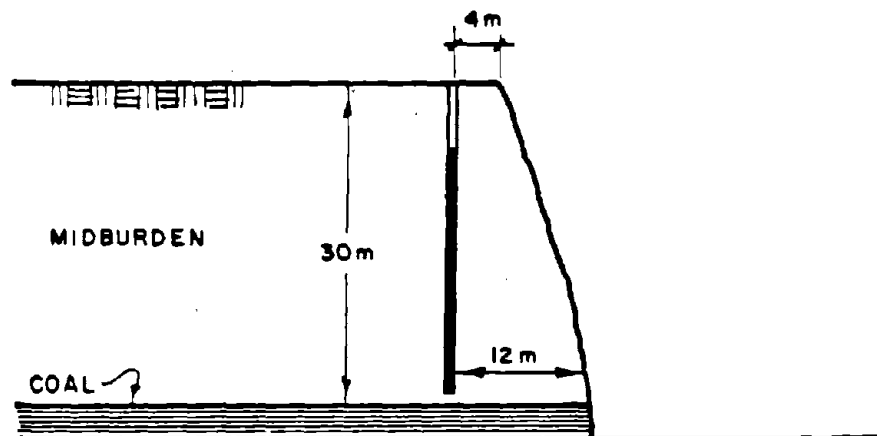
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(a) MIDBURDEN BLAST FRONT ROW DESIGN SHOWING ACCURATE CONTROL ON THE BURDEN WHEN PRESPLIT IS USED



(b) MIDBURDEN BLAST FRONT ROW DESIGN SHOWING POOR CONTROL BECAUSE OF BACKBREAK RESULTING IN VARYING BURDENS WHEN PRESPLIT IS NOT USED

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COMPARISON BETWEEN FRONT ROW
BURDEN WITH AND WITHOUT
PRESPLIT

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A good portion of oversize from any blast normally comes from the blast face. Casting the material using explosives throws the oversize clear of the dragline operation into an area which forms the base of the new spoil pile. This base usually has good drainage characteristics.

Finally, the use of the presplit line clearly defines the fragmented material for the dragline operator who no longer has to cut a batter for the highwall. At the same time the coal rib is very accurately defined resulting in lower coal losses.

5.3 Presplit Operation

The horizontal bedding and pronounced plane of weakness at the coal/overburden interface means that a single charge fully coupled at the bottom of the presplit hole is adequate. As stated previously, the entire overburden or midburden block moves on top of the coal by up to 1.5 feet.

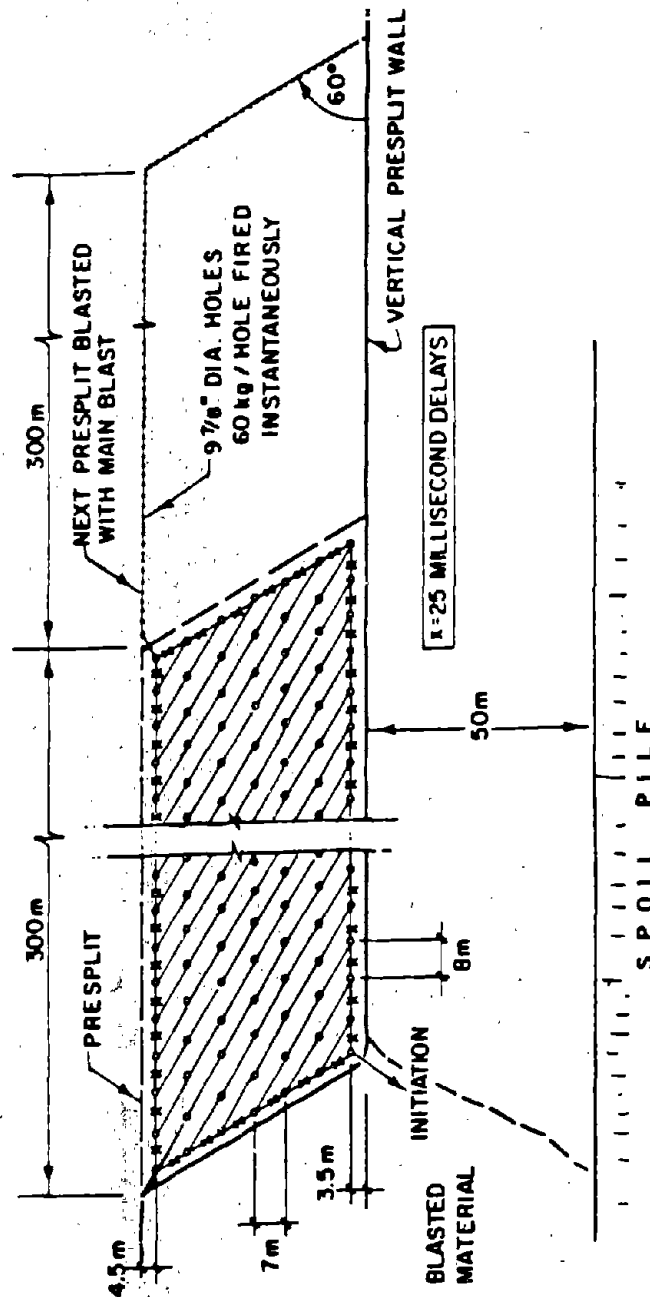
The presplit hole spacing for a 9 7/8-inch diameter blasthole varies from 3m to 6m at overseas operations, depending on the strength and competency of the material. Highly fractured softer materials require a 3m spacing, while competent higher strength materials, e.g., sandstones of compressive strength 12,000-15,000 psi, are satisfactorily presplit using a 5m-6m spacing.

Presplit holes are drilled to the coal and loaded with 60Kg (132 lbs) of explosives (usually slurry in wet conditions), the charge being placed on the hole bottom or suspended just above the bottom by 1 to 1 1/2m using a borehole liner. No stemming is used and, if possible, it is preferable to dewater. The presplit row, prior to any of the main blasthole pattern being drilled, is then fired instantaneously.

The presplit line is drilled completely around the proposed blast area, with the cutoff to the face being made at the blast cutoff angle (usually either 45 or 60 degrees with the face). Once an area has been presplit, the entire block of material should then be blasted or poor fragmentation results at the blast interfaces. This main overburden blast is shot in the normal manner with a back row standoff from the presplit line of 5m for a 9 7/8-inch hole. Figure 18 presents an overall plan view of the presplit and main blast showing the method of delaying and relative positions of the blasts.

Figure 19 shows presplit results above a single seam of coal and Figure 20 shows a multiple-seam operation. In Figure 20 the hole spacing change is evident. In the softer overburden above the upper seam of coal the hole spacing is 3m, whereas above the lower seam, the waste material is competent sandstone and the hole spacing is 5m.

Figure 21 shows the cut-off at the end of a midburden blast that has been designed at an angle of 60 degrees with the face.



PLAN VIEW OF TYPICAL PRESPLIT AND MAIN BLAST TIE-IN AT A DRAGLINE OPERATION

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FIGURE 19: PRESPLIT WALL AT THE END OF A CUT



FIGURE 20: PRESPLIT WALL IN A TWO SEAM OPERATION



FIGURE 21: ILLUSTRATION OF A MIDBURDEN BLAST CUT-OFF

5.4 Throw Blasting for a Single-Seam Operation

The prime prerequisite for successful operation is an adequate blast throw across the pit. This compensates for the reduced dragline output in the pull back mode of operation. If the pull back mode is assumed to be 60 percent as efficient as the conventional mode of operation, then, as a first approximation, blast-over should be of the order of 40 percent or more before the pull back mode should be considered.

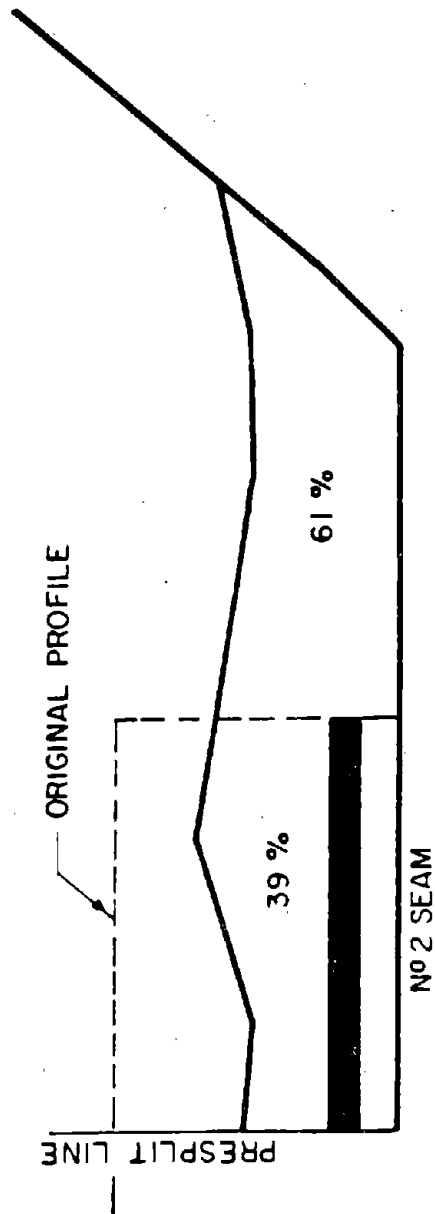
Figure 22 shows a typical cross-section through a throw blast for a single seam, and Figure 23 illustrates an actual blast in the foreground of the photograph. Figure 22 indicates more than 60 percent of the material can be thrown clear of the coal. The percentage blast over is related to the blast depth/width ratio as shown in Figure 24.

Two dragline operating positions are possible following this type of throw blast. Figure 25 illustrates a cross-section through the diagonal pull back method. Here the presplit wall falls within the operating radius of the machine. The pull back operation is therefore not performed in a direction normal to the highwall, but at an oblique angle defined by the machine operating radius and the pit width.

The dragline positions itself on the coal edge. Material is rolled back behind the dragline to construct a bench for the dragline. This material constitutes approximately 4 percent rehandle. A major advantage using this system is that the dragline can execute a low wall key cut, thereby increasing extraction and minimizing contamination.

Figure 26 shows the mode of dragline operation for the normal pull back method, so named because the dragline pull-back operation is conducted normal to the presplit highwall. The position of the dragline is defined by the operating radius.

As is shown in Figure 26, the dragline can almost produce a fully reclaimed spoil pile. If the selection of the dragline elevation is incorrect, additional material is spread behind the dragline on the dragline bench.



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SECTION THROUGH BLAST HAVING A 30° TIE-UP

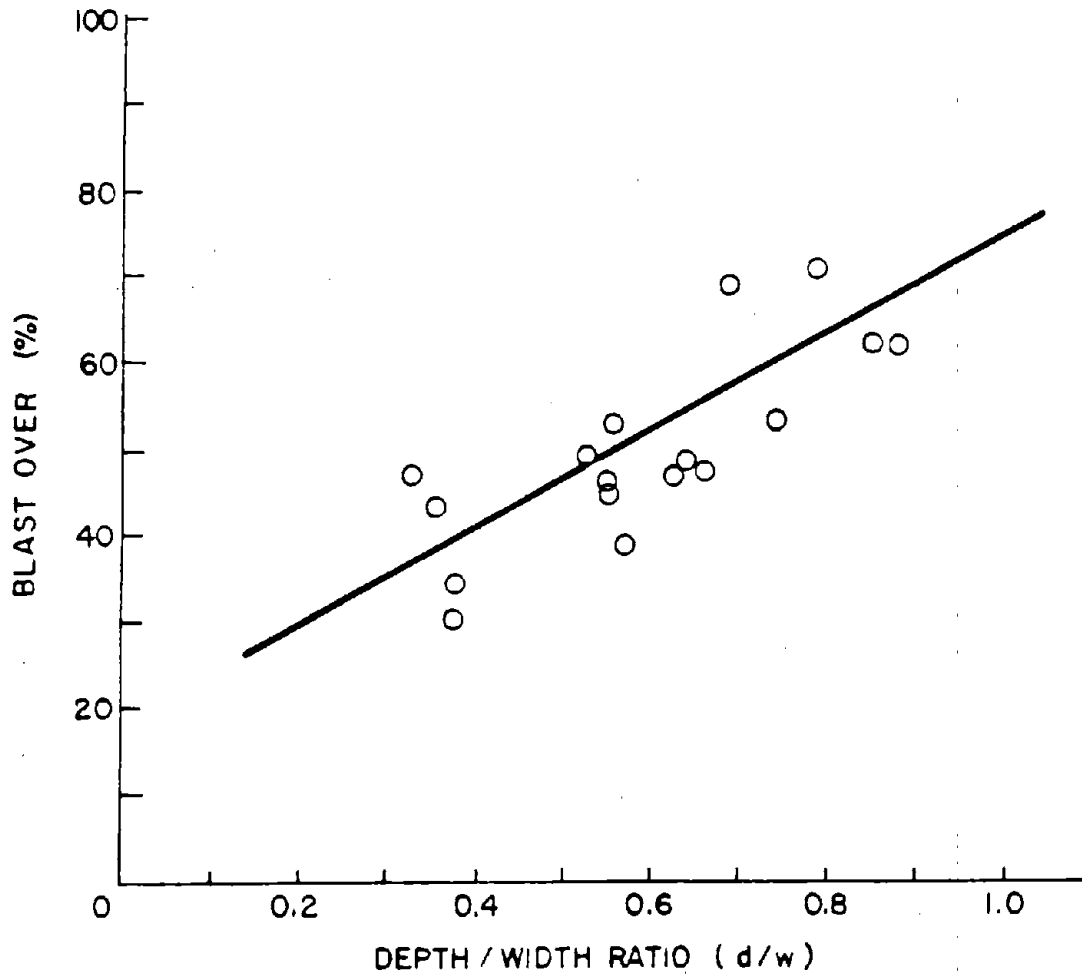
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FIGURE 23: ILLUSTRATION OF A THROW BLAST FOR A SINGLE SEAM OPERATION



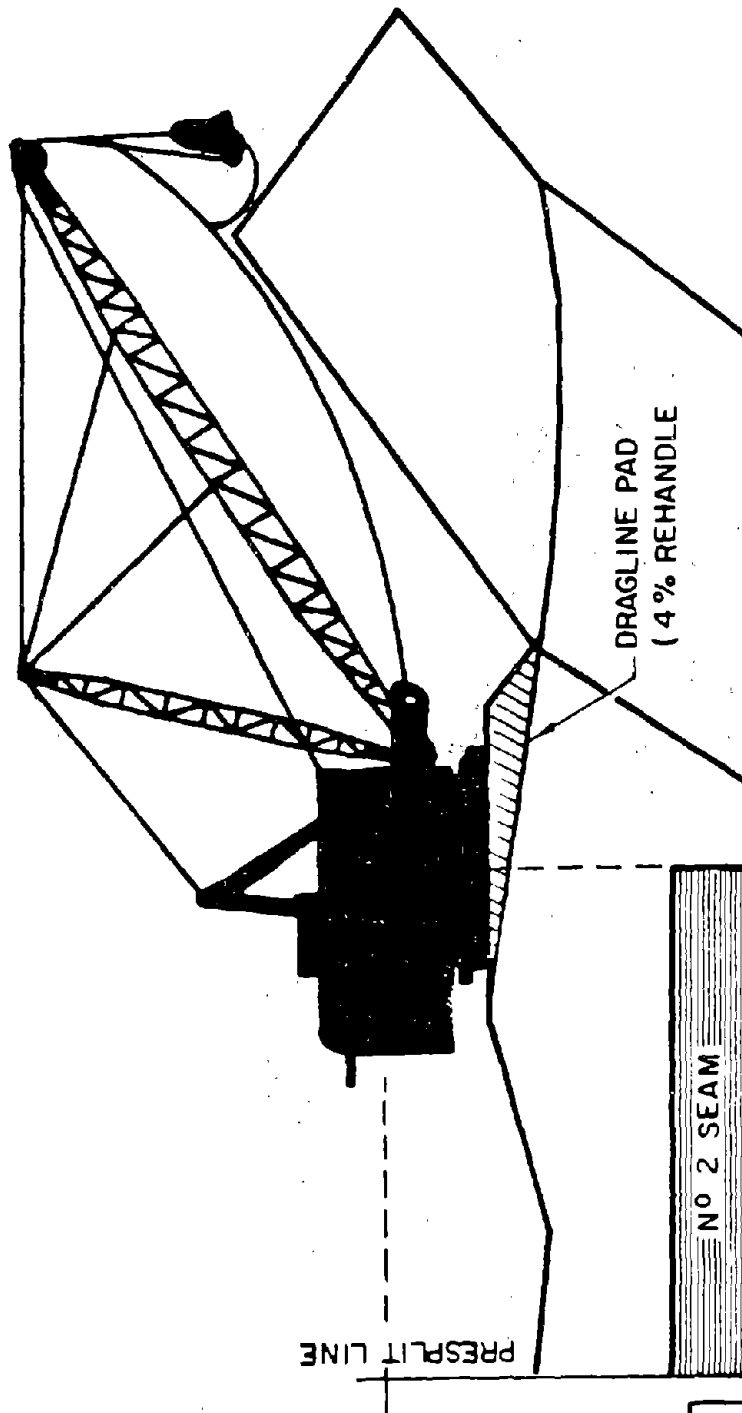
GRAPH SHOWING THE RELATIONSHIP
BETWEEN THE PERCENTAGE
BLAST-OVER AND DEPTH/WIDTH
RATIO (CROSBY, McDONALD
AND SMITH, 1982)

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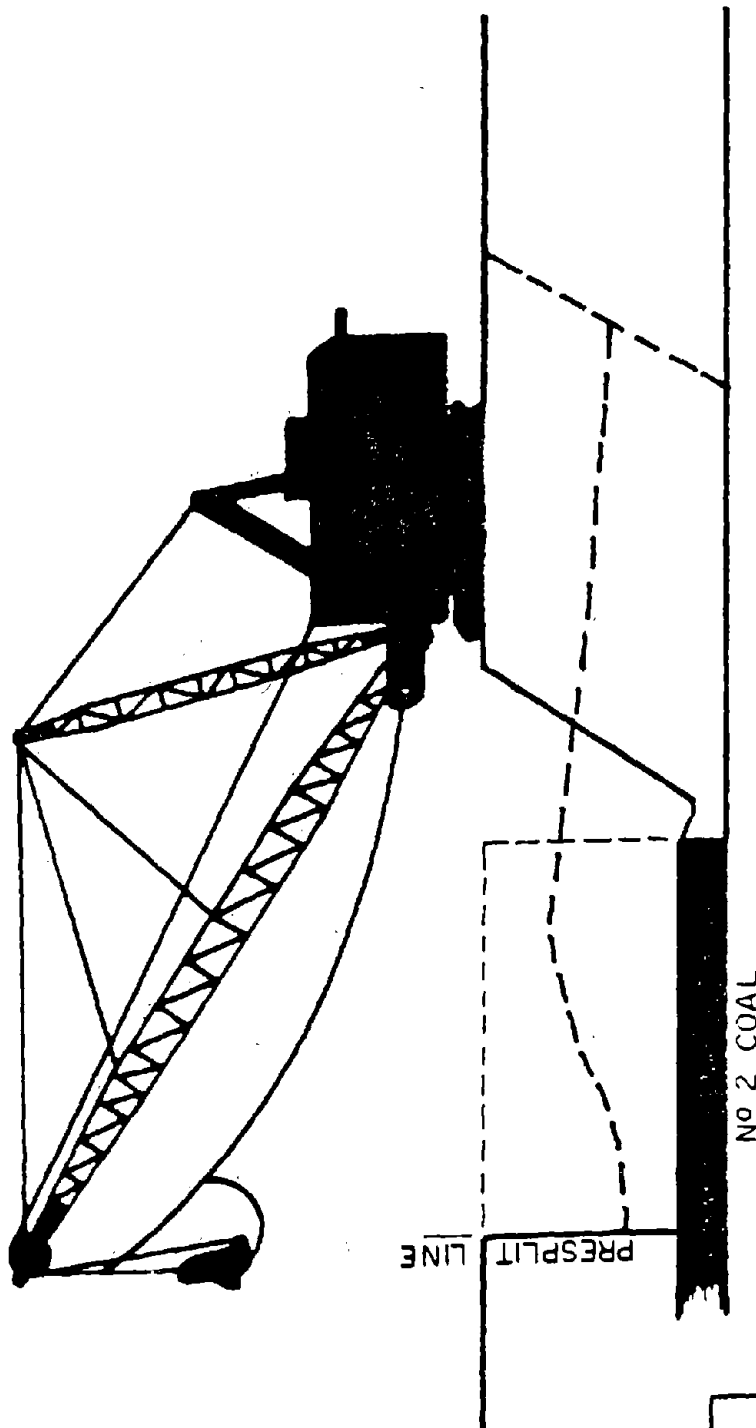
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DIAGONAL PULL BACK METHOD

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SECTION VIEW OF NORMAL
PULL BACK METHOD

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5.5 Comparison Between Single-Seam Dragline Operating Methods

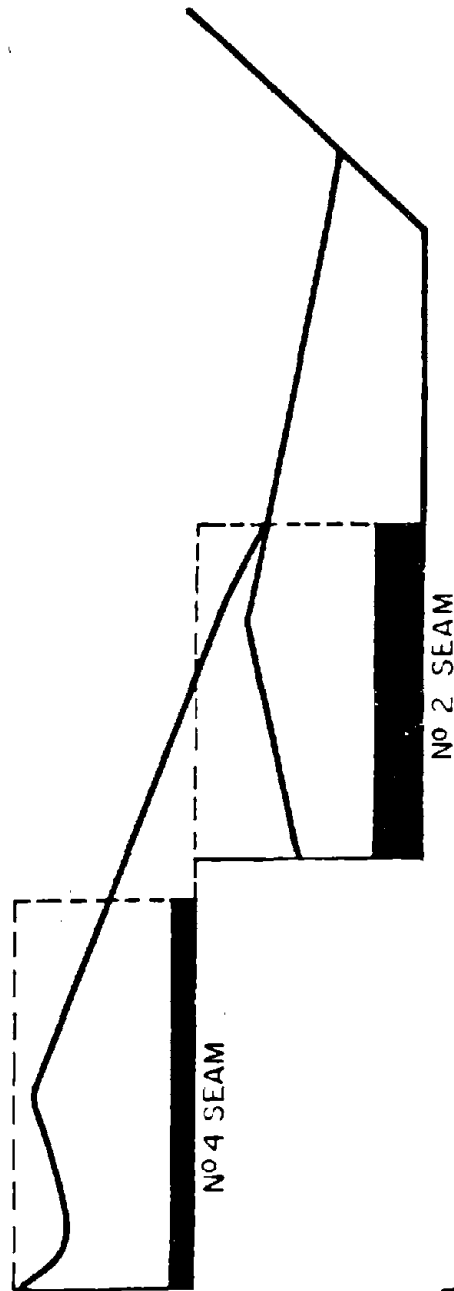
The sectional views indicated minimal rehandle (4 percent) for the diagonal pull back method and zero rehandle for the normal pull back method. In practice, the pad elevation for the diagonal pull back method has to be increased at the location of access ramps due to the physical constraints of the dragline. The additional rehandle incurred, because of ramps, requires a greater dumping height than in the areas outside the influence of these ramps. Increasing the dragline working bench height results in increased rehandle of the order of 15-20 percent. This 15-20 percent increase in rehandle is of the same order of magnitude as the expected increase in overall stripping efficiency and, therefore, tends to immediately preclude this system of dragline operation.

Actual dragline total movement rates for the diagonal method versus the normal method are usually lower. Therefore, even though a low wall void can be maintained using the diagonal technique, the normal pull back mode is preferred. This is particularly true when reclamation costs are included in the comparison.

5.6 Throw Blasting for a Two-Seam Operation

Figure 27 shows a typical cross-section after throw blasting both the overburden and midburden in a two-seam operation. Figure 28 is an illustration of a midburden blast prior to the blast-over of the upper overburden. Figure 29 shows the method of dragline operation used to uncover both seams of coal.

Figure 30A and 30B show two positions of dragline operation, both uncovering the lower seam. Figure 30A shows the key cutting operation and pad preparation for the final pull back. The final dragline location is shown in Figure 30B.



SECTION AFTER OVERBURDEN
AND MIDBURDEN BLASTS
FOR THE TWO SEAM OPERATION

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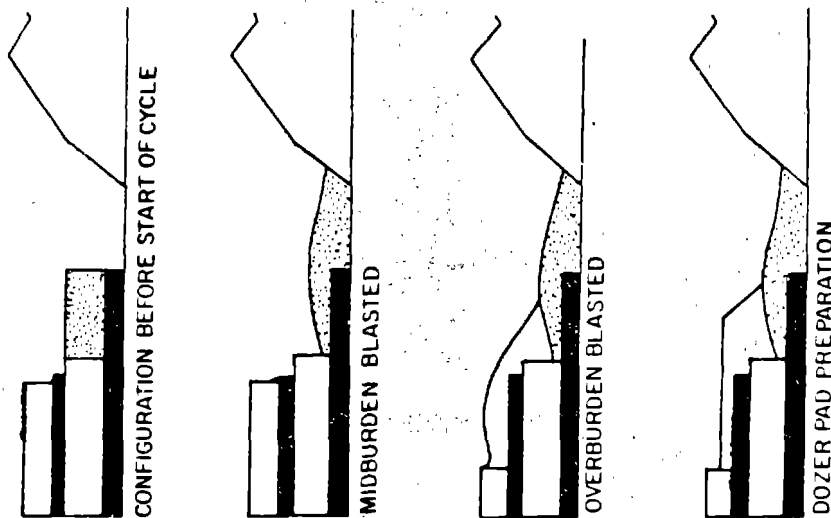
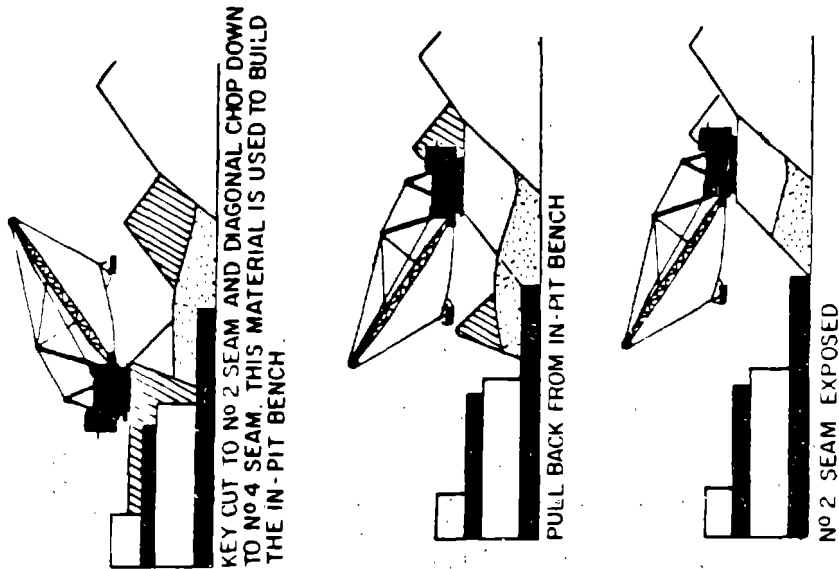
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FIGURE 28: MIDBURDEN THROW BLAST PRIOR TO OVERBURDEN BLAST AT A TWO SEAM OPERATION



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DOUBLE SEAM DRAGLINE OPERATION USING THROW BLASTING		
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FIGURE 30(a) INITIAL DRAGLINE POSITION UNCOVERING THE LOWER SEAM IN A TWO SEAM OPERATION



FIGURE 30(b) FINAL DRAGLINE POSITION UNCOVERING THE LOWER SEAM IN A TWO SEAM OPERATION

Finally, Figure 31 illustrates a close-up of the key cut made for the lower seam. In the distance the next midburden throw blast can be seen.

As well as producing productivity gains, throw blasting at two-seam operations provide other advantages.

The throw blast system of dragline operation effectively doubles the pit length because both the seams are exposed simultaneously. This allows correspondingly large in-pit coal inventories to be carried.

The throw blast system allows both seams to be mined simultaneously, thereby ensuring a consistent quality of raw coal feed for blending purposes.

The coal operation can be simplified if required. The dragline can spoil the upper seam onto the lower to reduce access requirements and coal loading shovel moves.

The blast-over results in competent hard rock forming the base of the final spoil pile, as opposed to a mixture of subsoil and weathered rock which forms the base when using conventional double seam methods. The material now forming the toe of the spoil pile exhibits better draining characteristics which improves the spoil pile stability.

Dragline scheduling is improved and made easier as compared to conventional mining sequences. Dragline deadheading is also reduced.

6.0 OLD BEN NO. 1 COAL MINE, WEST FIELD, TEST PROGRAM CASE STUDY

6.1 Historical West Field Mining Practice

The Old Ben Mining Company removes coal from its West Field operation at the rate of approximately 3200 tpd. The operation employs a Bucyrus-Erie 2570-W dragline for overburden stripping. The stripping dragline is aided by a Bucyrus-Erie 1150-B "pullback" dragline located atop the spoils (see Figures A-1 and A-2, Appendix A). The 2570-W has a 100-yd³ bucket and the 1150-B (pullback) has a 25-yd³ bucket. The 2570-W and 1150-B work 364 days/year, 3 shifts/day.

The explosive conditioning of overburden at the Old Ben has historically been by "buffer" blasting. One advantage of this method is that muck is not thrown into the pit and coal transport corridors within the pit are continuously operable. A possibly negative aspect of this technique is that the overburden, while fragmented at detonation, is so confined that geologic bedding structures may be only moderately disrupted. These semi-intact geologic surfaces form hard planes which may inhibit dragline bucket teeth penetration.

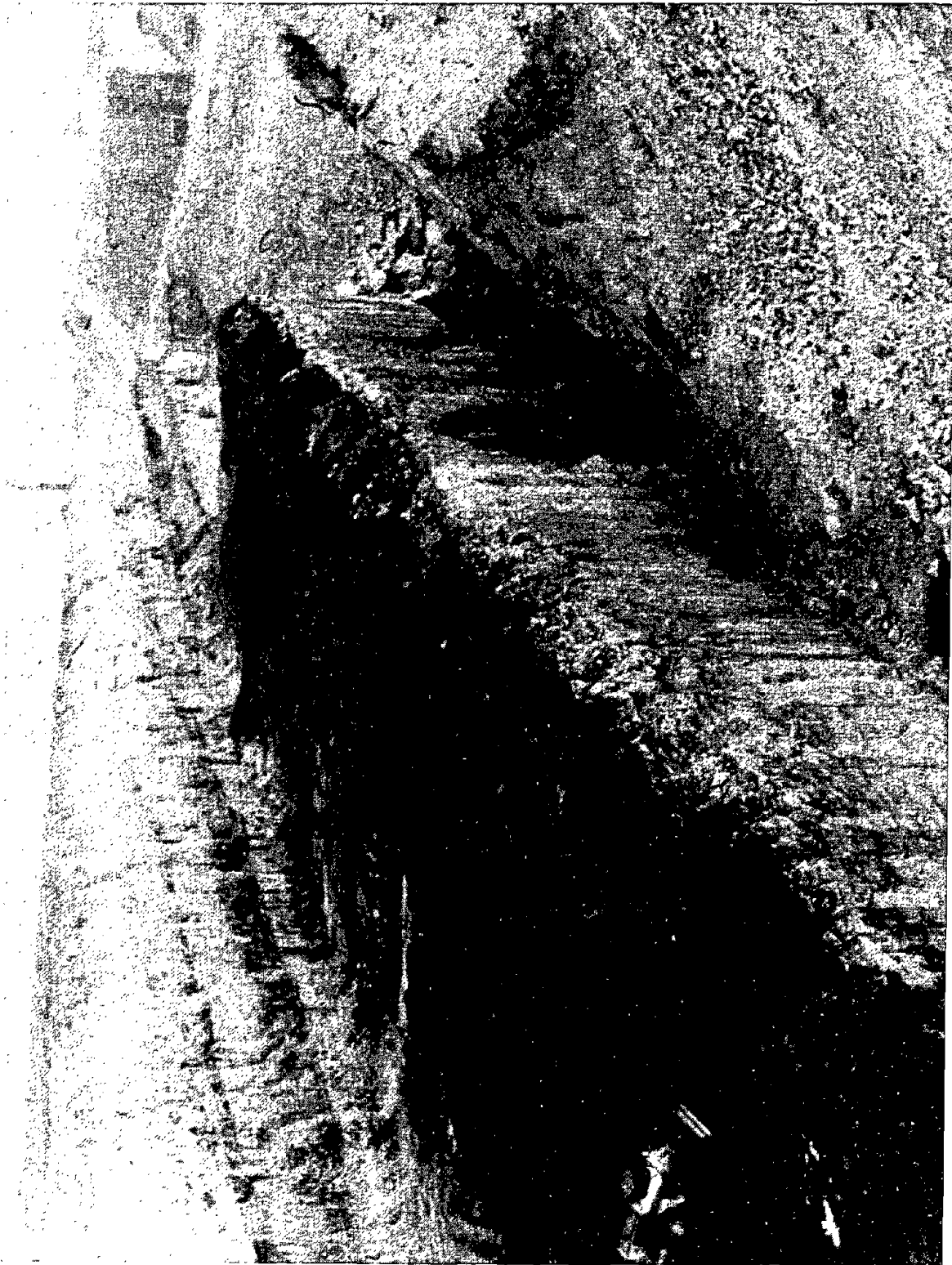


FIGURE 31: CLOSE-UP OF THE KEY CUT FOR THE LOWER SEAM OF A TWO SEAM OPERATION

The Old Ben historical blasting practice includes both full depth and satellite (shallow) holes. The full depth holes go from the bench level (the base of the No. 6. Coal seam, where present) to the top of the No. 5 Coal. Drill cuttings are placed in the bottom of the hole to protect the coal from deleterious blast effects. The drill hole is column or deck loaded with explosives to a prescribed collar height. The decked charges, when used, are positioned next to the hardest overburden strata to improve fragmentation. The cumulative explosives load per hole is controlled in the field by a "twenty times the hole depth" charging rule; i.e., a 100-foot hole would have approximately 2000 lbs. of AN/FO. Satellite holes are located between full depth holes and are designed to fragment a limestone caprock which is pervasive throughout the field. See Figure 32 for the historical blasting layout. The satellite holes are typically 30-35 feet deep and charged with 300 lbs. of AN/FO.

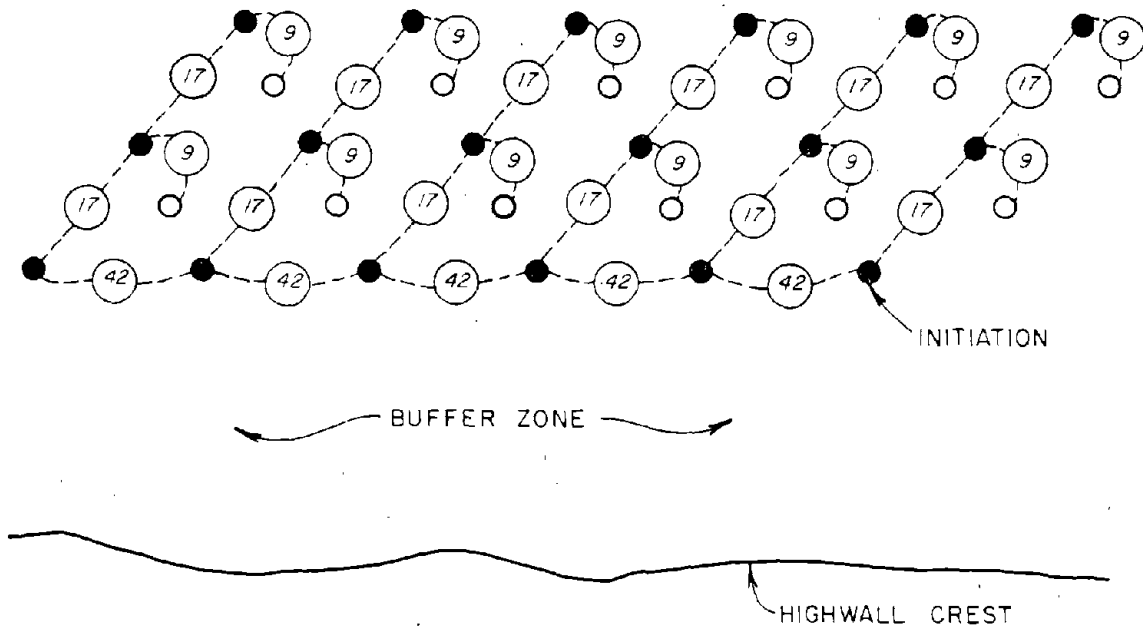
The West Field in some areas has a thin coal seam, the No. 6 Coal, lying some 15-30 feet below the topographic surface. This coal is uncovered by a separate overburden drilling program. The bulk of the West Field coal reserve is in the deeper lying No. 5 Coal seam. The No. 5 Coal is overlain by 100-150 feet of overburden or 85-125 feet of No. 6 - No. 5 Coal interburden.

A more complete description of West Field geology can be found in Appendix B.

The historical coal production sequence includes topsoil scraping and transportation to spoils side reclamation activities, No. 6 Coal overburden drilling and blasting, No. 6 Coal overburden removal by the 2570-W, No. 6 Coal extraction, bench leveling, No. 6 - No. 5 Coal interburden drilling and blasting, No. 5 Coal overburden stripping by the 2570-W, and No. 5 Coal extraction. The stripping cut and resultant pit width is nominally 100 feet. The spoiled overburden is redistributed by an 1150-B dragline and dozer combination to approximately its original contour. Ground water occurrences are sporadic in the field, but pit pumping is a nearly continuous operation due to the collection of surface runoff plus miscellaneous seepages. The West Field, if operated as a single broadly advancing face, would be nearly 10,000 feet in length. The operation, as existing, was working a 5000-foot-long face, with the remaining ground out of active production. The test area set aside for the blast design program was 2000 feet in length in a dry portion of the field.

Prior to the test program, buffer blasting was employed utilizing a 36 ft x 48 ft blast pattern (see Figure 32). The yield was 64 cyd/ft of hole. The powder factor was 0.33 lbs/cyd. Several decks were used in the blastholes with AN/FO explosive placed next to the hard layers.

The depth from the bench surface to the top of the explosive column varied with hole depth. This allowed further regulation of the powder factor as the depth of cover increased. The procedure did, however, result in the explosive being placed at distances well below caprock in many holes.

LEGEND

- FULL DEPTH DRILL HOLE
- SATELLITE DRILL HOLE
(TYPICALLY 30' DEPTH)
- ④2 HOOK-UP AND DELAY (msec.)

LOADING NOTES:

1. FULL DEPTH LOAD (lbs. ANFO) =
20x HOLE DEPTH/HOLE; 6 - 8
DECKS COMMON.
2. SATELLITE HOLE CHARGE (NOMINAL)
300 lbs/HOLE

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SCALE: 1"=50'								Job No 19943	

To break the limestone caprock consistently, satellite holes were drilled and loaded. These holes were 30 feet in depth and generally contained a toe load of 250 pounds of AN/FO plus a 50-pound column load at 15 feet below bench elevation. These charges were designed to crater through to the top surface, thereby fragmenting the caprock.

7.0 THROW BLAST DESIGN FOR THE FIELD TEST SITE

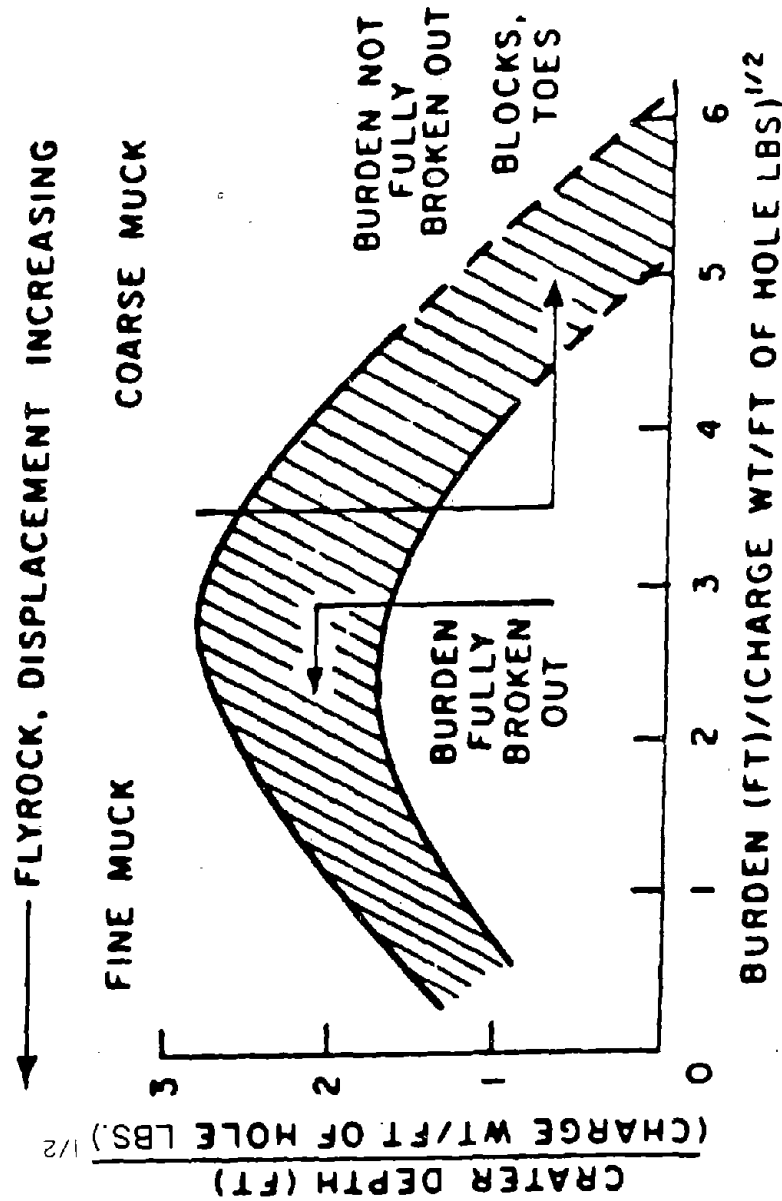
The field tests were designed to introduce the concept of blast throw to both reduce the quantity of material that must be moved by the dragline as well as increase the dragline performance by improving blast fragmentation. Dragline operation had to be maintained on the highwall side of the pit at the original highwall crest elevation to meet logistical dragline operating problems imposed by a short pit length and the upper seam of coal at the dragline tub elevation.

For the test program several changes were made to the blast design in order to achieve greater horizontal displacement and fragmentation. First, the buffer zone was removed for the 2000-foot test section. Second, the blast pattern was examined. Based on a scaled distance of $3.5 \text{ ft}/(\text{lbs})^{1/2}$ (square root scaling for a cylindrical column charge) a pattern of 40 ft x 40 ft, breaking 59 cyd/ft, was selected to be shot with a V-1 tie-in. Figure 33 shows the applicable crater relationships.

In order to consistently break the caprock, a constant collar depth was desired. It is assumed that the top eight charge diameters perform this task and acts as a spherical cratering charge. Based on a scaled depth of burial of $4.0 \text{ ft}/(\text{lbs})^{1/3}$ for the moderately strong limestone caprock, a constant explosives collar depth of 30 feet was selected.

The expected toe distances on the front row were calculated and ranged from 36 to 56 feet. AN/FO could be expected to break and displace a 36-foot toe at maximum. This was based on a scaled burial of $4.0 \text{ ft}/(\text{lbs})^{1/3}$. Since the strata directly above the coal were quite soft, the optimum could be greater than 4.0 and it was decided to start with AN/FO at the bottom of the blastholes and substitute a more energetic product if initial high-speed films indicated limited toe movement.

After the examination of the strata above the No. 5 Coal, a powder factor of 0.70 lbs/cyd (1.43 cyd/lb) was selected. For blastholes up to 100 feet in depth the powder factor was maintainable with a full column load. For 100 feet and greater, a deck was required to maintain the explosive consumption at 0.70 lbs/cyd. A collar load to the 30-foot level was maintained in almost all cases with the deck stemming placed next to a soft layer, below the collar load. Loading errors which placed the collar load to below the 30-foot level caused large fragmentation of the hard strata and thus inadvertently demonstrated the validity of the designed approach.



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THE CHANGE IN CRATER DEPTH
PRODUCED BY DIFFERENT BURDENS
ON CYLINDRICAL CHARGES
FIRED IN GRANITE

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With the above design it was possible to remove the satellite holes that were previously being drilled. This was due to the collar design which was designed to insure proper breakage through to the top surface on the designed pattern.

For the blast tie-in, millisecond delays of at least 1 millisecond per foot of burden and preferably 1-1/2 milliseconds, were to be used. Blasts were initially designed with a V-1 tie in. Subsequently, however, it was decided to shoot straight at the excavated pit to maximize the horizontal displacement. Each hole was separately delayed to reduce ground vibration effects.

The blast design is summarized in Table 2.

7.1 Mine Planning

A number of mine planning changes were necessary for the test section. The most significant was the preparation of the dragline bench grade.

TABLE 2

PRELIMINARY BLAST DESIGN
OLD BEN NO. 1 BLAST DESIGN PROGRAM

Item	
Blast Design	40 ft x 40 ft hole spacing
Collar	30 ft from surface
Explosives	AN/FO
Powder Factor	0.70 lbs/cyd
Decking	Only for holes above 100 feet
Delays	1 to 1 1/2 msec/ft of burden-- each hole
Surface Tie-In	Nonel trunkline and delays

Figure 34 demonstrates the profile of the blasted waste for a case where good horizontal displacement of the blast was achieved. The bench elevation drops considerably due to the displacement. Dragline grade preparation requires sufficient material from the side to be dozed over to provide a level bench and swing room for the dragline key cut. This requires either dozing material, possibly with the aid of blasting, to an intermediate lowered bench as shown in Figure 35 or refilling of the area to the original grade elevation. Normally, lowering the grade elevation would be preferred.

In this case, due to the caprock close to the bench surface and the proximity of easily moved material above the No. 6 Seam, it was decided to raise the bench to its original level by pushing material across to fill the depression caused by blasting (see Figures A-3, A-4 and A-5, Appendix A).

Other mine planning concerns included the sequencing of coal removal since access would be more limited, dragline procedures at the ends of the pit, and drill sequencing since the drills and dragline would now be on the same cut. It was possible to resolve these new requirements satisfactorily. These concerns are specific to any given operation and would have to be examined for each property.

7.2 Field Test Work

The field test consisted of two passes of a 2000-foot section of highwall or a total of 4000 feet (see Figure A-6, Appendix A). A total of fifteen separate blasts were shot.

For the first test period the pattern was set at 36 ft x 45 ft. This was different from the square pattern originally designed but was similar to previously drilled blasts and thought to be easier for the crews to drill.

The collar was set at 30 feet. No satellite holes were used.

In order to protect the coal from damage, 4 feet of backfill was placed in the front holes. In order to insure breakage of the limestone next to the coal this was reduced to 3 feet in the second row and 2 feet in the back row.

The blasts were tied-in such that each hole shot separately and directly toward the pit. Three rows of holes were required. Seventeen millisecond Nonel delays were used across the face with 42 millisecond delays inserted between the rows. A 100 millisecond delay was used between the last two holes on the front row, allowing the three holes down the side of the blast to detonate last in succession from front row to back.

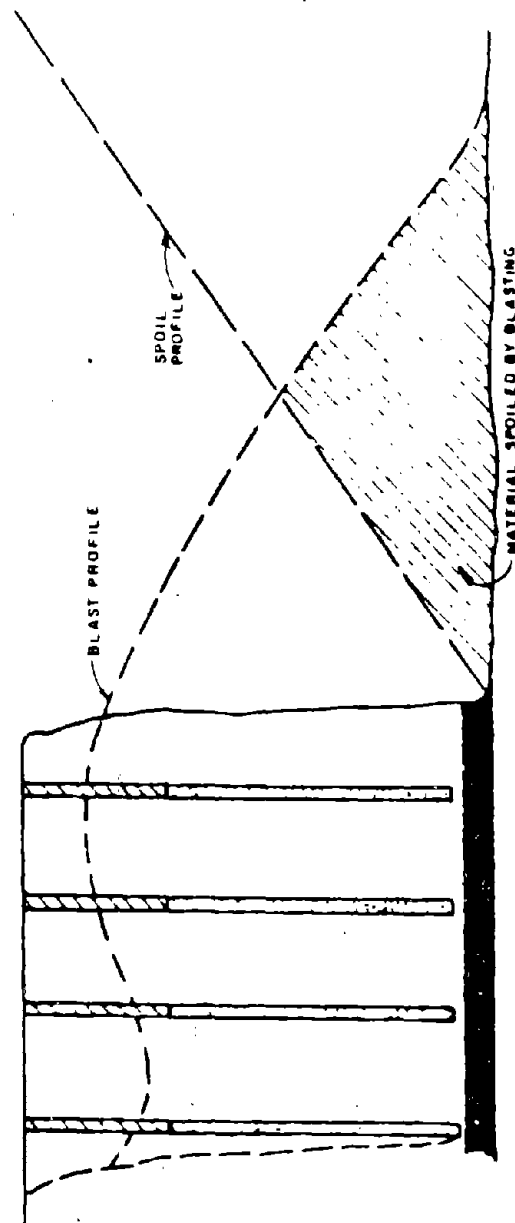


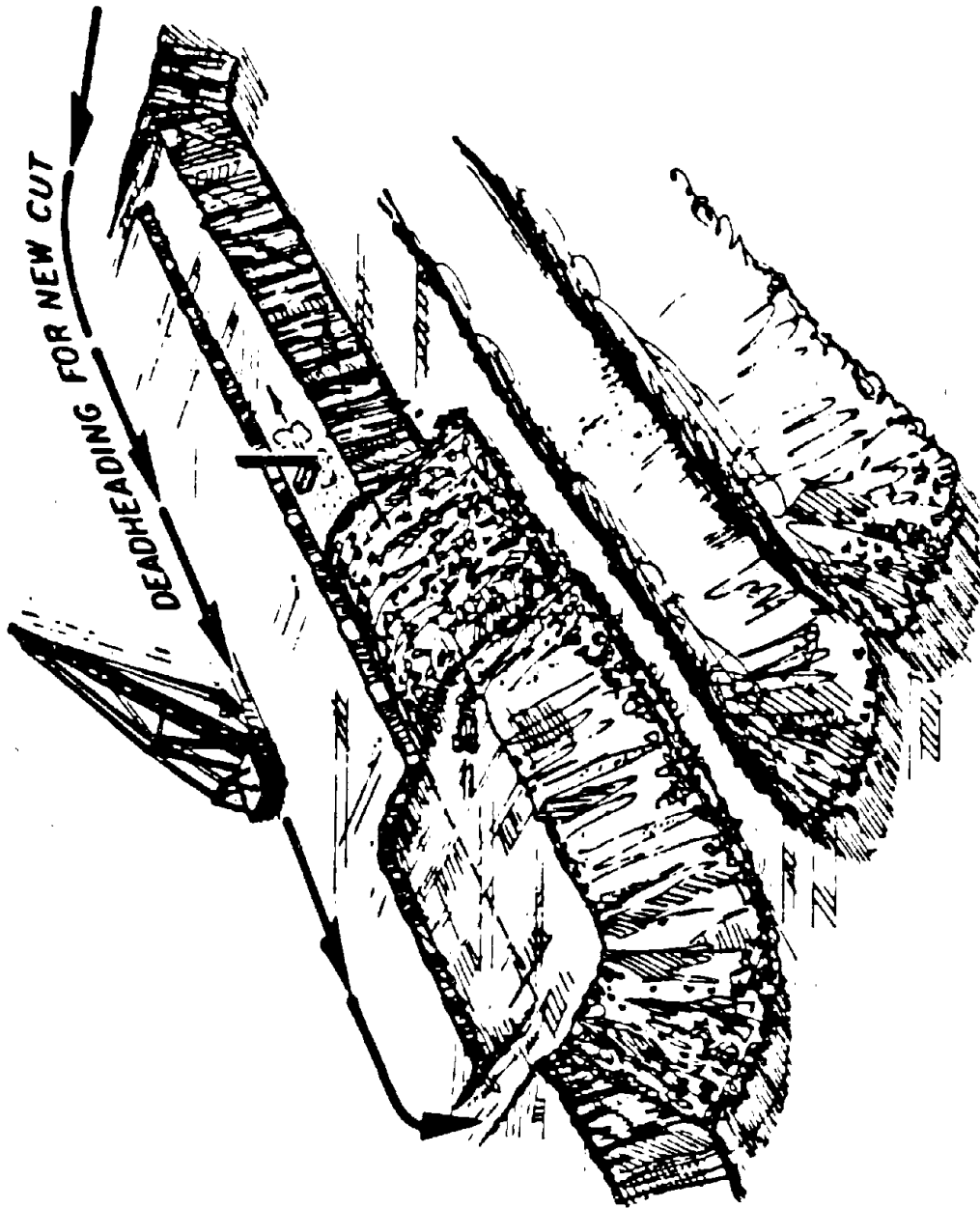
ILLUSTRATION OF OVERBURDEN SPOILED BY BLASTING ACTION

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DRAGLINE DEADHEADING SEQUENCE
WHEN TRACTOR USED TO LEVEL
BLASTED OVERBURDEN

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The result was a clean side wall. The drill could consistently be moved back on the first row of holes for the next blast with no need to adjust blasthole locations due to backbreak. A 200 millisecond delay was used down each hole to avoid cutoffs. Figure 36 shows a typical blast in plan view and Figure 37 illustrates the blast in cross-section. Blast layout, charging and sequencing details are shown on Figures A-7 to A-21, Appendix A.

The first test pass of 7 shots was uniformly drilled, column loaded, collared and delayed. The overburden ranged from 95-115 feet in depth. The column loading plus increasing hole depth (overburden increasing with each shot) led to an increasing powder factor. The overburden response, particularly with regard to muckpile overburden profile, percentage cast-over, seismicity, and fragmentation, was reviewed after each shot. Subsequent stripping of the first pass area revealed some serious coal chilling in the latter (powder factor approaching 1.0) shots.

The second pass incorporated a series of changes based on data gained from the first test. One change was to reduce the burden to 30 feet to better fragment the strata and to better match the pit geometry to avoid backbreak in the new highwall.

Initially, the spacing was set at 45 feet, generating a yield of 50 cyd/ft of blasthole. The collar was held at 30 feet and the powder factor at 0.7 lbs/cyd.

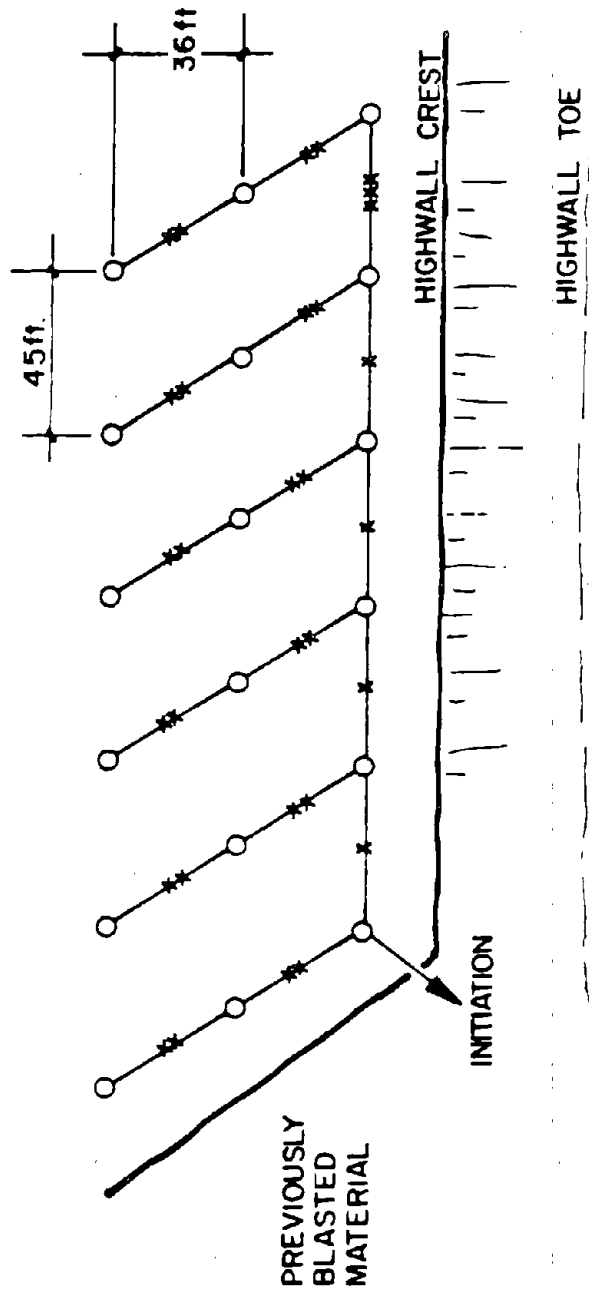
Observation of the first two blasts in Test 2 led to expansion of the spacings, first to 48 feet and subsequently to 50 feet. The burden remained at 30 feet in all cases. For blasts 5 through 8, the collars were adjusted to 26 feet, both to provide even better caprock breakage and because of the 50-foot spacing. The final four blasts had a yield of 56 cyd/ft.

Powder factors were also adjusted downward based on observation of the fragmentation achieved. Initially, in Test 2, the powder factor was 0.7 lbs/cyd. This was reduced gradually until the last blast was shot at a powder factor of 0.6 lbs/cyd. Reduction of the powder factor required stemming decks to be placed in the intermediate soft layers. It was essential that this be done accurately and this need was achieved.

The first row of holes was drilled as close to the face as possible. Toe distances were quite long in some cases. However, observation of films from the first test indicated good movement at the toe and AN/FO use was continued.

Blasts were again shot at the pit utilizing 17 millisecond delays across the face and 42 millisecond delays between rows. A 200 millisecond delay was used down each hole to insure no surface cutoffs would occur. The 100 millisecond delays between the last two holes on the front rows were again used.

x 17 Ms DELAYS
 xx 42 Ms DELAYS
 xxx 100Ms DELAYS



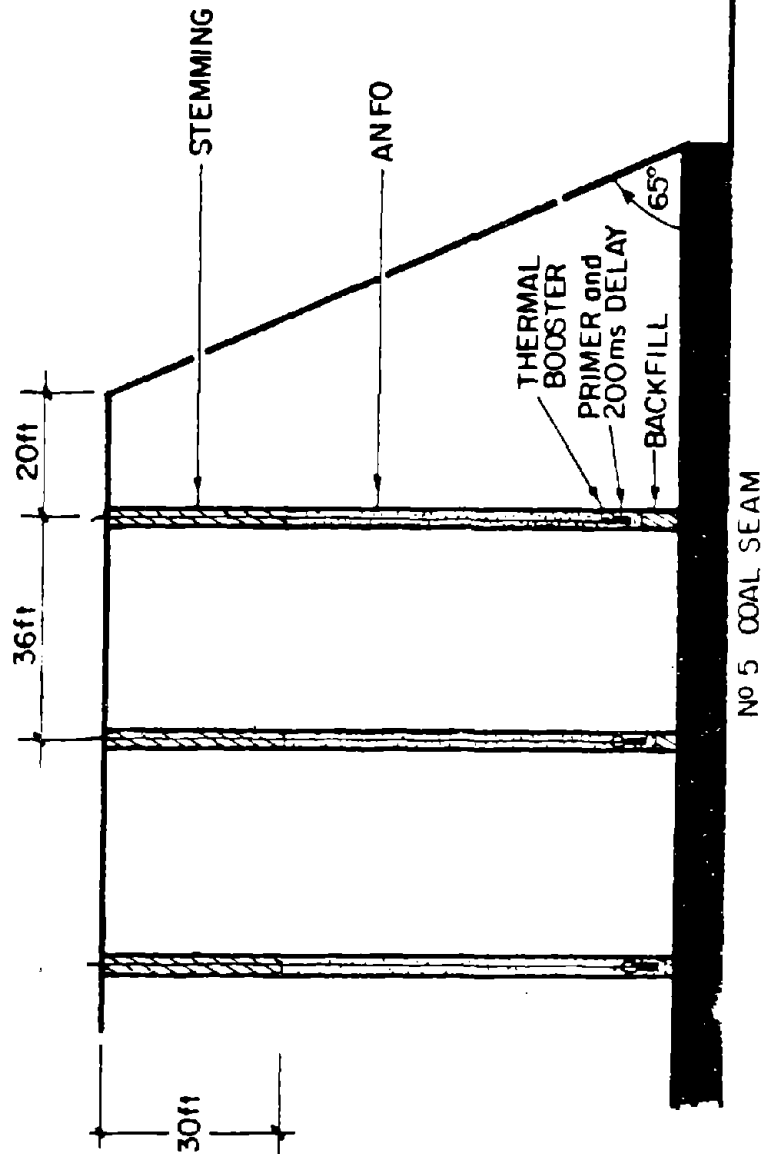
PLAN VIEW OF TEST BLAST
 NUMBER 1

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CROSS SECTION VIEW OF
TEST BLAST NUMBER 1

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One blast was shot on the diagonal, using 60 millisecond delays across the front and 17 millisecond delays between rows, to determine if an improved blast profile would occur. However, due to the restricted throw possible, fragmentation was observed to suffer and no additional blasts were tied-in in this way.

Table 3 is a summary of data for the second test program.

Figure 38 is a plan view of the blasts as finally designed and Figure 39 shows a typical cross-section.

The test blast program confirmed that muck casting, overburden fragmentation (including the limestone caprock without the use of satellite holes) and strata disruption were predictably achievable.

7.3 Test Results

A high-speed 16-mm camera was used to film most blasts. The camera was run at 400 frames per second in each case and the films studied with the use of a stop frame projector. The results of the film analyses are presented in Appendix C. Also included in the Appendix is a summary of the procedure used to perform the film analyses, tables showing the delay times, face velocities that were obtained, and a summary of the results from each individual film. The film summaries contain a blast plan showing hole numbers, delay times, firing times and delay between holes, and appropriate displacement-time graphs for the blast targets. Finally, statistical data on the delays are included.

Survey control errors gave problems in the film analysis. Surveying errors did not affect the delay time records, but did influence the face profiles and resulting face velocities. However, the velocity results reported in Appendix C are estimated to be within 15 percent.

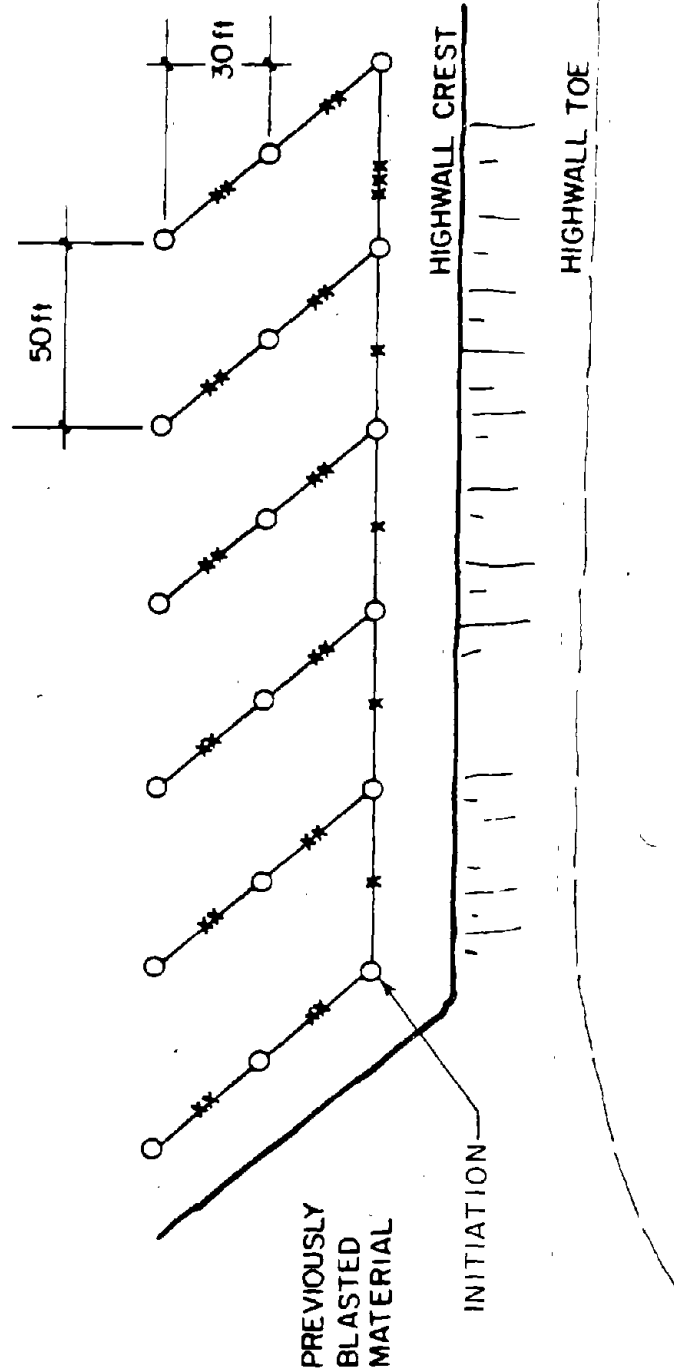
TABLE 3

DATA FOR EACH BLAST SHOT IN TEST PROGRAM 2
 OLD BEN NO. 1 BLAST DESIGN PROGRAM

Blast Number	Pattern (ft)	Collar (ft)	Powder Factor (lbs/cyd)	Front Row Delays (MS)*	Between Row Delays (MS)	Down Hole Delays (MS)
1	37 x 45	30	0.70	17	42	200
2	37 x 45	30	0.70	17	42	200
3	30 x 48	30	0.70	17	42	200
4	30 x 48	30	0.70	60	17	200
5	30 x 50	26	0.675	17	42	200
6	30 x 50	26	0.65	17	42	200
7	30 x 50	26	0.63	17	42	200
8	30 x 50	26	0.60	17	42	200

* 100 MS delay used between last two holes for each blast except No. 4.

x 17 Ms DELAYS
 xx 42 Ms DELAYS
 xxx 100 Ms DELAYS



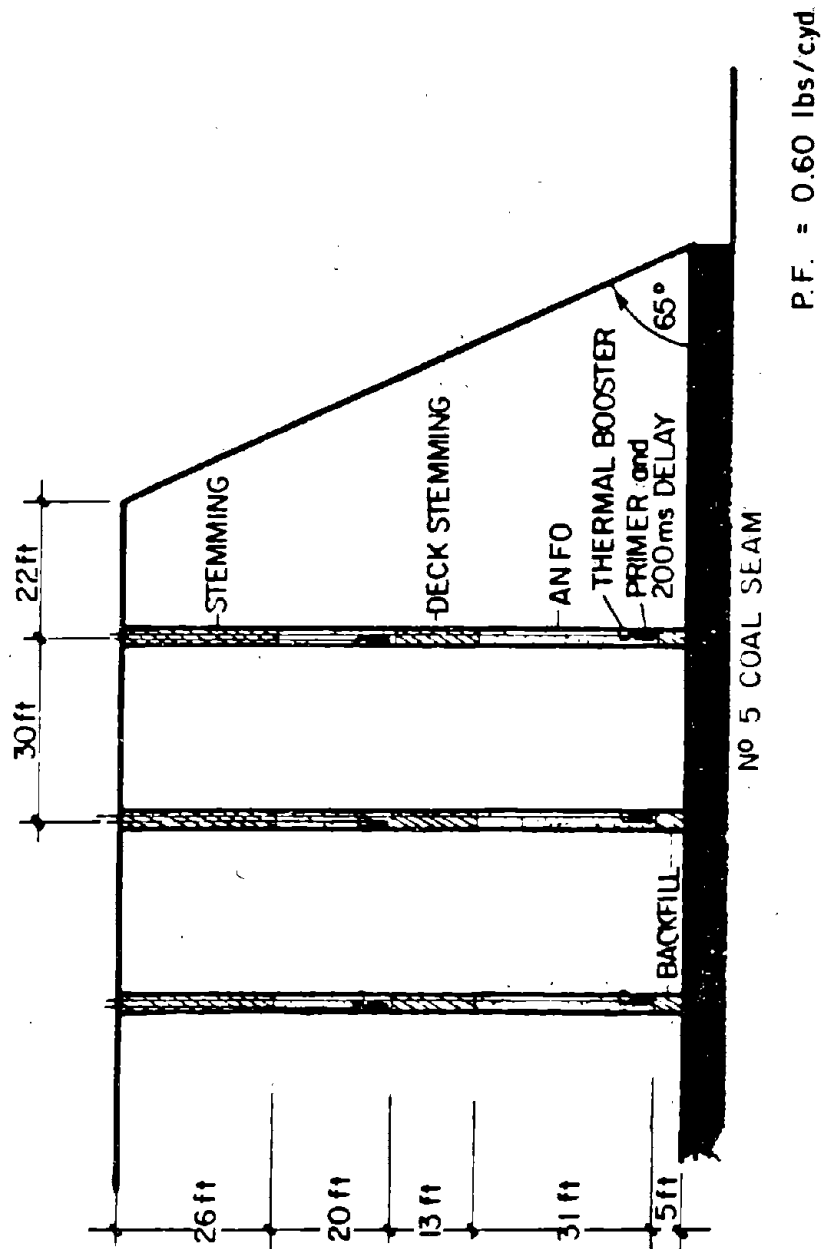
PLAN VIEW OF BLAST AS
 ULTIMATELY DESIGNED

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DRAWING NO 38 OF



CROSS SECTION VIEW OF THE FINAL BLAST DESIGN

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DRAWING NO	OF
39	

The films indicated good toe displacement in most cases. The material at the toe was observed to be ejected as much as 75 percent of the way across the pit floor, with waste from the body of the shot subsequently being cast over this first-moved portion of the blast.

The bench top was seen on the films to lift considerably more for the 26-foot collar loading. This correlated to the observed better fragmentation. No flyrock from the top was seen for any blast, which indicated good explosive confinement. At the collar loading of 26 feet it was noted that the top material was just on the verge of cratering out and generating flyrock.

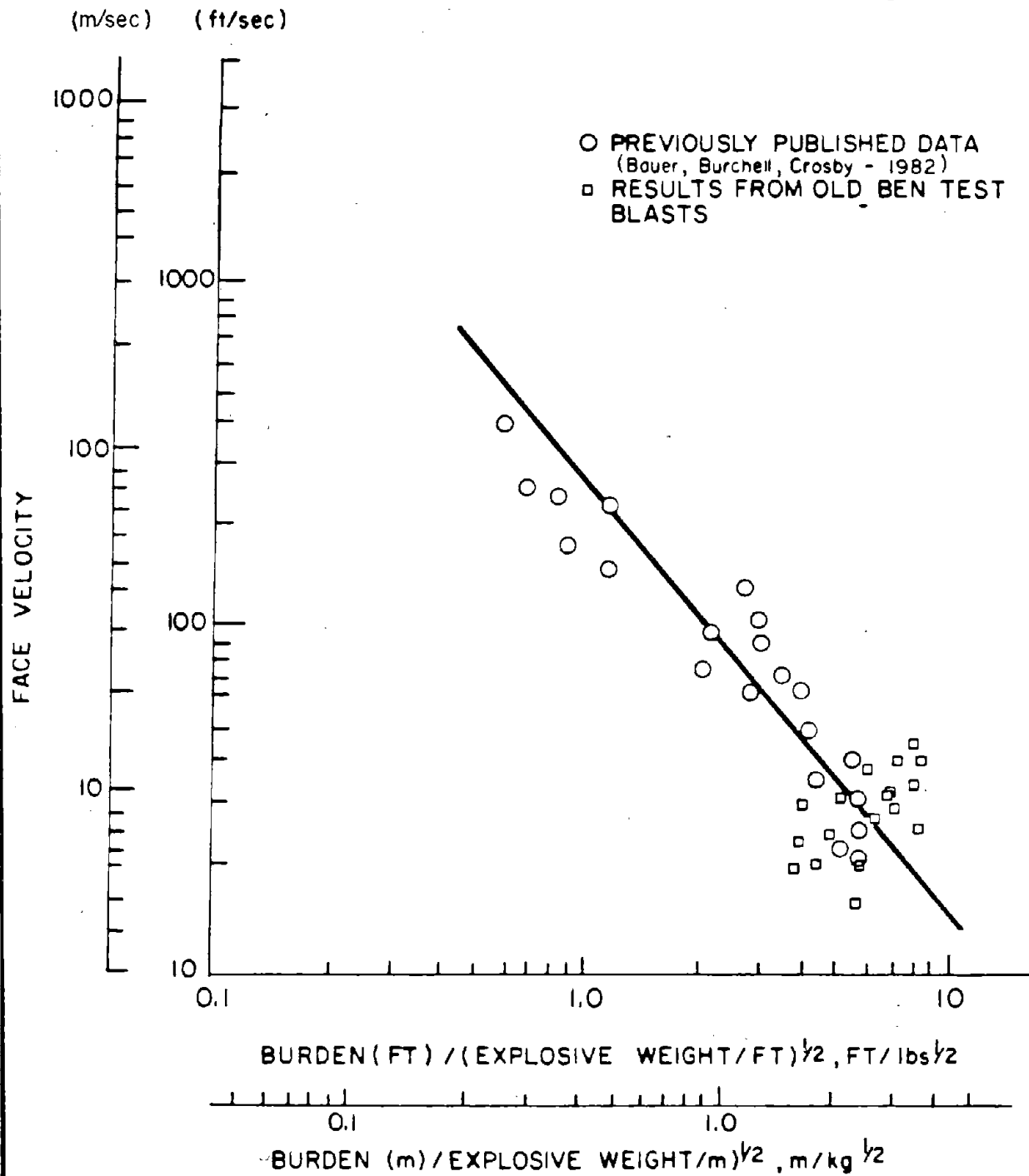
Figure 40 presents a plot of face velocity versus scaled burden. Data obtained from the test blasts have been superimposed on previously published data. Figure 40 shows that the results obtained were consistent with previous investigations. Calculations of the flyrock travel, using the observed velocities and trajectory angles, indicated a maximum range of just over 100 feet with an average of 75-80 feet. Figure 40 also illustrates the face velocities that can be experienced at lower scaled burdens, and in particular, illustrates the much higher velocity that can be obtained for the scaled burden of $2 \text{ ft/lbs}^{1/2}$ which would be used to maximize throw. Smaller scaled burdens which give even higher velocities are not recommended because of the small craters that are produced at these burdens using the large overburden drillhole sizes.

Analysis was made of the millisecond delay variation. The results were good with the holes firing in order in all cases. Some scatter about the nominal firing time was observed.

Statistical data obtained on the delay firing times are presented in Appendix C. Figure 41 presents the frequency distribution of the Nonel surface delay firing times for the 17 and 42 millisecond delays. The actual delay times were quite consistent, as anticipated, with the means being 21.7 Ms, 42.6 Ms and 95.3 Ms for the 17, 42 and 100 Ms units. The corresponding standard deviations were 2.51 Ms, 2.90 Ms, and 2.88 Ms, respectively. It should be noted that all of the blasts were double delayed which helped reduce the delay scatter. The surface delays were observed to perform as designed.

This was not true for the down-the-hole system. On a number of occasions the down-the-hole delay was observed to be very short, if not instantaneous. Using 100 Ms surface delays in certain rows gives the potential for downline cutoffs, which should be watched carefully in future work. An alternative down-the-hole system and priming system may be required.

The fragmentation was improved in all cases. Reduction of the powder factor to 0.60 lbs/cyd did not significantly affect this. Caprock fragmentation was improved, especially at a 26-foot collar, though no satellite holes were drilled. The muck piles were in a swelled, well-loosened state and it was observed that the dragline bucket had little difficulty in passing through the bank.



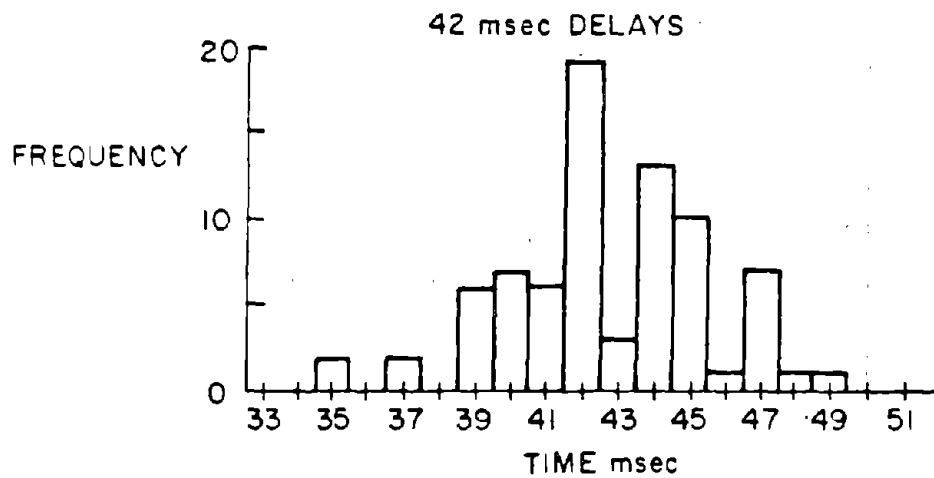
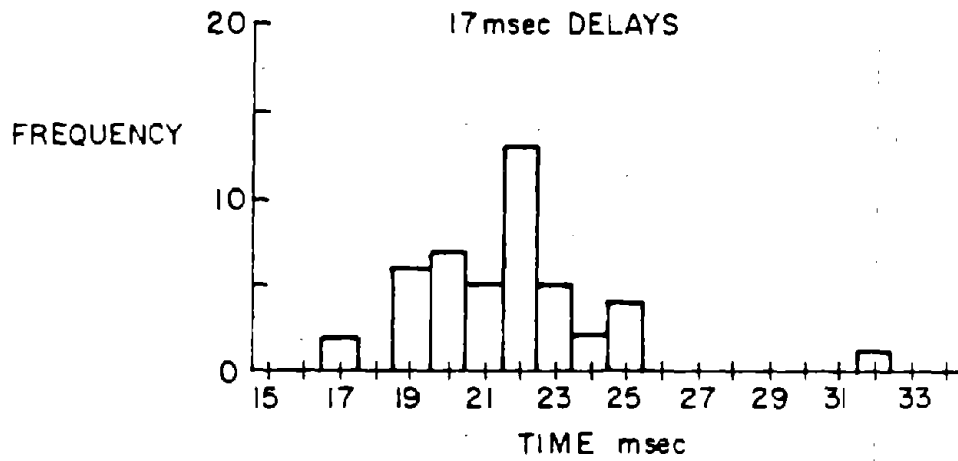
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FACE VELOCITY IN BENCH
BLASTING WITH VARIOUS
SCALED BURDENS

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DRAWING NO. 40 OF



FREQUENCY DISTRIBUTION
OF FIRING TIMES OF NONEL
SURFACE DELAYS (± 1 msec)

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41	

The bucket fill factors, as observed in the field, were good, with the bucket consistently being filled to capacity.

It was possible to cast a considerable amount of the material into the previously mined cut. The throw was fully across the pit and as much as 30 feet up the previous spoil. This resulted in a muck pile face angle of less than the angle of repose.

At the back of the blast the waste elevation dropped 25-30 feet on average. The heaped material in the center of the blast was in the range of 10-15 feet below the crest. The side wall muck at the end of the blast experienced a 15-30 foot drop.

Good throw was consistently achieved throughout the second test pass. This consistency was most easily achieved where the depth to the coal was 95-100 feet. For the portion of the test area where the strata ranged 100-110 feet in depth, the profiles were less consistent. This is likely indicative of increasing toe distance and might be alleviated by the use of a toe load of 15 percent aluminized AN/FO.

An observed result of the first test program was that the coal loss was higher than normally expected. This was due to cratering into the coal and subsequent removal by the dragline.

For the second test, 5 feet of backfill was employed in all holes. There was less loss of coal observed in this test section but some loss did occur, partially due to overlap of the last row of holes from the previous blast and partially due to the need for more backfill.

Perhaps the most important result was improved productivity. Analysis indicates that a rate increase in waste movement of 25 percent was achieved. Approximately 10 percent was due to dragline operation and 15 percent due to explosive casting of the material. (Twenty-five percent more coal was uncovered per unit of time.)

The 15 percent increase due to casting is positive, but an additional 10 percent gain may be possible with additional modifications to the system.

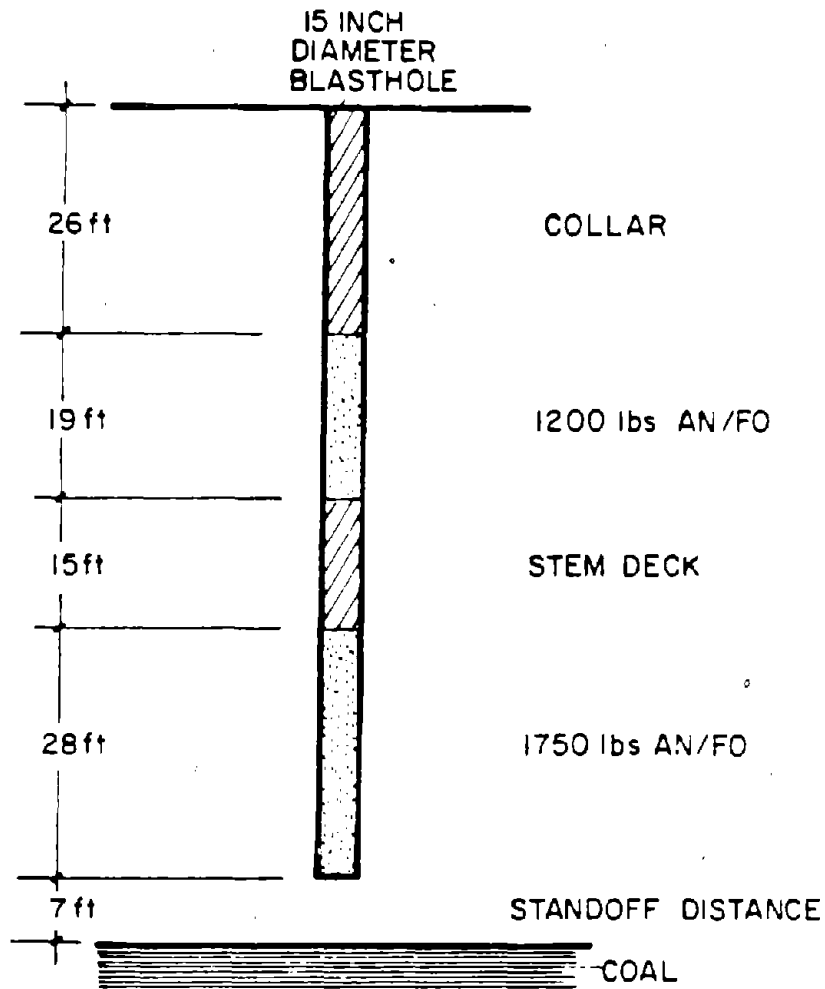
7.4 Proposed Optimum Blast Design for the Field Test Site

Based on the results of the throw and the resulting dragline productivity, it is estimated that the blast powder factor could be reduced to 0.55 lbs/cyd (1.82 cyd/lb) to give a close-to-optimum design. The other optimum blast parameters are presented in Table 4. A typical hole load is shown in Figure 42.

TABLE 4

PROPOSED OPTIMUM BLAST DESIGN PARAMETERS FOR THE TEST SITE,
OLD BEN NO. 1 COAL MINE, BLAST DESIGN PROGRAM

ITEM	VALUE
Blast Pattern	30 ft. x 50 ft.
Collar Height	26 ft.
Explosive	AN/FO
Powder Factor	0.55 lbs/cyd
Delays - Front row	17 Ms.
- Between rows	42 Ms.
- Down-the-hole	200 Ms.
Surface Tie-in	Nonel Trunkline and Delays



TYPICAL PROPOSED HOLE LOAD
FOR THE OPTIMUM BLAST DESIGN
FOR THE TEST SITE

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42	

To avoid coal loss due to blasting, more backfill is needed. This should be increased to 7 feet for each hole. At this level, in a 15-inch hole, the coal loss problem may be eliminated.

Additional casting is possible. Locating the first row closer to the face by changing the method of drill operation and using AL/AN/FO in the toe would help accomplish this. Longer delay times may also help in this regard. Such additional casting must be weighed against further increases in grade building.

A better highwall could be formed by presplitting the block to be shot. This can be effected with large diameter holes and a toe load of explosive. The holes are not stemmed and are fired prior to the production blast. Several operations using this technique in competent sandstone and shales have reported good results.

A possible improvement to the method at Old Ben would be to reduce the dragline elevation and shoot a dragline tail room blast. A drop in elevation of 30 feet would seem appropriate. This approach has been developed elsewhere and is described in Section 5.

Highwall presplitting and operating bench lowering, although recognized as possible elements in an optimized Old Ben casting program, were not field tested.

8.0 DISCOUNTED CASH FLOW ANALYSIS, OLD BEN NO. 1 COAL MINE WEST FIELD BLAST DESIGN PROGRAM

3.1 Introduction

The following Discounted Cash Flow (DCF) analyses are predicated on a number of test result "typical" approximations. These estimating efforts were required both because some pertinent data were unrecorded in the field and other recorded data, upon analysis, exhibited questionable accuracy. These difficulties arose largely as a result of the limited nature of the field program. The project's execution exemplified the learning curve theory of performance. Suggestions pertaining to improved data compilation approaches for use in similar blast design programs are made throughout this section.

Equipment identifications in the analyses accurately represent the Old Ben production fleet. Historical West Field availabilities and operating costs were used when available. When unavailable, derated manufacturers estimates were used. All analyses begin from common equipment, manpower and wage assumptions.

The stated purpose of the field test was twofold: (1) to investigate blast designs to improve dragline stripping rates; and (2) to investigate the coal production cycle and cost effects of the blast designs. The program concept was to develop blast design procedures which could result in improved dragline productivity and commensurate coal production

increases. Changes in blast design and extraction cycle were to use the equipment and personnel existing at the chosen test site. Interpretation of the recovered data from the field tests could, where warranted, suggest equipment and/or techniques that were not in use at the test site at the time of the field testing.

Within the context of the field test, an "optimized" blasting and production procedure was developed and compared, in a time value of money format, to the historical production sequence at the mine. These comparisons are after tax discounted cash flow internal rate of return analyses.

8.2 Summary Comparison of Discounted Cash Flow Analyses

The historical operating procedure and field test results were developed as three mutually exclusive extraction programs. These were:

- | | |
|------------|---|
| OPTION A - | The historical operating condition in the Old Ben West Field. |
| OPTION B - | Optimum blast design with associated overburden casting. Production level equal to historical practice. Dragline sizing at 0.75 of Option A due to materials cast and cycle improvements. Dozer highwall filling required. |
| OPTION C - | Optimum blast design with associated overburden casting. Dragline sized at 1.00 of Option A. Coal production increased to 1.25 of Option A due to beneficial effects of casting and cycle adjustments. Dozer highwall filling required. |

These programs were operated on a fixed reserve equal to 20 years of mining at the historical rate of 980,000 tons of raw coal per year. The results of the analyses are as follows:

TABLE 5

DISCOUNTED CASH FLOW RESULTS
OLD BEN NO. 1 COAL MINE, BLAST DESIGN PROGRAM

OPTION	INITIAL CAPITAL COST*	PROD. RATE. (TPY)	YEARS OF MINING REQUIRED	\$/TON WASHED COAL*	IRR*
A	\$53,000,000	980,000	20	\$28.12	9.1%
B	\$41,432,000	980,000	20	\$26.45	14.5%
C	\$53,457,000	1,225,000	16	\$25.75	14.5%

* Based on 1982 new equipment prices and wage rates, and a \$36/ton washed coal sale price.

The Old Ben operation exemplified the "stripping dragline constraint on production" circumstance which premised mine site selection. The reduced muck handling and observed dragline cycle improvements due to the blast redesigns, manifested as 25 percent more daily coal exposure, was fully exploitable with the existing production, reclamation, and preparation equipment (Option C).

The reduced materials handling and dragline cycle improvements of the "casting" technique, if employed at the historical production level of 980,000 tpy, would under-utilize the existing stripping dragline. The historical production rate improved blast design method developed in Option B assumed the replacement of the existing dragline with the next smallest model available from the manufacturer.

Additional improvements of the blast designs may be possible and, with resizing of some ancilliary equipment, further production increases and/or cost savings attainable. Final determination of the applicability of the technique at the Old Ben No. 1 Mine would not be possible until better clay fill stability, tub maintenance cost, and coal chilling data are developed.

8.3 DCF Elements and Assumptions

The test blast field program was limited to a pit area of approximately 2000 x 200 feet. This cut was accomplished in two 100-foot-wide passes of the dragline. The drilling and blasting for the program was undertaken on the bench 2-5 days prior to the stripping dragline activity. The two-pass program included seven separate shots (15-23 holes each) in Test No. 1, and eight shots (15-18 holes each) in Test No. 2. A period of approximately 10 days existed between the conclusion of Test No. 1 and the initiation of Test No. 2.

While a visible evaluation of muck fragmentation, highwall drop, and materials cast could be made immediately after a blast, other blast effects, such as backbreak, highwall stability, and coal condition were not assessable until the stripping dragline had excavated the muckpile.

Concurrent with the blast assessment-design activities was a production cycle effect monitoring and field methods adjustment program. Many of the machine operator personnel involved in the program implementation were 20-year men, i.e., adapted to historical methods and not inclined to restructure existing procedures. The partial adherence to existing methods skewed field generated data away from a pure test of the submitted blast and cycle designs. These data impacts were most pronounced in Test No. 1. Generally, Test No. 2 was more successful in yielding meaningful design oriented data.

Existing equipment hourly operating capacities and operating cost baselines were common throughout the analyses. Common baseline exceptions included dozer operating costs in the "hard" reclamation duty of Options B and C, increased job efficiency in the coal washing plant in Option C, and a dragline maintenance cost proportional to the increased swing counts of Options B and C. These modifications to the common bases were developed to depict, as accurately as possible, the practical response of the operation to the increased dozing, mining, and coal washing loads.

Twenty-year extrapolations based on limited field observation necessitated the establishment of boundary conditions. Conditions outside these boundaries were not addressed. The DCF analyses are not taken to represent the economics of a mining program in the Old Ben No. 1 Coal Company's West Field which has overburden and ground water conditions different from those encountered on the test bench. Rather, they are economic analyses and comparisons of test bench conditions and costs versus historical (year-to-date 1981) costs. It was deemed reasonable to limit this DCF comparative study to observed test results and documented history from adjacent, and presumed similar, strip mine cuts.

The DCF design data elements, with appropriate background discussions, are as follows:

- (1) The Size of Reserve and Annual Production Rate. The existing equipment in the West Field produces approximately 3200 tons/day. The coal mining operations run 6 days/week, 1 shift/day. An estimate of 980,000 TPY, 19.6 million tons in 20 years, was used in the fixed production DCF's. A 90 percent mining recovery factor was used in all DCF's.

In the casting DCF's, a coal recovery estimate of less than the historical 90 percent was visually assessed during the field test program. The quantification of coal losses attributable to the blasting techniques utilized in the program was not possible due to inaccuracies inherent in the original estimates of coal reserve in the 2000 x 200-foot area underlying the test blast bench. The coal observed to be lost was both the result of "spoiling", i.e. dragline excavated and placed onto the spoils pile, as well as intact coal seam left under the leading edge of the finished spoils pile. The spoiled coal was largely the result of coal "chilling" (shattering or pulverization) due to blasting. The chilled coal, having suffered structural damage, was soft enough for the dragline bucket teeth to penetrate during the final, or coal cleaning, portion of the stripping cycle. The problem might be reduced in either of two ways: (1) to station an observer in the pit to direct the dragline operator when coal cleaning was undertaken, or (2) to increase the bottom stemming (fill placed in the bottom of the drill hole to act as a buffer) from the 5 feet placed during the program (Test No. 2) to 7 feet. The latter approach to solving the chilling-spoiling portion of the observed coal losses was incorporated in the "optimum" blast design utilized in the DCF determinations.

The observation of coal seam at the base of the spoils pile reflected two problems. The stripping dragline operator, because of the explosive casting of muck across the pit, lost definite knowledge of where the far edge of the coal lay. The operator would begin a spoils pile at a distance estimated to allow for a full height spoils pile with its pit-side toe terminating at the base of the intact coal seam. In some cases, the spoils pile positioning was so poor that considerable rehandling of spoils material would still not expose the coal edge. An attempt was made to quantify the coal lost in this manner through pit surveys. These surveys indicated considerable lengths of coal edge loss, possibly up to 10 percent of the reserve in the area (pit surveys in the test area were later found questionable and coal loss projections from either the original or revised

[resurveyed] pit surveys were not accepted as representative of the field condition). A second factor involved in the coal edge loss observations was the physical limitation of the operating radius of the stripping dragline. The range diagrams developed for the project did not indicate any pit excavation difficulty, but the field observations of the as-developed pit suggested reach difficulty. These observations were further reinforced by pit width measurements, which, in some areas, were 20-40 percent wider than the original design. This wide pit-lost coal situation reflected two problems: (1) a backbreak from the blast program which was not immediately recognized; and (2) two shifts of stripping activity problems apparently related dragline positioning controls.

A solution to coal-edge loss lies in operator familiarity and resultant skills development in estimating the spoil pile to coal edge relationship. Range diagrams indicate operator skill can solve the problem. The blast backbreak and excessive pit widths are solvable by proper blast design (tighter row spacing to achieve a design pit width) and front row hole positioning (as close to the highwall crest as safety permits). The surveying control error was an isolated incident which, while contributing to the impression of excessive coal loss during the program, is not a flaw inherent to the blast technique or resultant production cycle.

The "optimum" blast designs will, in our judgment, with additional development and field shakedown, yield a coal recovery equal to the historical 90 percent.

- (2) Geology. The test bench area had overburdens of 95-125 feet pre-reclamation scraping of topsoil. The thicker overburden had remnants of the near-surface (15-25 feet from the surface) No. 6 Coal Seam. Extraction of this seam reduced the drilling bench level to 105± feet.

The No. 5 Coal seam, the principle deep-lying reserve in the area tested, tended to be horizontal with minor undulations. Little ground water was noted in either the pit or the drill holes on the bench.

The above described overburden, No. 5 and No. 6 Coal occurrences and ground water conditions were common to all the DCF analyses. See Appendix B for further descriptive geology.

- (3) Overburden Swell. Overburden swell is an important element in the casting analyses which was not accurately documented by the field surveying effort. Muckpile surveys were designed to section the centerline and sideline of individual blasts. The surveying activities were undertaken daily, with data stored

for subsequent analysis. The surveying data, upon reduction, was found to be consistently "long" (that is, the plotted data showed the muckpile toes running 10-20 feet beneath the spoils). This survey data, presumed to be the authoritative check for swell factor, was discarded. Field note observations of the individual blast highwall drop depths, muckpile angles of repose, and muckpile toe run-ups onto the spoils were averaged to yield a "typical" cross section. The resultant 1.25 "typical" swell factor fit a pre-program estimate of a 1.2-1.3 swell factor range. The 1.25 swell factor is identical to the historical swell factor observed at the mine.

Knowledge of the swell factor is important in understanding the incremental contribution to improved dragline efficiency made by various blast designs. The swell factor can change the volumetric requirement of the stripping machinery. Other elements such as diggability, fragmentation, and bucket fill factor being constant, an overburden swell of 1.30 would cause a 4 percent volumetric increase over a 1.25 swell with a proportional increase in the number of buckets required to expose a given area of coal. Conversely, a 1.20 swell factor would decrease the required number of swings by 4 percent against the 1.25 swell factor standard. The blast program swell factor of 1.25 and the historical swell factor of 1.25, plus the observed equality in bucket fill factors, led to a zero swell factor influence on the coal uncovering rate.

The percentage of material explosively "cast" into a position not requiring dragline handling was derived from the subjective (visual observation and estimation) profiling methodology. For a detailed assessment of this factor in similar undertakings, extensive sectioning and muckpile contouring surveying efforts should be undertaken following each blast. This surveying effort should be run for a number of individual blasts using identical powder factors, hole spacing, delay sequencing, etc. This repetition would allow for single blast aberrations, such as local geology or delay detonation vagaries, to be averaged. Single shot analysis for any given blast design, even with full and excellently implemented survey coverage, would be suspect for the above noted reason. The limited number of field test blasts undertaken in this program and the constant redesign of blasts to better ascertain technique trends lent itself to misinterpretations of this "single" event type. Averaging of the 15-blast sequence was considered as an appropriate method of depicting the generalized trend of the blast results.

- (4) Equipment Availability and Productive Activity Rating. Historical records for coal production in the Old Ben West Field accurately fix the equipment availability for drills, dozers and stripping dragline. Dragline productive activities are further documented as to number of cycles, time per cycle, down time causes and average swing angles. This body of historical information was used for the existing practice DCF analysis. An established dragline information recording system was in force during the test program and was used as the basis for the test program DCF extrapolations.

Equipment ancilliary to the drilling-blasting-stripping activities; i.e., reclamation, coal loading and transportation, and coal washing, was noted for size and historical availability. This information, combined with derated manufacturers performance recommendations and equipment operator observations was the basis for the operating-hour estimates used in the DCFs.

The drill performance was based on documented historical performance.

- (5) Equipment Operating Costs. For all equipment there existed either a historical or manufacturers guideline on operating costs. Manufacturers recommendations were subject to additional derating. Historical costs for the dragline were well documented.

The power consumption of the dragline was unsuccessfully monitored. The problems encountered included machine vibration leading to monitoring device failures, dust conditions in the monitoring area leading to recording pen failures, and the inaccessibility of the monitoring equipment for repairs except at a prescribed "no movement" interval on each shift. The operators stated that the muck dug similarly to the historical muck. Dragline power costs were estimated to be equal to the historical condition.

Cable wear could be a reduced cost under the blast design program. This would result from cable drag through the clay topping of the working bench as opposed to the normal shale and limestone topping. This potential maintenance benefit was not quantifiable in the short duration of the test program, hence, no cost savings were included in the DCF analyses.

A potentially influential cost element not defined during the field test program was the highwall fill working bench effect on dragline tub maintenance. The clay fill tended to deform, to compact as a slope towards the pit, during the excavation process, resulting in machine operation off the horizontal. The dragline initially (Test No. 1) walked back from its

operating position to place shale on the bench, allowed a dozer to level and smooth the pad, and to walk back onto the pad to resume operations. The indication of when to re-level the working bench was usually initiated by the dragline maintenance staff because of unusual sounds coming from the tub. In Test No. 2 it became standard practice to place a 5 to 10-foot cover of shale on the clay fill to stabilize the working bench. This technique was observed to improve the stability of the working bench. Mine personnel were doubtful that stability would be maintained in wet weather conditions. Tub failure could impact substantially the economics of the blast design program. The adequacy of the clay fill as a machine support, even with a shale cover, remained an unresolved concern. The DCF analysis for the maintenance cost of the dragline was assumed equal to the historical cost.

More detailed machine operating cost breakdown, with underlying assumptions, can be found in Appendix D.

- (6) Equipment Capacities. The existing drill, coal shovel, haulage, washing and reclamation equipment has additional capacity, either as additional shift time availability or under-utilization on the present shift schedule, to support the daily production of more coal. This situation, as noted in the introduction to this report, is common throughout the domestic strip coal mining industry.

A piece by piece equipment review of the non-dragline equipment at Old Ben indicated that a 1,225,000 ton/yr production level could be handled by the existing equipment without any increase in the number of shifts worked.

The increased wash plant utilization was one important assumption based primarily on operating personnel consultations.

- (7) Equipment Depreciation and Ownership Cost

Ownership cost for the equipment was taken to be:

$$\frac{(\text{Interest} + \text{Insurance} + \text{Taxes}) \times .5 \times \text{Equipment Capital Cost}}{\text{Depreciation Hours}}$$

Interest, either as a financing charge or as the opportunity cost of capital expended, was taken to be 16 percent (reflecting the prevailing interest rate in the spring of 1982), and both the Indiana State sales tax and insurance were estimated at 2 percent.

This formula or derivations very similar to it are common industry guides to the ownership costs for new equipment.

All equipment was straight line depreciated over a manufacturers recommended "average working condition" equipment life.

8.4 Other Blast Design Program Notes

Drilling and loading, dozing, dragline stripping, and coal extraction activities were undertaken on a three-shift basis in the test area. Blasting was limited to the daylight hours only.

The field procedure was to review a day's activities with Old Ben's blasting personnel and management at the conclusion of a day's shooting. The reviews were generally limited to discussions of cast, fragmentation, seismic and air blast recordings, and apparent backbreak. In the latter portion of the test passes, as the dragline advanced and the situation could be observed, discussions on coal recovery were included in the daily review sessions.

The drilling, loading, wiring and detonation activities during the daylight hours were comprehensively observed and field-note recorded. The physical status of each drill hole, both geologically and explosives procedurally, was used to: (1) possibly correlate blasting result aberrations to specific hole preparation problems, or (2) suggest modification to existing procedures. The bench monitoring yielded wet hole data, drilled coal incidents, and collaring misdeterminations which carried through and were identifiable in the blast results. New bench procedures for the operation were not developed during the field test effort.

Wet holes were isolated occurrences with less than 2 percent of the holes drilled during the tests being so classified. The wet holes presented the greatest difficulty for the historical technique. Pre-packaged AN/FO was dropped down the hole with poor knowledge of the continuity of the deck (bag hang-ups) or the integrity of the bag (splitting, etc.). The few wet holes which were shot did not seem to fragment or move the muck with the consistency of the dry holes. Driliner options could be one indicated bench technique improvement.

The dragline cycle was observed both night and day on a spot basis. Individual operator habits and resultant cycle element time study differences were apparent. A cycle observation included timing the muckpile dig, swing-hoist, dump, and swing return times, a bucket fill factor estimate, and other notables such as key cut positioning, clean coal function, etc. The cumulative results of swing records taken from the stripping dragline showed a marginal time-advantage for the blast program mucking cycle in spite of substantial lost time for coal edge uncovering activity throughout the test period. See Section 8.3 (1) for a description.

The dozing effort was surveyed for volumetric movement and monitored for operating hours. Twenty-four hour vigils from remote locations were employed as one means of determining the actual earth moving hours/shift versus the daily mine record. These observations yielded a nominal six-hour/shift active dozing effort. See Figure A-22, Appendix A, for plan and section views of dozer activity in Test 2.

Power consumption monitoring devices were placed on the drill rig, the stripping dragline, and the pullback dragline. The recorders were designed to yield power factor, instantaneous kilowatt hours demand, and 15 minute cumulative kilowatt hours demand. The recording instruments were checked twice daily for continuity of data retrieval. None of the recorders provided consistent records. The principle causes of failure were machine vibration and dust infiltration. The repeated pen and instrument failures ruled out the power consumption record keeping in this fashion.

Field surveying was undertaken on a daily basis to determine: (1) muckpile configuration, including toe location, for both the blast centerline and, on selected occasions, the blast endlines; (2) overburden movements (dozing); (3) overburden movements (stripping dragline); (4) high speed camera location (on spoils pile opposite the blasts); (5) seismograph locations; (6) highwall crest and toe locations; (7) spot checks on the angle of repose of spoils; and (8) spot checks on the pre-dragline stripping, post-bench filling and leveling, and highwall crest. The surveying data were compiled, but not reduced, during the test program. Subsequent data reduction efforts disclosed inconsistencies in the pit location, muckpile profiles, and overburden volume records. Part of the error was traced to an improperly located benchmark used at the mine. Additional error was traced to an improperly aligned grid system used for referencing other stations used in the surveying effort, and, in a few instances, erroneous transit readings were recognized and discarded. Final results were only approximately in accord with field notes from paced distances for pit width and backbreak, highwall drop estimations, etc. The surveying work was utilized for site plan illustrations and dozer-overburden movement determinations, but was discarded for muckpile profiles. Where used, the surveying data were taken only as a best available approximation of the field effort.

The clay-like No. 6 Coal overburden which was used as "highwall drop" bench fill was the first material excavated by the stripping dragline when coming into an area. Previous to the test program cycle, the No. 6 Coal overburden was "chopped" as the last item prior to dragline advance. The historical sequence allowed this softer material to be placed on the very top of a completed spoils pile. This topping approach was preferred by the reclamation dozer operators to the test program spoils piles which were topped with shale and limestone since these materials were more difficult to bulldoze to grade.

During the test program the dragline lacked an "out of bench" digging option. Historically, when dust would obscure the dragline operators vision of bench materials, the machine would rotate to the No. 6 overburden, take a bucket, spoil and return to bench excavation when the dust had cleared. The lack of No. 6 overburden, it having been 100 percent dozed onto the dropped highwall for a 200-foot distance back from the dragline centerline, led to periods of "no movement" while the operator waited for the dust to clear from the bench working area.

As originally designed for the program, a 15-foot stripping dragline offset away from the spoils was to be compensated for by adjustment in the pullback dragline elevation. A wider than expected pit, plus difficulty in "estimating" the required pullback dragline position, led to reach shortfalls. This situation was remedied by highwall reconstructions to move the BE 2570 closer to the spoils. This bench rebuilding approach required increased dozing effort.

The magnitude of the material movement was also larger than in the original design because of highwall drops greater than expected.

8.5 Field Test Program Tabulated Results and DCF Calculations

The following tabulations provide a summary of the field test program results, historical operating conditions, and 20-year DCF extrapolations.

TABLE 6

BE2570 STRIPPING DRAGLINE FIELD TEST AND HISTORICAL DATA

HISTORICAL (Monthly Average 1981 prior to August 1, 1981)

Yardage/Month	2,246,337
Total Swings/Month	24,233
Cubic Yards/Bucket	92.7
Seconds/Swing, overburden removal	67
Seconds/Swing bench removal	98
Seconds/Swing bench fill	104
Average Swing Angle, overburden removal	75°
Average Swing Angle, bench removal	187
Average Swing Angle, bench fill	154
% of Operating Time on overburden removal	70
% of Operating Time on bench removal	23
% of Operating Time on bench fill	6
Cubic Yards/Digging Hour	4503

TEST PERIOD STRIPPING (September 5, to 19, 1981; October 5-20, 1981)

Total Yardage Moved to Spoils	2,230,000
Yardage Moved by Dragline*	1,873,000
Total Swings*	20,250
Cubic Yards/Bucket**	92.7
Dozing (BCY; avg. Push Distance = 189 ft)****	271,000
Seconds/Swing, overburden removal	65.5
Seconds/Swing, bench removal	83.2
Seconds/Swing, bench fill	84.0
Average Swing Angle, overburden removal	76.9
Average Swing Angle, bench removal	129.7
Average Swing Angle, bench fill	137.5
% of Operating Time, overburden removal	90
% of Operating Time, bench removal	4
% of Operating Time, bench fill	6
Cubic Yards/Digging Hour (exc. cast)***	4974

* Estimated due to volumetric and dragline positioning surveying errors.

** Assumed fill factor equal to historic.

*** Full cycle productivity analysis, increased moving and maintenance time proportional to increased swing rate must be considered; see Table 8.

**** Extrapolated for 15 shots from Test Pass 2 data.

TABLE 7

DISCOUNTED CASH FLOW ANALYSES
OPTIONS
OLD BEN NO. 1 COAL MINE BLAST DESIGN PROGRAM

OPTION A - The historical operating condition in the Old Ben West Field

OPTION B - Optimum blast design with associated overburden casting.

Production level equal to historical practice. Dragline sizing at 0.75 of Option A due to materials cost and cycle improvements. Dozer highwall filling required. See Figure A-23, Appendix A.

OPTION C - Optimum blast design with associated overburden casting.

Dragline sized at 1.00 of Option A. Coal production increased to 1.25 that of Option A due to beneficial effects of casting and cycle adjustments. Dozer highwall filling required. See Figure A-23, Appendix A.

TABLE 8

DISCOUNTED CASH FLOW ANALYSES
DESIGN DATA
OLD BEN NO. 1 COAL MINE BLAST DESIGN PROGRAM

ITEM	OPTION		
	A	B	C
Size of Reserve (MT)	20.6	20.6	20.6
Coal Thickness (ft)	8	8	8
Coal Dip	Horizontal		
OB Thickness (ft)	114	114	114
Strip Ratio (BCY/ton)	16.8	16.8	16.8
Overburden Type	Shale,	Sandstone,	Limestone
Coal Density (lb/ft ³)	62.8	62.8	62.8
Overburden Swell Factor	1.25	1.25	1.25
Annual Production (TPY)	.98	.98	1.225
Per Cent Recovery (%) (Mining)	90	90	90
Per Cent Recovery (%) (Washing)	85	85	85
Annual Stripping Days	364	364	364
Stripping Shifts/Day	3	3	3
Annual Coal Loading Days	312	312	312
West Field Coal Loading Shifts/Day	1	1	1
OB Drilling Days/Year	312	312	312
OB Drilling Shifts/Day	2	2	2
Strip Equipment Availability (%)	70*	69**	67**
Strip Equipment Productive Activity (min/day)	1008	994	970
Dozer Availability (%)	70	70	70
Topsoil Stripping Depth (ft)	3	3	3
Casting Factor (%)	0	16	16
Type of Blast	Fragmentation	Cast	Cast
Blast Configuration	36' x 48'	30' x 50'	30' x 50'
Satellites	Yes	No	No

* Historical

** Machine availability based on increased swing count and repair difficulty estimates.

TABLE 9

DISCOUNTED CASH FLOW ANALYSES
 DRAGLINE PERFORMANCE DATA
 (Based on Test Bench Performance)
 OLD BEN NO. 1 COAL MINE BLAST DESIGN PROGRAM

<u>ITEM</u>	<u>OPTION</u>		
	A	B**	C**
OB Removal Min/Day	670	849	812
Bench Fill Min/Day	64	58	58
Bench Removal* Min/Day	219	29	28
OB Removal Swing Time (sec)	67	65	65
Bench Fill Swing Time (sec)	104	84	84
Bench Removal Swing Time (sec)	98	83	83
Moving Min/Day	58	58	72
BCY/Bucket	72	55	72
Yearly Coal Production (10^6 tons/year)	0.98	0.98**	1.225**

* No. 6 Overburden "Chop Down"

** 16% of overburden throw blasted into spoiled condition

TABLE 10

DISCOUNTED CASH FLOW ANALYSES
BLASTING CONSUMABLES COST ESTIMATES
OLD BEN NO. 1 COAL MINE BLAST DESIGN PROGRAM

PRODUCTION LEVEL: Option A, 980,000 tpy
Option B, 980,000 tpy
Option C, 1,225,000 tpy

COAL RECOVERY: 90% All Options

COAL RESERVE: 20,600,000 tons

DRILLING: 750 ft of 15" Ø drillhole/8-hour shift;
2 shifts/day, 312 days/yr

BLASTHOLE PATTERNS: Option A, 36' x 48' with 30' satellite holes between rows
Option B, 30' x 50'; no satellites
Option C, 30' x 50'; no satellites

BLASTHOLE LOADING SUMMARY POWDER FACTORS: Option A, 3.0 BCY/lb explosive
Option B, 1.8 BCY/lb explosive
Option C, 1.8 BCY/lb explosive

EXPLOSIVE & ACCESSORY SUPPLY COSTS

c) AN/FO - Pre-Oiled	\$170/ton	\$0.085/lb
d) Dow Thermal Primer	\$ 40.50/100 lb	\$8.10/20 lb
e) Cast Primer		\$2.55/each
f) Downhole Delay		\$1.10/each
g) Primacord - 40 grain/foot	\$ 69.20/1000ft	\$0.069/ft
h) Nonel Surface Delay		\$2.50/each
i) Nonel Trunk line	\$ 16.50/500 ft	\$0.033/ft

TABLE 10
(Continued)

OPTION A
(95 FT HOLE)

ITEM	NO.	COST REGULAR HOLE	NO.	COST SATELLITE HOLE	TOTAL COST
AN/FO	18381b	\$156.23	3001b	\$14.17	\$170.40
Thermal Primer	3	24.30	1	4.50	28.80
Cast Primer	3	7.65	1	1.42	9.07
Downhole Delay	3	3.30	1	0.61	3.91
Primacord	90 ft	6.21	30 ft	1.15	7.36
Surface Delay	2	5.00	1	1.39	6.39
Nonel Trunkline	60 ft	1.98	20 ft	0.37	2.35
TOTAL PER HOLE					\$228.28

Note: The costs entered for the satellite hole are 5/9 of the actual cost since only 5/9 satellite holes are required per regular blasthole.

The cost per BCY is $\frac{\$228.28}{6080 \text{ BCY}} = \underline{\underline{\$0.038/\text{BCY}}}$

Total Blast
Consumables Cost/yr = $\left(\frac{980,000 \text{ tpy}}{.90 \text{ recovery}}\right) \left(\frac{16.8 \text{ BCY}}{\text{ton}}\right) \left(\frac{\$0.038}{\text{BCY}}\right) \approx \$695,000$

TABLE 10
(Continued)

OPTIONS B AND C
(95 FT HOLE)

ITEM		PROPOSED
An/FO - 2900 lb.hole	@ 0.085/lb	\$246.50
Thermal Primer	2 @ \$8.10 ea.	16.20
Cast Primer	2 @ \$2.55 ea.	5.10
Downhole Delay	2 @ \$1.10 ea.	2.20
Primacord	2 lines/hole-140 ft	9.66
Surface Delays	2 @ \$2.50 ea.	5.00
Nonel Trunkline	60-feet/hole including lead-in line	<u>1.98</u>
	TOTAL/HOLE	\$286.64

The Cost per BCY is $\frac{\$286.64}{5278\text{BCY}} = \$0.054/\text{BCY}$

OPTION B

$$\text{Total Blast Consumables /Yr} = \left(\frac{980,000 \text{ TPY}}{.90 \text{ recovery}} \right) \left(\frac{16.8 \text{ BCY}}{\text{ton}} \right) \left(\frac{\$.054}{\text{BCY}} \right) \approx \$985,000$$

OPTION C

$$\text{Total Blast Consumables/Yr} = \left(\frac{1,225,000 \text{ TPY}}{.90 \text{ recovery}} \right) \left(\frac{16.8\text{BCY}}{\text{ton}} \right) \left(\frac{\$.054}{\text{BCY}} \right) \approx \$1,235,000$$

TABLE 11

DISCOUNTED CASH FLOW ANALYSES
CAPITAL INVESTMENT AND DEPRECIATION SUMMARY ESTIMATES
OLD BEN NO. 1 COAL MINE, BLAST DESIGN PROGRAM

ITEM	NO.	CAP. COST (\$)	ST. LINE DEPRECIATION YEARS	YEARLY CHARGE TO WEST FIELD OPERATION** ($\text{\$}$)		
				OPTION		
				A	B	C
2570 Stripping Dragline	1	34,000,000	20	1,530,000		1,530,000
1570 Stripping Dragline	1	22,000,000	20		990,000	
61R Drill	1	1,400,000	10	126,000	126,000	126,000
1150 Pullback Dragline	1	8,000,000	20	360,000	360,000	360,000
erex S24 Reclamation Scraper*	1	400,000	3	55,500	55,500	55,500
Flat Allis HD-31 Ground Dozer (with stripping dragline)	1	360,000	10	32,400	32,400	32,400
Caterpillar D-9 Reclamation Dozer	1	450,000	10	40,500	40,500	40,500
Caterpillar D-9 Bench Preparation Dozer	1	450,000	4		101,250	101,250
BE 150B Coal Shovel*	1	1,500,000	20	37,500	37,500	37,500
Michigan 175 Front-end Loader*	1	250,000	5	20,750	20,750	20,750
Rimpull 120-ton Coal Haulers*	4	1,800,000	7	103,000	103,000	103,000
Lube Truck, Mechanics Truck, Welding Truck, Electricians Truck, Supply Truck	1 ea.	130,000	5	11,500	11,500	11,500
Water Truck*	1	77,000	10	3,250	3,250	3,250
Explosives Truck	1	47,000	5	8,500	8,500	8,500
Pick-up Trucks*	4	44,000	3	6,600	6,600	6,600
Crane*	1	99,000	10	4,250	4,250	4,250
Pit Pump Systems	2	18,000	10	1,500	1,500	1,500
Caterpillar 14G Grader*	1	175,000	10	7,500	7,500	7,500
Cable	-	150,000	5	27,000	27,000	27,000
Washing Plant & Load-out Facility	-	5,400,000	10	486,000	486,000	486,000
Power System (substations, connection boxes)	-	550,000	20	24,750	24,750	24,750
Explosive Storage Facility	-	16,000	20	800	800	800
Offices and Washhouse	-	44,000	20	2,200	2,200	2,200
Shop and Warehouse	-	75,000	20	3,750	3,750	3,750
Oil and Fuel Storage Facilities	-	15,000	20	800	750	750
Floodlights, Communications	-	25,000	10	2,250	2,250	2,250
Truck Scales*	-	30,000	10	1,350	1,350	1,350
Site Preparation, Roadways	-	55,000	20	2,750	2,750	2,750
Field Indirects inc., Engineering, Contingency, Overhead and Administration, and Working Capital	-	-	-	145,000	125,000	150,000
TOTALS				3,045,350	2,586,600	3,151,600

* Denotes one shift in West Field and one shift chargeable to other Old Ben activity per operating day.

** Tires excluded; prorated salvage value included, as appropriate, for unused portion of last machine in sequence; 10% salvage value used for expended machinery. Non-equipment capitalizations assumed to have zero salvage value.

TABLE 12

DISCOUNTED CASH FLOW ANALYSES
MANNING TABLE AND WAGE ESTIMATES
OLD BEN NO. 1 COAL MINE BLAST DESIGN PROGRAM

Personnel	Manpower/Option			Wages/man/day*	West Field Cost/Year/Option		
	A	B	C		A	B	C
Production							
Dragline Operator ¹	3	3	3	124	135,400	135,400	135,400
Dragline Oiler ¹	3	3	3	124	135,400	135,400	135,400
Dragline Electrician ¹	1	1	1	117	42,600	42,600	42,600
Pullback Operator ¹	3	3	3	124	135,400	135,400	135,400
Pullback Oiler ¹	3	3	3	124	135,400	135,400	135,400
Ground Dozer Operator ¹	3	3	3	117	127,800	127,800	127,800
Driller ²	2	2	2	114	71,100	71,100	71,100
Driller Helper ²	2	2	2	110	68,600	68,600	68,600
Explosives Truck Driver ²	2	2	2	114	71,100	71,100	71,100
Explosives Truck Helper ²	2	2	2	110	68,600	68,600	68,600
Bench Prep Dozer Opr. ¹	0	3	3	117	0	127,800	127,800
Reclamation Scraper Opr. ^{**2}	2	2	2	117	36,500	36,500	36,500
Reclamation Dozer Opr. ²	1	1	1	117	36,500	36,500	36,500
Coal Shovel Operator ^{**2}	2	2	2	124	38,700	38,700	38,700
Coal Shovel Oiler ^{**2}	2	2	2	120	37,400	37,400	37,400
Front-End Loader Opr. ^{**2}	2	2	2	117	36,500	36,500	36,500
Pitman ^{**2}	2	2	2	110	34,300	34,300	34,300
Truckdrivers (coal) ^{**2}	8	8	8	117	146,000	146,000	146,000
Shooter ²	2	2	2	114	71,100	71,100	71,100

TABLE 12
(Continued)

Personnel	Manpower/Option			Wages/man/day*	West Field Cost/Year/Option		
	A	B	C		A	B	C
Maintenance							
Grader Operator**2	2	2	2	117	36,500	36,500	36,500
Water Truck Operator**2	2	2	2	110	34,300	34,300	34,300
Lube Truck Driver**2	3	3	3	117	54,800	54,800	54,800
Mechanics**2	6	6	6	117	109,600	109,600	109,600
Supply Truck Driver**2	3	3	3	110	51,500	51,500	51,500
Electricians**2	3	3	3	117	54,800	54,800	54,800
Welders**2	3	3	3	117	54,800	54,800	54,800
Scale Operator**2	2	2	2	110	34,300	34,300	34,300
Utility Man**2	6	6	6	110	103,000	103,000	103,000
				Annual Salary			
Salary							
Superintendent**	1	1	1	50,000	25,000	25,000	25,000
Pit Foreman	1	1	1	38,000	38,000	38,000	38,000
Bench Foreman	3	3	3	28,000	84,000	84,000	84,000
Maintenance Superintendent**	1	1	1	36,000	18,000	18,000	18,000
Maintenance Foreman**	3	3	3	26,000	39,000	39,000	39,000
Electricians							
Superintendent**	1	1	1	36,000	18,000	18,000	18,000
Electricians Foreman**	3	3	3	26,000	39,000	39,000	39,000
Mining Engineer**	2	2	2	33,000	33,000	33,000	33,000
Office Manager**	1	1	1	26,000	13,000	13,000	13,000

Woodward-Clyde Consultants

TABLE 12
(Continued)

Personnel	Manpower/Option			Annual Salary	West Field Cost/Year/Option		
	A	B	C		A	B	C
Safety Inspector**	1	1	1	24,000	12,000	12,000	12,000
Purchasing Agent**	1	1	1	24,000	12,000	12,000	12,000
Timekeeper**	1	1	1	20,000	10,000	10,000	10,000
Bookkeeper**	1	1	1	18,000	9,000	9,000	9,000
Warehouseman**	6	6	6	18,000	54,000	54,000	54,000
Surveyors**	2	2	2	16,000	16,000	16,000	16,000
Blasting Superintendent**	1	1	1	36,000	18,000	18,000	18,000
Total					2,400,000	2,527,800	2,527,800

* Figures are averaged for shift differentials; figures include vacation set-aside.

** Denotes half-time on West Field Operation.

1 364 days/yr

2 312 days/yr

TABLE 13

DISCOUNTED CASH FLOW ANALYSES
CAPITALIZATION SCHEDULE**
OLD BEN NO. 1 COAL MINE BLAST DESIGN PROGRAM

OPTION A
(\$1000)

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Item																				
BE 2570 D/L	(34,000)									(1,260)										3,400
BE 61R Drill	(1,400)																			140
BE 1150 D/L	(8,000)																			800
Terex Scraper**	(200)		(166)						(166)						(166)			(166)		56
P.A. HD-31 Dozer	(360)									(324)										36
Cat. D-9 (Recl.) Dozer	(450)									(405)										45
Cat. D-9 (Bench) Dozer	NA																			
BE 1508 Coal Shovel**	(750)																			75
Mich. 175 P.E. Loader**	(125)									(104)					(104)					10
Rimpull Coal Haulers** (4)	(900)						(720)							(720)						103
Lube... Service Trucks**	(65)									(58)					(58)					6
Water Truck**	(38)									(32)										3
Explosive Truck	(47)									(42)					(42)					4
Pick-Up Trucks**	(22)		(20)						(20)			(20)			(20)			(20)		7
Crane**	(50)									(38)										4
Pit Pump Systems	(18)									(15)										2
Cat. 14G Grader**	(88)									(75)										8
Cable	(150)									(135)					(135)					15
Wash Plant	(5,400)									(4,860)										540
Power Systems	(550)																			55
Expl. Stg. Facility	(15)																			---
Offices & Warehouse	(44)																			---
Shop & Warehouse	(75)																			---
Fuel Stg. Facilities	(15)																			---
Floodlights, Communications	(25)																			---
Truck Scales**	(15)									(22)					(22)					2
Site Preparation	(55)									(14)					(14)					2
Contingency, Misc.	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)
TOTALS	(53,001)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	(145)	5,168

* 10% salvage deducted from all expended replacement equipment except for final equipment item in series where salvage value is taken as undepreciated sum.
** Capital cost and salvage proportioned to West Field activity level of 50%.

TABLE 14

DISCOUNTED CASH FLOW ANALYSES
CAPITALIZATION SCHEDULE*

OLD BEN NO. 1 COAL MINE BLAST DESIGN PROGRAM

OPTION B
(\$1000)

Item	Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
BE 1570 D/L	(2,200)									(1,260)											2,200
BE 61R Drill	(1,400)																				140
BE 1150 D/L	(8,000)																				800
Terex Scraper**	(200)		(166)				(166)			(166)		(166)				(166)			(166)		56
P.A. HD-31 Dozer	(360)										(324)										36
Cat. D-9 (Recl.) Dozer	(450)									(405)											45
Cat. D-9 (Bench) Dozer	(450)			(405)					(405)				(405)			(405)					45
BE 1508 Coal Shovel**	(750)																				75
Mich. 175 F.E. Loader**	(125)					(104)				(104)					(104)						10
Rimpull Coal Haulers** (4)	(900)							(720)							(720)						103
Lube...Service Trucks**	(65)					(58)				(58)					(58)						6
Water Truck**	(38)									(32)											3
Explosive Truck	(47)					(42)				(42)					(42)						4
Pick-Up Trucks**	(22)		(20)				(20)			(20)			(20)			(20)			(20)		7
Crane**	(50)									(38)											4
Pit Pump Systems	(18)									(15)											2
Cat. 14G Grader**	(88)									(75)											8
Cable	(150)					(135)				(135)						(135)					15
Wash Plant	(5,400)									(4,860)											540
Power Systems	(550)																				55
Expl. Stg. Facility	(15)																				---
Offices & Warehouse	(44)																				---
Shop & Warehouse	(75)																				---
Fuel Stg. Facilities	(15)																				---
Floodlights, Communications	(25)																				---
Truck Scales**	(15)									(22)											2
Site Preparation	(55)									(14)											2
Contingency, Misc.	(125)	(125)	(125)	(125)	(125)	(125)	(125)	(125)	(125)	(125)	(125)	(125)	(125)	(125)	(125)	(125)	(125)	(125)	(125)	(125)	(125)
TOTALS	(41,432)	(125)	(125)	(311)	(530)	(464)	(311)	(845)	(530)	(311)	(7,529)	(125)	(716)	(125)	(845)	(650)	(530)	(125)	(311)	(125)	4,033

* 10% salvage deducted from all expended replacement equipment except for final equipment item in series where salvage value is taken as undepreciated sum.

** Capital and salvage proportioned to West Field activity level of 50%.

TABLE 15

DISCOUNTED CASH FLOW ANALYSES
CAPITALIZATION SCHEDULE*

OLD BEN NO. 1 COAL MINE BLAST DESIGN PROGRAM

Item	Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
											OPTION C (\$1000)											
BE 2570 D/L		(34,000)										(1,260)						6,120				
BE 61R Drill		(1,400)																504				
BE 1150 D/L		(8,000)																1,440				
Terex Scraper**		(200)			(166)			(166)			(166)			(166)			(166)	111				
P.A. RD-31 Dozer		(360)										(324)						130				
Cat. D-9 (Recl.) Dozer		(450)										(405)						162				
Cat. D-9 (Bench) Dozer		(450)			(405)					(405)				(405)				45				
BE 150B Coal Shovel**		(750)																150				
Mich 175 P.E. Loader**		(125)					(104)					(104)					(104)	83				
Rimpull Coal Haulers** (4)		(900)					(720)									(720)		515				
Lube...Service Trucks**		(65)					(58)					(58)					(58)	46				
Water Truck**		(38)										(32)						13				
Explosive Truck		(47)					(42)					(42)					(42)	34				
Pick-Up Trucks**		(22)			(20)			(20)			(20)			(20)			(20)	14				
Crane**		(50)										(38)						17				
Pit Pump Systems		(18)										(15)						6				
Cat. 14G Grader**		(88)										(75)						30				
Cable		(150)					(135)					(135)					(135)	108				
Wash Plant		(5400)										(4860)						1944				
Power Systems		(550)																99				
Expl. Stg. Facility		(15)																--				
Offices & Warehouse		(44)																--				
Shop & Warehouse		(75)																--				
Fuel Stg. Facilities		(15)																--				
Floodlights, Communications (25)												(22)						9				
Truck Scales**		(15)										(14)						5				
Site Preparation		(55)																--				
Contingency, Misc.		(150)	(150)	(150)	(150)	(150)	(150)	(150)	(150)	(150)	(150)	(150)	(150)	(150)	(150)	(150)	(150)	(150)				
TOTALS		(53,457)	(150)	(150)	(336)	(555)	(489)	(336)	(870)	(555)	(336)	(7534)	(150)	(741)	(150)	(870)	(675)	11435				

* 10% salvage deducted from all expended replacement equipment except for final equipment item in series where salvage value is taken as undepreciated sum.

** Capital and salvage proportioned to West Field activity level of 50%.

TABLE 17

DISCOUNTED CASH FLOW ANALYSIS
OWNERSHIP AND OPERATING COST ESTIMATES
(Excludes Depreciation, Operator and Maintenance Labor)
OLD BEN NO. 1 COAL MINE BLAST DESIGN PROGRAM

Option B

Item	West Field				West Field				Opr. \$ depr. hr	Own. \$ depr. hr	Opr. \$ depr. hr	Own. + Oper. \$ yr
	Depr. Hr YR	NA	NA	NA	Depr. Hr YR	NA	NA	NA				
BE 2570-W D/L	8736	NA	NA	NA	6468	413.06	251.83	NA	305.82	NA	NA	4,872,000
BE 1570-W D/L	8736	8736	4900	167.51	4900	167.51	91.58	93.96	93.96	1,621,000	NA	1,621,000
BE 1150-B D/L	4992	4992	3400	75.00	3400	75.00	28.04	51.08	51.08	395,000	NA	395,000
BE 61R Drill	8736	8736	2000	35.00	2000	35.00	4.12	8.01	8.01	106,000	NA	106,000
HD-31 Ground Dozer	2496	2496	1872	53.50	1872	53.50	8.01	40.12	40.12	120,000	NA	120,000
Terex S-24 Scraper	2496	2496	1800	49.00	1800	49.00	18.03	35.34	35.34	133,000	NA	133,000
Cat. D-9 Recl. Dozer	8736	8736	5500	53.00	5500	53.00	5.15	33.37	33.37	337,000	NA	337,000
Cat. D-9 Bench Prep. Dozer	2496	2496	1640	48.20	1640	48.20	30.05	31.67	31.67	154,000	NA	154,000
BE 150B Coal Shovel	2496	2496	1640	30.00	1640	30.00	5.01	19.71	19.71	62,000	NA	62,000
Michigan 175 Loader	2496	2496	1250	18.00	1250	18.00	1.54	9.01	9.01	26,000	NA	26,000
Water Truck	4368	4368	2184	30.00	2184	30.00	1.49	15.00	15.00	72,000	NA	72,000
Service Trucks	9984	9984	7200	224.00	7200	224.00	9.01	40.38	40.38	493,000	NA	493,000
Rimpull 120-Ton Coal Trucks (4)	4992	4992	2200	11.92	2200	11.92	0.94	5.25	5.25	31,000	NA	31,000
Explosives Truck	4368	4368	3300	8.97	3300	8.97	0.50	6.78	6.78	32,000	NA	32,000
Pick-up Trucks (4)	2496	2496	1550	22.00	1550	22.00	3.50	13.66	13.66	43,000	NA	43,000
Cat 14G Grader												
Total												8,497,000

TABLE 18

DISCOUNTED CASH FLOW ANALYSIS
OWNERSHIP AND OPERATING COST ESTIMATES
(Excludes Depreciation, Operator and Maintenance Labor)
OLD BEN NO. 1 COAL MINE BLAST DESIGN PROGRAM

Option C

Item	West Field West Field					
	Depr. Hr		Opr. Hr		\$	
	YR	YR	YR	YR	op-hr	depr. hr
BE 2570-W D/L	8736	6359	NA	NA	550.71	389.19
BE 1570-W D/L	NA	NA	NA	NA	NA	NA
BE 1150-B D/L	8736	6000	6000	6000	167.51	91.58
BE 61R Drill	4992	4300	4300	4300	75.00	28.04
HD-31 Ground Dozer	8736	2500	2500	2500	35.00	4.12
Terex S-24 Scraper	2496	2340	2340	2340	53.50	8.01
Cat. D-9 Recl. Dozer	2496	2250	2250	2250	49.00	18.03
Cat. D-9 Bench Prep. Dozer	8736	6115	6115	6115	53.00	5.15
BE 150B Coal Shovel	2496	2050	2050	2050	48.20	30.05
Michigan 175 Loader	2496	2050	2050	2050	30.00	5.01
Water Truck	2496	1250	1250	1250	18.00	1.54
Service Trucks	4368	2184	2184	2184	30.00	1.49
Rimpull 120-Ton Coal Trucks (4)	9984	9000	9000	9000	232.00	9.01
Explosives Truck	4992	2200	2200	2200	11.92	0.94
Pick-up Trucks (4)	4368	3300	3300	3300	8.97	0.50
Cat 14G Grader	2496	1550	1550	1550	22.00	3.50
Total						
						11,026,000

6,902,000

NA

1,805,000

462,000

124,000

145,000

155,000

369,000

174,000

74,000

26,000

72,000

612,000

31,000

32,000

43,000

TABLE 19

DISCOUNTED CASH FLOW ANALYSES
ESTIMATED ANNUAL OPERATING COSTS
OLD BEN NO. 1 COAL MINE BLAST DESIGN PROGRAM
(Assumed \$36/Ton Coal Sale Price)

	OPTION A	OPTION B	OPTION C
LABOR (40% benefits included)	3,360,000	3,539,000	3,539,000
DEPRECIATION	3,045,000	2,587,000	3,152,000
OPERATING AND OWNERSHIP COSTS (machinery)	9,903,000	8,497,000	11,026,000
COAL WASH COST/RAW TON** (exc. depr.)	3,425,000	3,425,000	4,116,000
BLASTING CONSUMABLES (exc. rig & steel costs)	695,000	985,000	1,235,000
ROYALTIES* (10% of \$36/ton sale price; 85% washed coal recovery)	2,999,000	2,999,000	3,748,000
Total \$/op-year	23,427,000	22,032,000	26,816,000
Total washed coal production/op-year	833,000	833,000	1,041,250
Total \$/ton of washed coal	\$28.12	\$26.45	\$25.75

* Assumed

** Wash plant job efficiency increased from 31% to 38% assumed in Option C; but consumables/ton unchanged; Option A = \$3.49/raw ton; Option B \$3.49/raw ton; Option C = \$3.37/raw ton

TABLE 20

DISCOUNTED CASH FLOW ANALYSES
ANNUAL CASH FLOW
OLD BEN NO. 1 COAL MINE BLAST DESIGN PROGRAM
(Assumed \$36/Ton Coal Sale Price)

	OPTION A	OPTION B	OPTION C
Total Revenues	29,988,000	29,988,000	37,485,000
Less Costs	<u>23,427,000</u>	<u>22,032,000</u>	<u>26,816,000</u>
Net After Costs	6,561,000	7,956,000	10,669,000
Less Depreciation	<u>3,045,000</u>	<u>2,587,000</u>	<u>3,152,000</u>
Net After Depreciation	3,516,000	5,369,000	7,517,000
Less State Taxes (2% of net after depreciation)	<u>70,000</u>	<u>107,000</u>	<u>150,000</u>
Net After State Tax	3,446,000	5,262,000	7,367,000
Less Depletion*	<u>1,723,000</u>	<u>2,631,000</u>	<u>3,374,000</u>
Pretax Net Profit	1,723,000	2,631,000	3,993,000
Less Federal Income Tax (48%)	<u>827,000</u>	<u>1,263,000</u>	<u>1,917,000</u>
Net After Taxes	896,000	1,368,000	2,076,000
Plus Depreciation	3,045,000	2,587,000	3,152,000
Plus Depletion	1,723,000	2,631,000	3,374,000
Operating Cash Flow	5,664,000	6,586,000	8,602,000

* Statutory coal depletion allowance

TABLE 21

DISCOUNTED CASH FLOW ANALYSES
SUMMARY TABLE
OLD BEN NO. 1 COAL MINE BLAST DESIGN PROGRAM

OPTION A
(\$1000)

Year	Investment	Operating Income	Depr. & Depl	Taxable Income	Taxes (48%)	Net Profit After Tax	Cum. Net Cash Flow	Annual Net Cash Flow	Trial			Trial			
									5% Interest Rate		15% Interest Rate		Present		
									Factor	Value	Factor	Value	Factor	Value	
-2															
-1	(53,001)						(53,001)	(53,001)	1.0000	(53,001)	1.0000	(53,001)	1.0000	(53,001)	
1	(145)	29,988	4,768	1,723	827	896	(47,482)	5,519	0.9768	5,391	0.9332	5,150	0.9332	5,150	
2	(145)	29,988	4,768	1,723	827	896	(41,963)	5,519	0.9295	5,130	0.8115	4,479	0.8115	4,479	
3	(331)	29,988	4,768	1,723	827	896	(36,630)	5,333	0.8852	4,721	0.7056	3,763	0.7056	3,763	
4	(145)	29,988	4,768	1,723	827	896	(31,111)	5,519	0.8431	4,653	0.6136	3,386	0.6136	3,386	
5	(484)	29,988	4,768	1,723	827	896	(25,931)	5,180	0.8029	4,159	0.5336	2,764	0.5336	2,764	
6	(331)	29,988	4,768	1,723	827	896	(20,598)	5,333	0.7647	4,078	0.4640	2,474	0.4640	2,474	
7	(765)	29,988	4,768	1,723	827	896	(15,699)	4,899	0.7283	3,568	0.4034	1,976	0.4034	1,976	
8	(145)	29,988	4,768	1,723	827	896	(10,180)	5,519	0.6936	3,828	0.3508	1,936	0.3508	1,936	
9	(331)	29,988	4,768	1,723	827	896	(4,847)	5,333	0.6605	3,522	0.3050	1,626	0.3050	1,626	
10	(7,529)	29,988	4,768	1,723	827	896	(6,712)	(1,865)	0.6291	(1,173)	0.2652	(494)	0.2652	(494)	
11	(145)	29,988	4,768	1,723	827	896	(1,193)	5,519	0.5991	3,306	0.2306	1,273	0.2306	1,273	
12	(331)	29,988	4,768	1,723	827	896	4,140	5,333	0.5706	3,043	0.2006	1,070	0.2006	1,070	
13	(145)	29,988	4,768	1,723	827	896	9,659	5,519	0.5434	2,999	0.1744	962	0.1744	962	
14	(765)	29,988	4,768	1,723	827	896	14,558	4,899	0.5175	2,535	0.1516	743	0.1516	743	
15	(670)	29,988	4,768	1,723	827	896	19,552	4,994	0.4929	2,462	0.1319	659	0.1319	659	
16	(145)	29,988	4,768	1,723	827	896	25,071	5,519	0.4694	2,591	0.1146	632	0.1146	632	
17	(145)	29,988	4,768	1,723	827	896	30,590	5,519	0.4471	2,468	0.0997	550	0.0997	550	
18	(331)	29,988	4,768	1,723	827	896	35,923	5,333	0.4258	2,271	0.0867	462	0.0867	462	
19	(145)	29,988	4,768	1,723	827	896	41,442	5,519	0.4055	2,238	0.0754	416	0.0754	416	
20	5,168	29,988	4,768	1,723	827	896	52,274	10,832	0.3862	4,183	0.0655	709	0.0655	709	
Totals	(61,006)	599,760	95,360												(18,465)
															12,972

$$\text{DCF Increment} = \frac{12,972}{12,972 + 18,405} \times 10\% = 4.13$$

DCF Rate of Return 9.1

TABLE 22

DISCOUNTED CASH FLOW ANALYSES
SUMMARY TABLE
OLD BEN NO. 1 COAL MINE BLAST DESIGN PROGRAM

OPTION B
(\$1000)

Year	Investment	Operating Income	Depr. & Depl	Taxable Income	Taxes (48%)	Net Profit After Tax	Cum. Net Cash Flow	Annual Net Cash Flow	Trial			Trial		
									10% Interest Rate Factor	10% Interest Rate Present Value	15% Interest Rate Factor	15% Interest Rate Present Value		
-2									(New Equipment On-Site and Operational)					
-1	(41,432)						(41,432)	(41,432)	1.0000	(41,432)	1.0000	(41,432)		
1	(125)	29,988	5,218	2,631	1,263	1,368	(34,971)	6,461	0.9538	6,162	0.9332	6,029		
2	(125)	29,988	5,218	2,631	1,263	1,368	(28,510)	6,461	0.8671	5,602	0.8115	5,243		
3	(311)	29,988	5,218	2,631	1,263	1,368	(22,235)	6,275	0.7882	4,946	0.7056	4,428		
4	(530)	29,988	5,218	2,631	1,263	1,368	(16,179)	6,056	0.7166	4,340	0.6136	3,716		
5	(464)	29,988	5,218	2,631	1,263	1,368	(10,057)	6,122	0.6514	3,988	0.5336	3,267		
6	(311)	29,988	5,218	2,631	1,263	1,368	(3,782)	6,275	0.5922	3,716	0.4640	2,912		
7	(845)	29,988	5,218	2,631	1,263	1,368	1,959	5,741	0.5384	3,091	0.4034	2,316		
8	(530)	29,988	5,218	2,631	1,263	1,368	8,015	6,056	0.4894	2,964	0.3508	2,124		
9	(311)	29,988	5,218	2,631	1,263	1,368	14,290	6,275	0.4449	2,792	0.3050	1,914		
10	(7,529)	29,988	5,218	2,631	1,263	1,368	13,347	(943)	0.4045	(381)	0.2652	(250)		
11	(125)	29,988	5,218	2,631	1,263	1,368	19,808	6,461	0.3677	2,376	0.2306	1,490		
12	(716)	29,988	5,218	2,631	1,263	1,368	25,678	5,870	0.3343	1,962	0.2006	1,178		
13	(125)	29,988	5,218	2,631	1,263	1,368	32,139	6,461	0.3039	1,963	0.1744	1,127		
14	(845)	29,988	5,218	2,631	1,263	1,368	37,880	5,741	0.2762	1,586	0.1516	870		
15	(650)	29,988	5,218	2,631	1,263	1,368	43,816	5,936	0.2511	1,490	0.1319	783		
16	(530)	29,988	5,218	2,631	1,263	1,368	49,872	6,056	0.2283	1,382	0.1146	694		
17	(125)	29,988	5,218	2,631	1,263	1,368	56,333	6,461	0.2075	1,341	0.0997	644		
18	(311)	29,988	5,218	2,631	1,263	1,368	62,608	6,275	0.1887	1,184	0.0867	544		
19	(125)	29,988	5,218	2,631	1,263	1,368	69,069	6,461	0.1715	1,108	0.0754	487		
20	4,033	29,988	5,218	2,631	1,263	1,368	79,688	10,619	0.1559	1,655	0.0655	696		
Totals	(52,032)	599,760	104,360								11,835	(1,220)		

DCF Rate of Return 14.5

$$\text{DCF Increment} = \frac{11,835}{1220 + 11,835} \times 5\% = 4.53$$

TABLE 23

DISCOUNTED CASH FLOW ANALYSES
SUMMARY TABLE
OLD BEN NO. 1 COAL MINE BLAST DESIGN PROGRAM

OPTION C
(\$1000)

Year	Investment	Operating Income	Depr. & Depl	Taxable Income	Taxes (48%)	Net Profit After Tax	Cum. Net Cash Flow	Annual Net Cash Flow	Trial			Trial				
									10% Interest Rate		15% Interest Rate		10% Interest Rate		15% Interest Rate	
									Factor	Present Value	Factor	Present Value	Factor	Present Value	Factor	Present Value
-2									(New Equipment On-Site and Operational)							
-1	(53,457)						(53,457)	(53,457)	1.0000	(53,457)	1.0000	(53,457)	1.0000	(53,457)		
1	(150)	37,485	6,526	3,993	1,917	2,076	(45,005)	8,452	0.9538	8,062	0.9332	7,887	0.9332	7,887		
2	(150)	37,485	6,526	3,993	1,917	2,076	(36,553)	8,452	0.8671	7,329	0.8115	6,859	0.8115	6,859		
3	(336)	37,485	6,526	3,993	1,917	2,076	(28,287)	8,266	0.7882	6,515	0.7056	5,832	0.7056	5,832		
4	(555)	37,485	6,526	3,993	1,917	2,076	(20,240)	8,047	0.7166	5,766	0.6136	4,938	0.6136	4,938		
5	(489)	37,485	6,526	3,993	1,917	2,076	(12,127)	8,113	0.6514	5,285	0.5336	4,329	0.5336	4,329		
6	(336)	37,485	6,526	3,993	1,917	2,076	(3,861)	8,266	0.5922	4,895	0.4640	3,835	0.4640	3,835		
7	(870)	37,485	6,526	3,993	1,917	2,076	3,871	7,732	0.5384	4,163	0.4034	3,119	0.4034	3,119		
8	(555)	37,485	6,526	3,993	1,917	2,076	11,918	8,047	0.4894	3,938	0.3508	2,823	0.3508	2,823		
9	(336)	37,485	6,526	3,993	1,917	2,076	20,184	8,266	0.4449	3,678	0.3050	2,521	0.3050	2,521		
10	(7,534)	37,485	6,526	3,993	1,917	2,076	21,252	1,068	0.4045	432	0.2652	283	0.2652	283		
11	(150)	37,485	6,526	3,993	1,917	2,076	29,704	8,452	0.3677	3,108	0.2306	1,949	0.2306	1,949		
12	(741)	37,485	6,526	3,993	1,917	2,076	37,565	7,861	0.3343	2,628	0.2006	1,577	0.2006	1,577		
13	(150)	37,485	6,526	3,993	1,917	2,076	46,017	8,452	0.3039	2,568	0.1744	1,474	0.1744	1,474		
14	(870)	37,485	6,526	3,993	1,917	2,076	53,749	7,732	0.2672	2,066	0.1516	1,172	0.1516	1,172		
15	(675)	37,485	6,526	3,993	1,917	2,076	61,676	7,927	0.2511	1,990	0.1319	1,046	0.1319	1,046		
16	11,435	37,485	6,526	3,993	1,917	2,076	81,713	20,037	0.2283	4,574	0.1146	2,296	0.1146	2,296		
17																
18																
19																
20																
Totals	(55,919)	599,760	104,416							13,540				(1517)		

$$\text{DCF Increment} = \frac{13,540}{1,517 + 13,540} \times 5\% = 4.5$$

DCF Rate of Return 14.5

9.0 THROW BLASTING ANALYSIS SENSITIVITY

The throw blasting technique demonstrated at the Old Ben No. 1 Coal Mine documented increased overburden removal rates with the same equipment. The principle additional components required for the increased productivity were blasting agents and dozing effort.

Table 24 presents the drilling and blasting unit costs which were obtained from Old Ben Coal Mine No. 1 drilling and blasting personnel. The costs are current as of October 1981.

The existing blast pattern prior to the test program used 15-inch diameter holes on a 36 ft x 48 ft pattern and 30-foot-depth fill-in satellite holes. The blast powder factor was maintained at 0.33 lb/cyd. Table 25 presents the calculated drill and blast cost per bank cubic yard based on a highwall height of 95 feet. The overall cost is shown to have been \$0.065/cyd for that particular highwall height.

The proposed "optimum" drill and blast pattern is 30 ft x 50 ft with no satellite holes and a powder factor of 0.55 lbs/cyd. Table 26 shows that this blast design has a corresponding cost of \$0.081/cyd; an increase of 24.6 percent. Neither Tables 25 or 26 include blast labor cost which is estimated at less than 10 percent of the total.

Table 26 shows the effect of increasing the drill penetration rate and AN/FO cost by 10 percent. The development of these increases is shown in Table 27.

Table 26 shows that a 10 percent change in an AN/FO price results in a corresponding 6 percent change in the overall drill and blast costs. A 10 percent increase in drill penetration is shown to produce a 2.5 percent reduction in overall costs.

TABLE 24

DRILL AND BLAST UNIT COSTS

Drilling Costs

a) Drill Cost/Foot - Excludes ownership and drillbits but includes labor, electricity, maintenance, drillstems, etc.	\$1.35/foot
b) Bit Cost - Based on a 15-inch bit with a cost of \$6,948 each and a bit life of 43,000 feet.	<u>\$0.16/foot</u>
Total Drilling Cost	<u>\$1.51/foot</u>

Explosive & Accessory Supply Costs

c) AN/FO - pre-oiled	\$170/ton	\$0.085/lb
d) Dow Termal Primer	\$40.50/100 lb	\$8.10/20 lb
e) Cast Primer		\$2.55/each
f) Downhole Delay		\$1.10/each
g) Primacord - 40 grain.foot	\$69.20/1000 ft	\$0.069/foot
h) Nonel Surface Delay		\$2.50/each
i) Nonel Trunkline	\$16.50/500 ft	\$0.033/foot

TABLE 25

DRILL AND BLAST COSTS FOR THE EXISTING METHOD OF DRILL
AND BLAST PRIOR TO THE TEST PROGRAM

Item	No.	Cost Regular Hole	No.	Cost Satellite Hole	Total Cost
Drilling Cost	95'	\$128.25	30'	\$22.50	\$150.75
Bit Cost	95'	15.20	30'	2.67	17.87
AN/FO	1838#	156.23	300#	14.17	170.40
Thermal Primer	3	24.30	1	4.50	28.80
Cast Primer	3	7.65	1	1.42	9.07
Downhole Delay	3	3.30	1	0.61	3.91
Primacord	90'	6.21	30'	1.15	7.36
Surface Delay	2	5.00	1	1.39	6.39
Nonel Trunkline	60'	1.98	20'	0.37	<u>2.35</u>
Total Per Hole					<u>\$396.90</u>

Note: The costs entered for the satellite hole are 5/9 of the actual cost since only 5/9 satellite holes area required per regular blasthole.

$$\text{The cost per BCY is } \frac{\$396.90}{6080 \text{ BCY}} = \underline{\underline{\$0.065/\text{BCY}}}$$

TABLE 26

COST OF PROPOSED DRILLING AND BLASTING METHOD AND SENSITIVITY

Item	Proposed	Drill Penetration Rate +10%	AN/FO Cost +10%
a) Drilling Cost	\$128.25	\$116.59	\$128.25
b) Bit Cost	15.20	15.20	15.20
c) AN/FO - 2900 lb/hole	246.50	246.50	271.15
d) Thermal Primer 2 @ \$8.10 each	16.20	16.20	16.20
e) Cast Primer 2 @ \$2.55 each	5.10	5.10	5.10
f) Downhole Delay 2 @ \$1.10 each	2.20	2.20	2.20
g) Primacord - 2 lines/hole - 140 ft	9.66	9.66	9.66
h) Surface Delays 2 @ \$2.50 each	5.00	5.00	5.00
i) Nonel Trunkline - 60 ft/hole including lead-in line	1.98	1.98	1.98
Total/Hole	\$430.39	\$418.73	\$454.74
For 55.56 BCY/foot x 95 feet =			
5278 BCY/hole : Cost Per BCY	<u>\$0.081</u>	<u>\$0.079</u>	<u>\$0.086</u>
Percent Change from Existing Costs	+24.6	+21.5	+32.2
Percent Change from Proposed Costs	-	-2.5	+6.2

The labor cost of loading is estimated at \$40/hole.

The cost of blasting accessories, Items d through i are \$40.14.

Both these are relatively small compared to the total cost and, thus, a 10% change would affect the total by less than 1%.

TABLE 27

DRILLING COST AND AN/FO CHANGES AS USED IN TABLE 26

a) Drill Cost (excluding bits)	\$1.35/foot
b) Bit Cost	<u>0.16/foot</u>
Total Drilling Cost	\$1.51/foot

c) Drill Footage/Shift is 704 ft.

$$\begin{aligned} \therefore \text{Drill Cost/Shift} &= \$1.35 \times 704 = \$ 950.40/\text{shift} \\ \text{Bit Cost/Shift} &= \$0.16 \times 704 = \$ \underline{112.64/\text{shift}} \end{aligned}$$

$$\text{Total Drilling Cost} = \$1,063.04/\text{shift}$$

If the penetration rate is increased 10% then
the footage/shift is 774 feet.

$$\begin{aligned} \therefore \text{Drill Cost/Shift} &= \$950.40 : 774 = \$1.23/\text{ft.} \\ \text{Bit Cost/Foot is Constant} &= \underline{0.16/\text{ft.}} \end{aligned}$$

$$\text{Total Drilling Cost} = \$1.39/\text{ft.}$$

If the AN/FO cost is increased 10%, the cost increases from \$170/ton to \$187/ton or \$0.0935/lb.

Throw blasting, with a 10 percent increase in AN/FO cost, is shown to add \$.021/BCY, or approximately \$.40/ton of coal (16.8 BCY/ton stripping ratio; 90 percent recovery) over the historical blasting agent component of mine production. This \$.40/ton represents an increase of 1.5 percent in the washed coal product cost. The incremental \$.093/ton attributable to the 10 percent increase in AN/FO costs represents only 4 percent of the total unit cost savings evidenced by the adoption of the throw blasting technique (Option C versus Option A).

The highwall fill dozing effort, if doubled, represents an additional yearly cost of \$600,000 (includes depreciation, ownership, operating supplies and operator). In the Option C 1,225,000 tons per year production mode, this represents approximately a 2 percent increase in washed coal production cost. The \$.49/ton doubled dozing effort cost increase represents 20 percent of the washed coal cost saving demonstrated by the increased (Option C) production technique.

The Old Ben analysis was influenced to a considerable degree by other existing and/or assumed test conditions and results. Included here are:

- 1) The increased coal production was attainable without additional production equipment or coal washing capacity. The comparison of Option C (1,225,000 tons of coal/yr) and Option A (the existing 980,000 tons of coal/yr production rate) was, to a degree, a comparison between under-utilized (dragline constrained) and fully utilized equipment. Such comparisons favor the more efficient operation.
- 2) The two-seam mining method "chop-down" elimination, although a demonstrated productivity improvement in the area of the test bench, would not be realized throughout the West Field since the No. 6 Coal seam occurs only sporadically.

Although common to all programs, the ownership cost and subsequent cash flow determinations are strongly influenced by the existing discount rate. In periods of high interest rate charges, the "savings" attributable to decreased capitalization approaches (such as Option B) are considerable. In a lower "cost of capital" environment, these "savings" contribute less to the discounted cash flow determination. The resulting advantage of the machine replacement in Option B is reduced by up to 20 percent (i.e., at a 7 percent interest rate versus the 16 percent used).

Less well defined, and subject to legislative fiat, are depreciation scheduling and tax credit effects. It is possible that accounting manipulations such as the proposed 3-5-7 depreciation schedule or additional energy investment tax credits could off-set the attractiveness of the reduced capitalization Option B approach.

10.0 RECOMMENDATIONS AND CONCLUSIONS

The Old Ben field test of the casting technique yielded qualified conclusions. The findings were tentative due to the inability in the short duration test program to quantify the dragline tub maintenance cost impact. The dragline tub was observed to sink, unevenly, into the claylike fill which had been dozed onto the post-blast highwall void to reconstruct the working bench. A shale cap, placed as a dragline rehandle function, of 5-10 ft thickness, was demonstrated to slow the rate of settlement. The clay fill, even when shale capped, might not support the dragline during periods of wet weather.

The qualified conclusions are as follows:

- (1) A geology-related drill pattern, charging, and blast sequencing plan may make it possible to utilize muck casting for many stripping capacity production constrained strip mines. This approach, whether worked from a reconstructed bench, a lowered bench with tailroom excavation, or from a spoils side pull-back configuration, may yield improved coal production rates based on reduced materials handling and/or cycle efficiency improvements.
- (2) The increased explosives and dozing (highwall filling) costs incurred by the casting approach were, in the Old Ben experiment, more than offset by the increased revenues of the increased rate of coal production.
- (3) The casting technique, for a given production rate, can lead to a downsizing of the primary stripping dragline. This represents a capitalization and ownership savings which, at Old Ben, more than offset the increased costs of drilling, blasting and dozing.

The Old Ben field test program indicated substantial increased coal production and cost cutting potential. During the course of the program, mine operators overseas, particularly in South Africa, have increasingly adopted and improved casting techniques for strip coal mines. The favorable results of the Old Ben program, combined with the evolving positive overseas experience, indicate the emergence of a significant production technique.

It is pertinent that the casting technique is not limited to the highwall dragline operating mode or the working bench reconstruction approaches used at Old Ben. The Old Ben fill material (No. 6 Coal overburden) was of questionable stability. In light of the substantial investment in the BE2570, it would seem questionable whether the casting program, as developed in the field, would be routinely employed. A wet spell might saturate the fill with serious results. Old Ben however, due to the rapid evolution of the casting technique, has choices other than highwall filling. As noted in this report, there is reason to expect the lowered highwall approach to be feasible. This multiplicity of options, one of which solves the clay fill stability problem, would allow Old Ben, or any other investigating mine operator, to take a limited test program such as this project's 15-shot sequence and extrapolate the results to a mine specific casting technique.

A basic knowledge of the pattern and explosives requirement to achieve casting, plus a review of coal production cycle elements such as pit haulage options, dragline turnaround approaches, and geologic-hydrologic influences, can be the basis for a feasibility analysis of the evolving casting approaches. For most strip coal mines, one or more of these approaches might be found attractive, and a casting program could be further developed. The Old Ben field test represents a coal field specific

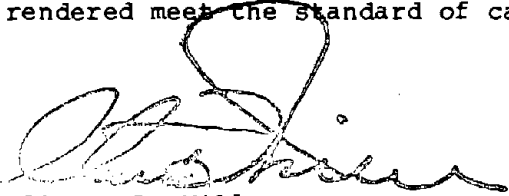
geology-blasting-casting approach, which, when combined with the details of the West Field coal occurrence and overburdens, may form the basis for additional studies of the other casting options. The clay fill instability potential which qualified the conclusions on applicability of the technique demonstrated at Old Ben, is not necessarily a "fatal flaw" for the casting approach. Rather, the clay fill problem requires a reapproach to the technique integration, possibly leading to further field studies on the lowered bench technique.

In light of the substantial cost and production benefits potentially accruable to the widespread exploitation of the casting techniques, the following recommendations are made:

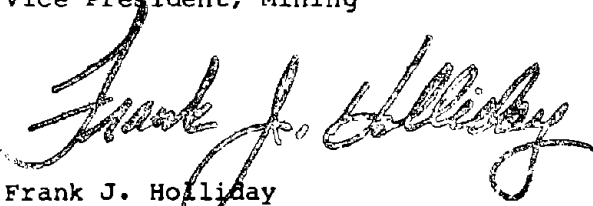
- (1) A review of strip coal mining operations employing the casting technique is suggested. This would include development of range diagrams, pit haulage networks, turn-around methods, dragline performance criteria, highwall dozing effort, if required, pertinent blast related geology, and discussions with mine personnel on problems, prospective solutions, and the history of the technique development. Such a project would analyze foreign experiences with casting to broaden domestic knowledge on casting approaches.
- (2) That further testing of the casting technique at the Old Ben No. 1 Coal Mine in Oakland City, Indiana be considered. This effort would further quantify fill stability relationships, coal protection stemming, blast designs and throw, and introduce modified casting techniques, i.e., the lowered bench single-seam approach. Further work with the Old Ben operation is desirable since an explosives utilization-geologic response has been developed as part of this study. A Phase III development of a more optimized and possibly modified technique would allow for a complete systems integration at a domestic mining operation. If, for reasons presently unforeseen, the Old Ben No. 1 Mine does not choose to further develop casting at its No. 1 Mine, a similar strip mine operation should be solicited for involvement.

GENERAL INFORMATION

Professional judgments on the results of field test work are presented in this report. These are based partly on our evaluations of technical information gathered and partly on our understanding of the operations being considered. All estimates are based on observations of a short duration test program and under no circumstances should be construed as other than tentative. We do not guarantee the performance of any project in any respect, only that our engineering work and judgments rendered meet the standard of care of our profession.



Clinton L. Miller
Vice President, Mining



Frank J. Holliday
Vice President/Principal

APPENDIX A
OVERSIZE FIGURES

APPENDIX A

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- FIGURE A-10 - OLD BEN NO. 1 BLAST DESIGN PROGRAM, TEST 1 BLAST 5
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APPENDIX A

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(Continued)

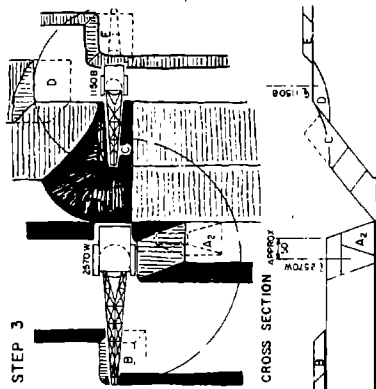
FIGURE A-19 - OLD BEN NO. 1 BLAST DESIGN PROGRAM, TEST 2 BLAST 6

FIGURE A-20 - OLD BEN NO. 1 BLAST DESIGN PROGRAM, TEST 2 BLAST 7

FIGURE A-21 - OLD BEN NO. 1 BLAST DESIGN PROGRAM, TEST 2 BLAST 8

FIGURE A-22 - OLD BEN NO. 1 BLAST DESIGN PROGRAM DOZER OVERBURDEN
BEFORE AND AFTER SECTIONS

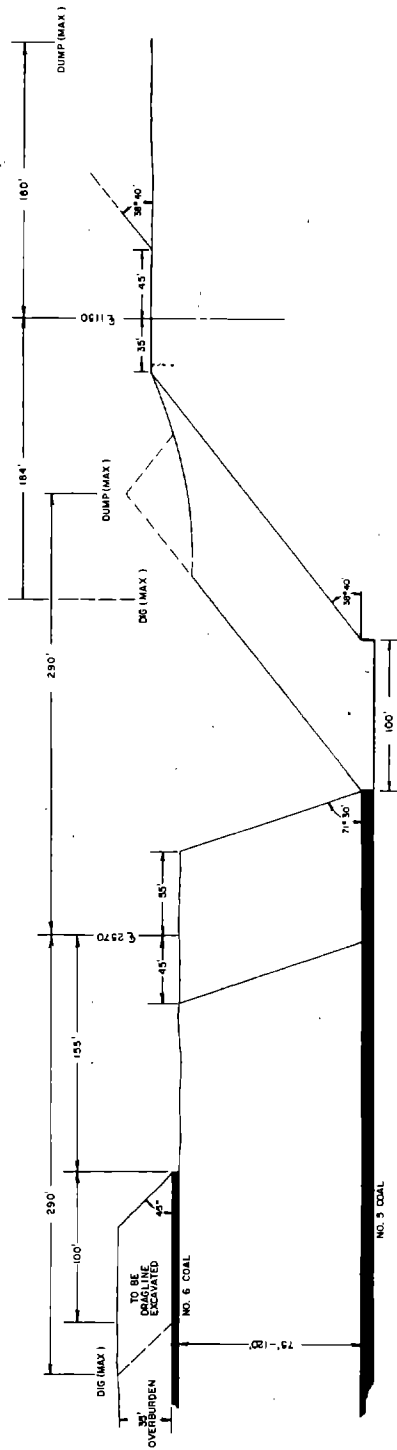
FIGURE A-23 - OLD BEN NO. 1 BLAST DESIGN, POST BLAST MUCKPILE (TYPICAL)
CENTERLINE TEST #2: DRAGLINE WORKING BENCH REBUILD
(TYPICAL) TEST #2



STEP THREE

1150-B CONTINUES REMOVING MATERIAL PREVIOUSLY SPOILED BY THE 2570-W (AREA C) AND USES MATERIAL TO COMPLETE BEACH (AREA D) AND FINAL SPOIL (AREA E).

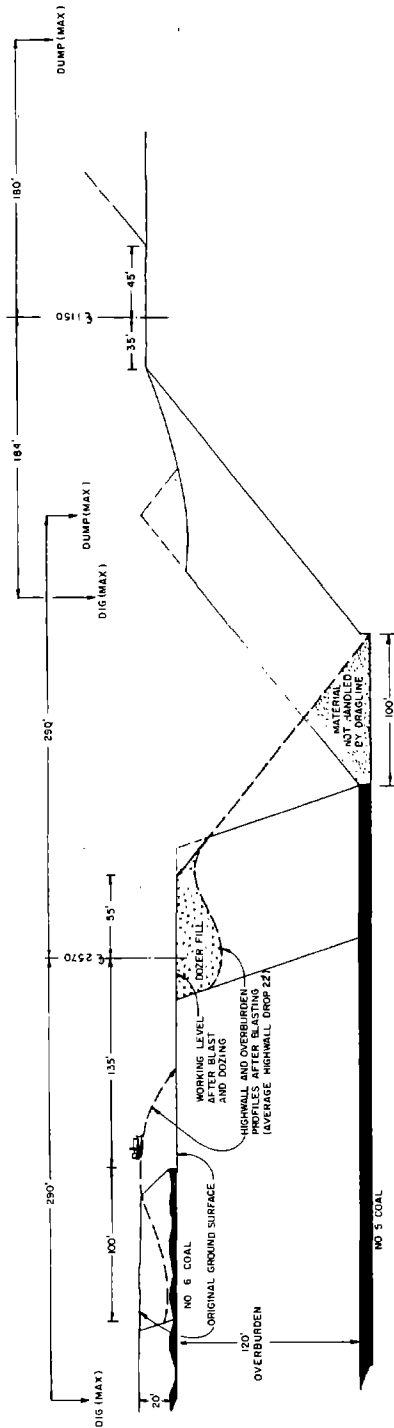
NO SCALE
SOURCE: OLD BEN COAL CO
A-1



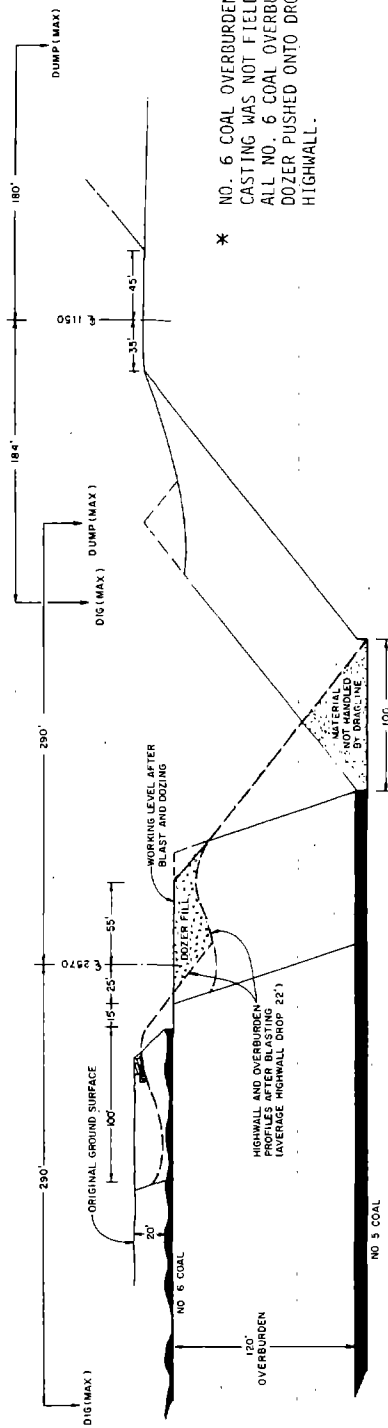
EXISTING EXISTING DRAGLINE OPERATIONS

DRAWN BY		DATE	CHECKED BY	DATE
AS CONSTRUCTED - CONNECTIONS COMPLETE				
BY		R E V I S I O N S		
NO	DESCRIPTION	DATE	BY	
		SCALE 1" = 80'		
		JOB NO. 19943		
		DRAWING NO. OF		
		A - 2		

RANGE DIAGRAMS FOR
OLD BEN NO 1 COAL MINE
EXISTING DRAGLINE OPERATION
WOODWARD - CLYDE CONSULTANTS
Consulting Engineers
Denver, Colorado



NO. 6 COAL * OPTION NO. 1 - OPEN BENCH HIGHWALL DROP - OVERBURDEN THROW (20' OVERBURDEN COAL NO. 6, 120' OVERBURDEN COAL NO. 5, 22' AVERAGE HIGHWALL DROP)

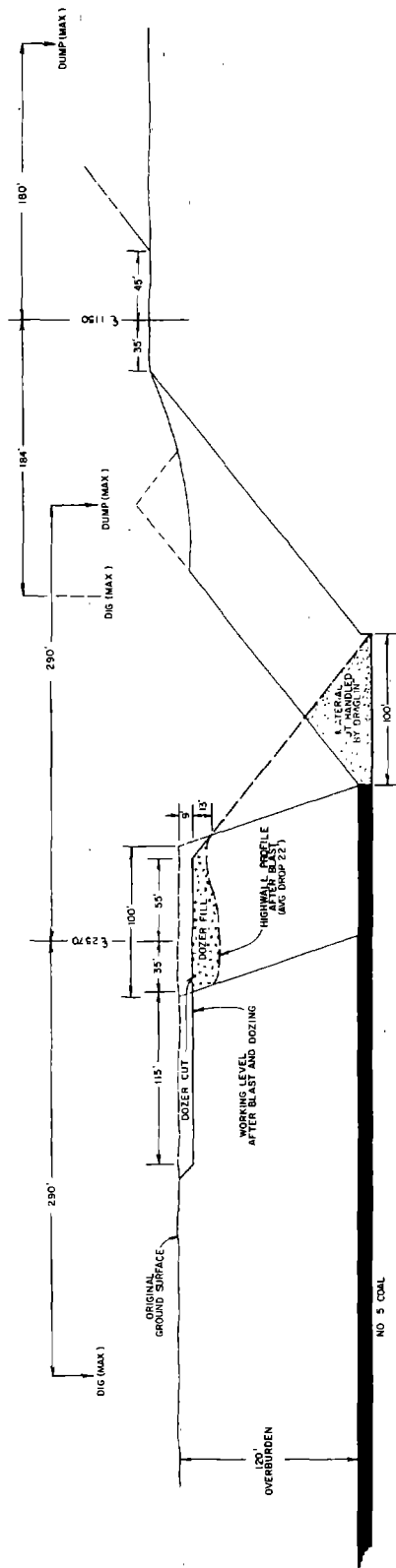


* NO. 6 COAL OVERBURDEN EXPLOSIVES CASTING WAS NOT FIELD TESTED. ALL NO. 6 COAL OVERBURDEN WAS DOZER PUSHED ONTO DROPPED HIGHWALL.

NO. 6 COAL * OPTION NO. 2 - MINIMAL OPEN BENCH HIGHWALL DROP - OVERBURDEN THROW (20' OVERBURDEN COAL NO. 6, 120' OVERBURDEN COAL NO. 5, 22' AVERAGE HIGHWALL DROP)

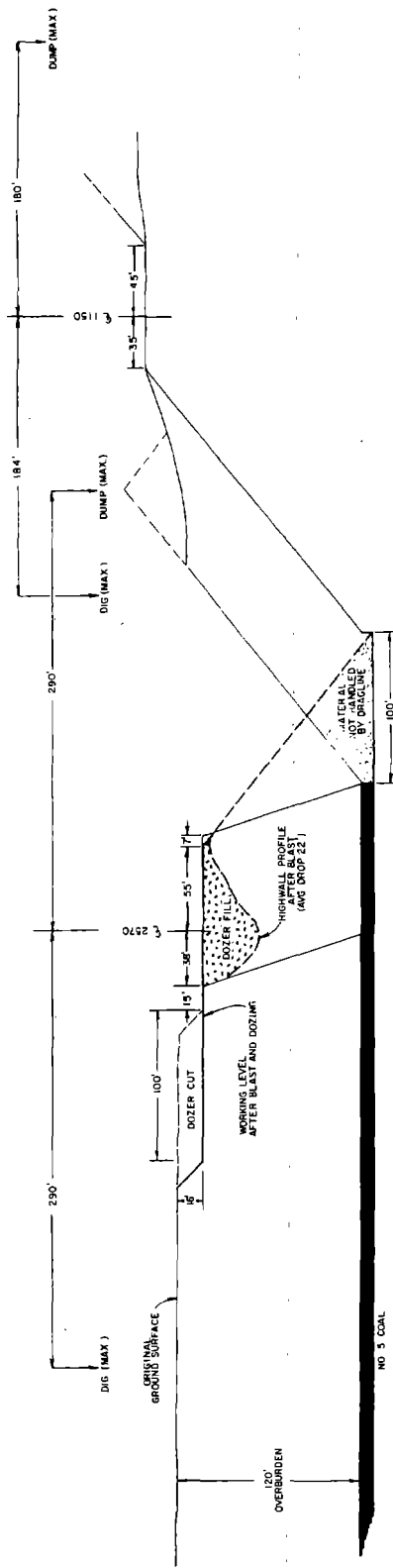
DATE	7/22/81	CHECKED BY	
BY	AS CONSTRUCTED - CORRECTIONS COMPLETE	DATE	
REVISIONS		DATE	
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RANGE DIAGRAMS FOR OLD BEN NO. 1 COAL MINE, PROPOSED TWO-SEAM OPERATION WOODWARD - CLYDE CONSULTANTS Consulting Engineers Denver, Colorado SCALE 1" = 60' DRAWING NO. A-3			

[illegible]



120' OVERBURDEN

DESIGNED INITIAL CUT SEQUENCE
HIGHWALL DROP
OVERBURDEN LEVELING (120' OVERBURDEN, 22' HIGHWALL DROP)



120' OVERBURDEN

DESIGNED ULTIMATE CUT SEQUENCE
HIGHWALL DROP
OVERBURDEN LEVELING (120' OVERBURDEN, 22' HIGHWALL DROP)

DESIGNED BY	J. L. L.	CHECKED BY	DATE	1/22/01
BY	AS CONSTRUCTED - CORRECTIONS COMPLETE	DATE		
NO	DESCRIPTION	DATE	BY	
RANGE DIAGRAM FOR CUL BEN NO 1 COAL NINE NO 5 COAL PROPOSED SINGLE- SEAM OPERATIONS: 120' OVERBURDEN WOODWARD - CLYDE CONSULTANTS Consulting Engineers Denver, Colorado SCALE 1" = 80' DRAWING NO. 4-5 259 NO. 49543				

A

□ CAMERA
10/5/81

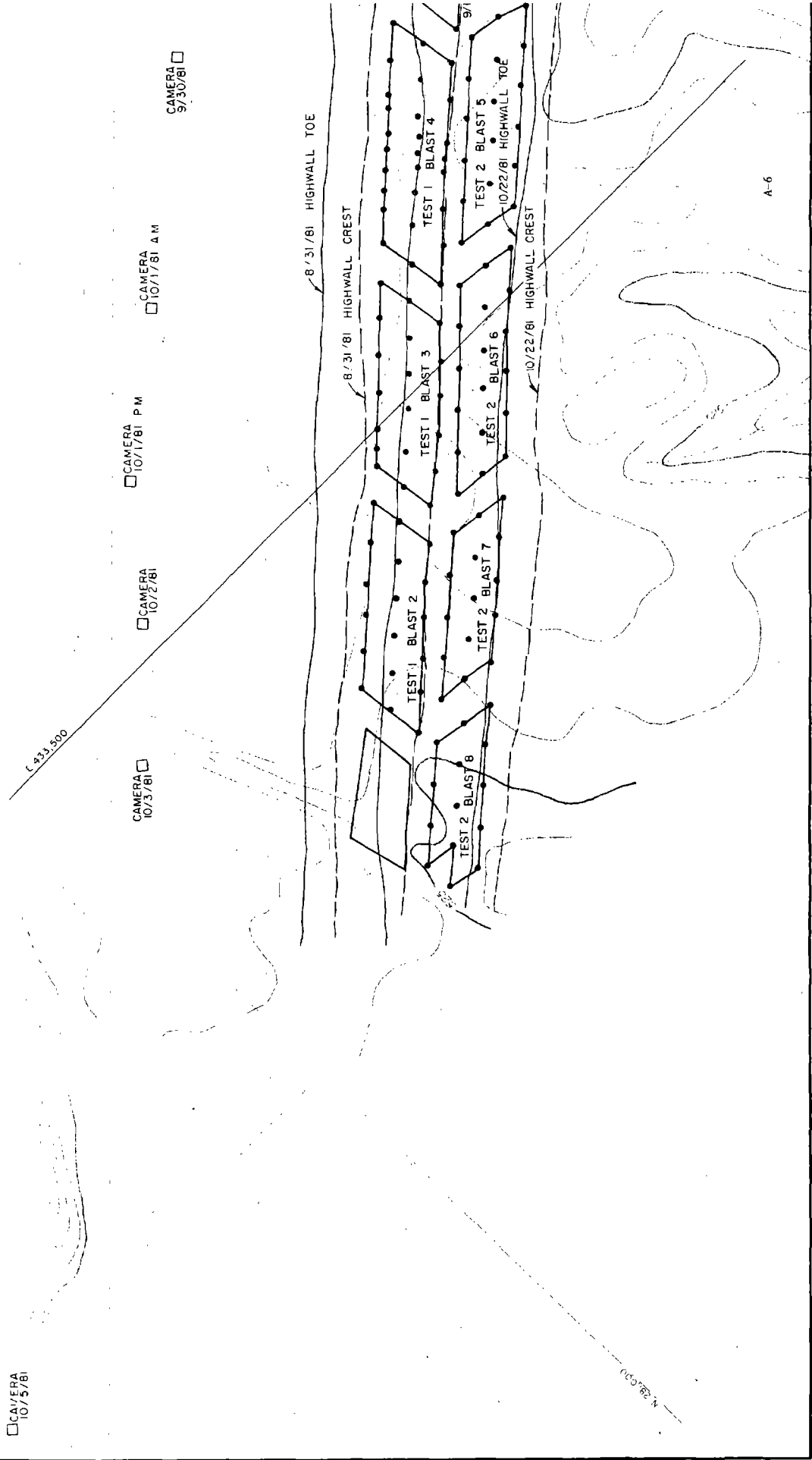
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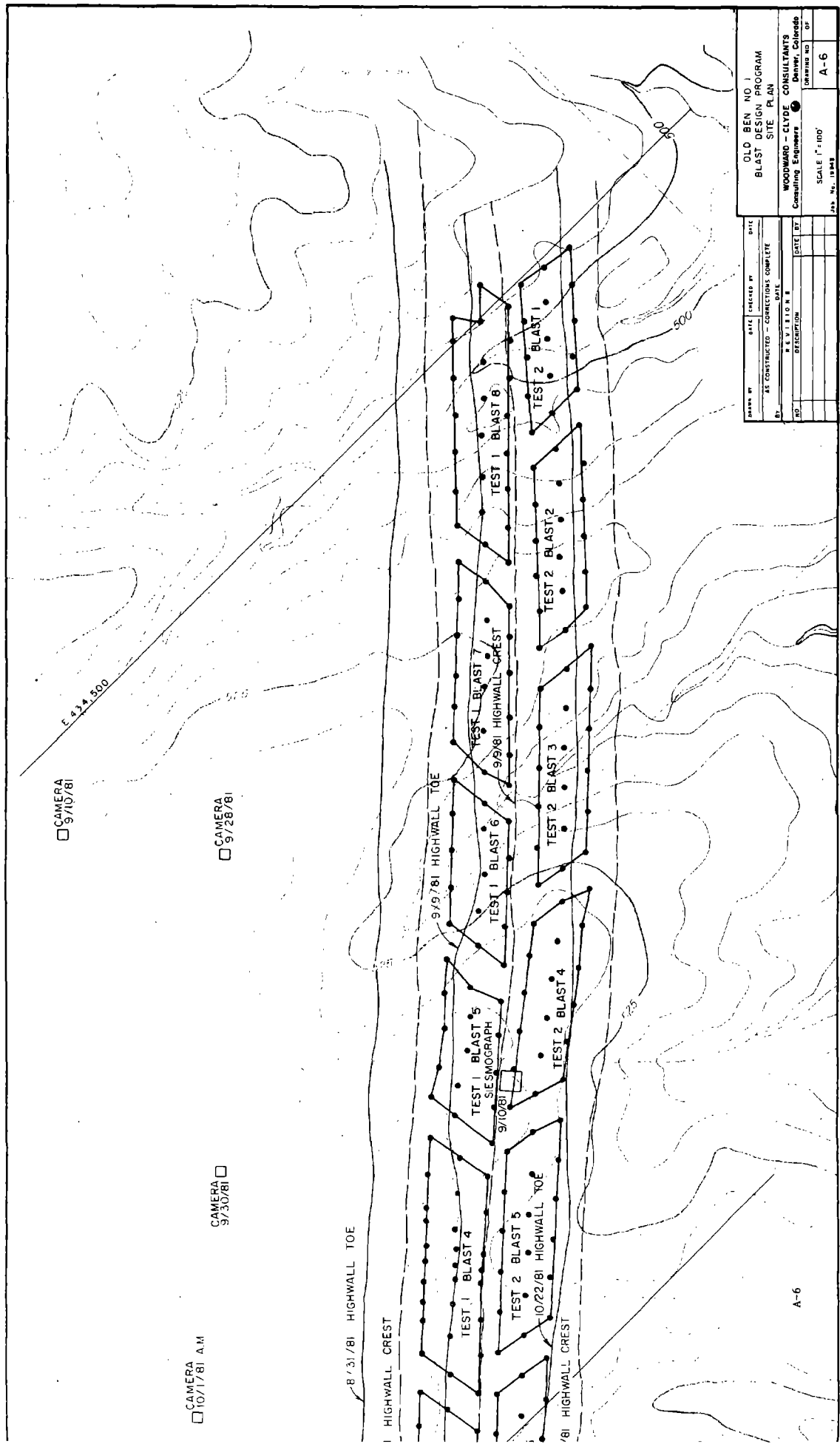
□ CAMERA
10/2/81

□ CAMERA
10/1/81 P.M.

□ CAMERA
10/17/81 A.M.

□ CAMERA
9/30/81





□ CAMERA
9/10/81

□ CAMERA
9/28/81

□ CAMERA
10/11/81 A.M.

CAMERA
9/30/81 □

8/31/81 HIGHWALL TOE

1 HIGHWALL CREST

9/9/81 HIGHWALL TOE

TEST 1 BLAST 5
SEISMOGRAPH

9/10/81

TEST 2 BLAST 5
10/22/81 HIGHWALL TOE

7/81 HIGHWALL CREST

TEST 1 BLAST 7
9/9/81 HIGHWALL CREST

TEST 1 BLAST 6

TEST 1 BLAST 8

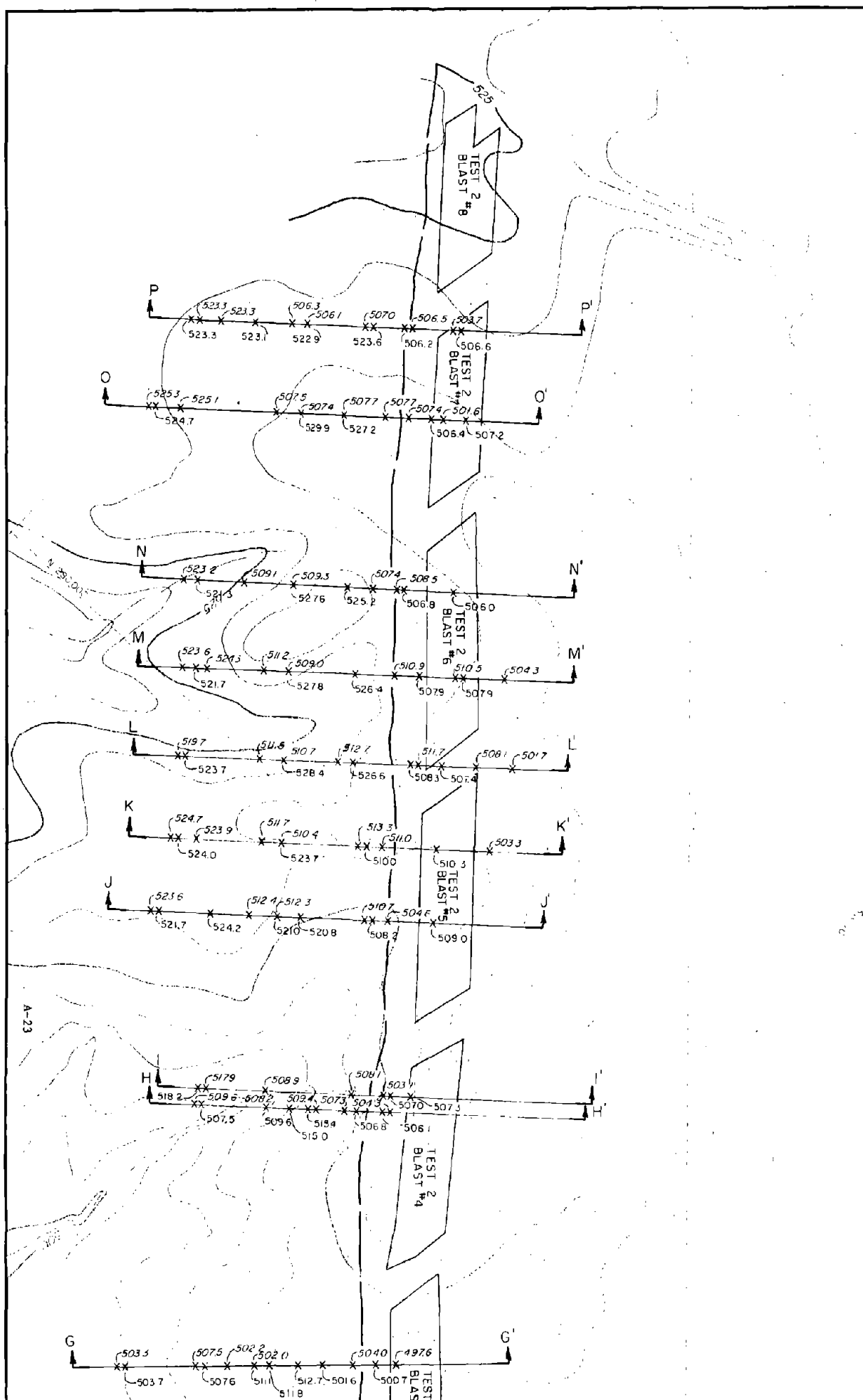
TEST 2 BLAST 3

TEST 2 BLAST 2

TEST 2 BLAST 1

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AS CONSTRUCTED - CORRECTIONS COMPLETE		DATE	
BY		DATE	
DESCRIPTION		DATE BY	
JOB NO. 18443		SCALE 1" = 100'	
DRAWING NO. 01		A-6	

OLD BEN NO. 1
BLAST DESIGN PROGRAM
SITE PLAN
WOODWARD - CLYDE CONSULTANTS
Consulting Engineers
Denver, Colorado



22-1A

LEGEND

- D.H. NO.
- DEPTH
- ANFO LOAD (lbs.)
- COLLAR

INITIATION SEQUENCE

HOOK-UP AND DELAY (msec.)

HM TOE (PRE-BLAST)

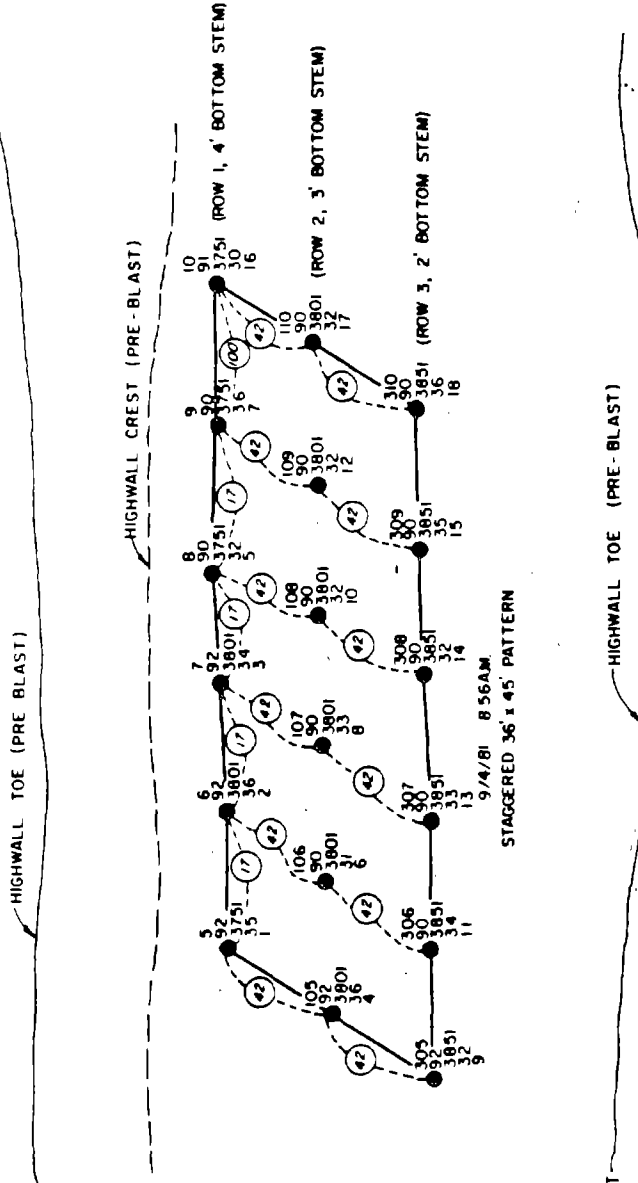
HM CREST (PRE-BLAST)

HM TOE (POST-BLAST)

HM CREST (POST-BLAST)

BLAST SIDE LINE AREA OF INFLUENCE

DRILL HOLE LOCATION AND POST-BLAST ELEVATION



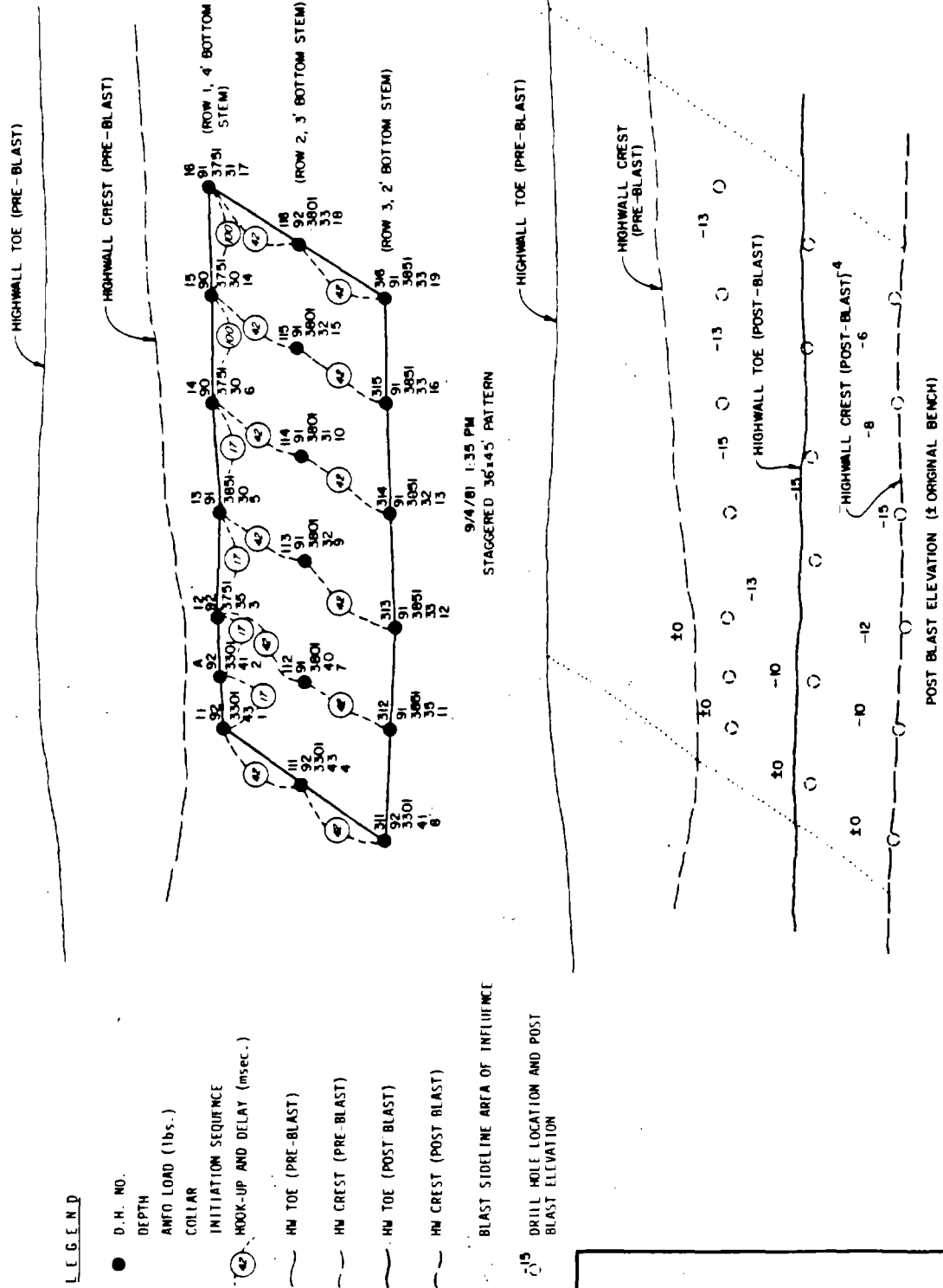
OLD BEN NO. 1
BLAST DESIGN PROGRAM
TEST 1 BLAST 2

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers Denver, Colorado

SCALE: 1" = 62.5'
Job No. 19943

DRAWING NO. OF
A-7

NOTE: EXTRA HOLES IN PATTERN FOR BITTER
CAP ROCK FRAGMENTATION. LATER
TESTS DID NOT INCLUDE THESE HOLES.



LEGEND

● D.H. NO.

DEPTH

ANFO LOAD (lbs.)

COLLAR

INITIATION SEQUENCE

HOOK-UP AND DELAY (msec.)

HW TOE (PRE-BLAST)

HW CREST (PRE-BLAST)

HW TOE (POST-BLAST)

HW CREST (POST-BLAST)

BLAST SIDELINE AREA OF INFLUENCE

DRILL HOLE LOCATION AND POST
BLAST ELEVATION

DRAWN BY		DATE		CHECKED BY		DATE	
AS CONSTRUCTED - CORRECTIONS COMPLETE							
BY		DATE					
R E V I S I O N S							
NO	DESCRIPTION					DATE	BY

OLD BEN NO. 1
BLAST DESIGN PROGRAM
TEST 1 BLAST 3

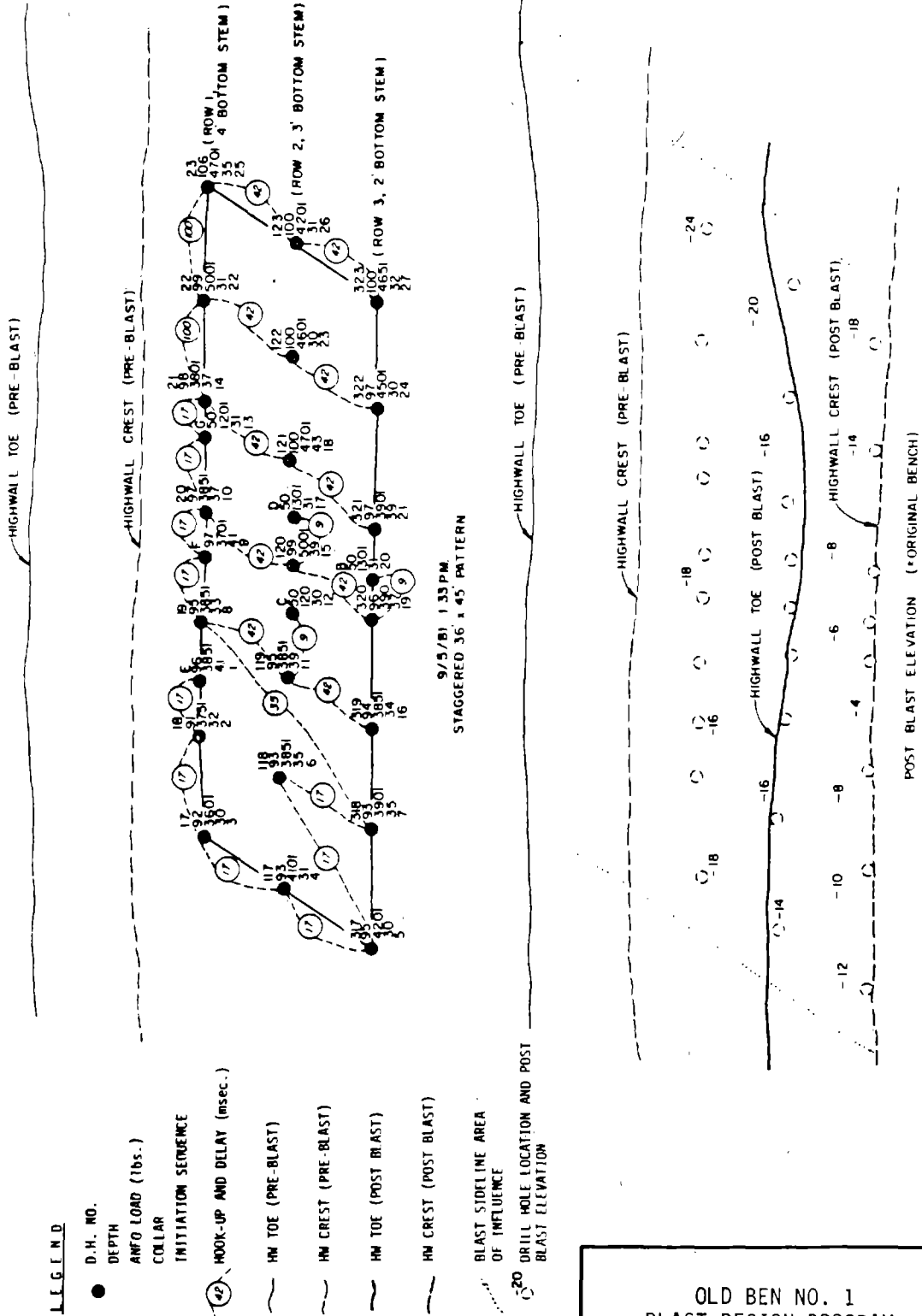
WOODWARD - CLYDE CONSULTANTS
Consulting Engineers Denver, Colorado

SCALE: 1" = 62.5'

Job No. 19943

DRAWING NO. OF
A-8

NOTE: EXTRA HOLES IN PATTERN FOR
BITTER LAP ROCK FRAGMENTATION.
LATER IT'S DID NOT INCLUDE
THESE HOLES.



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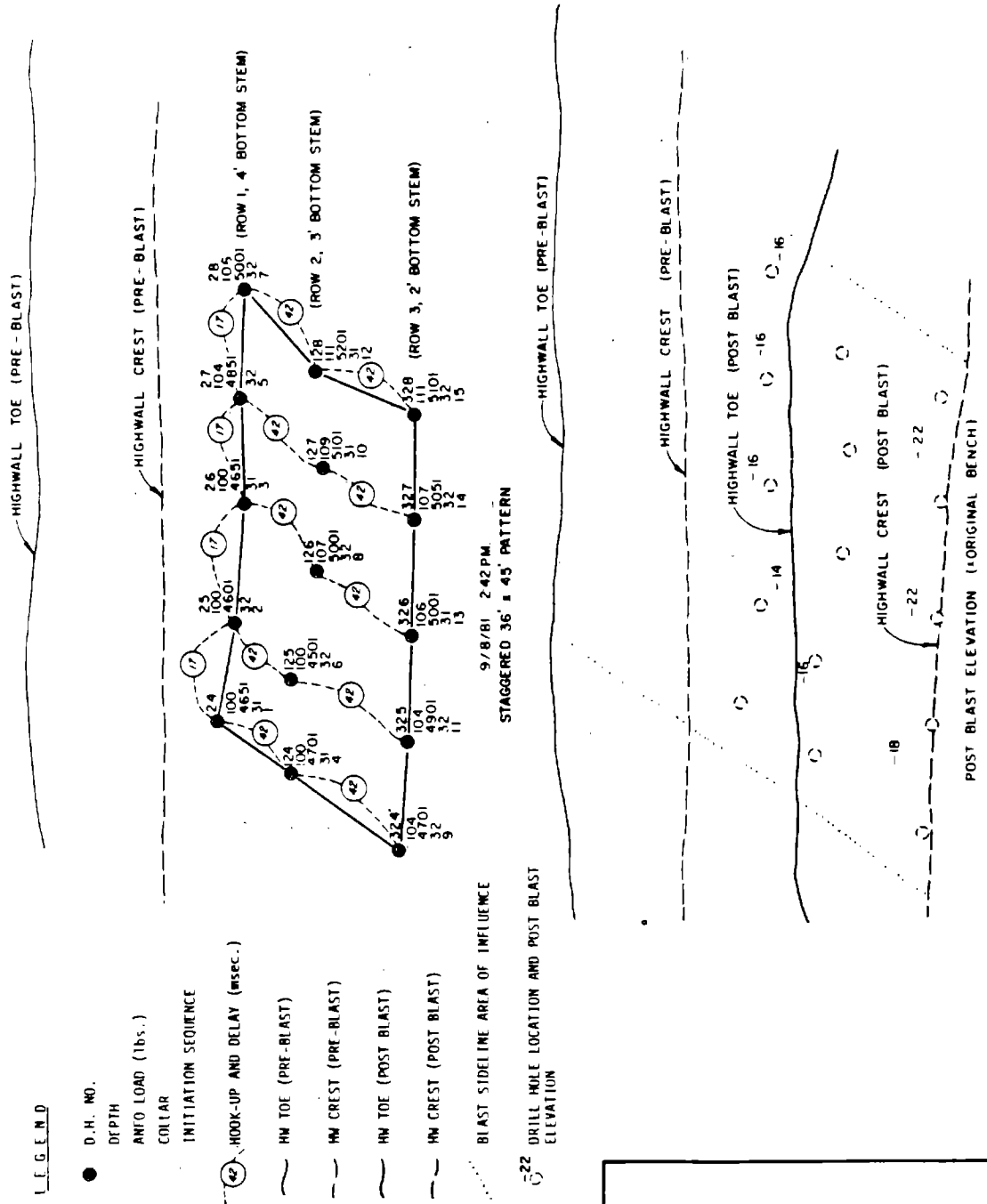
OLD BEN NO. 1
BLAST DESIGN PROGRAM
TEST 1 BLAST 4

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers ● Denver, Colorado

SCALE: 1" = 62.5'

Job No. 19943

DRAWING NO. OF
A-9



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OLD BEN NO. 1
BLAST DESIGN PROGRAM
TEST 1 BLAST. 5

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers Denver, Colorado

SCALE: 1" = 62.5'

Job No. 19943

DRAWING NO	OF
A-10	

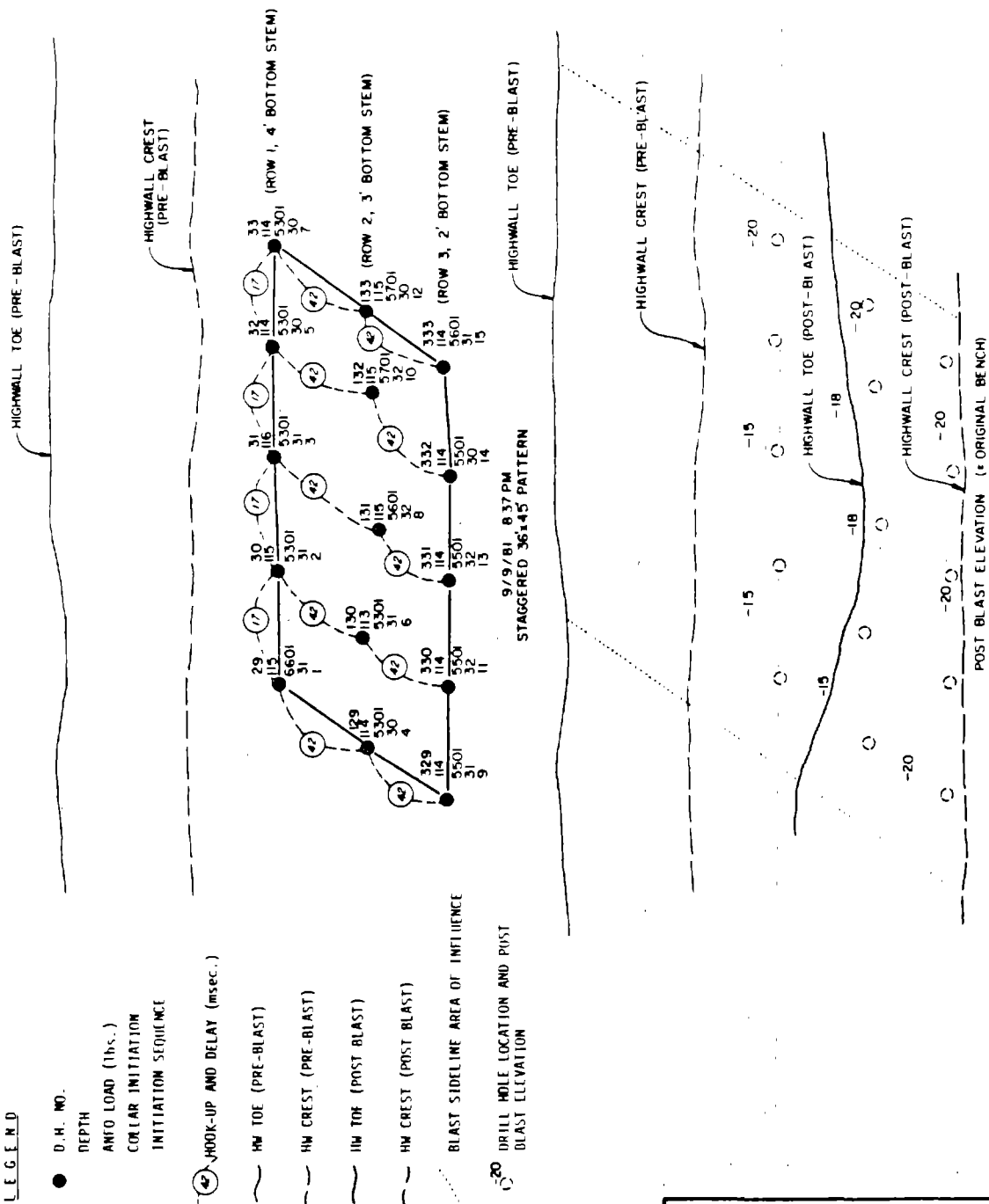
LEGEND

D. H. NO. _____
 DEPTH _____
 ANFO LOAD (lbs.) _____
 COLLAR INITIATION _____
 INITIATION SEQUENCE _____

 (42) HOOK-UP AND DELAY (msec.) _____
 _____ HW TOE (PRE-BLAST) _____
 _____ HW CREST (PRE-BLAST) _____
 _____ HW TOE (POST BLAST) _____
 _____ HW CREST (POST BLAST) _____

 BLAST SIDELINE AREA OF INT. _____

 DRILL HOLE LOCATION AND
 BLAST ELEVATION



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AS CONSTRUCTED - CORRECTIONS COMPLETE						
BY _____		DATE _____				
R E V I S I O N S						
NO	DESCRIPTION				DATE	BY

OLD BEN NO. 1
BLAST DESIGN PROGRAM
TEST 1 BLAST 6

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers • Denver, Colorado

SCALE: 1" = 62.5'

Job No. 19943

A-11

LEGEND

● D.H. NO.

DEPTH

ANFO LOAD (lbs)

COLLAR INITIATION

SEQUENCE

④2 HOOK-UP AND DELAY (msec.)

— HW TOE (PRE-BLAST)

— HW CREST (PRE-BLAST)

— HW TOE (POST-BLAST)

— HW CREST (POST-BLAST)

— BLAST SIDELINE AREA OF INFLUENCE

○ DRILL HOLE LOCATION AND POST-BLAST ELEVATION

HIGHWALL TOE (PRE-BLAST)

HIGHWALL CREST (PRE-BLAST)

HIGHWALL TOE (PRE-BLAST)

HIGHWALL CREST (PRE-BLAST)

HIGHWALL TOE (POST-BLAST)

HIGHWALL CREST (POST-BLAST)

POST-BLAST ELEVATION (+ ORIGINAL BENCH)

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AS CONSTRUCTED - CORRECTIONS COMPLETE							
BY		DATE					
REVISIONS							
NO	DESCRIPTION					DATE	BY

OLD BEN NO. 1
BLAST DESIGN PROGRAM
TEST 1 BLAST 7

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers Denver, Colorado

SCALE: 1" = 62.5'

Job No. 19943

DRAWING NO. OF

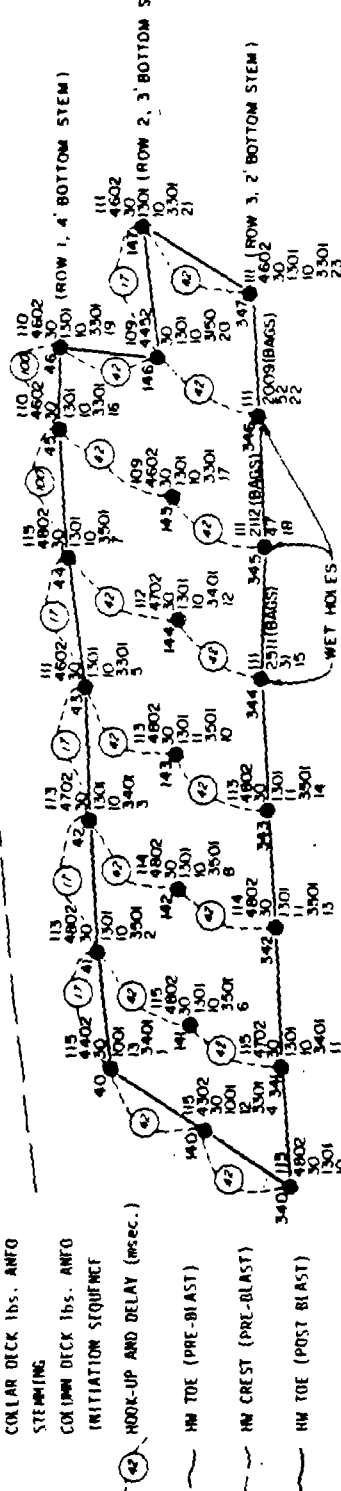
A-12

LEGEND

- D.H. NO. ● DEPTH
- TOTAL ANFO LOAD (lbs.)
- COLLAR
- COLLAR DECK lbs. ANFO
- STEMMING
- COLUMN DECK lbs. ANFO
- INITIATION SEQUENCE
- HOOK-UP AND DELAY (msec.)
- HW TOE (PRE-BLAST)
- HW CREST (PRE-BLAST)
- HW TOE (POST-BLAST)
- HW CREST (POST-BLAST)
- BLAST SIDE LINE AREA OF INFLUENCE
- 20 DRILL HOLE LOCATION AND POST-BLAST ELEVATION

HIGHWALL TOE (PRE-BLAST)

HIGHWALL CREST (PRE-BLAST)



OLD BEN NO. 1
BLAST DESIGN PROGRAM
TEST 1 BLAST 8

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers Denver, Colorado

SCALE: 1" = 62.5'

Job No. 19943

DRAWING NO. A-13

OF

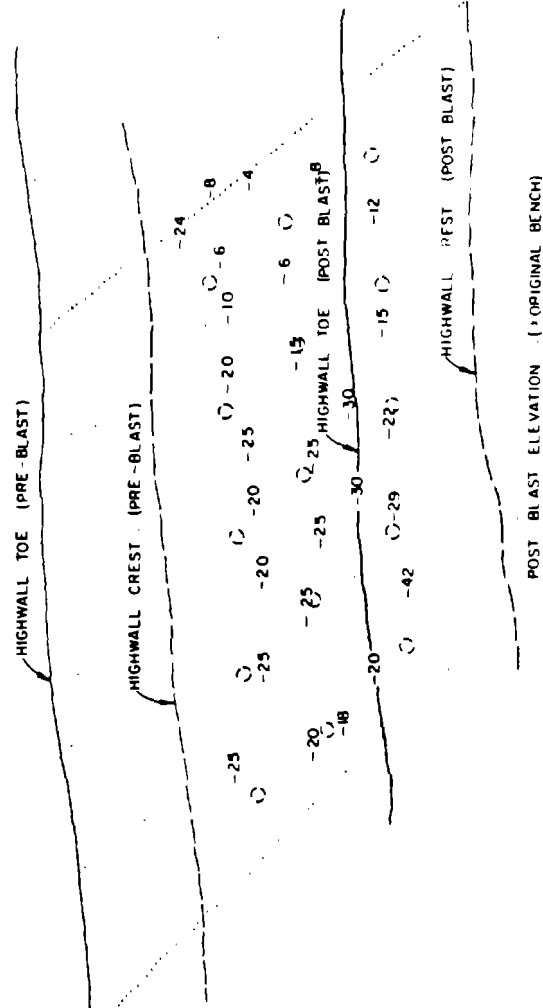
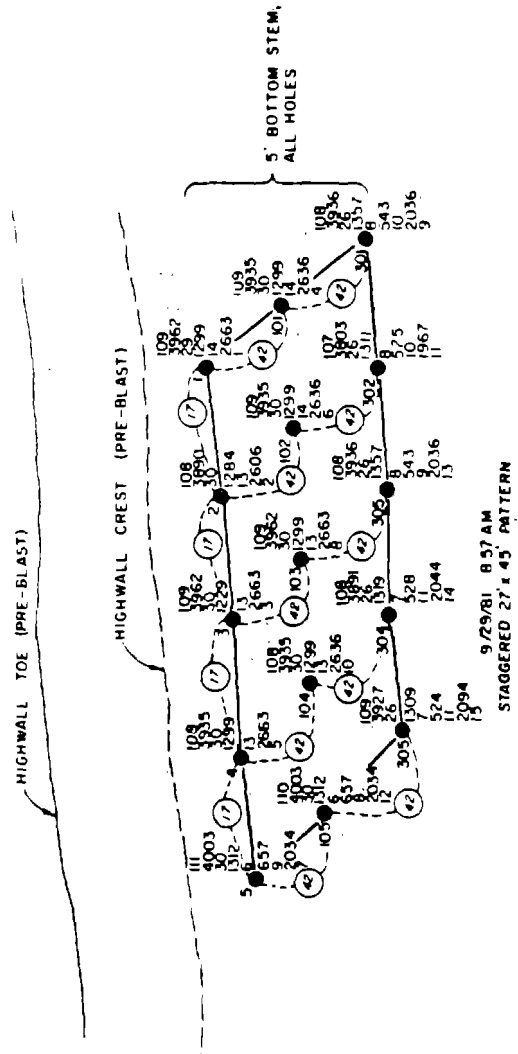
LEGEND

D.H. NO. ●

- DEPTH
- TOTAL ANFO LOAD (lbs)
- COLLAR
- COLLAR DECK lbs ANFO
- STEMMING ft.
- COLUMN DECK lbs ANFO (IF REQ'D)
- STEMMING ft.
- COLUMN DECK lbs ANFO (IF REQ'D)
- INITIATION SEQUENCE
- HOOK-UP AND DELAY (msec.)
- HM TOE (PRE-BLAST)
- HM CREST (PRE-BLAST)
- HM TOE (POST-BLAST)
- HM CREST (POST-BLAST)

BLAST SIDELINE AREA OF INFLUENCE

○ 20' DRILL HOLE LOCATION AND POST-BLAST ELEVATION



OLD BEN NO. 1
BLAST DESIGN PROGRAM
TEST 2 BLAST 1

BY	DATE	CHECKED BY	DATE
AS CONSTRUCTED - CORRECTIONS COMPLETE			
NO	DESCRIPTION	DATE	BY

WOODWARD - CLYDE CONSULTANTS	
Consulting Engineers	Denver, Colorado
SCALE: 1" = 62.5'	DRAWING NO. A-14
Job No. 19943	OF

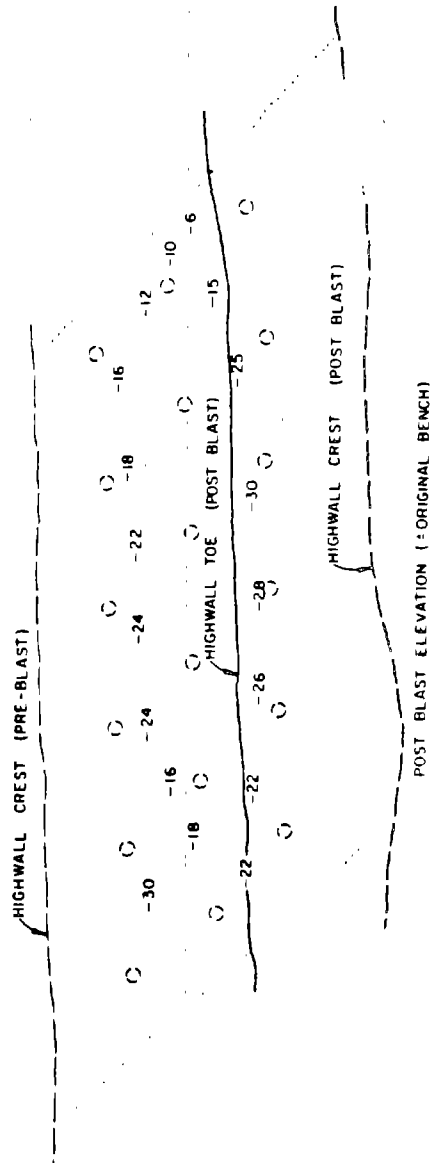
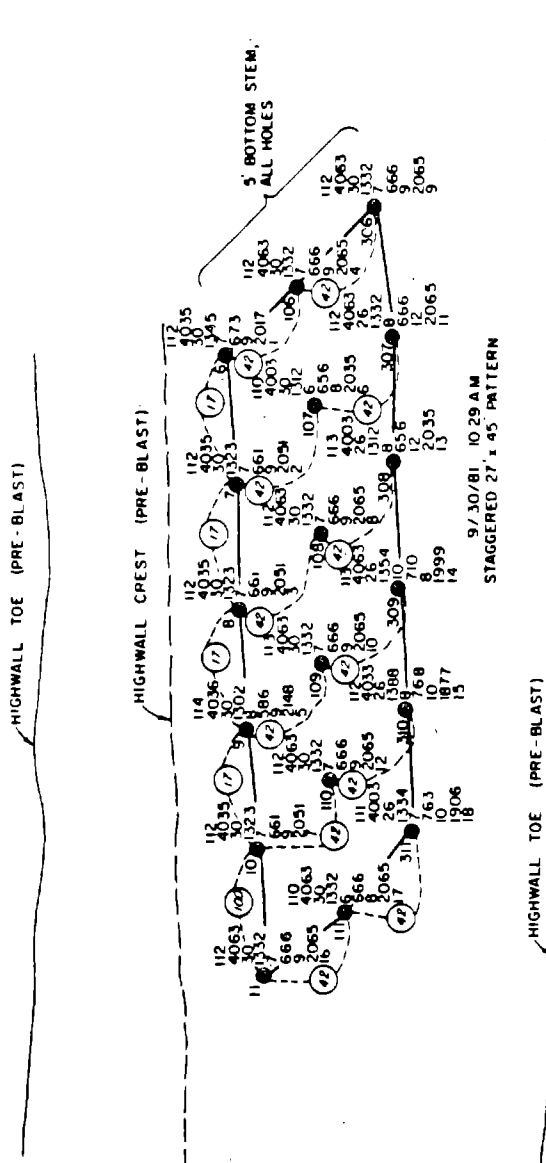
LEGEND

D.H. NO. ●

- DEPTH
- TOTAL ANFO LOAD (lbs)
- COLLAR
- COLLAR DECK lbs ANFO
- STEMMING, ft.
- COLUMN DECK lbs ANFO (IF REQ'D)
- STEMMING, ft.
- COLUMN DECK lbs ANFO (IF REQ'D)
- INITIATION SEQUENCE
- HOOK-UP AND DELAY (msec.)
- HW TOE (PRE-BLAST)
- HW CREST (PRE-BLAST)
- HW TOE (POST BLAST)
- HW CREST (POST BLAST)

BLAST SIDELINE AREA OF INFLUENCE

DRILL HOLE LOCATION AND POST BLAST ELEVATION



OLD BEN NO. 1
BLAST DESIGN PROGRAM
TEST 2 BLAST 2

DRAWN BY		DATE		CHECKED BY		DATE	
AS CONSTRUCTED - CORRECTIONS COMPLETE							
BY		DATE					
REVISIONS							
NO	DESCRIPTION					DATE	BY

WOODWARD - CLYDE CONSULTANTS	
Consulting Engineers	Denver, Colorado
SCALE: 1" = 62.5'	DRAWING NO. OF
Job No. 19943	A-15

LEGEND

D.H. NO. 1

DEPTH

TOTAL ANFO LOAD (lbs)

COLLAR

COLLAR DECK lbs ANFO

STEMMING ft.

COLLAR DECK lbs ANFO (IF REQ'D)

STEMMING ft. (IF REQ'D)

COLLAR DECK lbs ANFO (IF REQ'D)

STEMMING ft. (IF REQ'D)

INITIATION SEQUENCE

HOOK-UP AND DELAY (msec.)

5' BOTTOM STEM, ALL HOLES

HW TOE (PRE-BLAST)

HW CREST (PRE-BLAST)

HW TOE (POST-BLAST)

HW CREST (POST-BLAST)

BLAST SIDELINE AREA OF INFLUENCE

20 DRILL HOLE LOCATION AND POST-BLAST ELEVATION

HIGHWALL TOE (PRE-BLAST)

HIGHWALL CREST (PRE-BLAST)

HIGHWALL CREST (PRE-BLAST)

HIGHWALL CREST (POST-BLAST)

POST-BLAST ELEVATION (* ORIGINAL BENCH)

DRAWN BY _____		DATE _____		CHECKED BY _____		DATE _____	
AS CONSTRUCTED - CORRECTIONS COMPLETE							
BY _____				DATE _____			
R E V I S I O N S							
NO	DESCRIPTION					DATE	BY

OLD BEN NO. 1
BLAST DESIGN PROGRAM
TEST 2 BLAST 3

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers Denver, Colorado

SCALE: 1" = 62.5'

Job No. 19943

DRAWING NO. CF

A-16

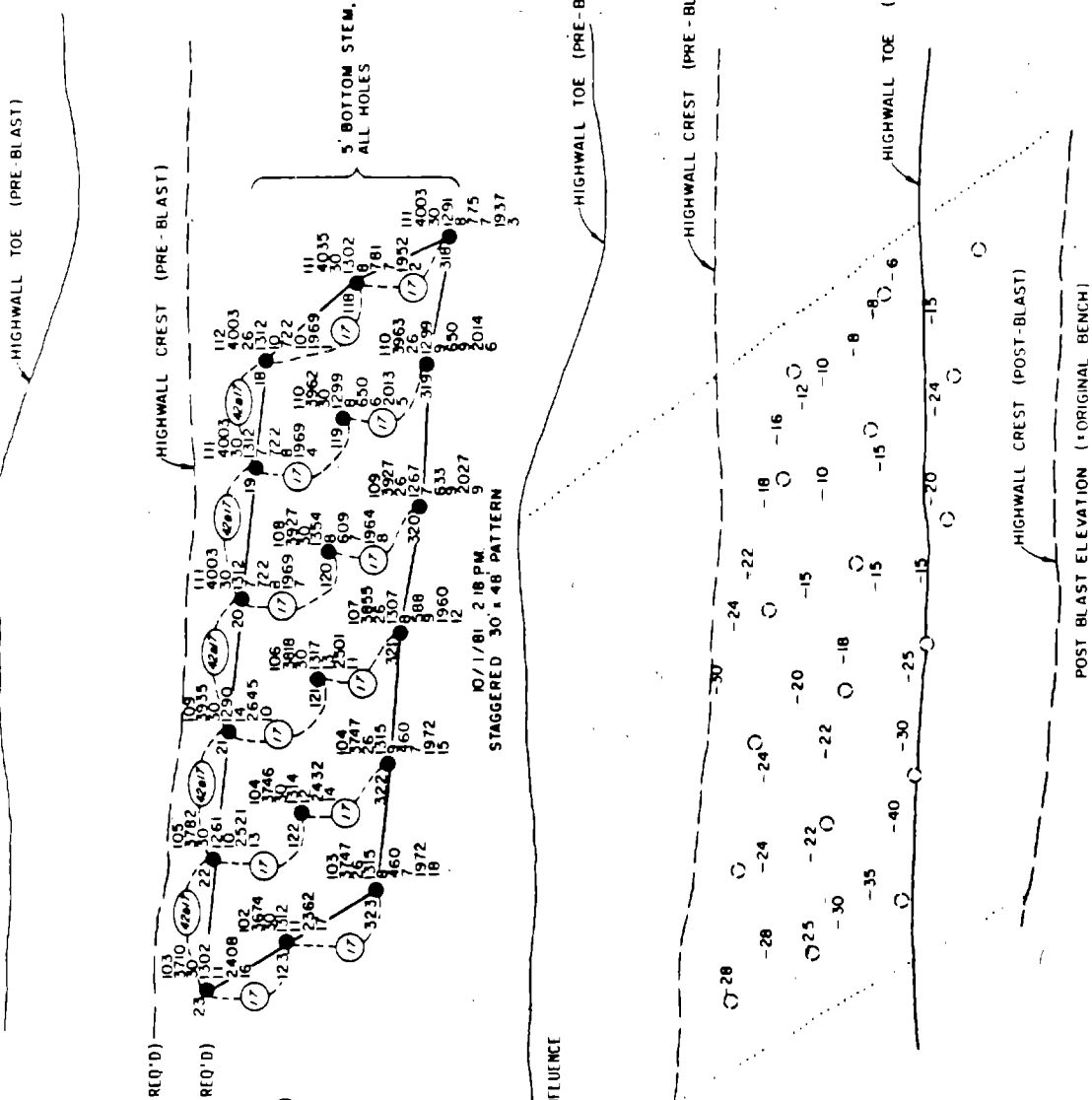
LEGEND

D.H. NO. ●

- DEPTH
- TOTAL ANFO LOAD (lbs.)
- COLLAR
- COLLAR DECK lbs ANFO
- STEMMING FL.
- COLLAR DECK lbs ANFO (IF REQ'D)
- STEMMING FL.
- COLLAR DECK lbs ANFO (IF REQ'D)
- INITIATION SEQUENCE
- HOOK-UP AND DELAY (msec.)
- HW TOE (PRE-BLAST)
- HW CREST (PRE-BLAST)
- HW TOE (POST-BLAST)
- HW CREST (POST-BLAST)

BLAST SIDELINE AREA OF INFLUENCE

○ 20" DRILL HOLE LOCATION AND POST-BLAST ELEVATION



DRAWN BY		DATE		CHECKED BY		DATE	
AS CONSTRUCTED - CORRECTIONS COMPLETE							
BY		DATE					
REVISIONS							
NO.	DESCRIPTION					DATE	BY

OLD BEN NO. 1
BLAST DESIGN PROGRAM
TEST 2 BLAST 4

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers ● Denver, Colorado
SCALE: 1" = 62.5'
Job No. 19943
DRAWING NO. OF
A-17

LEGEND

D.H. NO. ●

DEPTH

TOTAL ANFO LOAD (lbs)

COLLAR

COLLAR DECK lbs ANFO

STEMMING ft.

COLUMN DECK lbs ANFO (IF REQ'D)

STEMMING ft.

COLUMN DECK lbs ANFO (IF REQ'D)

INITIATION SEQUENCE

④2 HOOK-UP AND DELAY (msec.)

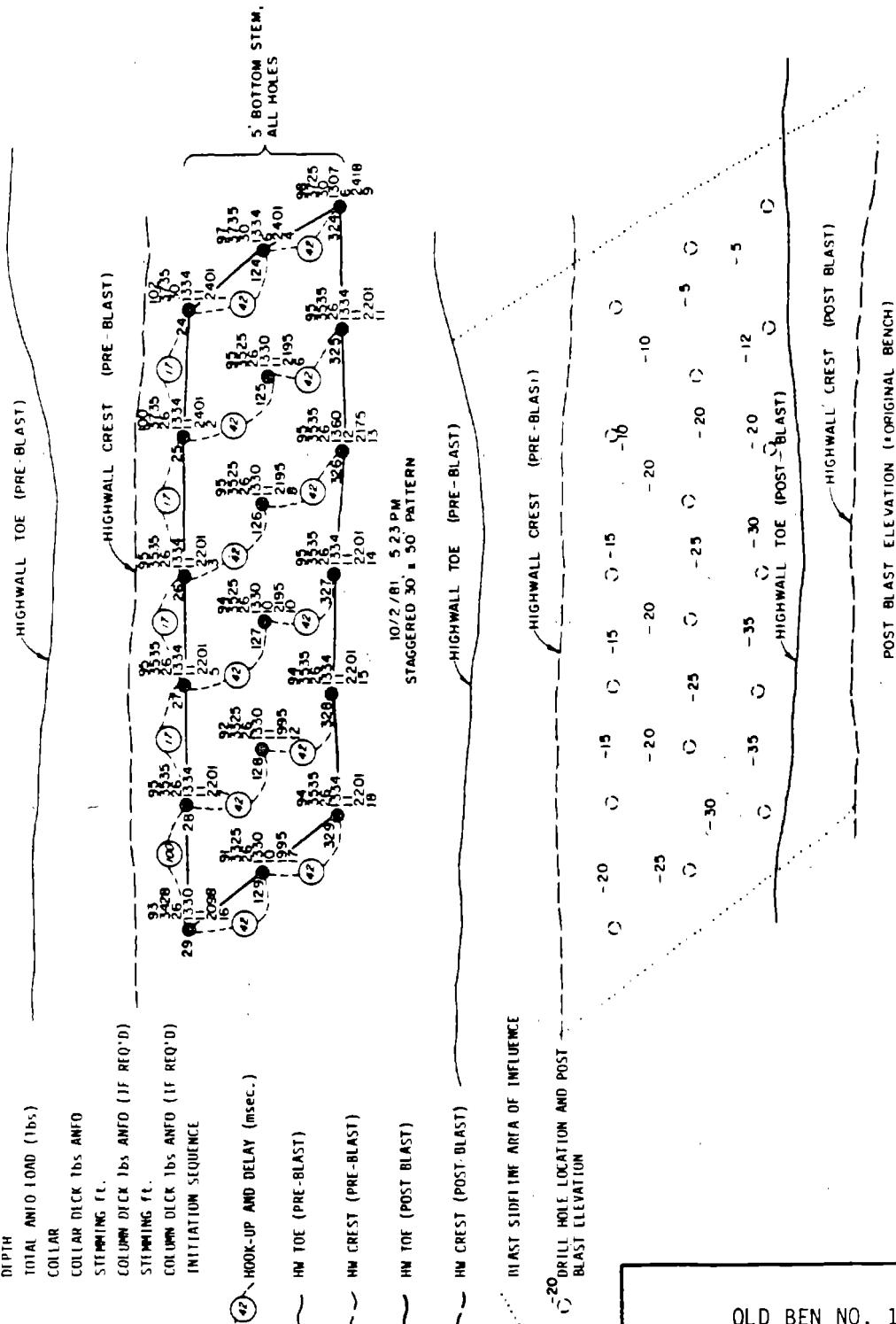
HW TOE (PRE-BLAST)

HW CREST (PRE-BLAST)

HW TOE (POST-BLAST)

HW CREST (POST-BLAST)

BLAST SLOPELINE AREA OF INFLUENCE

○ 20 DRILL HOLE LOCATION AND POST
BLAST ELEVATION

DRAWN BY		DATE		CHECKED BY		DATE	
AS CONSTRUCTED - CORRECTIONS COMPLETE							
BY		DATE					
REVISIONS							
NO	DESCRIPTION					DATE	BY

OLD BEN NO. 1
BLAST DESIGN PROGRAM
TEST 2 BLAST 5

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers ● Denver, Colorado

SCALE: 1"=62.5'
Job No. 19943

DRAWING NO. OF
A-18

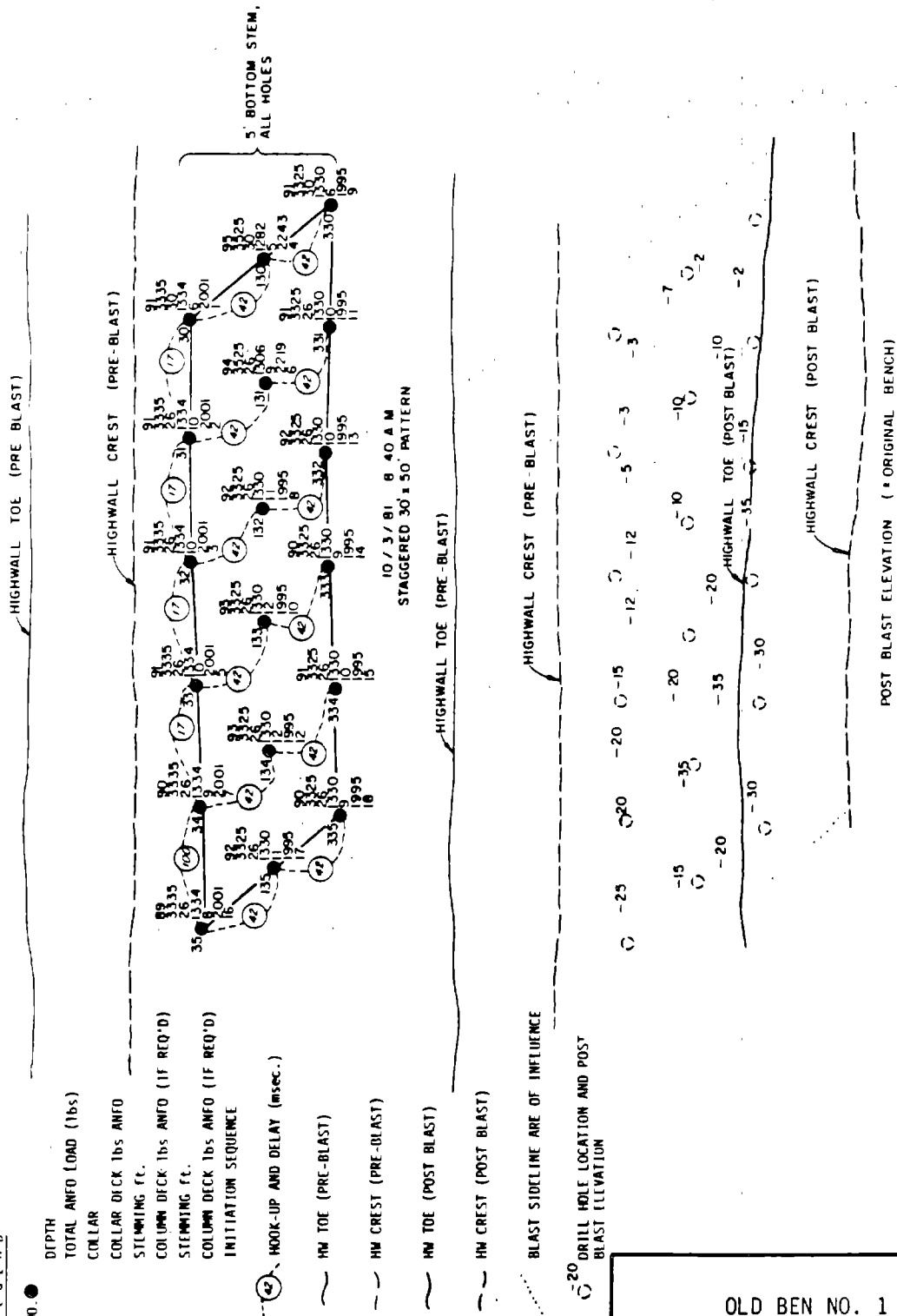
LEGEND

D.H. NO. ●

- DEPTH
- TOTAL ANFO LOAD (lbs)
- COLLAR
- COLLAR DECK lbs ANFO
- STEMMING ft.
- COLUMN DECK lbs ANFO (IF REQ'D)
- STEMMING ft.
- COLUMN DECK lbs ANFO (IF REQ'D)
- INITIATION SEQUENCE
- HOOK-UP AND DELAY (msec.)
- HW TOE (PRE-BLAST)
- HW CREST (PRE-BLAST)
- HW TOE (POST-BLAST)
- HW CREST (POST-BLAST)

BLAST SIDELINE ARE OF INFLUENCE

○ -20 DRILL HOLE LOCATION AND POST BLAST ELEVATION



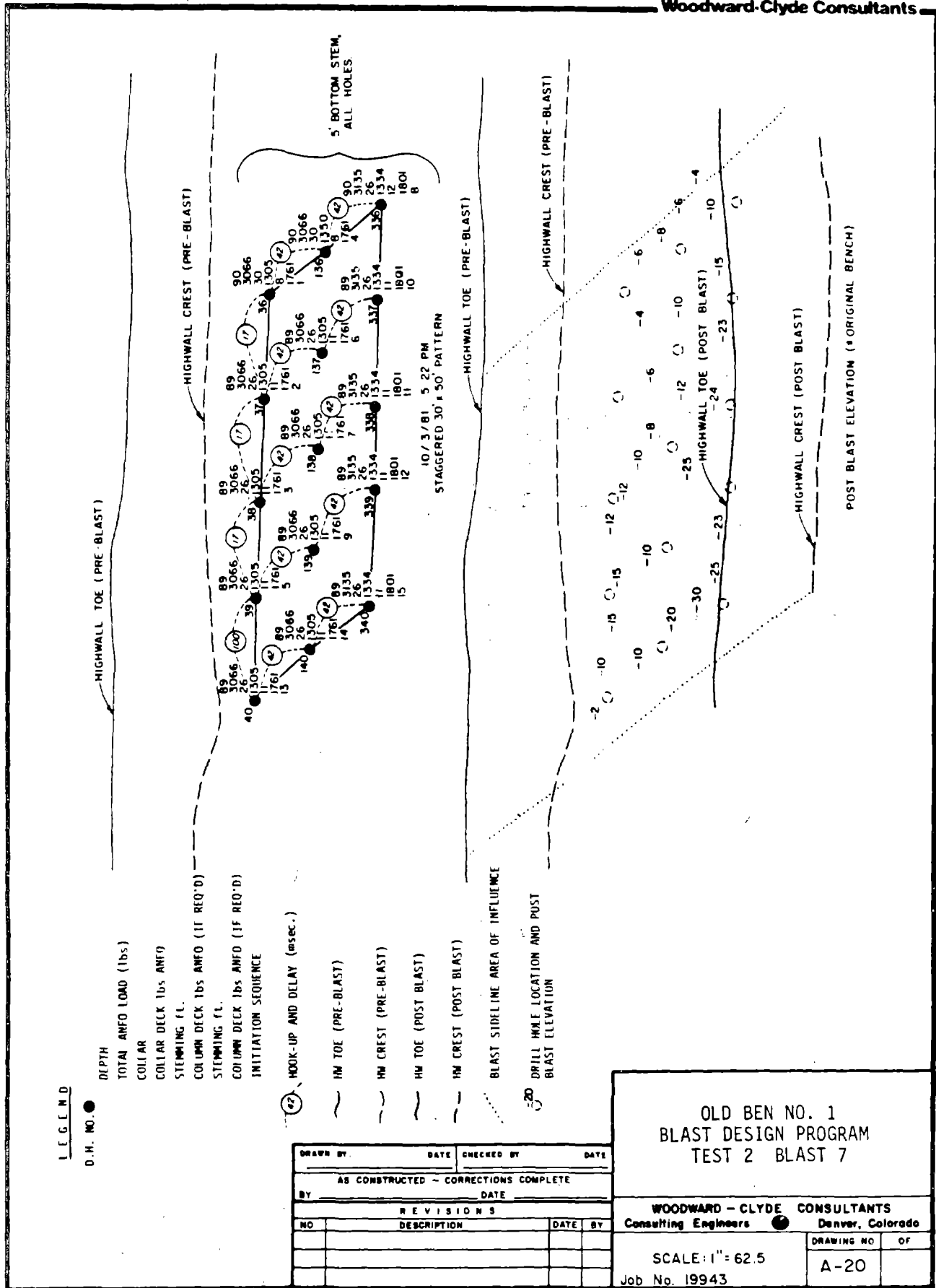
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BY		DATE					
R E V I S I O N S							
NO	DESCRIPTION					DATE	BY

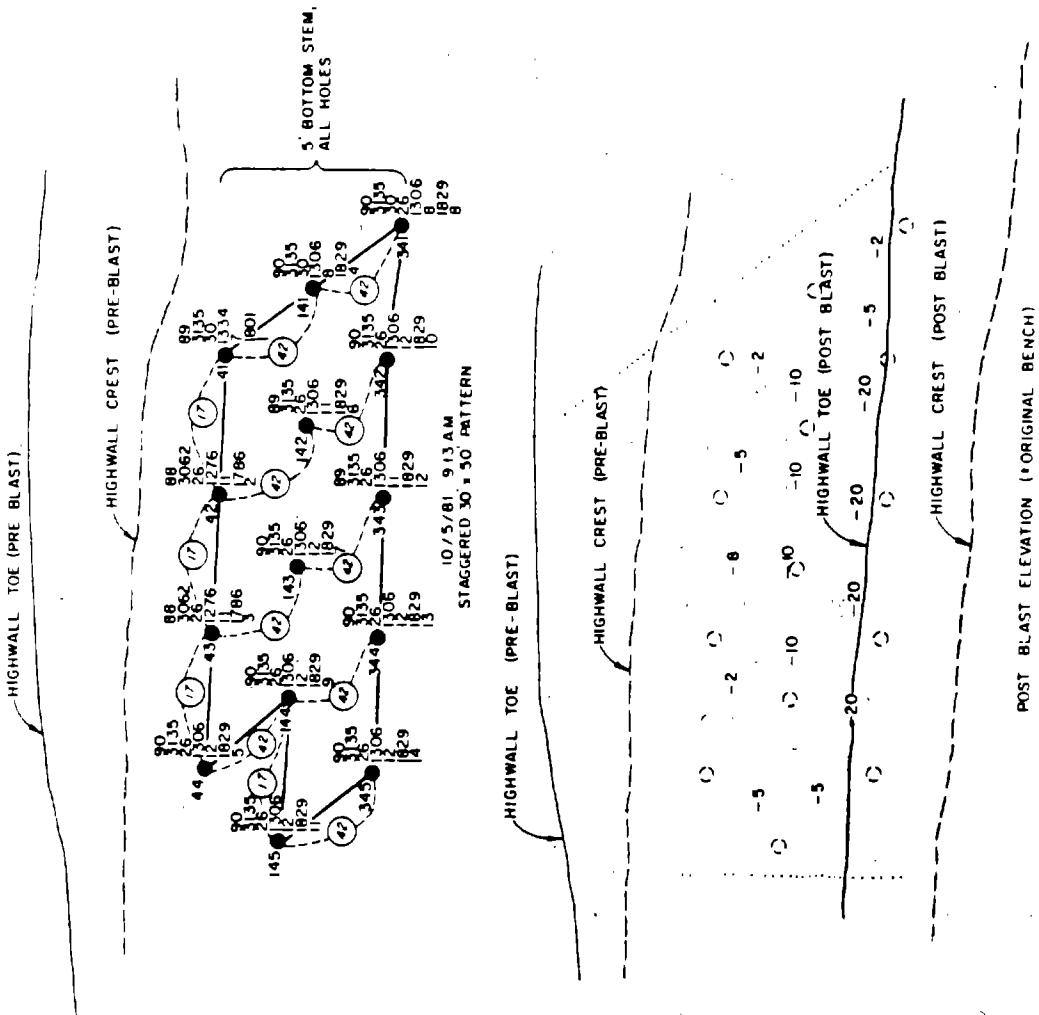
OLD BEN NO. 1
BLAST DESIGN PROGRAM
TEST 2 BLAST 6

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers Denver, Colorado

SCALE: 1" = 62.5'
Job No. 19943

DRAWING NO OF
A-19





LEGEND

D.H. NO. ●

- DEPTH
- TOTAL ANFO LOAD (lbs)
- COLLAR
- COLLAR DECK lbs ANFO
- STEMMING ft.
- COLUMN DECK lbs ANFO (IF REQ'D)
- STEMMING ft.
- COLUMN DECK lbs ANFO (IF REQ'D)
- INITIATION SEQUENCE

④2 HOOK-UP AND DELAY (msec.)

— HW TOE (PRE-BLAST)

— HW CREST (PRE-BLAST)

— HW TOE (POST-BLAST)

— HW CREST (POST-BLAST)

BLAST SIDELINE AREA OF INFLUENCE

DRILL HOLE LOCATION AND POST-BLAST ELEVATION

DRAWN BY		DATE		CHECKED BY		DATE	
AS CONSTRUCTED - CORRECTIONS COMPLETE							
BY		DATE					
REVISIONS							
NO	DESCRIPTION					DATE	BY

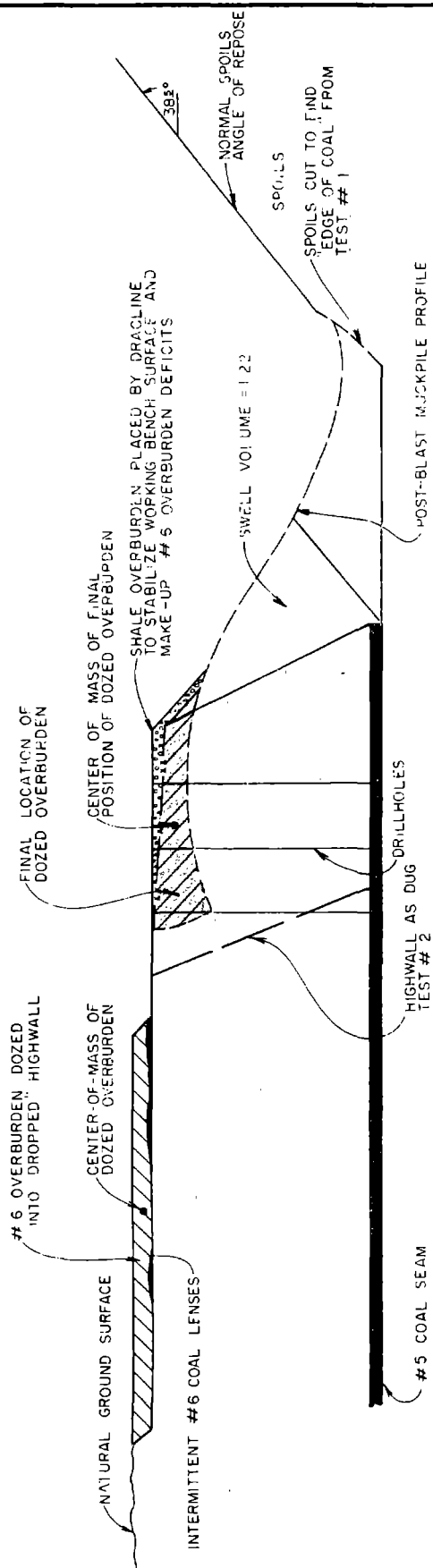
OLD BEN NO. 1
BLAST DESIGN PROGRAM
TEST 2 BLAST 8

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers Denver, Colorado

SCALE: 1"=62.5'

Job No. 19943

DRAWING NO. OF
A-21



(AS OBSERVED)

DATE	CHECKED BY	DATE	OLD BEN HILL BLAST DESIGN
AS CONSTRUCTED - CORRECTIONS COMPLETE	DATE	DATE	POST-BLAST MUCKPILE (TYPICAL) CENTERLINE
REVISIONS	DATE	DATE	TEST #2 DRACLINE WORKING BENCH
DESCRIPTION	DATE	DATE	REBUILD (TYPICAL) TEST #2
NO	DATE	DATE	WOODWARD - CLYDE CONSULTANTS
			Consulting Engineers
			Denver, Colorado
			SCALE 1" = 50'
			DRAWING NO. OF
			A-23

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APPENDIX B

HIGHWALL DESCRIPTIONS AND REGIONAL GEOLOGIC FRAMEWORK
OLD BEN COAL COMPANY NO 1 MINE
WEST FIELD BLAST DESIGN PROGRAM

Geologic data submitted in support of a Department of Energy study entitled "Blast Designs to Improve Dragline Stripping Rates, Phase II."

APPENDIX B

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Figure B-2 Composite Stratigraphic Section, West Field, Old Ben Coal Company No. 1 Mine	B-3

**HIGHWALL DESCRIPTIONS AND REGIONAL GEOLOGIC FRAMEWORK
OLD BEN COAL COMPANY NO. 1 MINE
WEST FIELD BLAST DESIGN PROGRAM**

1.0 INTRODUCTION

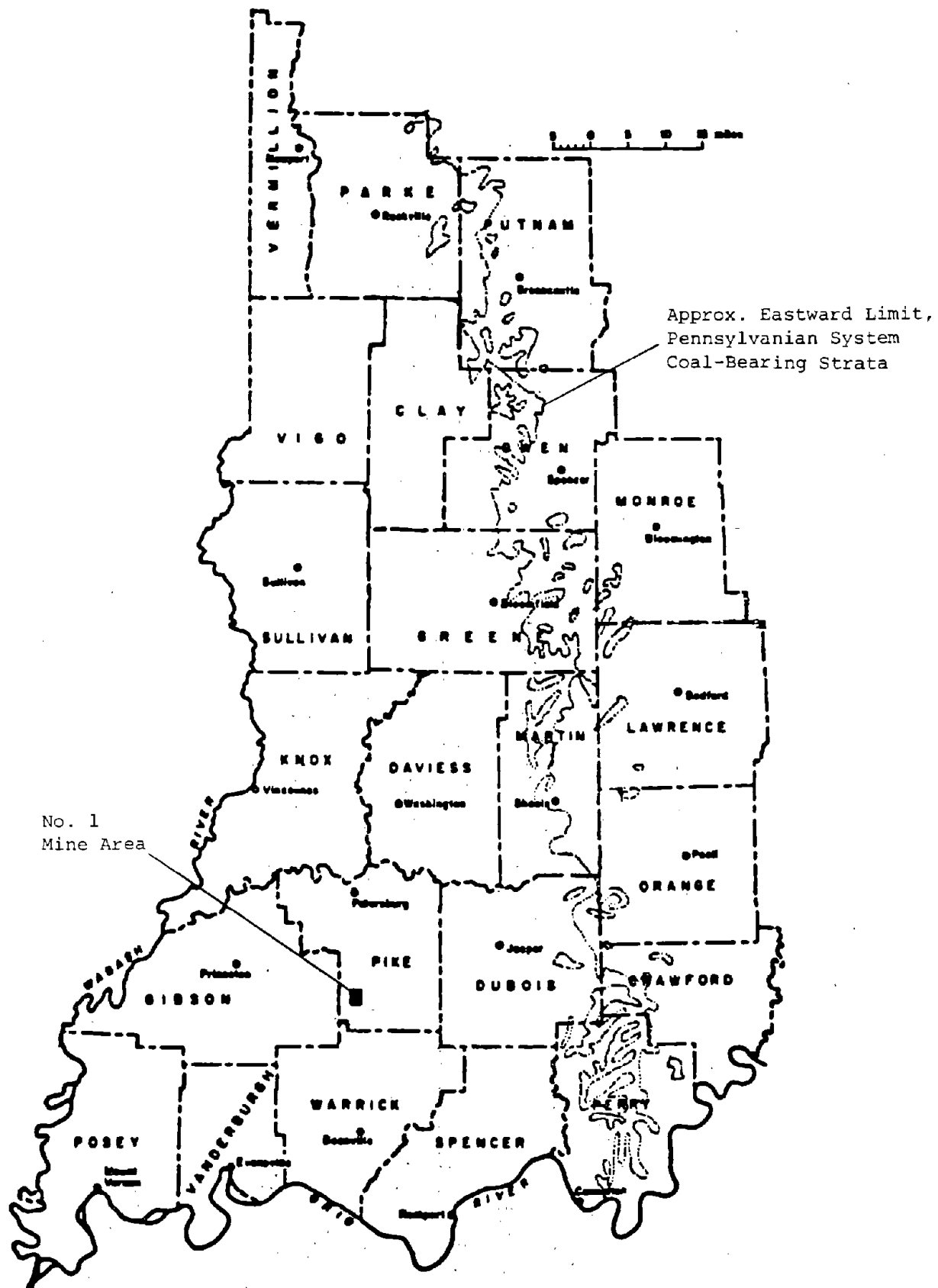
The West Field highwall of Old Ben Coal Company No. 1 mine is located in Sections 9, 15 and 16, approximately 30 miles southeast of Vincennes and 22 miles northeast of Evansville in Pike County, Indiana (Fig. B-1). It is served by an interconnecting network of all-weather state highways. The principal north-south route is State Route 61, located 1 mile east of the field. East-west routes are State Route 64 and Interstate Highway 64, located 4 miles north and 4 miles south respectively. Three miles to the west is State Route 57 which runs southwesterly towards Evansville. Coal from Old Ben Coal Company's No. 1 Mine is shipped by truck on these highways and by rail via a spur of the Algers, Winslow and Western line which connects with the Southern Railway at Oakland City, Indiana.

The mine is located in a relatively densely populated agricultural and rural area. The nearest towns are Oakland City (population 3,300) 4 miles northwest, Arthur (population about 500) 4 miles north, and Spurgeon (population about 500) 1 mile southeast. Southwestern Indiana has a temperate climate with about 38 inches of annual rainfall.

West Field has a maximum relief of 100 feet and an average elevation of 475 feet. The area is located several miles south of the farthest advance of Wisconsin-age glaciation and consists of rolling lowlands containing aggraded streams and broad valleys. West Field is in the Ohio River drainage basin. Most runoff drains to the south and west into Pigeon Creek, which flows south to the Ohio River. The extreme northern edge of West Field is drained by northward flowing streams into the Patoka and the Wabash rivers.

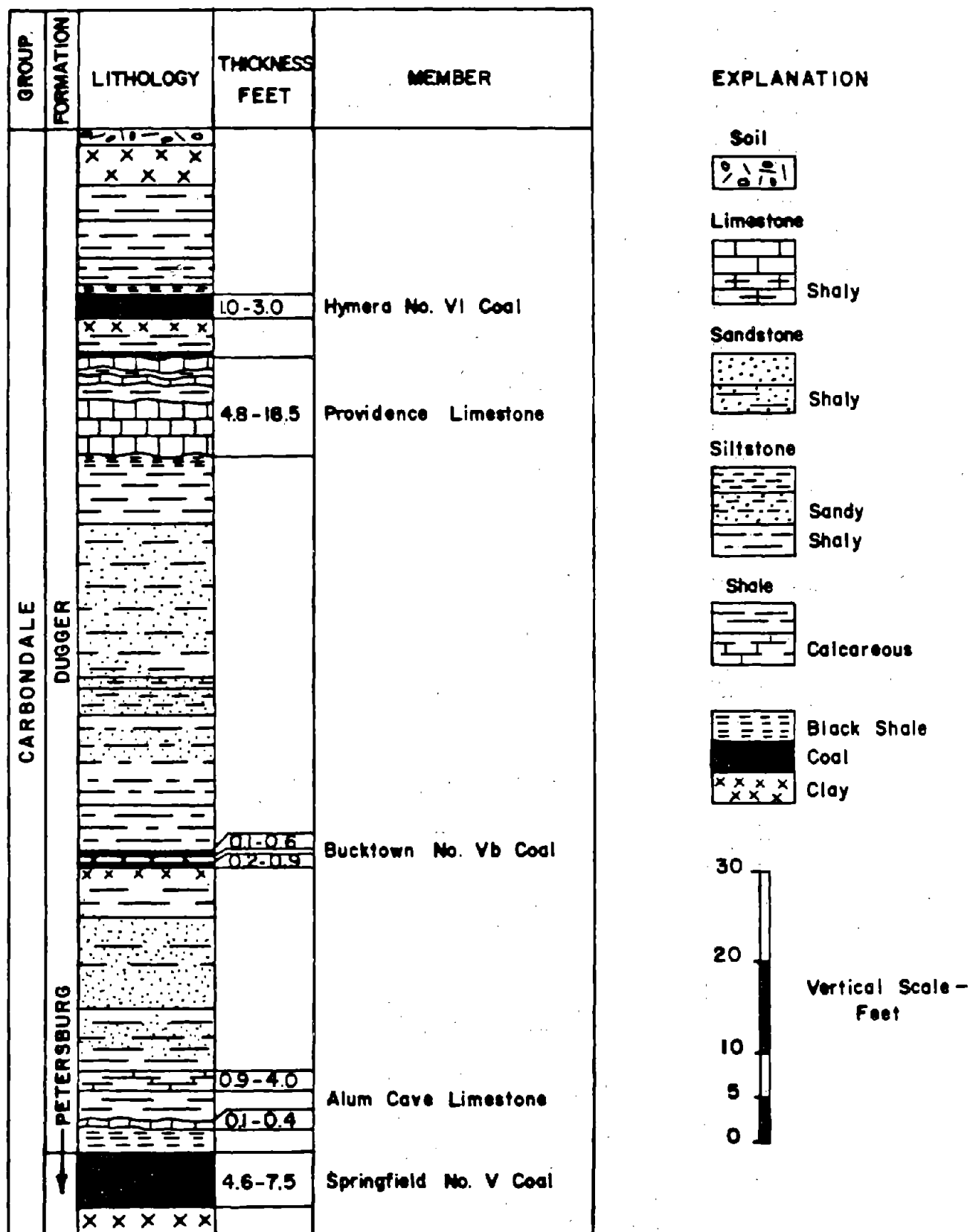
2.0 REGIONAL GEOLOGY AND COAL OCCURRENCE

The West Field of the Old Ben Coal Company's No. 1 Mine lies at the eastern margin of the Illinois Basin Coal Field which underlies much of Illinois and extends into western Kentucky and western Indiana. The basin contains nearly flat-lying or gently dipping beds of sedimentary rock. These beds have a southwesterly dip of 20 feet to 30 feet per mile. (less than one-degree) toward the center of the basin. The highwall lithologies in West Field are within the Middle Pennsylvanian Carbondale Group (Dugger and Petersburg Formations). The Carbondale Group consists of sandstone, siltstone and shale with lesser amounts of clay, coal and limestone (Fig. B-2).



Job No. 19943

LOCATION MAP. OLD BEN COAL COMPANY Fig. B-1
NO. 1 MINE, PIKE COUNTY,
SOUTHWESTERN INDIANA



COMPOSITE STRATIGRAPHIC SECTION, WEST FIELD,
OLD BEN COAL COMPANY NO. 1 MINE

The mine is located one-mile west of outcrops of the Springfield (V) Coal Member and the Hymera (VI) Coal Member. Lower coals in the Carbondale Group occur at greater depths in the mine area and extend farther east. Indiana's coals are ranked high-volatile bituminous B or C. Typical analyses of the Springfield (V) Coal and Hymera (VI) Coal appear in Table B-1.

3.0 FIELD METHODS

Overburden rock samples and lithology descriptions contained in this report are of the West Field highwall which trends northwestward and is being advanced southwestward down-drainage. At the time of field work (10-15-81), the West Field highwall had been divided into two 4400-foot-long highwalls with an access road separating the southeastern half (inactive for 1 1/2 years) from the northwestern half (active). Rock strata of Middle-Pennsylvanian age are well exposed in the highwall; however, the highwall was generally inaccessible for close inspection except at the slopes formed by the access roads which crossed the pits or along the haul roads which extended into the pits. Since no single location allowed access to the entire length of the highwall from the surface down to the Springfield (V) Coal, three separate descriptions of the highwall were completed.

4.0 LOCAL STRATIGRAPHY

The highwall at West Field contains five named members of the Dugger and Petersburg Formations. They are in ascending order: Springfield (V) Coal, Alum Cave Limestone, Bucktown (Vb) Coal, Providence Limestone, and Hymera (VI) Coal. A smut zone occurs locally at the horizon of the Herrin Coal. Although no site-specific work has been completed by the Indiana Geological Survey concerning correlations of strata of the West Field highwall, identification of the beds can be readily completed by use of published stratigraphic columns of the Indiana Geological Survey (Figure B-2).

The primary coal being mined at the Old Ben Coal Company's No. 1 Mine is the major productive seam of Indiana, the Springfield (V) Coal Member, from which nearly 50 percent of Indiana's annual tonnage is produced. It is the uppermost bed of the Petersburg Formation and is regionally persistent. It ranges in thickness from 4 to 13 feet and has an average thickness of about 6 feet. The floor of this coal is locally clay, shale or sandstone; the clay and shale are present in the West Field northwest pit. Strata lying directly above the coal are typically black shale, although sandstone or gray shale occurs locally. One lens of fairly well-bedded, gray shale, 40 feet long and up to 3 feet thick, occurs at the southeast end of the southeast pit.

The Alum Cave Limestone Member typically occurs 3 to 10 feet above the Springfield (V) Coal and most commonly consists of two very fossiliferous limestones separated by thin layers of calcareous shale (Figure B-2).

Table B-1
 Typical Coal Analyses
 Old Ben Coal Co. No. 1 Mine, West Field

	<u>Moisture</u>	<u>Fixed Carbon</u>	<u>Volatile Matter</u>	<u>Ash</u>	<u>Sulfur</u>	<u>BTU</u>
Springfield (V) Coal Member	9%	40%	41%	10%	3-6%	11,600
Hymera (VI) 10,800 Coal Member		11%	41%	34%	9-18%	2.6-3.7%
Old Ben Coal Co. Mine No. 1 Coal	12%	--	--	10%	3.6%	11,400

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Driller logs indicate this unit to be 0.2 to 8.0 feet thick with an average thickness of 2.5 feet. The lower Alum Cave Limestone Member as observed at West Field is thin and argillaceous. It grades upward into calcareous shale. The upper limestone is a thick, distinct, and persistent bed that can be traced along the entire length of the West Field highwall. This limestone is commonly unfractured in contrast to the other strata of the highwall.

The Bucktown (Vb) Coal Member appears as two thin, persistent beds that can be traced along the entire highwall in West Field. The Bucktown (Vb) Coal typically occurs 10 to 67 feet above the Springfield (V) Coal. The average interval of separation is 30 feet. The shale, siltstone and sandstone above the Bucktown (Vb) Coal showed only limited lateral variations in composition and thickness except at the southeast end of the southeast pit.

The Herrin Coal Member is a coaly shale or smut zone locally in the West Field highwall. This coaly horizon is present at the base of the Providence Limestone.

The Providence Limestone Member typically occurs as one to five limestone beds with calcareous shale or clay interbeds. This member, which thins to the north and south of Pike County, averages 20 feet in thickness in West Field. It occurs as only 3 beds, with the lowest of the three being the thickest. Except for the apparent replacement by sandstone in the southeast pit, the limestone is consistent in thickness. The Providence Limestone Member may be very weathered or eroded in parts of West Field as indicated by the absence of limestone in several drillers logs.

The Hymera (VI) Coal Member has also been called the Dugger Coal and Lower Millersburg Coal. The Hymera (VI) Coal is more than six feet thick to the north of Pike County, but within the West Field it varies from 1 to 3 feet in thickness.

5.0 WEST FIELD HIGHWALL LITHOLOGY DESCRIPTION

Field description of Old Ben Coal Company's No. 1 Mine highwall lithology as seen in the West Field pit.

<u>Depth in Feet</u> <u>To Base of Unit</u>	<u>Thickness</u> <u>In Feet</u>	<u>Lithologic Description</u>
--	------------------------------------	-------------------------------

Measured Section 1:

?	?	Soil - a thin, unknown amount of surface material had been removed.
---	---	---

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<u>Depth in Feet To Base of Unit</u>	<u>Thickness In Feet</u>	<u>Lithologic Description</u>
CARBONDALE GROUP		
Dugger Formation		
4.5	4.5	Clay - light brown to yellow-orange, gray zones on vertical fractures, brown root remains throughout, especially on the fractures, damp, soft but firm, is easily broken into irregular clumps, becomes firmer and develops bedding towards base, grades into . . .
8.5	4.0	Shale - light brown with greenish and yellowish zones, gray coloration on vertical fractures, brown root remains throughout, well bedded, soft but firm, easily broken along bedding, well preserved plant remains (leaves and stems) on bedding, occasional red or yellow iron concretion or iron rich shale band becoming more numerous towards base, becomes harder as it grades into . . .
12.5	4.0	Shale - light brown and gray with gray faces on vertical fractures, very firm, bedding is less defined than the above unit, well preserved fern leaves and stems (<u>Neuropteris Scheuchzeri</u> and <u>Percopteris sp.</u>) on bedding surfaces, breaks into irregular planar pieces. Occasional red iron concretions, grades into . . .
15.5	3.0	Shale - yellow-brown, well-bedded, firm, occasional iron concretions, well preserved fern leaves and stems throughout, breaks into irregular pieces, rapid gradation. . .
16.5	1.0	Shale - black, very carbonaceous, very soft but firm, well bedded, over 50% coaly layers, occasional gray shale stringers, crumbles easily, rapid gradation. . .

<u>Depth in Feet To Base of Unit</u>	<u>Thickness In Feet</u>	<u>Lithologic Description</u>
19.0	2.5	Coal - <u>Hymera No. VI Coal Member</u> also known as the lower Millersburg coal, black, blocky, banded, bright, although less so than the No. V coal, pyrite crystals on cleats. . .
23.0	4.0	Shale - covered interval, unable to measure the No. VI coal underclay, approximate thickness taken from drillers logs and field estimation of this interval. . .
Measured Section 2:		
23.3	0.3	Coal - black, bright, blocky, banded, very fractured, 40% vitrain, thickness varies from 0.2" to 0.4", sharp basal contact. . .
23.4	0.1	Limestone - <u>Providence Limestone Member</u> - light gray with pinkish-tan weathered surface, calcareous, fine, massive to poorly bedded, very fractured (from previous blasting?), very hard, breaks with difficulty (rock hammer) into irregular, angular blocks, no fossils seen, varies in thickness from 1.5' to 2.0'. very irregular contacts. . .
25.4	2.0	Shale - gray, well bedded, soft, no fossil or plant impressions seen, breaks easily into small pieces, irregular but sharp contact with. . .
26.4	1.0	Limestone - <u>Providence Limestone Member</u> - light gray, weathers to dark tan or pinkish-tan, calcareous, crystalline, one massive bed, very hard, occasional brachiopod shell seen, occasional cavities (0.1') throughout, breaks with difficulty (rock hammer) into irregular, sharp, angular pieces, ledge forming, occa- sional vertical fracture (from previous blasting?), sharp contact with. . .

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<u>Depth in Feet To Base of Unit</u>	<u>Thickness In Feet</u>	<u>Lithologic Description</u>
28.3	1.9	Shale - Upper 1.1' is gray to greenish gray with brown discoloration on vertical fractures, well bedded, soft, breaks into small angular pieces. Middle 0.5' is dark gray to black, well bedded, very soft, damp, gummy, carbonaceous. Lower 0.3' is gray to greenish gray, well bedded, very soft, damp, gummy, a few plant remains seen. All contacts with this unit are gradational but contact is sharp and irregular with. . .
34.3	6.0	Limestone - <u>Providence Limestone Member</u> - gray with pinkish-tan weathered surface, calcareous crystalline, very hard, occasional brachiopod shell, massive to poorly bedded, vertical fractures, ledge forming and continuous like the above two limestones (although thickness varies from 5.5' to 6.3'), sharp contact with. . .
41.8	7.5	Shale - gray to dark gray, 85% covered, upper section is exposed - a very well bedded shale, . weathers into small, angular plates, firm but easily broken. Waxy near upper contact with limestone, silty and harder near base, non-calcareous, some areas along the highwall have a black shale with carbonaceous material present in upper 1.0', <u>Herrin Coal Member Horizon</u> , distinct contact with. . .
58.8	17.0	Shale and Sandstone - This unit consists of alternating and interbedded layers of shale, siltstone and sandstone with the shales being very silty to sandy and the sandstone being silty to argillaceous. Bedding is irregular in places but distinct units (especially the more massive ones) can be traced across the highwall. Colors vary from light gray to gray with occasional thin pink layers in the massive beds.

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<u>Depth in Feet To Base of Unit</u>	<u>Thickness In Feet</u>	<u>Lithologic Description</u>
		Shales occur as sections of thin, platy beds (0.1' thick) which are hard but easily broken, to thicker, harder beds (1.3' thick). The shales are micaceous and represent approximately 80% of this sequence. Sandstone ranges from well bedded, isolated layers to fairly massive, traceable beds. Thinner beds (0.3' to 0.8') are fine grained, argillaceous, silty, fairly cemented, micaceous and can be easily split by a hammer. Thicker (0.2' to 1.0') beds, of which there are approximately 11 at 2.0', 3.0', 3.7', 5.6', 6.4', 6.8', 7.3', 9.0', 10.2', 11.0' and 13.2' from the base of this unit, are silty and fine grained but appear very hard and well cemented, thus producing massive, very angular, sharp pieces when broken. Very slightly calcareous, occasional ironstone concretions, few plant remains, rapid gradation into. . .
59.8	1.0	Sandstone - light gray, very fine to medium grained, silty to argillaceous, micaceous, massive appearance but can be broken into planar pieces, loosely cemented, varies from 0.6' to 1.1', distinct bedding plane with. . .
62.8	3.0	Siltstone - light gray, dark gray and carbonaceous on bedding, very silty, hard well bedded with somewhat irregular flaser bedding, breaks into planar pieces, upper 0.8' is hard and sandy with fewer plant remains, carbonaceous pieces or layers, distinct contact with. . .
72.8	10.0	Shale and Siltstone - gray to light gray, very well bedded, very even bedding, 0.02' to 0.03' micaceous, firm, hard. Will part on bedding but only the thinner pieces will break in two, very few fractures, carbonaceous plant impressions throughout, especially on bedding planes, occasional clay-ironstone concretions (0.2'), ledge forming, gradational contact becomes overall softer and less silty into. . .

<u>Depth in Feet To Base of Unit</u>	<u>Thickness In Feet</u>	<u>Lithologic Description</u>
77.8	5.0	Shale and Siltstone - dark gray to light gray, micaceous, firm to hard, well to very well bedded, weathered shale breaks into small platy bits. Lower 0.5' is very dark gray, carbonaceous and softer than the rest of unit. 0.1' vitrain band rides 0.2' out of underlying coal with gray shale separation. Iron concretions 0.02' to 0.4' in size throughout, 80% covered, slope forming, sharp contact with. . .
78.3	0.5	Coal - <u>Bucktown No. Vb Coal Member</u> - black, blocky, banded, yellow sulfur weathering on cleats, hard, well bedded with developed cleats, brown crusts (weathered calcite) on cleats, 30% vitrain as thin bands, minor pyrite as disseminated crystals, sharp contact with. . .
78.8	0.5	Clay - dark gray to greenish gray, soft, very well bedded although irregular, lower 0.3' contains thin coal layers and is carbonaceous, sharp contact with. . .
79.4	0.6	Coal - <u>Bucktown No. Vb Coal Member</u> - black, blocky banded, yellow sulfur weathering on fractures and cleats, hard, gray shale lens 0.01' to 0.03' thick extends through the bed. Less than 20% vitrain as thin bands, fusain 10%, upper 0.02' shaley, minor pyrite crystals, sharp contact with. . .
Measured Section 3:		
84.9	5.5	Shale and Sandstone - This unit contains approximately 75% silty, gray shale and 25% argillaceous, tan, fine grained sandstone in an interbedded relationship with all layers occurring less than 0.5' thick as lenses or discontinuous beds. The shales are light gray to gray, very well bedded, soft but firm, silty, occasionally sandy with carbonaceous bits and layers throughout, gummy underclay with root impressions

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<u>Depth in Feet</u> <u>To Base of Unit</u>	<u>Thickness</u> <u>In Feet</u>	<u>Lithologic Description</u>
		in upper 1.0', lower 2.5' becomes silty with frequent thin (0.01' to 0.4') sandstone lenses, wavy sandstone layers develop laterally varying in thickness but remaining fairly continuous, gray, weathers light tan or yellow, fine to medium grained, micaceous, argillaceous with carbonaceous layers and flaser throughout, two sandstone layers seem most prominent, upper sandstone layer is 20% gray shale lenses and is 0.5' thick, lower sandstone is 0.5' thick, easily spilt by hammer, distinct contact with. . .
94.9	10.0	Sandstone - light gray, weathers yellow to pinkish yellow, fine grained, argillaceous to very argillaceous in spots, micaceous, carbonaceous bits and occasional carbonaceous or coaly lenses (0.01') throughout. Minor cross-bedding, flasers present, massive to well bedded, weathers into rounded blocks, ledge forming, 20 to 40% gray shale interbeds, can easily be chipped, rapid gradation into. . .
101.9	7.0	Shale - dark gray, very well bedded, silty with 20% numerous thin sandstone lenses, carbonaceous layers throughout, varies in thickness from 6' to 8', water seeps from this unit all along highwall, sharp contact. . .
103.9	2.0	Limestone - <u>Alum Cave Limestone Member</u> - gray, weathers tan and reddish or black, calcareous, very fine grained, hard, dense, massive, forms a very continuous prominent layer, varies in thickness from 0.9' to 2.0' with local (5.0' to 15.0') thickening to 3.0' or 4.0', very few brachiopod shells, breaks with difficulty into irregular angular blocks, distinct, sharp contact with. . .

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<u>Depth in Feet</u> <u>To Base of Unit</u>	<u>Thickness</u> <u>In Feet</u>	<u>Lithologic Description</u>
107.6	3.7	Shale - dark gray, very well bedded, silty with 20% thin sandstone lenses, occasional carbonaceous piece or layer, water bearing, gradually becomes calcareous and fossiliferous towards base, grades into. . .
108.0	0.4	Limestone - <u>Alum Cave Limestone Member</u> - dark gray, argillaceous to very argillaceous, fossiliferous with brachiopod fragments and crinoid ossicles, massive, very variable in thickness with a 0.4' maximum, irregular contacts, distinct basal contact. . .
110.9	2.9	Shale - black, carbonaceous, very well bedded, hard, very few brachiopods, contains numerous clay-ironstone concretions which are spherical to flat ellipsoidal, 0.5' to 2.5' long, occasionally projecting into the No. V coal, very few thin sandstone lenses, breaks into 1' to 3' flat plates or blocks, varies locally from 1.5' to 3.5', lower 0.4' is soft and coaly, distinct contact with Petersburg Formation
117+	6'+	Coal - <u>Springfield No. V Coal Member</u> - black, bright, blocky, banded, abundant vitrain, calcite on cleats, pyrite appears as small crystals on cleats, as irregular lenses up to 0.2' thick and as hard, pyritic, vertical shale lenses, plant impressions and fusain on bedding, exposed coal has well developed cleats and is fractured, easily broken with hammer.
?	?	Seat rock - varies laterally from a gray to greenish gray, soft, gummy under-clay to a brown to gray, hard, sandy, carbonaceous shale that weathers yellow-tan.

6.0 WEST FIELD HIGHWALL PHOTOGRAPHS

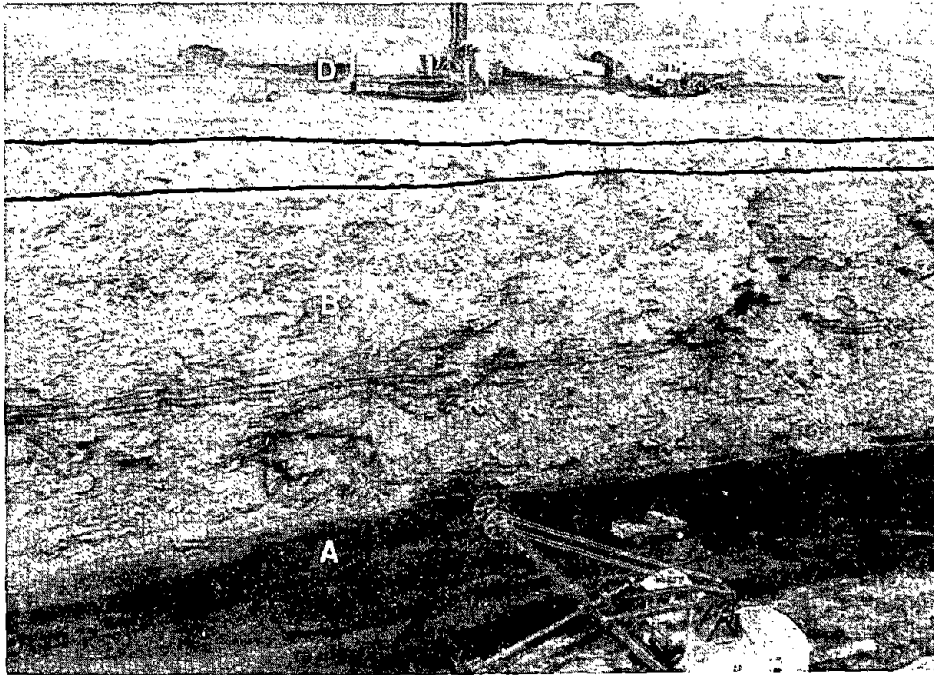


Fig. B-3

West Field highwall; members in ascending order:

- A - Springfield (V) Coal (Petersburg Formation),
- B - Bucktown (Vb) Coal (Dugger Formation),
- C - Providence Limestone Member (Dugger Formation),
- D - Hymera (VI) Coal (Dugger Formation).

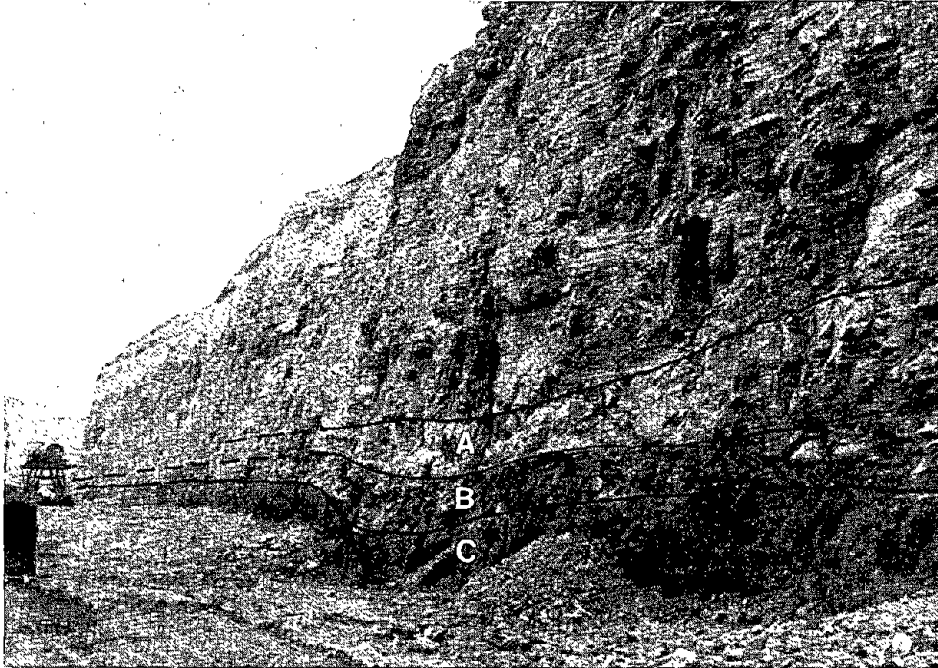


Fig. B-4

Local roll or dip (4 feet amplitude) in the Springfield (V)
Coal in the West Field Northwest pit:

- A - Alum Cave Limestone Member,
- B - Ironstone concretions in Black Shale,
- C - Springfield (V) Coal.

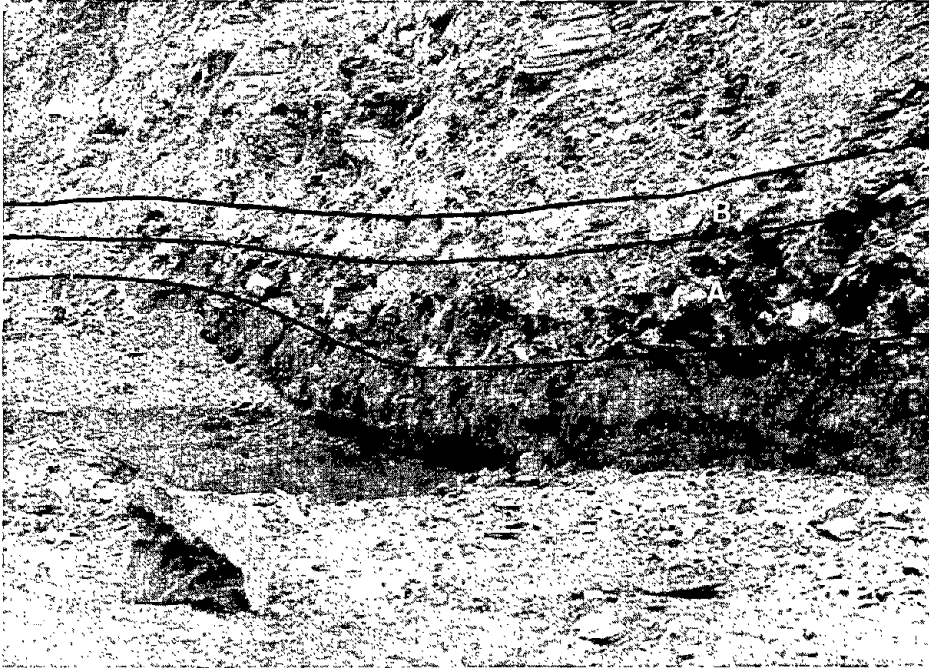


Fig. B-5

Four-foot roll (dip) in Springfield (V) Coal in the West Field northwest pit. The overlying black shale:
A - thickens at these rolls and each bed above has a matching dip. This reflection of the Springfield (V) Coal roll could be traced as high as the Bucktown (Vb) Coal although less pronounced;
B - Alum Cave Limestone Member.

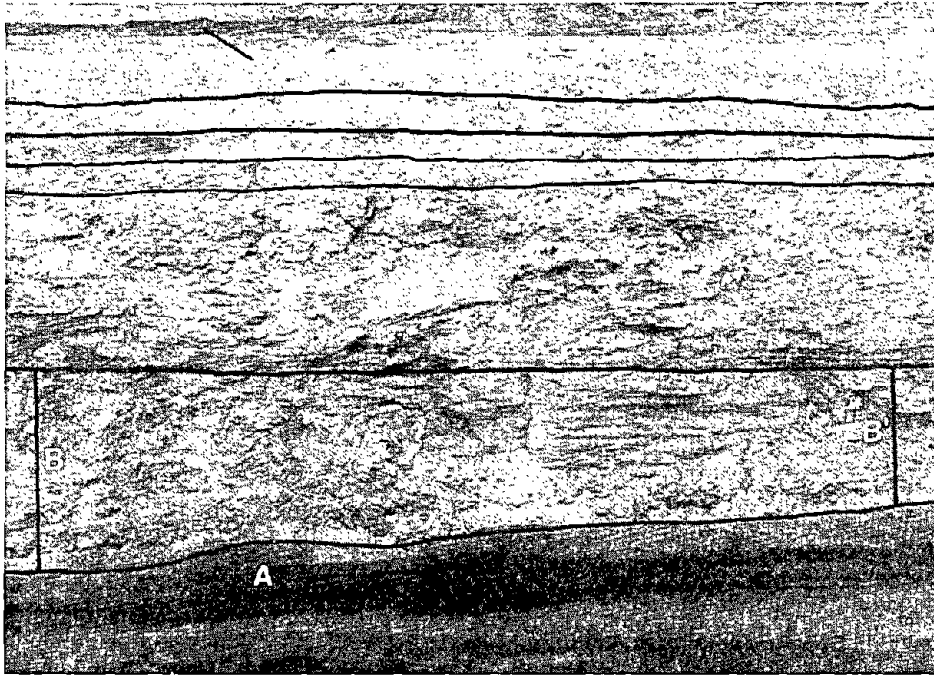


Fig. B-6

West Field highwall, location of 6th test-blast:

A - Springfield (V) Coal dipping southeasterly;

B and B' - V Coal - Vb Coal interval thins to the northwest. Most of the thinning along the 200-foot length of this exposure occurred in the shales ten feet above the coal;

C - Providence Limestone Member layers;

D - Hymera (VI) Coal.

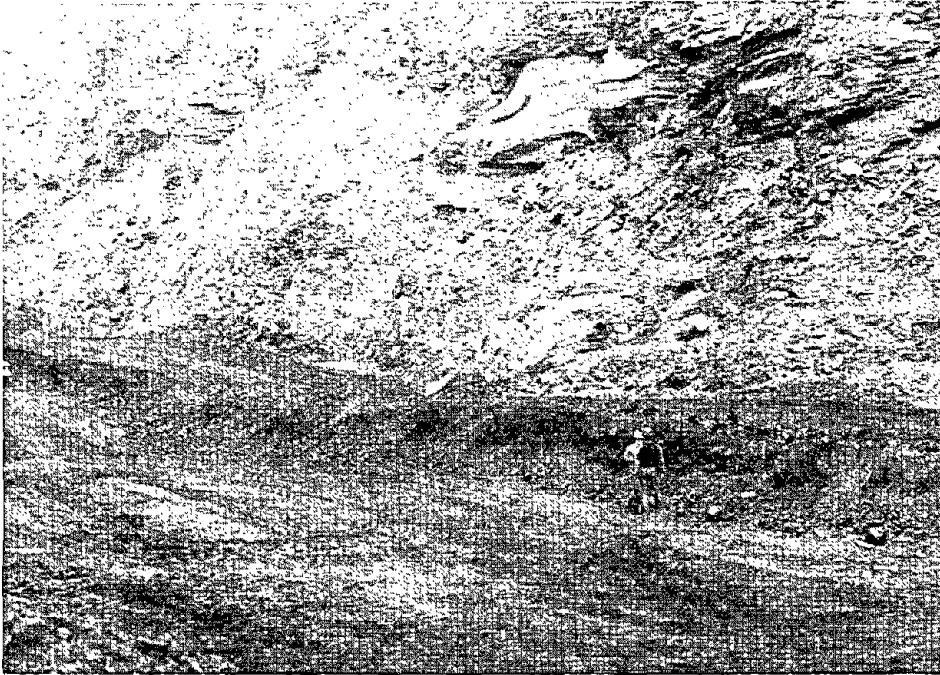


Fig. B-7

West Field highwall, showing southeasterly dip of the
Springfield (V) Coal.

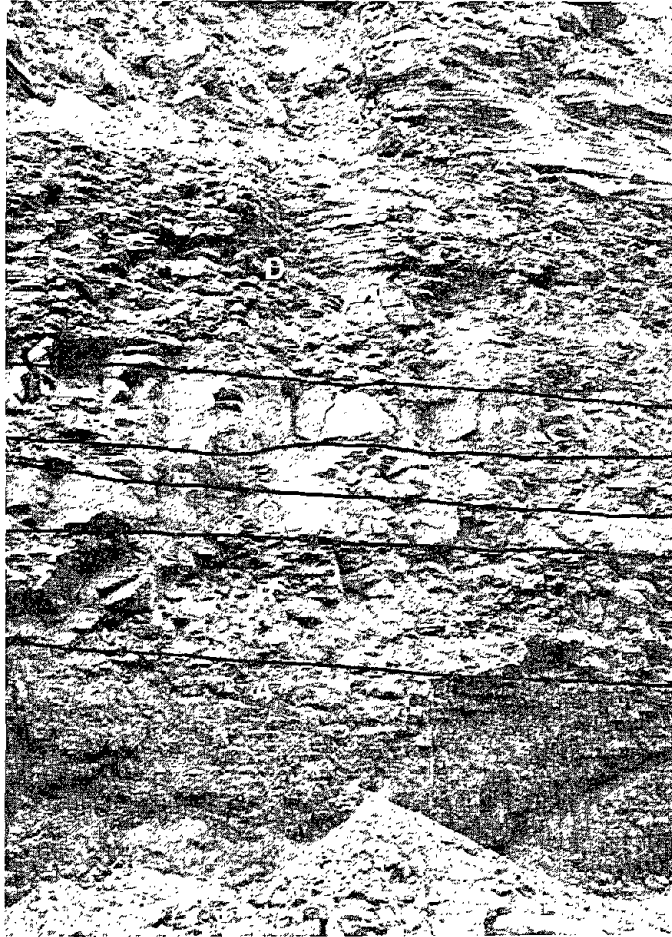


Fig. B-8

Lower highwall lithology:

- A - Springfield (V),
- B - Black Shale,
- C and C' - Alum Cave Limestone Member, and
- D - unnamed sandstone and shale.

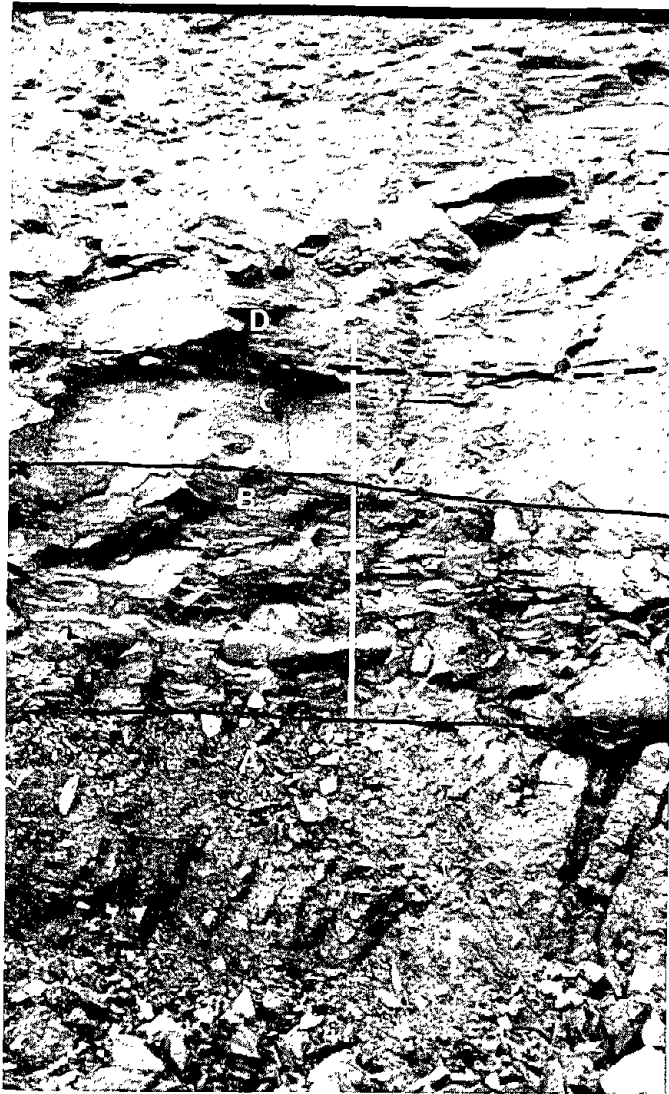


Fig. B-9

Lower highwall lithology:

A - Springfield (V) Coal,

B - Black Shale,

C - Alum Cave Limestone Member,

D - unnamed sandstone and shale (ruler is 6 feet).

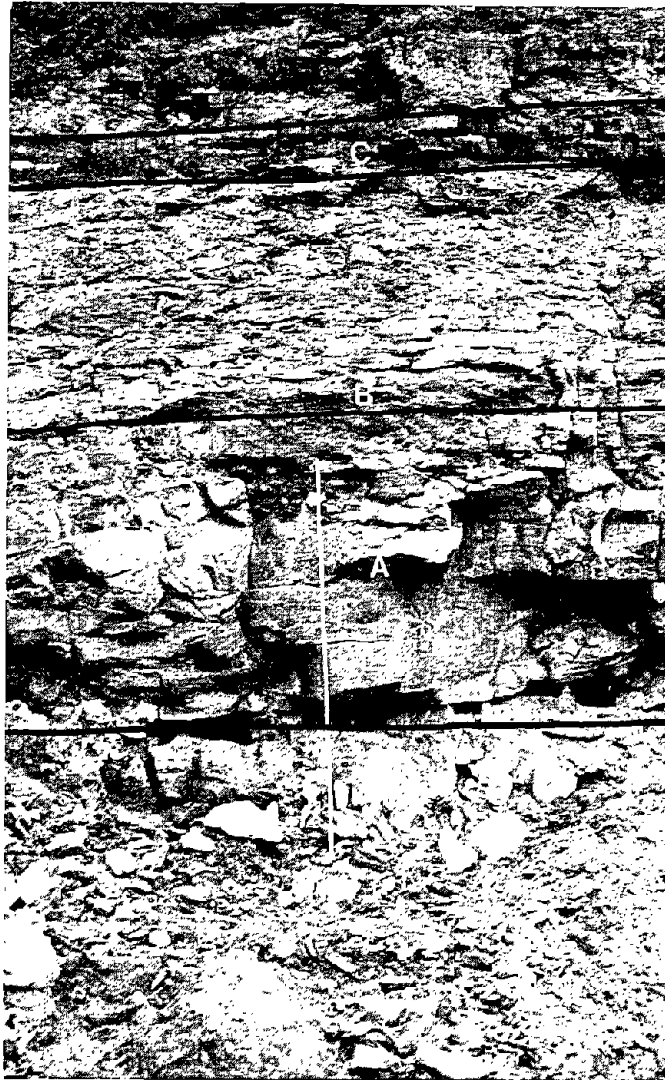


Fig. B-10

Middle highwall lithology:

A - Sandstone of sample H,

B - Unnamed shale,

C - Bucktown (Vb) Coals (ruler is 6 feet). Fifty percent of West Field drillers logs indicate two Bucktown (Vb) Coals. Several drillers logs indicated very thin or no coal at this horizon.

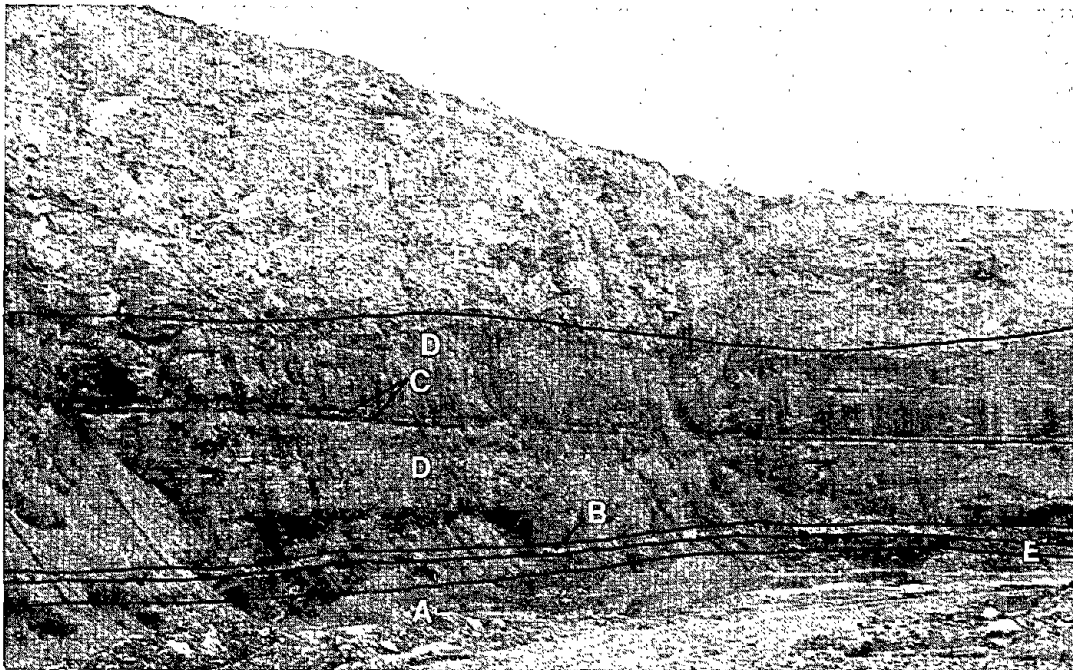


Fig. B-11

West Field highwall in the southeast pit:

- A - Springfield (V) Coal,
- B - Alum Cave Limestone Member,
- C - Bucktown (Vb) coal.
- D - Unnamed sandstone and shale,
- E - Bucktown (Vb) coal.
- D - Light gray, fine to medium grained sandstone sequence cuts down from 30 feet to within 10 feet of the Bucktown (Vb) Coal (E)

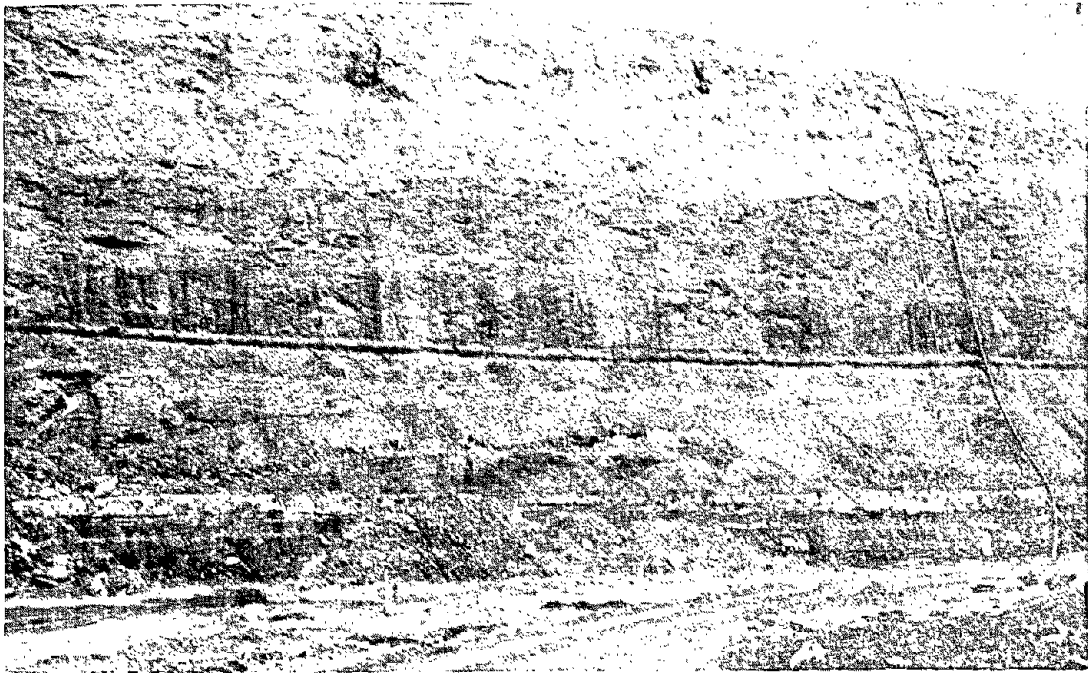


Fig. B-12

Northwest extension of Fig. B-11 with the same lithologies present except sandstone at top is thinner. This sandy sequence is about 300 feet thick and replaces the Providence Limestone Member at the highwall top.



Fig. B-13

Hymera (VI) Coal overburden in the middle of the inactive southeast pit. Interbedded layers of sandstone and shale (0.5 to 1.0 feet thick). Drillers logs indicate the composition of Hymera (VI) Coal overburden to be equal amounts of shale and sandstone throughout West Field.



Fig. B-14

Hymera (VI) Coal overburden, Sandstone
lens in shale (ruler is 6 feet).

APPENDIX C
BLAST FILM ANALYSIS

APPENDIX C

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BLAST FILM ANALYSIS

1.0 PROCEDURES FOR BLAST FILM ANALYSIS

Between September 9, 1981 and October 5, 1981, nine high-speed films were taken of test blasts at the Old Ben No. 1 Mine. The films taken are as indicated in Table C-1. Delay times were obtained for all blasts but face velocities could only be obtained for four blasts. This was due to:

- (a) blast 1-6 was filmed for delay time interval determination and as such had no face markers which could be tracked to follow the face trajectory,
- (b) blast 2-2 had only 1 marker which could be observed,
- (c) blasts 2-5, 2-6, 2-8 came out rather dark, making it difficult to observe the marker trajectories accurately.

Mine survey locations were questionable and many did not correlate well with the analysis results. To perform an accurate analysis, the camera location must be known. The distance to the blast, the angle between the face and camera, and the elevation angle are used to correct the observed displacements (on the screen) to determine the true displacement. If these values are not accurately known, the results can be in error by a considerable amount. Using known reference markers on the blast, the supplied blast patterns, and surveyed camera location, did not yield a good correlation, making camera location adjustment necessary.

Several checks had to be made to obtain the correlation between the known and observed distances on the screen. Taking this into account, the results are estimated to be accurate within 15 percent.

TABLE C-1
SUMMARY OF BLAST FILMS

Date	Blast No.	Time	Results Obtained
9- 9-81	1-6	8:37 am	Delay times
9-11-81	1-8	2:20 pm	Delay times & Face Velocities
9-29-81	2-1	9:00 am	Delay times & Face Velocities
9-30-81	2-2	10:29 am	Delay times
10-1-81	2-3	9:05 am	Delay times & Face Velocities
10-1-81	2-4	2:18 pm	Delay times & Face Velocities
10-2-81	2-5	5:15 pm	Delay times
10-3-81	2-6	8:40 am	Delay times
10-5-81	2-8	9:20 am	Delay times

The delay times were obtained by observing the initiation of the Primacord downlines which were themselves initiated by the Nonel surface lines. This technique allows the surface delay times to be determined but not the downhole times. The number of frames observed between the initiation of each hole was recorded and knowing the time between frames, the surface delay times were derived. There is the possibility that the cord is initiated anytime between the two frames, thus giving an accuracy of ± 1 millisecond.

The method of motion analysis was based on the USBM Report of Investigation - 5584 (Blair, 1960). Basically, the method for determining the face velocities consists of tracking the trajectories of fluorescent orange colored wooden markers which were suspended over the face and noting the elapsed time and their position. The initiation of the first hole was defined as "zero" time and all detonation times and marker position times were the elapsed time (msec) from "zero time."

Once the marker position was known on the screen the coordinates (Y and Z) were recorded with respect to the optic axis or line of sight from the camera to the blast. Using the plan views and the blast pattern, the distance from the camera to the location position, where the optic axis intersects the face, was then found. This also allowed the angle between the camera and the face and the elevation or camera angle of tilt to be determined. These data were then used to derive the scale factors at the different locations on the blast and to make the corrections necessary to convert the coordinates observed on the screen to the true coordinates. The converted coordinates were then plotted on a cross-section profile and the horizontal displacements were obtained for each marker. They were then plotted against time to obtain the average horizontal velocity. From the profile, the initial angle of trajectory was obtained, and with the horizontal velocity, the true velocity was calculated.

To obtain the scaled burden, the distance from the marker to the explosive column was measured and then divided by the weight of explosive per foot of borehole to the half power (i.e., distance/(wt. of exp. per foot)^{1/2}). Using AN/FO in 15-inch diameter boreholes, the weight of explosive per foot was constant at 63 lbs, thus to the half power it was 7.94.

The time of first movement of the face was difficult to determine visually due to the camera lens used, the distance to the shot, and the difficulty in resolving small amounts of movement on the screen. The minimum length on the screen that can be resolved was approximately 0.1 inch, which in conjunction with an average scale factor of 1 inch = 14 feet in the films, yielded a minimum resolvable true distance of 1.4 feet. The average marker velocity was 30 feet per second, thus 47 milliseconds are required for the marker to move the 1.4 feet (0.1 inch on the screen) before motion can be detected. For this reason, visual determination of first movement was approximate. In another approach, using the displacement versus time graphs, first movement was determined by

locating the intersection of the displacement line with the time axis, but this did not provide very consistent results.

2.0 RESULTS OF THE ANALYSIS OF HIGH-SPEED FILMS OF OLD BEN TEST BLASTS

The following results include delay times of the surface delays, face marker velocities, face profiles, scaled burden and face velocity relationships and general comments on each shot. Due to the discrepancies in the survey data, the velocity results were not as accurate as desired, but the values obtained are reasonable for the scaled burdens of the blasts. The velocity versus time graphs were plotted using the horizontal component of the trajectory. The true velocity was found by using the angle of trajectory. This was done to reduce the effect gravity might have in the marker velocity since the velocity was relatively low to begin with and gravity may have caused a significant acceleration.

TEST BLAST 1-6

Shot 9-9-81 8.37 am Camera Speed 407 frames/second

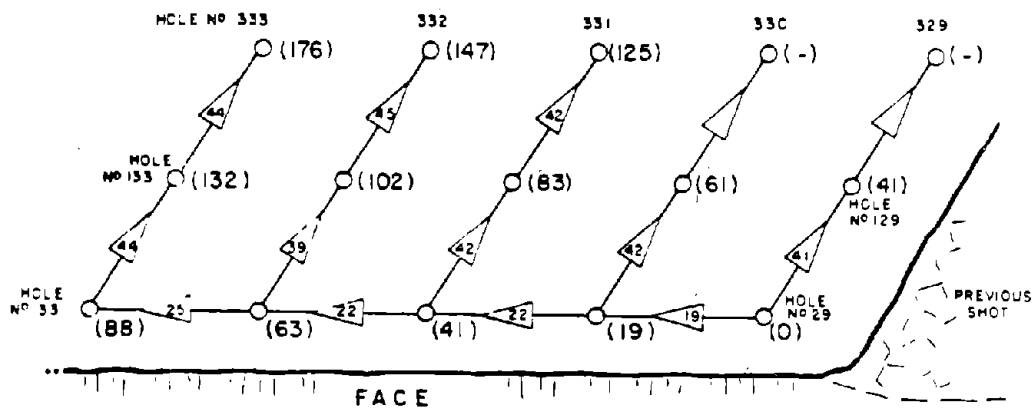
15 Holes 36 ft x 45 ft pattern 30-ft to 32-ft collars

The delay times are shown on Figure C-1. The detonation times of Holes No. 329 and 330 could not be determined due to their being obscured by the holes in front which detonated first, and the low angle from which the films were taken. There were 17 msec delays used between rows and 42 msec delays used between holes within a row. The downhole delays could not be determined except that face movement was first observed in one area of the face at approximately 70 msec after the initiation of the blast. This indicates that one or more of the downhole delays (200 msec) fired instantaneously or shortly thereafter, or that the downline initiation system initiated the hole(s) before the downhole delay fired.

There were no face markers on this blast so the face velocities could not be determined. Large pieces or fragments of the rock face could be used if their position and distance from the camera were known, but in this blast there were no such fragments.

This blast as well as the others was very well contained and little if any flyrock or excessive top movement was observed.

OLD BEN TEST BLAST 1-6
 SHOT 9-9-81 8:37 AM
 PATTERN 36x45 15" DIAM 30'-32' COLLARS



() NO IN BRACKETS IS HOLE DETONATION TIME (msec)

△ INDICATES DELAY TIME BETWEEN HOLES (msec)

(-) INDICATES DETONATION TIME COULD NOT BE DETERMINED

○ INDICATES HOLE POSITION

BLAST 1-6 DELAY TIMES

DRAWN BY M.R.E.L.		DATE		CHECKED BY		DATE	
AS CONSTRUCTED - CORRECTIONS COMPLETE							
BY		DATE					
REVISIONS							
NO	DESCRIPTION			DATE	BY		

WOODWARD - CLYDE CONSULTANTS
 Consulting Engineers Denver, Colorado

Job No. 19943

DRAWING NO	OF
C-1	

TEST BLAST 1-7

Shot 9-10-81 9:05 am

Due to communication problem between the camera operator and the blaster, the camera was started after the blast had been initiated and movement had commenced. No data could be obtained from this film.

TEST BLAST 1-8

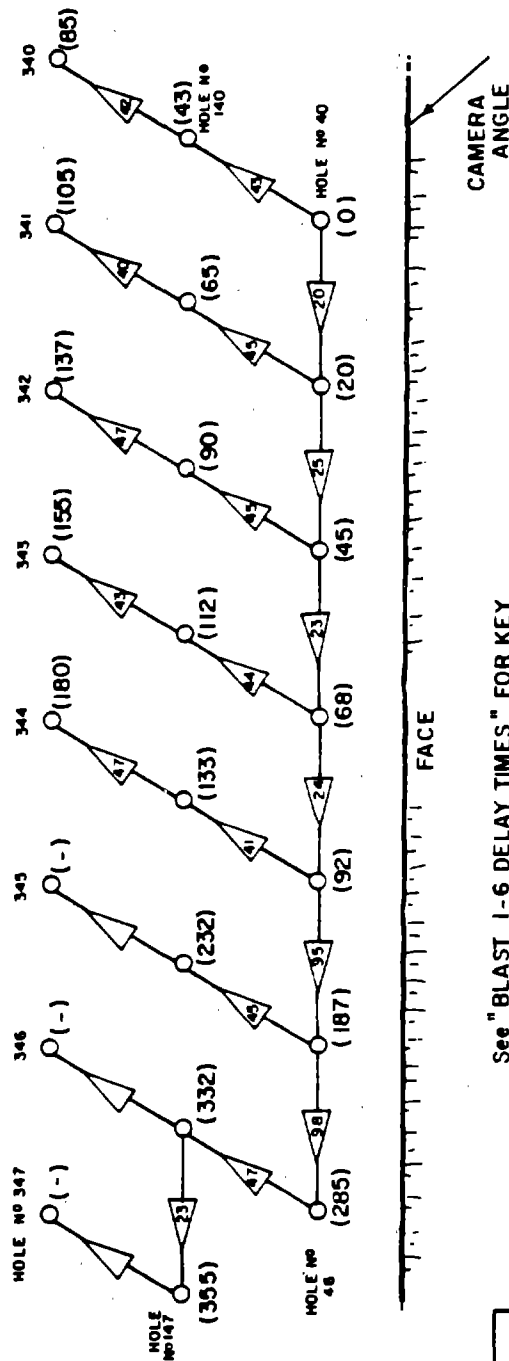
Shot 9-11-81 2:20 pm Camera speed 400 frames/second
23 Holes 36 ft x 45 ft pattern 30-ft collars

The 17 and 42 msec delays were used similarly to blast 1-6, but the delay between Holes No. 44-45 and 45-46 was 100 msec. The observed delay times are shown in Figure C-2. It should be noted that using the 100 msec delays allowed Hole No. 344 to initiate before Holes No. 45, 145 and 345. If the downhole delay (200 msec) is accurate there should be no problem, but if it fires early, then there is a very real chance of a cut-off. This is especially true since No. 344 is in the back row, which means it has more tendency to produce vertical movement than holes nearer the face. This is also true of Holes No. 345 and 346.

The data obtained from the face markers allowed the face profile and the horizontal face velocities to be determined. They are shown on Figures C-3 to C-6.

Figure C-3 shows the face profile at 500 msec intervals. These are the elapsed times from the initiation of the blast and not from first movement. The markers were placed in front of Hole No. 42 which was initiated on surface at 45 msec, thus it is assumed that the downhole delays initiated 200 msec later at 245 msec. The observed first movement of the markers was at approximately 335 msec, which would be expected. The profile also shows how well the top of the blast was contained as the top marker (No. 1) continued to drop as the blast progressed. The horizontal velocity of this marker was only 14.5 ft/sec as per Figure C-4. With an angle of trajectory of 27 degrees from the horizontal, the true velocity was 16 ft/sec. The other markers (Figure C-5 and C-6) had higher true velocities of 20, 27, 31 and 28 ft/sec.

OLD BEN TEST BLAST NO 1-8
SHOT 9-11-81 2:20 PM
36 x 45 PATTERN 30' COLLARS



See "BLAST 1-6 DELAY TIMES" FOR KEY

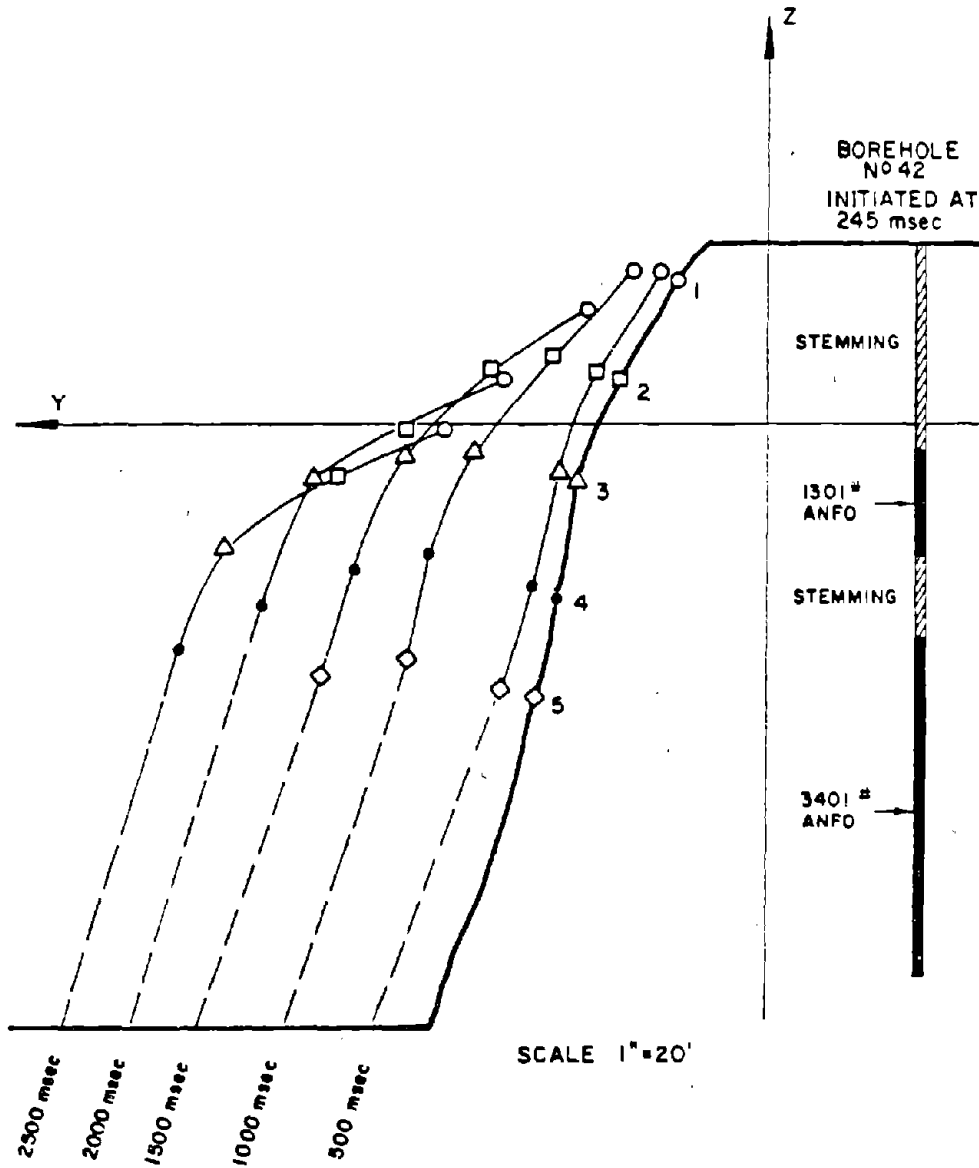
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AS CONSTRUCTED - CORRECTIONS COMPLETE								
BY		DATE						
R E V I S I O N S								
NO.	DESCRIPTION						DATE	BY

BLAST 1-8 DELAY TIMES

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Job No. 19943

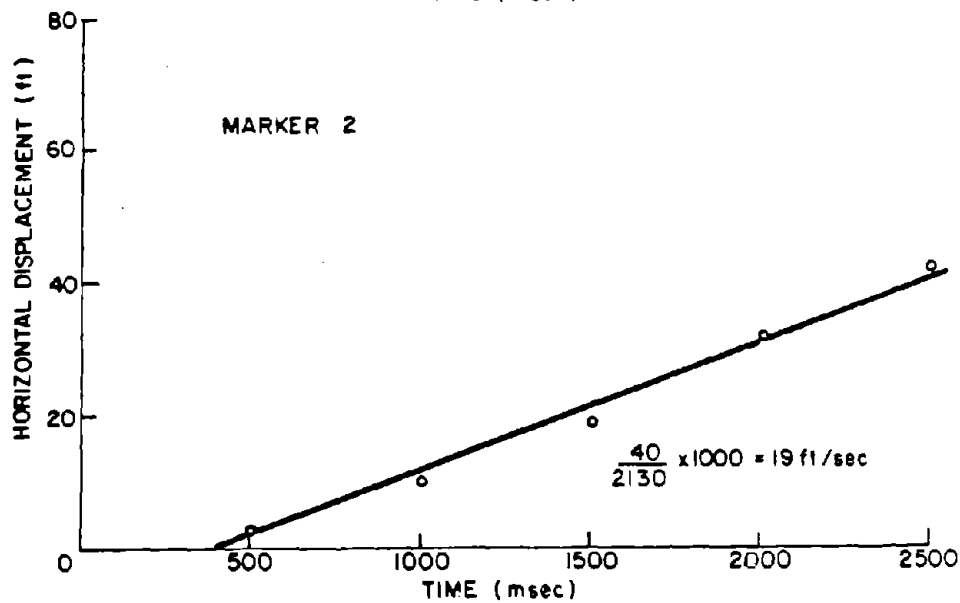
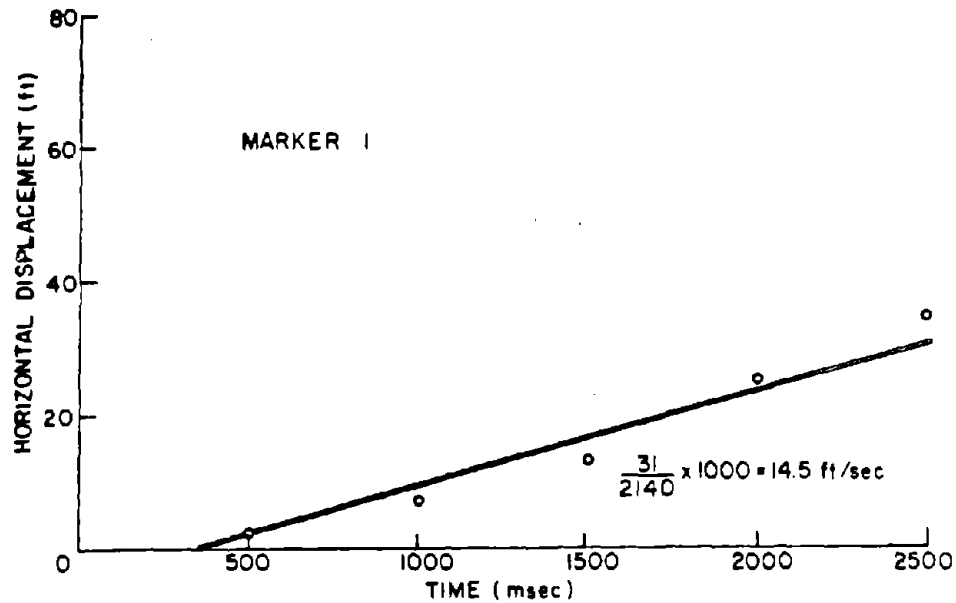
DRAWING NO	OF
C-2	



DRAWN BY		M.R.E.L.		DATE		CHECKED BY		DATE	
AS CONSTRUCTED - CORRECTIONS COMPLETE									
BY		DATE							
REVISIONS									
NO.	DESCRIPTION						DATE	BY	

FACE PROFILE
BLAST NO. 1-8
OLD BEN

WOODWARD - CLYDE CONSULTANTS	
Consulting Engineers	Denver, Colorado
Job No. 19943	DRAWING NO. OF
	C-3



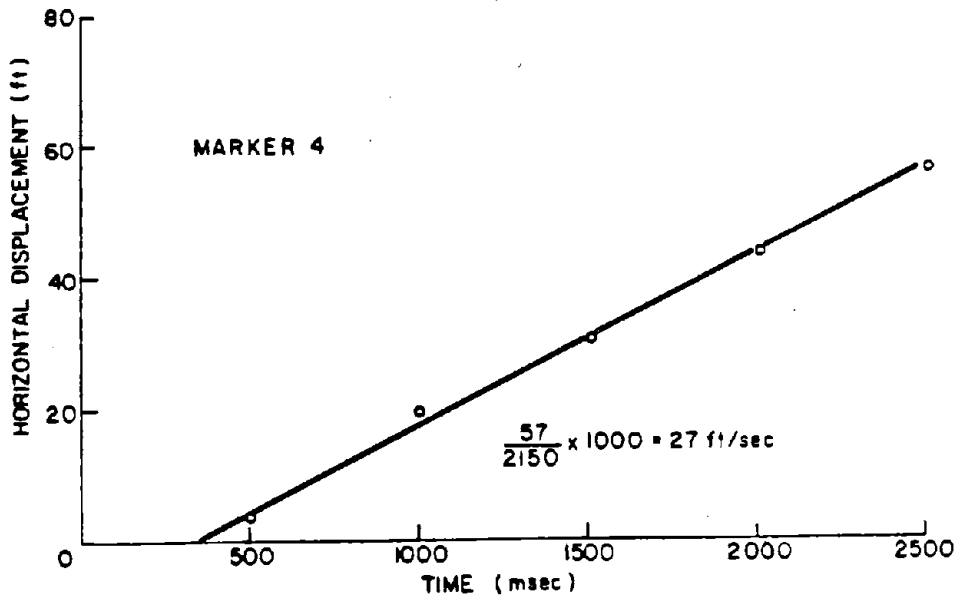
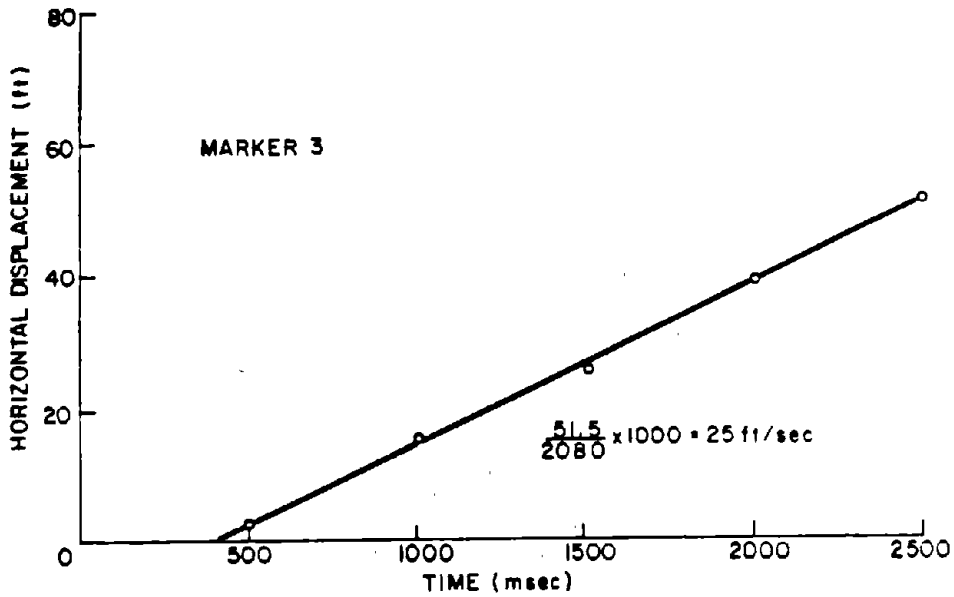
DRAWN BY	M.R.E.L.	DATE	CHECKED BY	DATE
AS CONSTRUCTED - CORRECTIONS COMPLETE				
BY	DATE			
REVISIONS				
NO.	DESCRIPTION	DATE	BY	

FACE MARKER VELOCITIES FOR
 OLD BEN NO. 1
 TEST BLAST 1-8
 9-11-81 2:20PM

WOODWARD - CLYDE CONSULTANTS
 Consulting Engineers Denver, Colorado

Job No. 19943

DRAWING NO	OF
C-4	



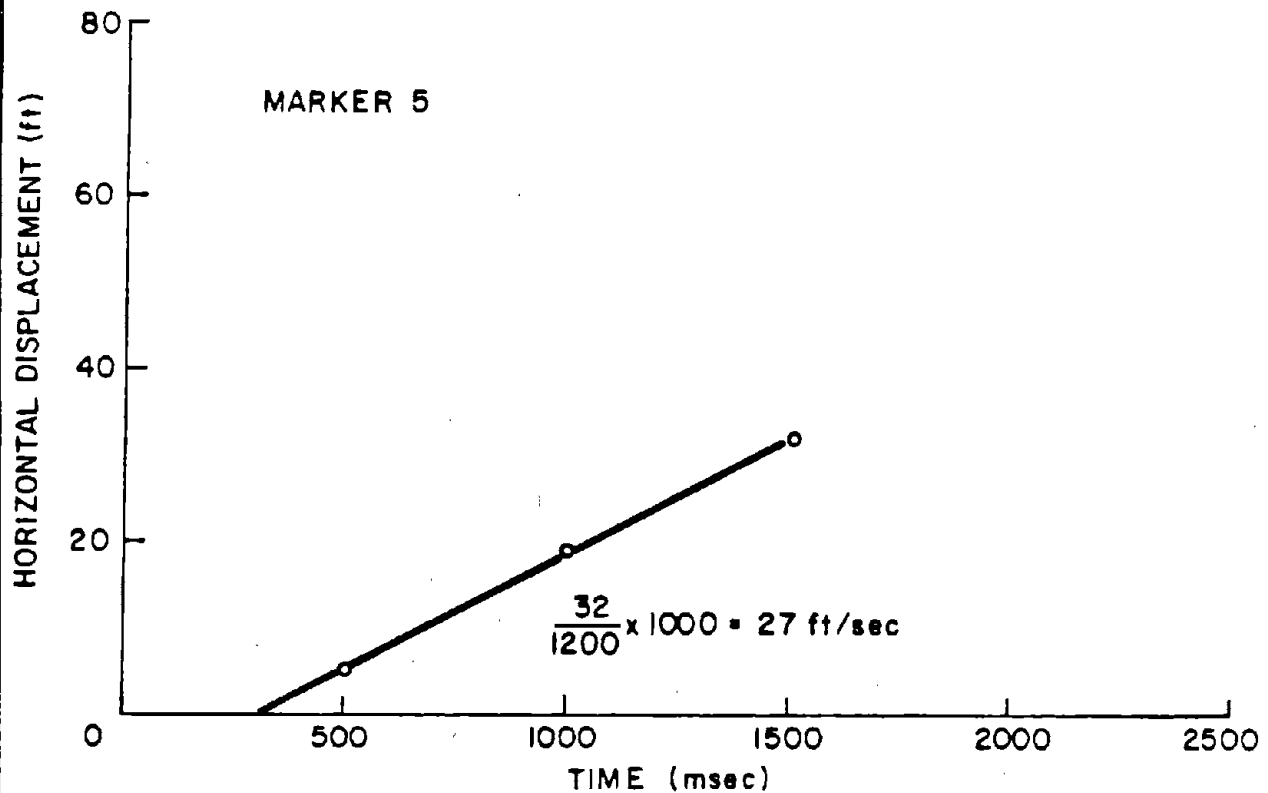
DRAWN BY: M.R.E.L.		DATE		CHECKED BY		DATE	
AS CONSTRUCTED - CORRECTIONS COMPLETE							
BY		DATE					
REVISIONS							
NO	DESCRIPTION			DATE	BY		

FACE MARKER VELOCITIES
FOR TEST BLAST 1-8
9-11-81 2:20PM

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers Denver, Colorado

Job No. 19943

DRAWING NO.	OF
C-5	



DRAWN BY <u>M.R.E.L.</u>		DATE _____		CHECKED BY _____		DATE _____	
AS CONSTRUCTED - CORRECTIONS COMPLETE							
BY _____		DATE _____					
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NO	DESCRIPTION				DATE	BY	

FACE MARKER VELOCITY FOR
TEST BLAST 1-8
9-11-81 2:20PM

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers ● Denver, Colorado

Job No. 19943

DRAWING NO	OF
C-6	

TEST BLAST 2-1

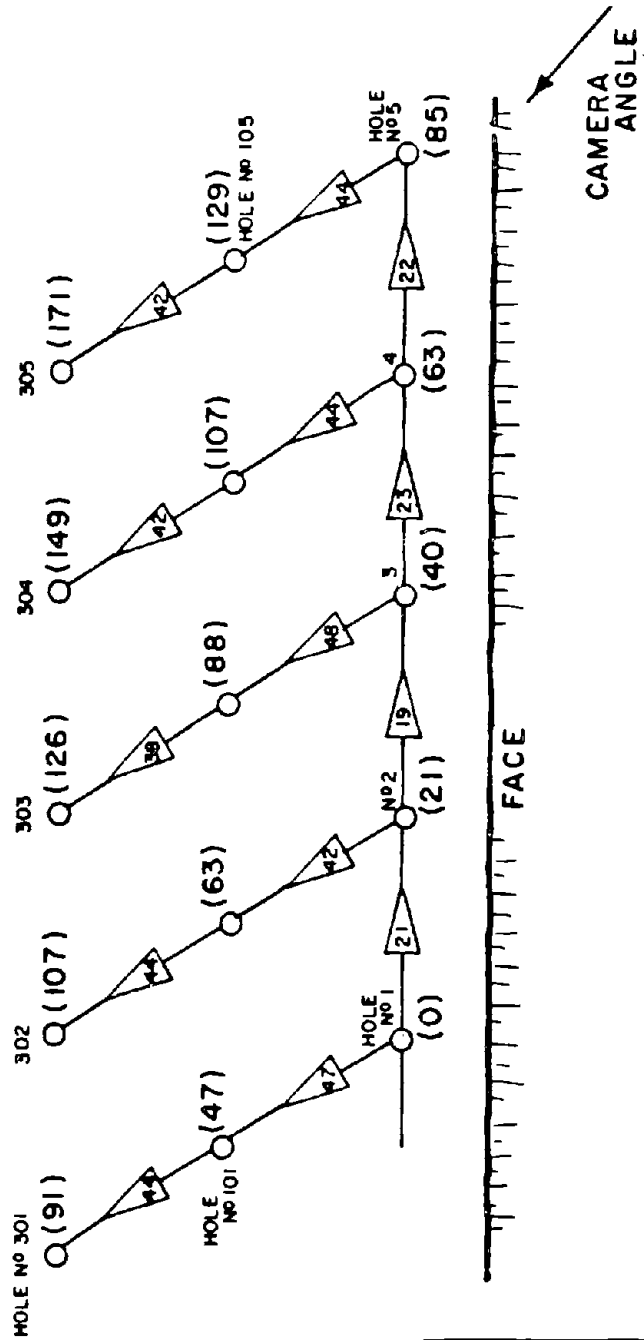
Shot 9-29-81 9:00 am Camera speed 314 frames/second

15 Holes 30 ft x 45 ft pattern 30-ft collars

The surface delay times are indicated on Figure C-7. There was some movement of the face at 156 msec which indicates premature initiation of a hole. Due to the camera angles, which were approximately 30 degrees to the face, it was difficult to establish which hole it was. There should not have been movement before approximately 240 msec if the downhole delays were correct and considering the 40 to 60-foot burdens.

Figure C-8 shows the marker trajectories and the face profile at 1000 and 1500 msec. Unfortunately, some of the markers were obscured before the movement was complete but the velocities could be determined as in Figures C-9 and C-10. Again, the velocities increased nearer the toe and were higher than those of Blast 1-8. Marker M4 has an unusual trajectory which may be due to interference of the wooden marker by collision with rock fragments. The horizontal component, which is plotted in Figure C-10, is consistent at 35 ft/sec, which at an angle of 25 degrees from the horizontal, yields a true velocity of 39 ft/sec.

OLD BEN BLAST No 2-1
 SHOT 9-29-81 9 AM.
 30 x 45 PATTERN 30' COLLARS



See "BLAST 1-6 DELAY TIMES" FOR KEY

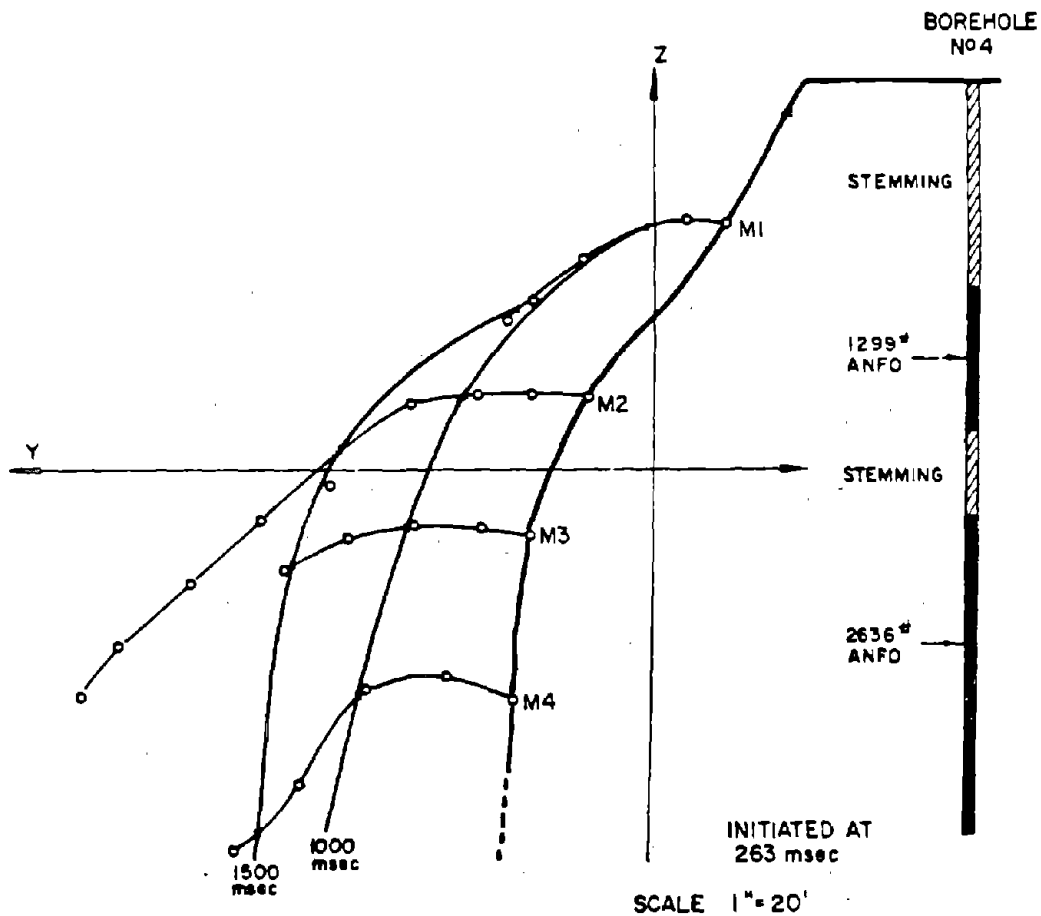
DRAWN BY	M.R.E.L.	DATE	CHECKED BY	DATE
AS CONSTRUCTED - CORRECTIONS COMPLETE				
BY		DATE		
REVISIONS				
NO	DESCRIPTION	DATE	BY	

BLAST 2-1 DELAY TIMES

WOODWARD - CLYDE CONSULTANTS
 Consulting Engineers Denver, Colorado

Job No. 19943

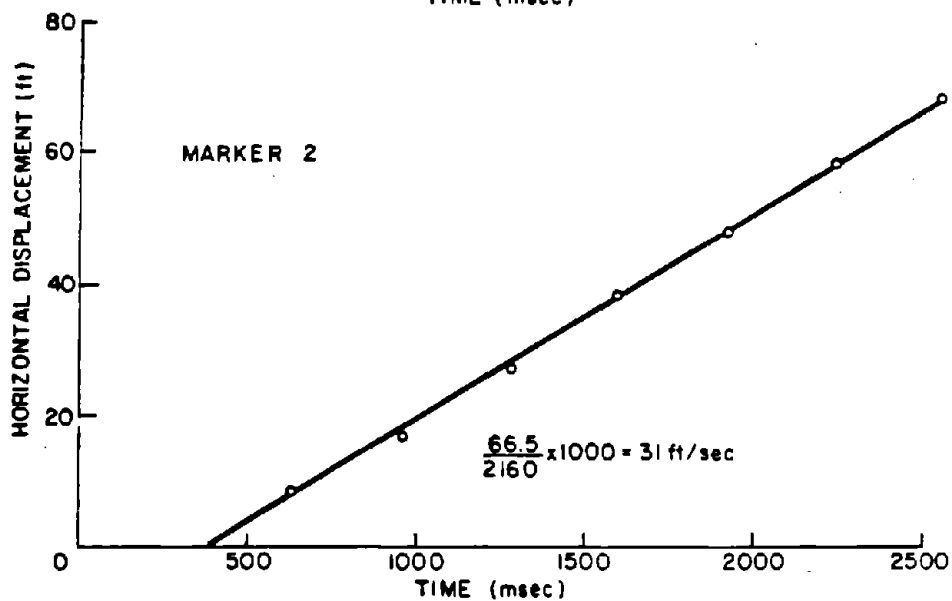
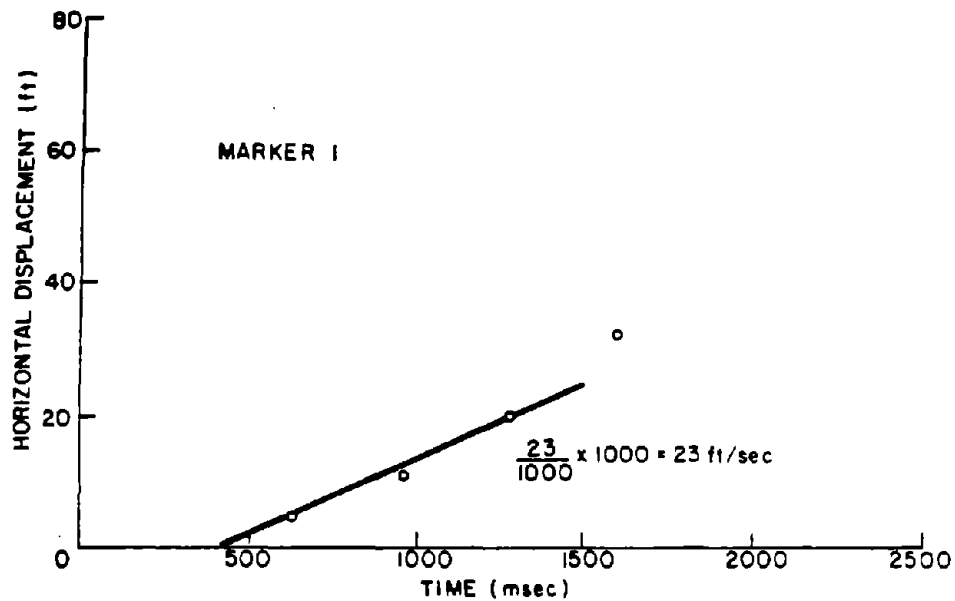
DRAWING NO. OF
 C-7



FACE PROFILE OF
TEST BLAST 2-1

DRAWN BY		M.R.E.L.		DATE		CHECKED BY		DATE			
AS CONSTRUCTED - CORRECTIONS COMPLETE											
BY		DATE									
REVISIONS											
NO		DESCRIPTION						DATE		BY	

WOODWARD - CLYDE CONSULTANTS		
Consulting Engineers	Denver, Colorado	
Job No. 19943	DRAWING NO	OF
	C-8	



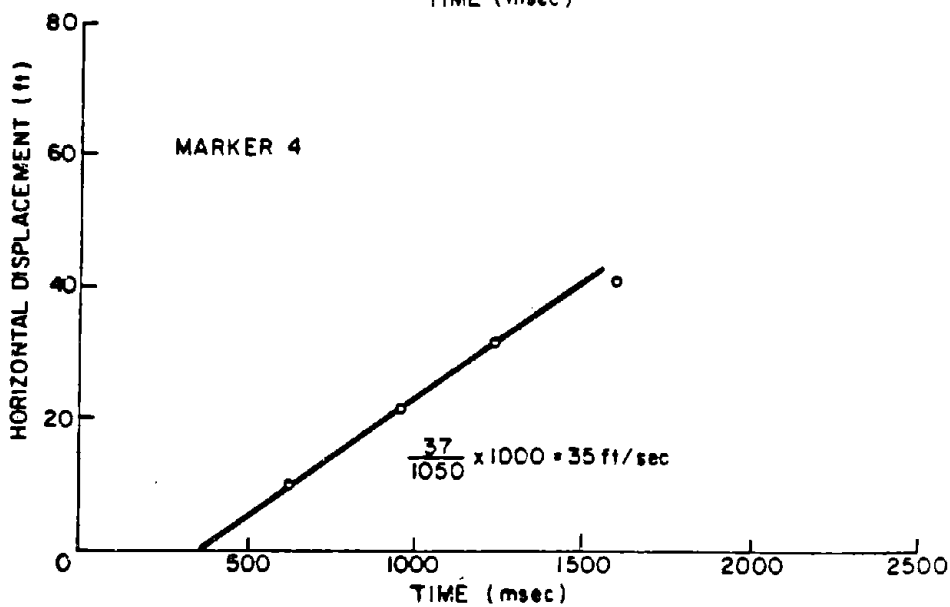
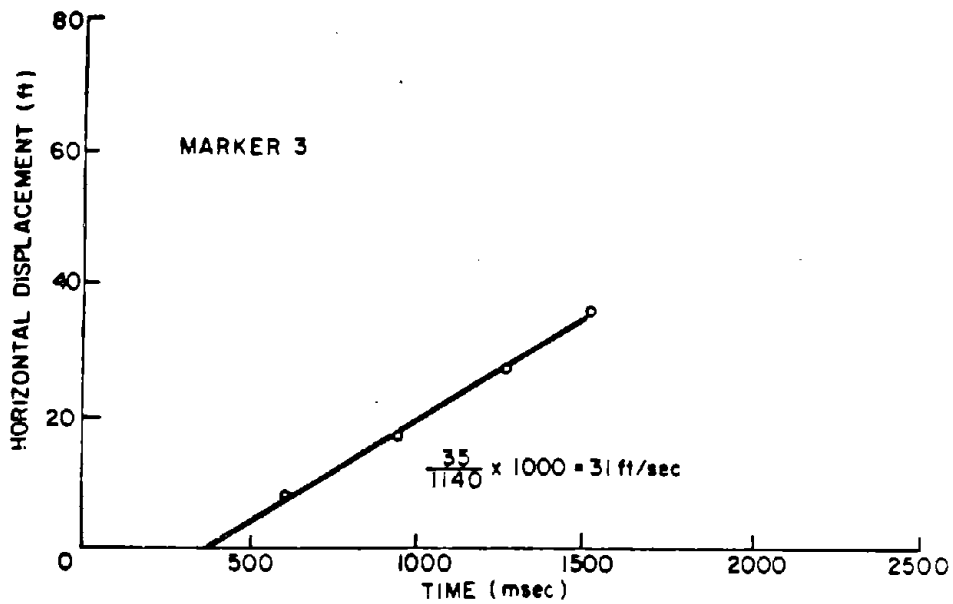
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AS CONSTRUCTED - CORRECTIONS COMPLETE							
BY		DATE					
REVISIONS							
NO.	DESCRIPTION					DATE	BY

FACE VELOCITIES FOR
TEST BLAST NO. 2-1

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers ● Denver, Colorado

Job. No. 19943

DRAWING NO	OF
C-9	



DRAWN BY M.R.E.L.		DATE		CHECKED BY		DATE	
AS CONSTRUCTED - CORRECTIONS COMPLETE							
BY		DATE					
REVISIONS							
NO.	DESCRIPTION					DATE	BY

FACE VELOCITIES FOR
TEST BLAST NO. 2-1

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers ● Denver, Colorado

Job No. 19943

DRAWING NO	OF
C-10	

TEST BLAST 2-2

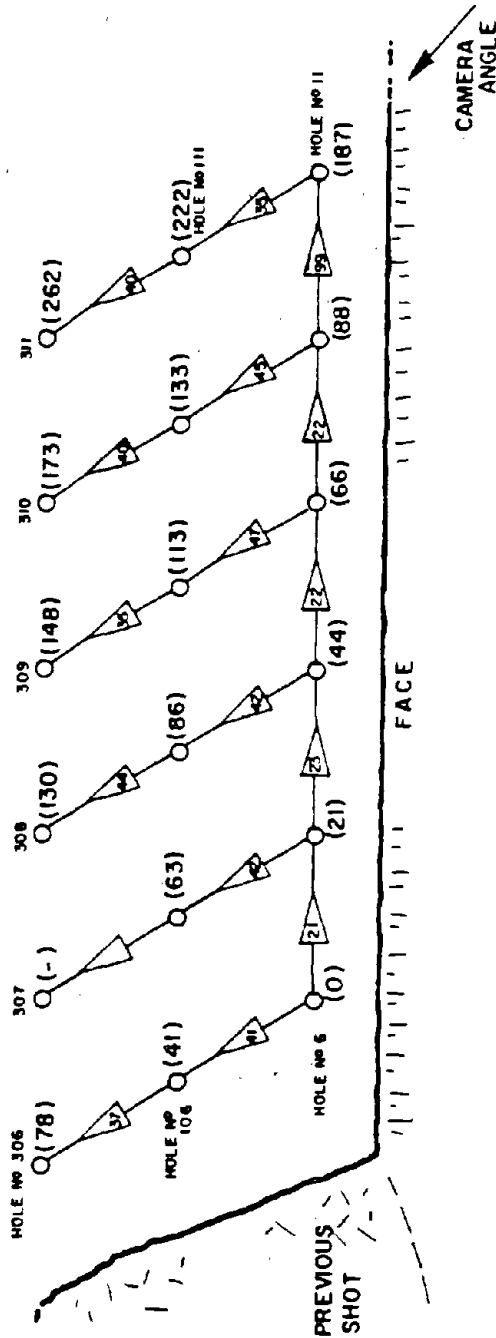
Shot 9-30-81 10:29 am Camera speed 403 frames/second

18 Holes 30 ft x 45 ft pattern 30 ft-collars

The delay times for this blast are shown on Figure C-11 and in this shot, a 100 msec delay was used between Holes No. 10 and 11. There was movement of the face observed approximately 80 msec after the blast was initiated, which indicates premature initiation of a hole.

Only 1 marker could be tracked due to the large amount of dust and smoke which occurred at the face soon after initiation. This one marker was located about midway down the face and the horizontal velocity shown on Figure C-12 was 30 ft/sec. This gave a true velocity of 35 ft/sec due to angle of 35 degrees from the horizontal, which is consistent with the results from Blast 2-1.

OLD BEN FILM NO 2-2
SHOT 9-30-81 10:30 AM
PATTERN 30' x 45' 30' COLLARS

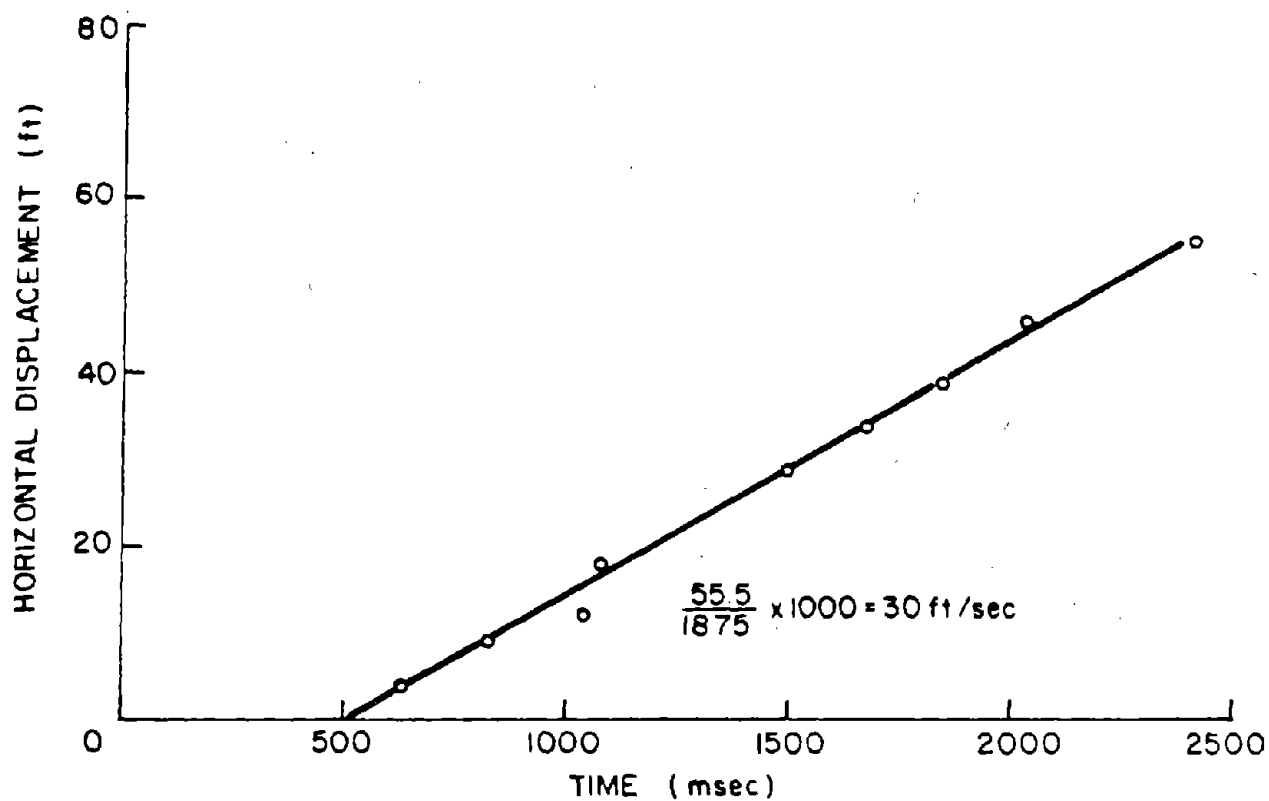


See "BLAST 1-6 DELAY TIMES" FOR KEY

DRAWN BY		M.R.E.L.		DATE		CHECKED BY		DATE	
AS CONSTRUCTED - CORRECTIONS COMPLETE									
BY		DATE							
REVISIONS									
NO	DESCRIPTION						DATE	BY	

BLAST 2-2 DELAY TIMES

WOODWARD - CLYDE CONSULTANTS		
Consulting Engineers		
Denver, Colorado		
Job No. 19943		DRAWING NO. OF
		C-II



DRAWN BY M.R.E.L.		DATE		CHECKED BY		DATE	
AS CONSTRUCTED - CORRECTIONS COMPLETE							
BY		DATE					
REVISIONS							
NO	DESCRIPTION			DATE	BY		

FACE VELOCITIES FOR
TEST BLAST NO. 2-2

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers ● Denver, Colorado

Job No. 19943

DRAWING NO	OF
C-12	

TEST BLAST 2-3

Shot 10-1-81 9:05 am Camera speed 405 frames/second

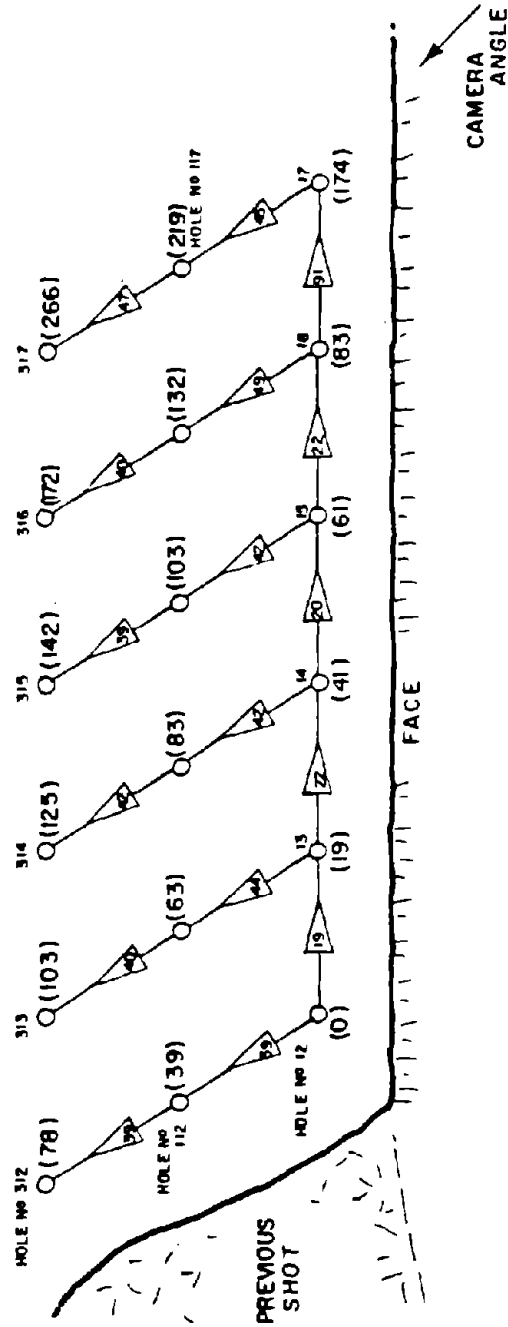
18 Holes 30 ft x 48 ft pattern 30-ft collars

The delay times are shown on Figure C-13, and again a 100 msec delay was used between the last two rows (Holes No. 16 and 17).

A total of five markers were tracked and are shown on Figure C-14. This blast had three decks. From the face profile, Marker 3 appears to have begun movement before the others. This indicates that possibly one of the decks initiated before the other two. The downline was initiated at 62 msec and thus, the hole is assumed to initiate at 262 msec after the first hole was initiated. The horizontal velocities are shown on Figures C-15, C-16, and C-17, and Marker 3 on Figure C-16 does have a velocity consistent with the others. The true velocities for M1 through M5 were 31, 37, 39, 44 and 25 ft/sec, respectively. Marker 5 is low and this may be due to the fact that the marker could not be tracked very far and thus the velocity, (the slope of the horizontal displacement versus the time), was based on only 2 points.

The top marker velocity was initially 26 ft/sec and was consistent with the top movement seen in Blast 2-1, which was 25 ft/sec. Both of these markers had the same scaled distance.

OLD BEN FILM NO 2-3
 SHOT 10-1-81 9:05AM
 PATTERN 30' x 48' 30' COLLAR

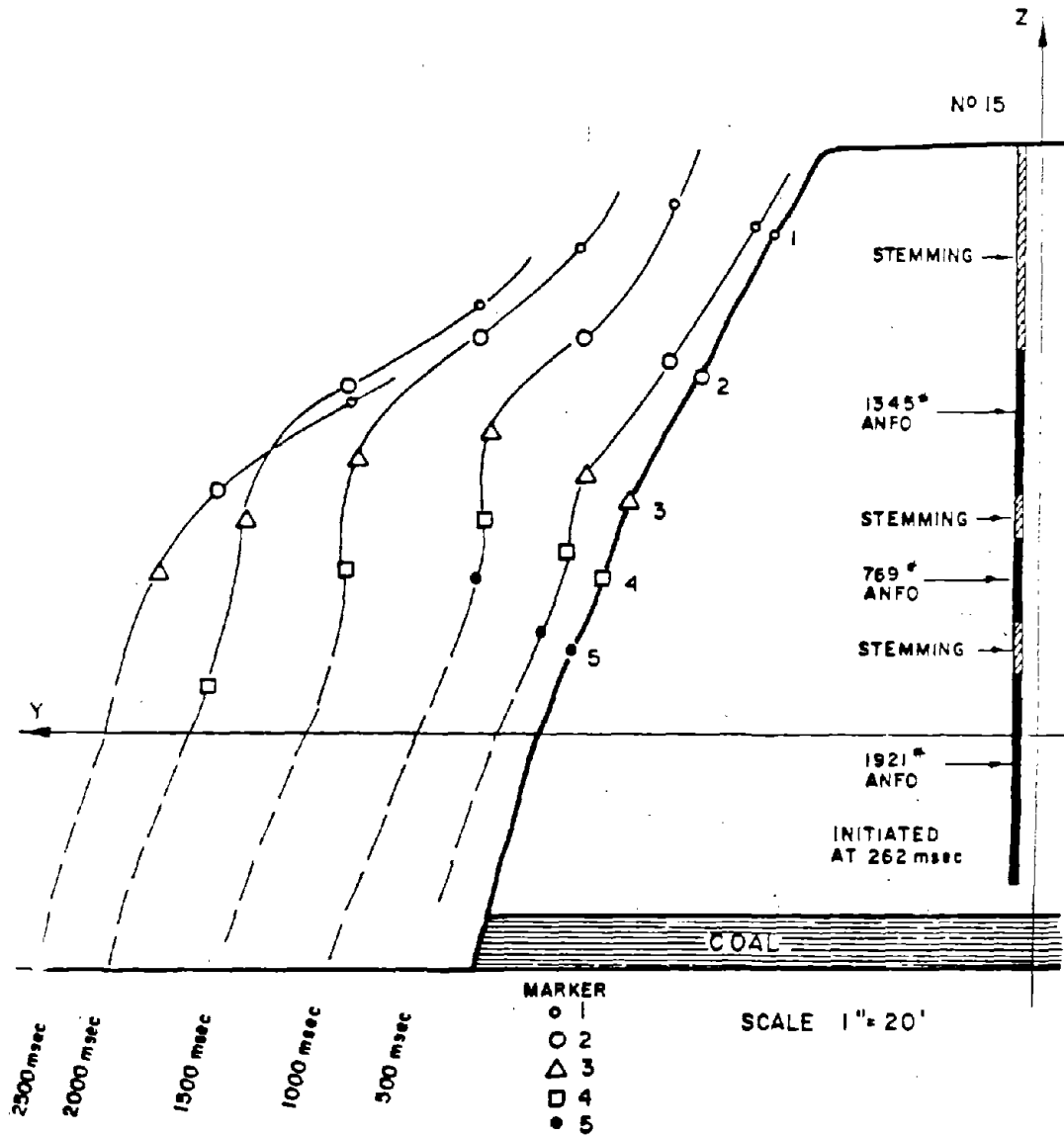


See "BLAST 1-6 DELAY TIMES" FOR KEY

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AS CONSTRUCTED - CORRECTIONS COMPLETE							
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BLAST 2-3 DELAY TIMES

WOODWARD - CLYDE CONSULTANTS			
Consulting Engineers		Denver, Colorado	
Job No. 19943		DRAWING NO	DF
		C-13	



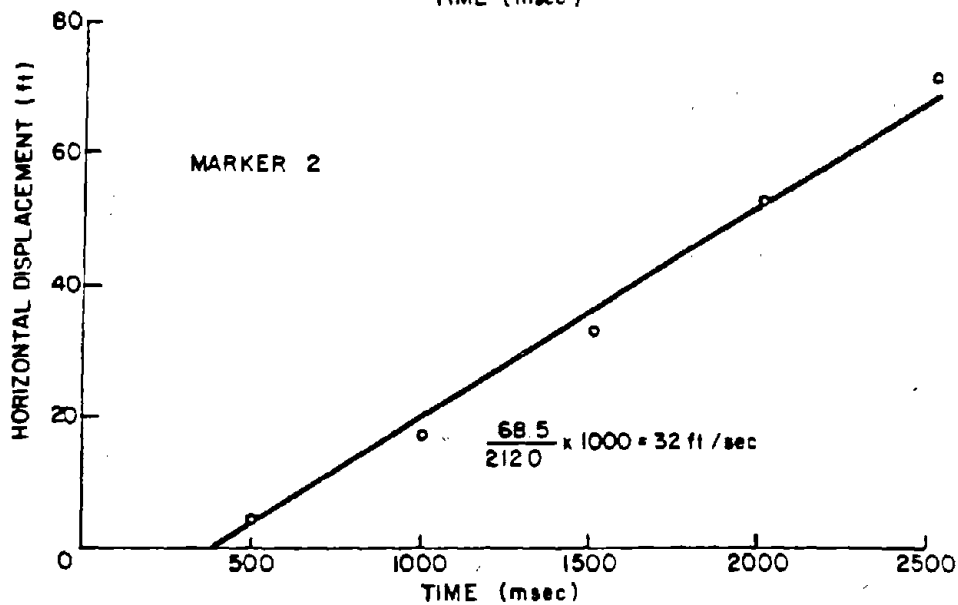
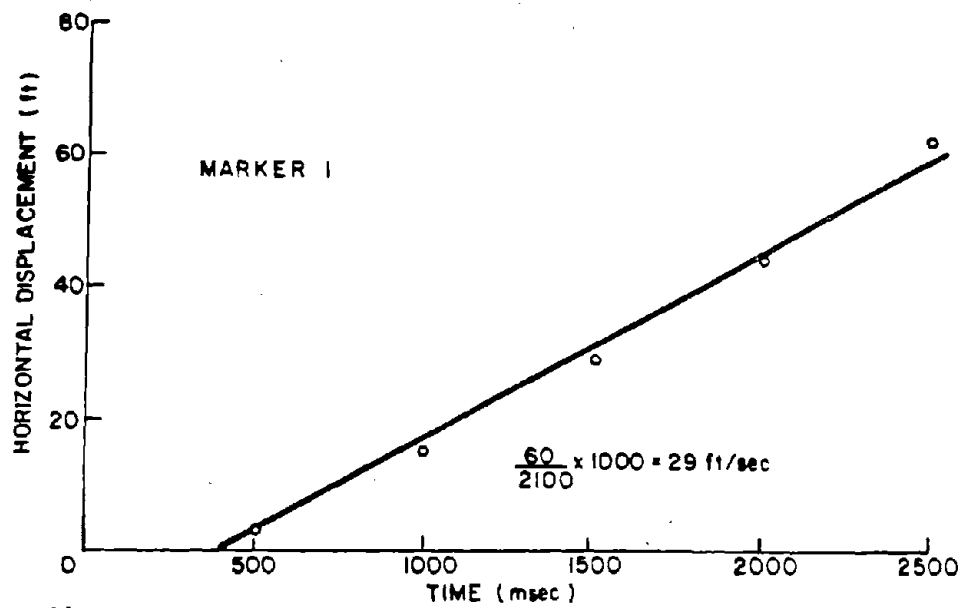
FACE PROFILE
BLAST 2-3
10-1-81 9:05 AM

DRAWN BY		M.R.E.L.		DATE		CHECKED BY		DATE			
AS CONSTRUCTED - CORRECTIONS COMPLETE											
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NO.		DESCRIPTION						DATE		BY	

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers Denver, Colorado

Job No. 19943

DRAWING NO. OF
C-14



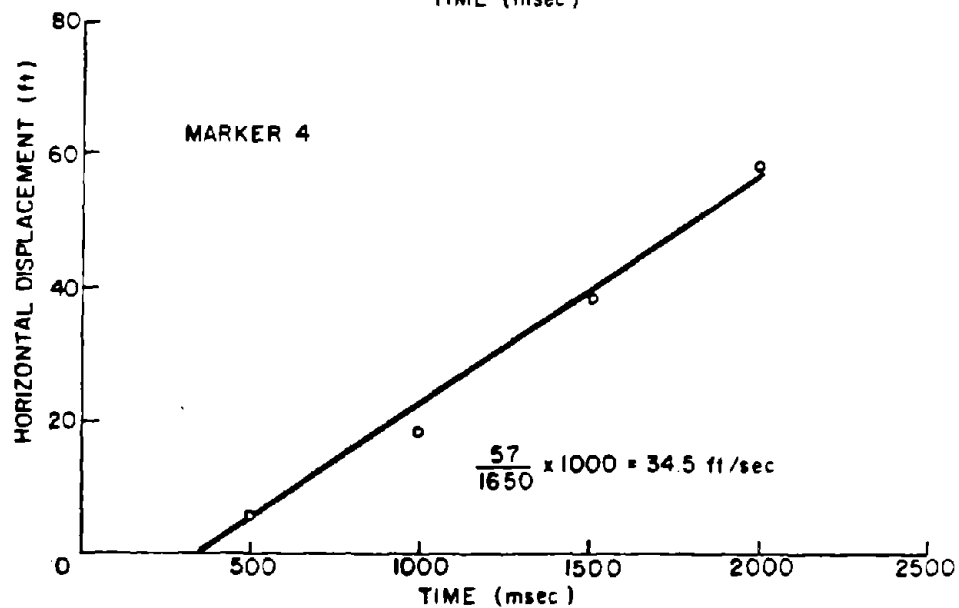
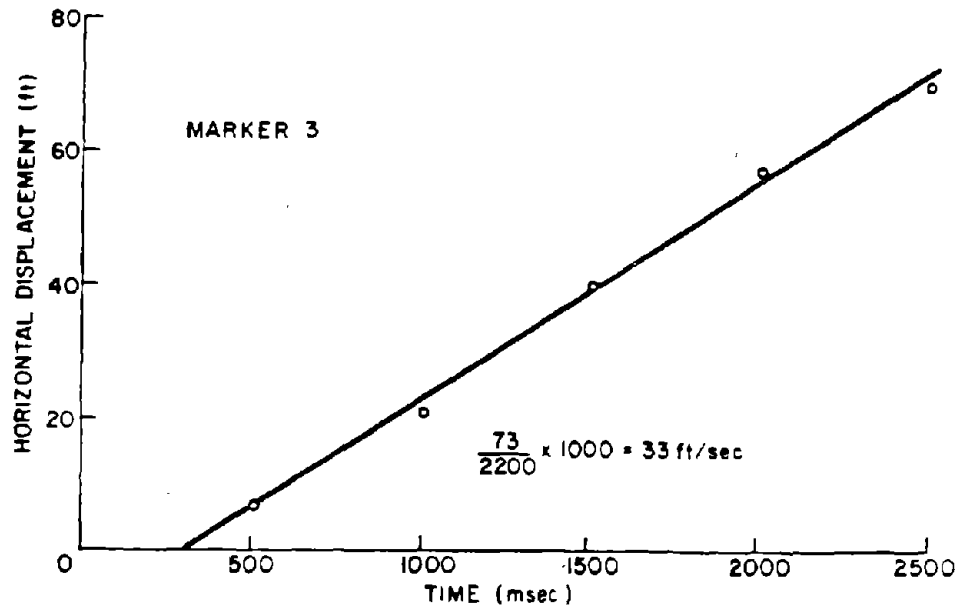
DRAWN BY M.R.E.L.		DATE		CHECKED BY		DATE	
AS CONSTRUCTED - CORRECTIONS COMPLETE							
BY		DATE					
REVISIONS							
NO	DESCRIPTION			DATE	BY		

FACE VELOCITIES FOR
TEST BLAST 2-3

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers Denver, Colorado

Job No 19943

DRAWING NO	OF
C-15	



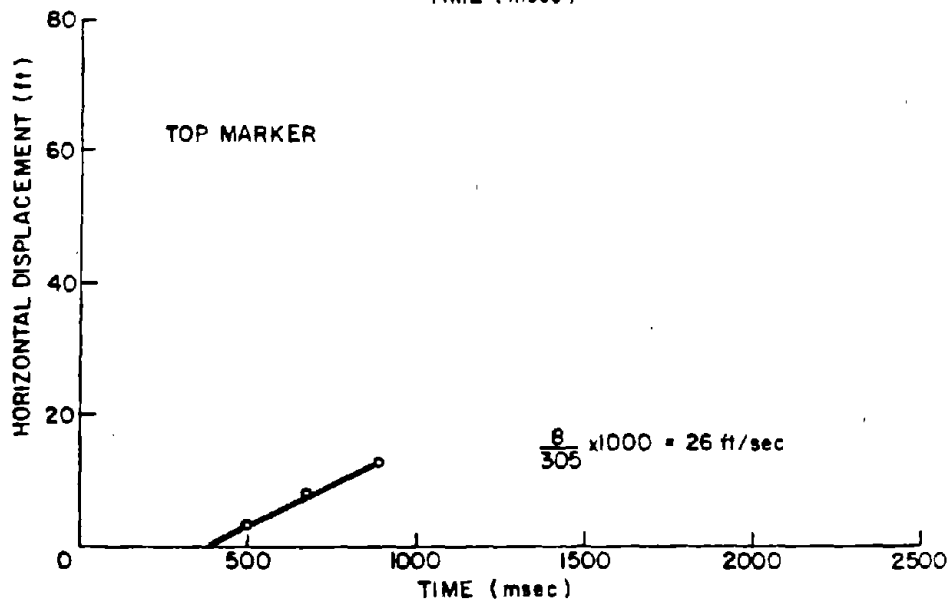
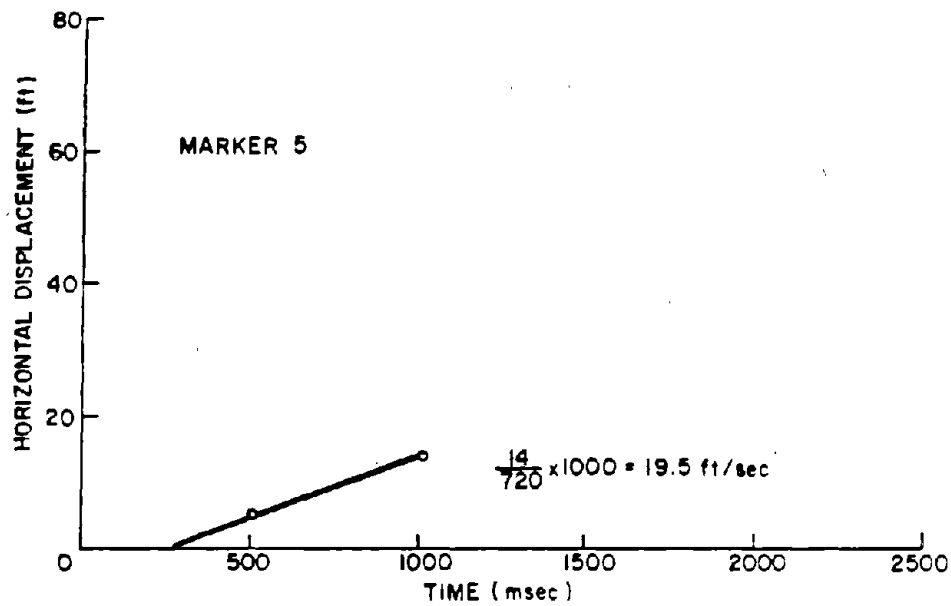
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AS CONSTRUCTED - CORRECTIONS COMPLETE							
BY		DATE					
REVISIONS							
NO	DESCRIPTION			DATE	BY		

FACE VELOCITIES FOR TEST BLAST 2-3

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers ● Denver, Colorado

Job No. 19943

DRAWING NO	OF
C-16	



DRAWN BY M.R.E.L.		DATE		CHECKED BY		DATE	
AS CONSTRUCTED - CORRECTIONS COMPLETE							
BY		DATE					
REVISIONS							
NO	DESCRIPTION					DATE	BY

FACE VELOCITIES FOR
TEST BLAST NO. 2-3

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers ● Denver, Colorado

Job No. 19943

DRAWING NO	OF
C-17	

TEST BLAST 2-4

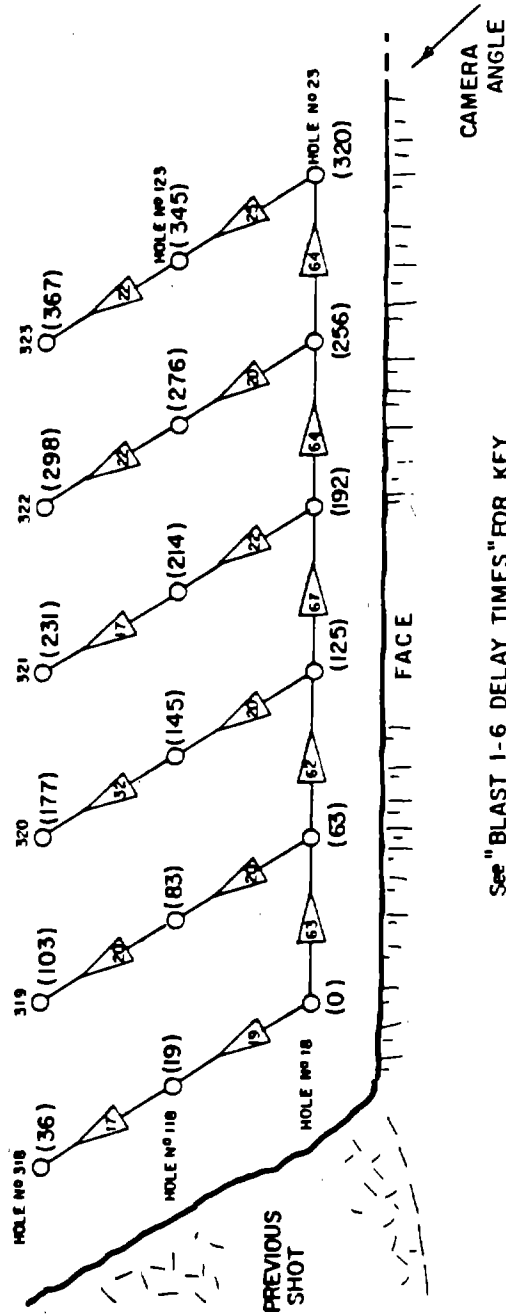
Shot 10-1-81 2:18 pm Camera speed 403 frames/second
18 Holes 30 ft x 48 ft pattern 30-ft collars

The delay pattern for this blast was different from the previous ones since a 17 and 42 msec delay was used together to produce a 59 msec delay between rows and 17 msec delays were used between holes within a row. The observed delay times are given on Figure C-18.

There were two lines of markers suspended over the face at different locations. The first line consisted of Markers M1, M2 and M3, which are shown on Figure C-19. The top marker, M1, could not be tracked for a considerable portion of the trajectory and the horizontal velocity as shown on Figure C-20 indicates this. The true velocity of 19 ft/sec is consistent with the upper marker in the previous blasts. The other markers on Figure C-20 and C-21 gave true velocities of 23 and 29 ft/sec, respectively. These markers were located in front of Hole No. 22. The second line of markers were located in front of Hole No. 20. They provided little data except for M4, as the others could not be initially tracked. For this reason a face trajectory profile could not be drawn for either Line 1 or 2. The horizontal velocity of M4 is shown on Figure C-23. It had a true velocity of 20 ft/sec.

The films of Test Blasts 2-5, 2-6 and 2-8 were underexposed slightly and did not allow the markers to be tracked. Generally, these blasts did appear to be similar to the previous ones and no unusual top motion or flyrock was observed. The delay times were obtained for each.

OLD BEN FILM NO 2-4
 SHOT 10-1-81 2:18 PM
 PATTERN 30' x 48' 30' COLLARS



See "BLAST 1-6 DELAY TIMES" FOR KEY

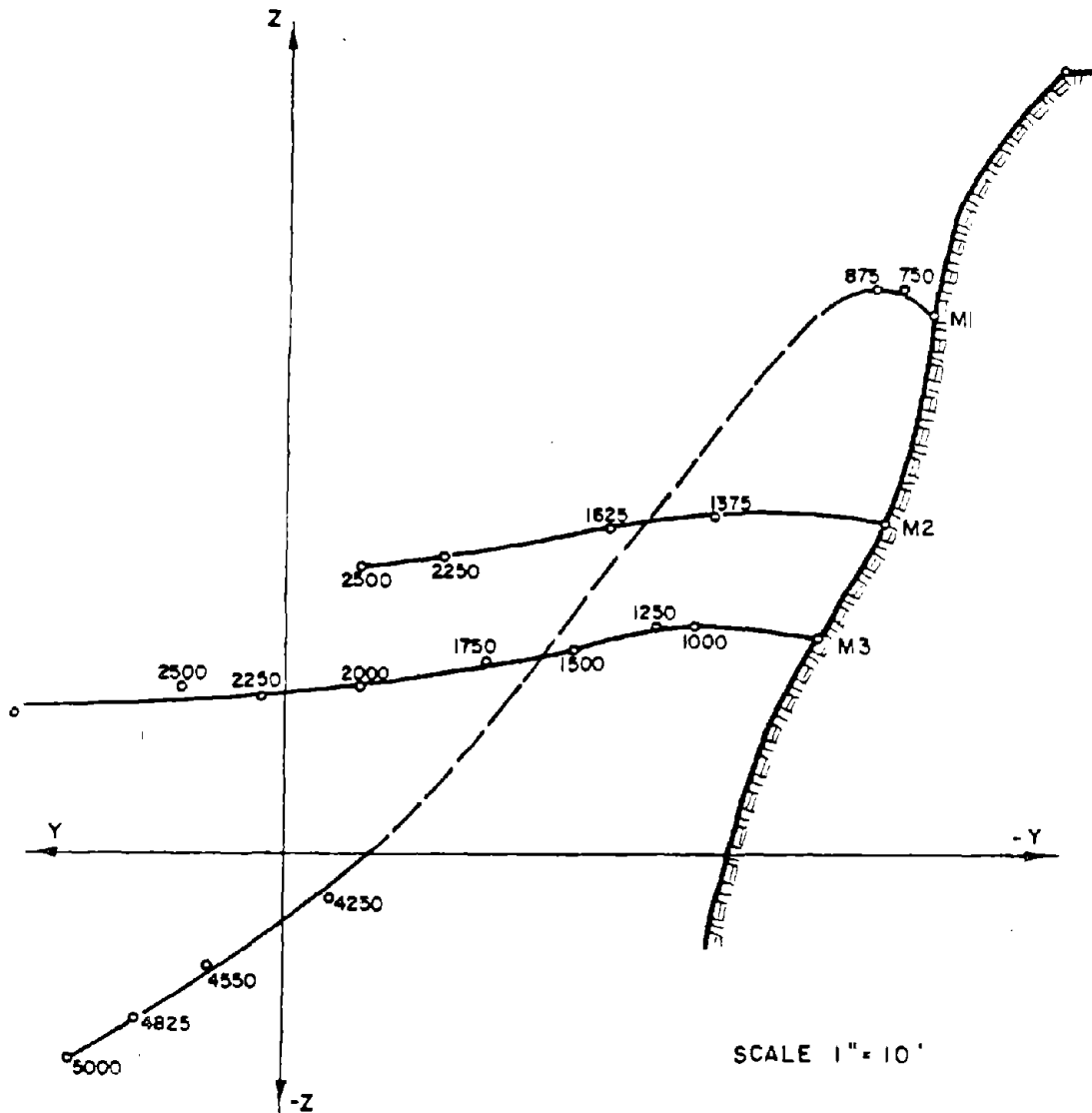
BLAST 2-4 DELAY TIMES

DRAWN BY M.R.E.L.		DATE		CHECKED BY		DATE	
AS CONSTRUCTED - CORRECTIONS COMPLETE							
BY		DATE					
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WOODWARD - CLYDE CONSULTANTS
 Consulting Engineers Denver, Colorado

Job No. 19943

DRAWING NO. C-18
 OF



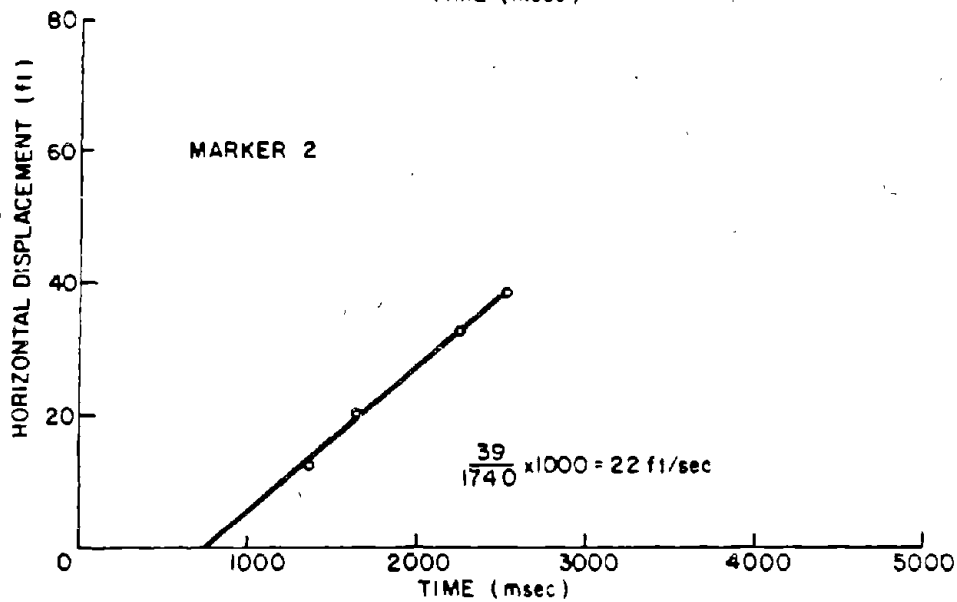
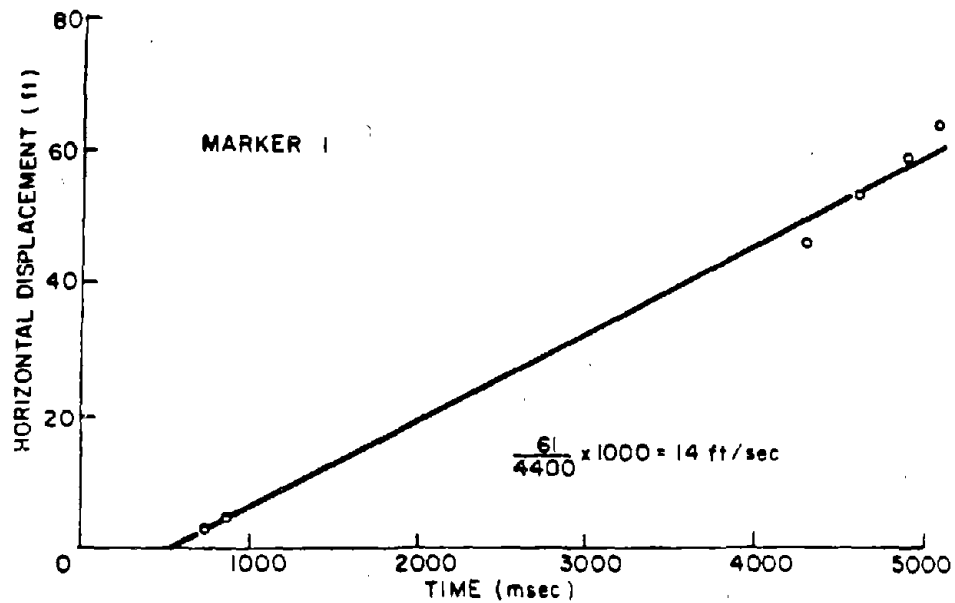
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AS CONSTRUCTED - CORRECTIONS COMPLETE									
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MARKER TRAJECTORIES BLAST 2-4

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers Denver, Colorado

Job No. 19943

DRAWING NO	OF
C-19	



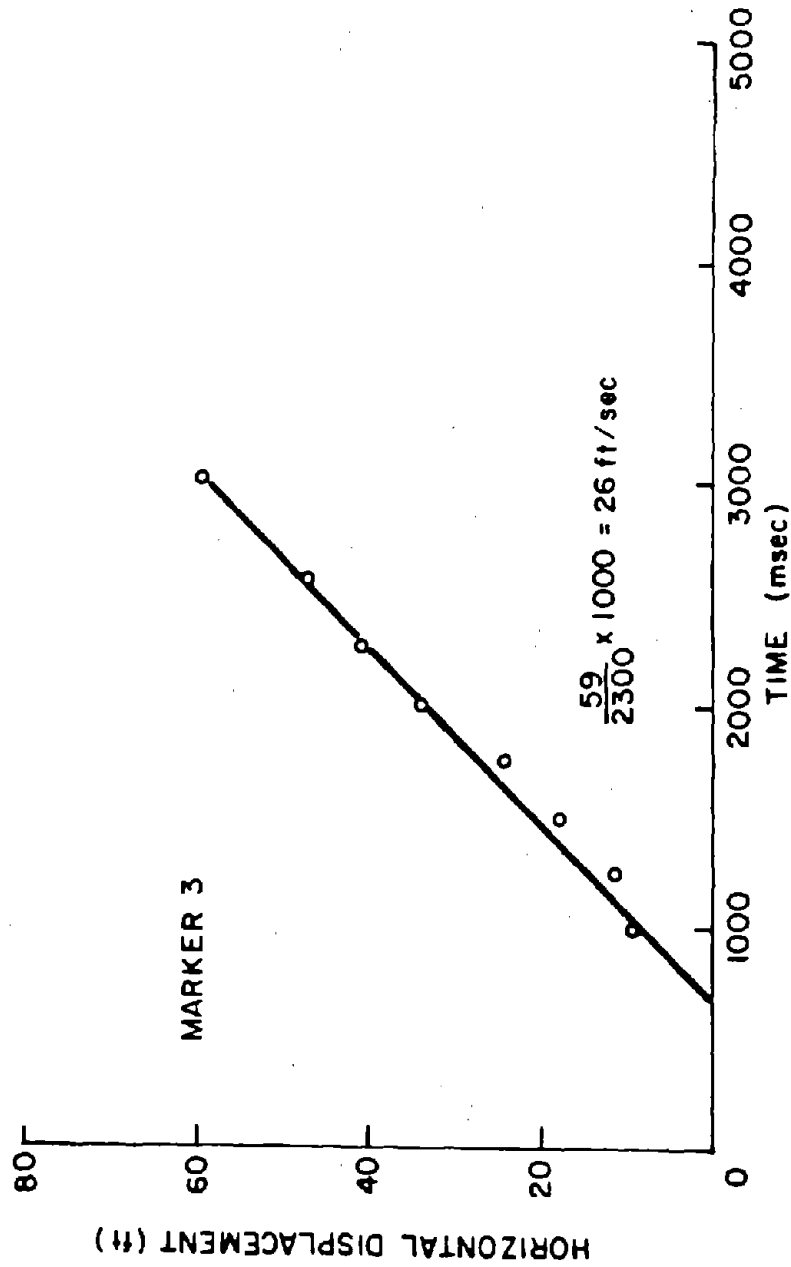
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AS CONSTRUCTED - CORRECTIONS COMPLETE							
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REVISIONS							
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FACE VELOCITIES FOR
TEST BLAST NO. 2-4

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers Denver, Colorado

Job No. 19943

DRAWING NO	OF
C-20	



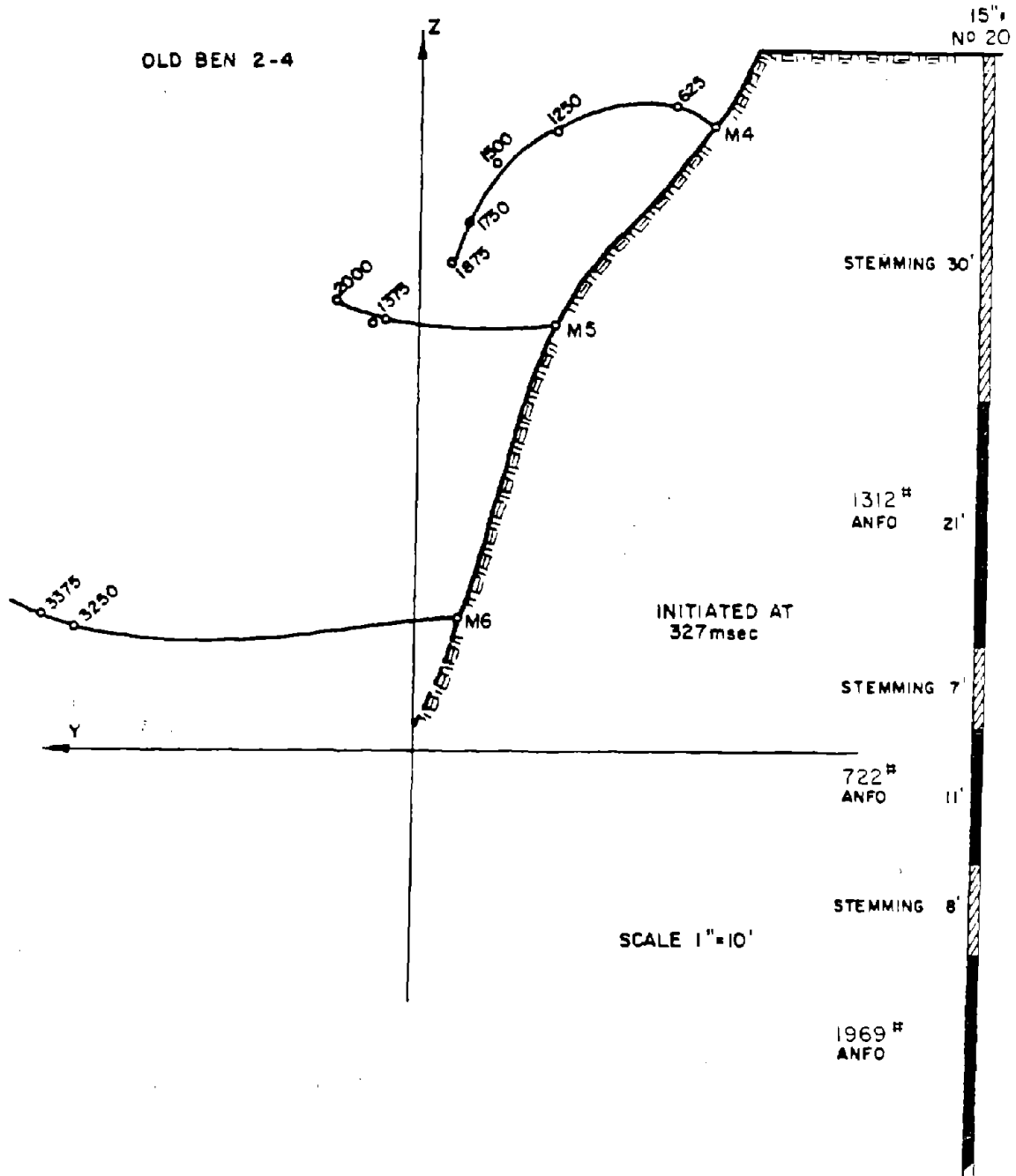
DRAWN BY M.R.E.L. DATE		CHECKED BY DATE	
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NO.	DESCRIPTION	DATE	BY

FACE VELOCITIES FOR
TEST BLAST NO. 2-4

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers ● Denver, Colorado

Job No. 19943

DRAWING NO	OF
C-21	



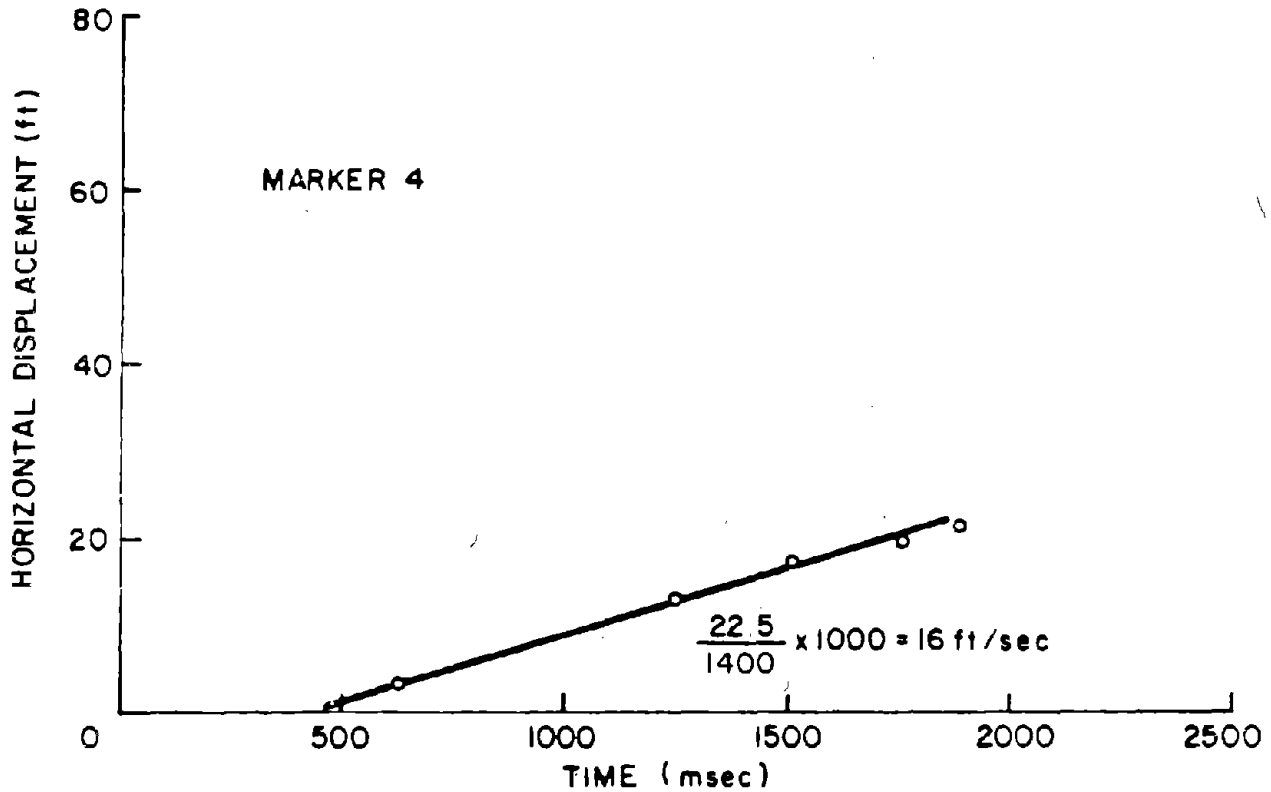
DRAWN BY		M.R.E.L.		DATE		CHECKED BY		DATE	
AS CONSTRUCTED - CORRECTIONS COMPLETE									
BY		DATE							
R E V I S I O N S									
NO	DESCRIPTION						DATE	BY	

MARKER TRAJECTORIES BLAST 2-4

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers Denver, Colorado

Job No. 19943

DRAWING NO. OF
C-22



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AS CONSTRUCTED - CORRECTIONS COMPLETE							
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FACE VELOCITIES FOR
TEST BLAST NO. 2-4

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers Denver, Colorado

Job No. 19943

DRAWING NO.	OF
C-23	

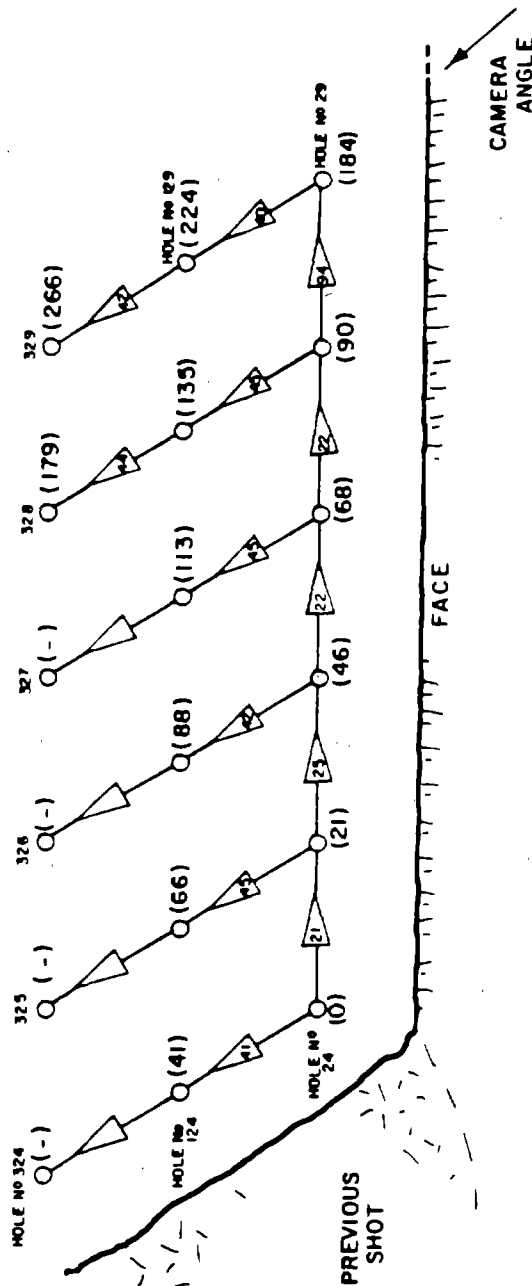
TEST BLAST 2-5

Shot 10-2-81 5:15 pm

18 Holes 30 ft x 50 ft pattern 26-ft collars

This shot had a 100 msec delay between the last row. Several delay times in the back row were obscured by the firing of the holes in front of them as shown in Figure C-24.

FILM NO 2-5
SHOT 10-2 -81 5:15 PM
30' x 50' 26' COLLARS



See "BLAST 1-6 DELAY TIMES" FOR KEY

DRAWN BY		M.R.E.L.		DATE		CHECKED BY		DATE			
AS CONSTRUCTED - CORRECTIONS COMPLETE											
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NO		DESCRIPTION						DATE		BY	

BLAST 2-5 DELAY TIMES

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers Denver, Colorado

Job No. 19943

DRAWING NO	OF
C-24	

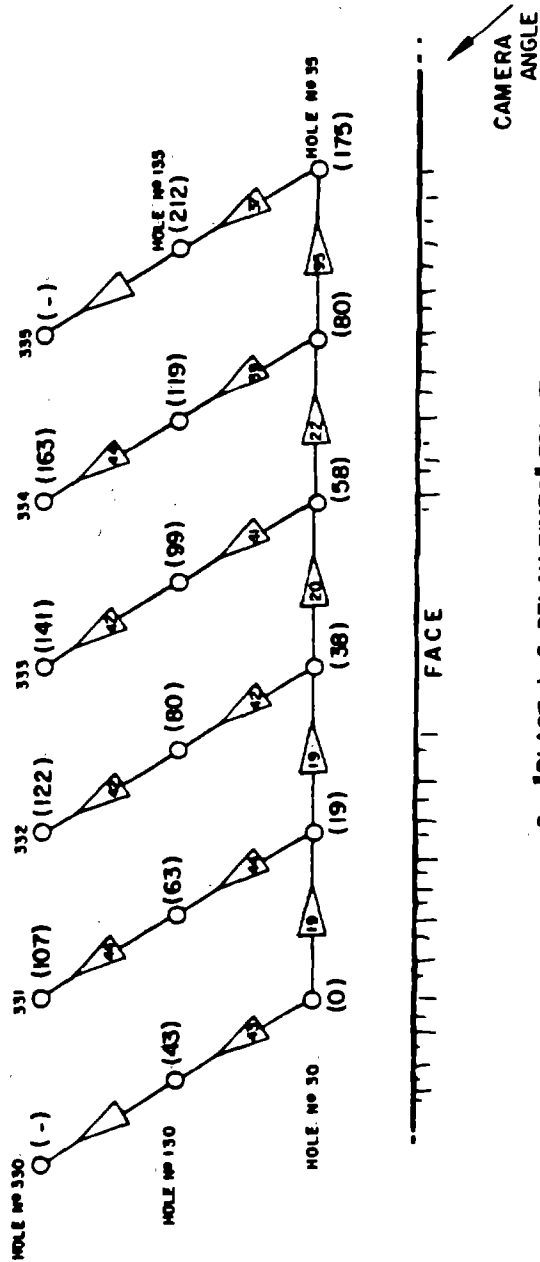
TEST BLAST 2-6

Shot 10-3-81 8:40 am

18 Holes 30 ft x 50 ft pattern 26-ft collars

This blast was similar to 2-5 and the delay times are shown on Figure C-25.

OLD BEN FILM NO 2-6
SHOT 10-3-81 8:40 AM
PATTERN 30'x50' 26' COLLAR



See "BLAST 1-6 DELAY TIMES" FOR KEY

DRAWN BY		M.R.E.L.		DATE		CHECKER BY		DATE	
AS CONSTRUCTED - CORRECTIONS COMPLETE									
BY		DATE							
REVISIONS									
NO.	DESCRIPTION							DATE	BY

BLAST 2-6 DELAY TIMES

WOODWARD - CLYDE CONSULTANTS
Consulting Engineers Denver, Colorado

Job No. 19943

DRAWING NO	OF
C-25	

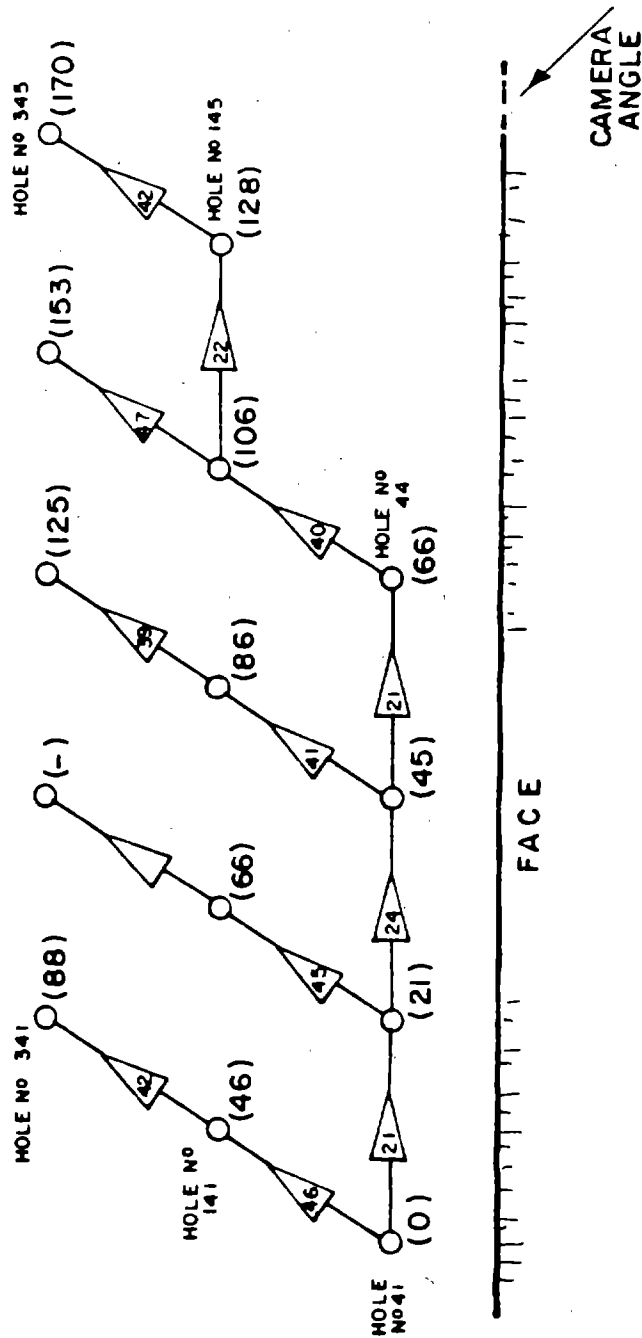
TEST BLAST 2-8

Shot 10-5-81 9:20 am

14 Holes 30 ft x 50 ft pattern 26-ft collars

Delay times are shown on Figure C-26.

OLD BEN FILM NO 2-8
 SHOT 10-5-81 9:20 AM
 30 x 50 26 ft COLLAR



See "BLAST 1-6 DELAY TIMES" FOR KEY

DRAWN BY		M.R.E.L.		DATE		CHECKED BY		DATE	
AS CONSTRUCTED - CORRECTIONS COMPLETE									
BY		DATE							
R E V I S I O N S									
NO	DESCRIPTION						DATE		BY

BLAST 2-8 DELAY TIMES

WOODWARD - CLYDE CONSULTANTS
 Consulting Engineers Denver, Colorado

Job No. 19943

DRAWING NO	OF
C-26	

TABLE C-2
SUMMARY OF DELAY TIMES

17 msec	N = 45
	$\bar{X} = 21.7$ msec
	$\sigma = 2.51$
	High = 32
	Low = 17
	Range = 15
42 msec	N = 78
	$\bar{X} = 42.6$ msec
	$\sigma = 2.90$
	High = 49
	Low = 35
	Range = 14
100 msec	N = 6
	$\bar{X} = 95.3$
	$\sigma = 2.88$
	High = 99
	Low = 91
	Range = 8
17 + 42 = 59 msec	N = 5
	$\bar{X} = 64.0$
	$\sigma = 1.87$
	High = 76
	Low = 62

TABLE C-3

SUMMARY OF DATA FROM FACE MARKERS

Blast	Marker No.	Horizontal Velocity F/S	Trajectory Angle From Horizontal	True Velocity F/S	Scaled Burden
1-8	1	14.5	27	16	5.5
	2	19	19	20	5.7
	3	25	24	27	6.3
	4	27	30	31	6.7
	5	27	16	28	7.1
2-1	1	23	20	24	4.7
	2	31	12	32	6.9
	3	31	18	33	7.9
	4	35	25	39	8.3
	Top		90	25 (vert.)	7.5
2-2	1	30	31	35	-
2-3	1	29	21	31	5.0
	2	32	29	37	5.9
	3	33	33	39	7.3
	4	34.5	38	44	7.7
	5	19.5	39	25	8.3
	Top		90	26 (vert.)	7.5
2-4	1	14	42	19	3.7
	2	22	18	23	3.8
	3	26	25	29	4.4
	4	16	38	20	4.0

APPENDIX D

MACHINE OPERATING COSTS

DISCOUNTED CASH FLOW ANALYSIS
OWNERSHIP AND OPERATING COST ESTIMATES
(Excluding Operator, Maintenance labor, and Depreciation)
OLD BEN NO. 1 COAL MINE BLAST DESIGN PROGRAM

APPENDIX D

TABLE OF CONTENTS

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DISCOUNTED CASH FLOW ANALYSES, OWNERSHIP AND OPERATING COST ESTIMATES (Excluding Operator, Maintenance Labor, and Depreciation) OLD BEN NO. 1 COAL MINE BLAST DESIGN PROGRAM	
BE 2570-W Dragline	D-1
BE 1570-W Dragline	D-2
BE 1150-B Dragline	D-3
BE 61R Drill	D-4
Terex S-24 Reclamation Scraper	D-5
Fiat-Allis HD 31 Ground Dozer	D-6
Cat D-9 Reclamation Dozer	D-7
Cat. D-9 Bench Preparation Dozer	D-8
BE 150 B Coal Shovel	D-9
Michigan 175 Front-End Loader (with Coal Shovel)	D-10
Rimpull 120-ton coal Hauler (1)	D-11
Lube, Mechanics, Welding, Electrician, and Service Trucks	D-12
Water Truck	D-13
Explosives Truck	D-14
Pick-up Trucks (4)	D-15
Cat. 14G Grader	D-16

MACHINE OPERATING COSTS

DISCOUNTED CASH FLOW ANALYSES
OWNERSHIP AND OPERATING COST ESTIMATE
(Excluding Operator, Maintenance labor, and Depreciation)
OLD BEN NO.1 COAL MINE BLAST DESIGN PROGRAM

BE 2570-W Dragline

OWNERSHIP COST

Capital Cost: \$34,000,000
Annual Depreciation Hours: 8736

$$\text{Ownership Cost} = \frac{(.20)(.5)(\$34,000,000)}{(8736)} = \$389.19/\text{depr. hr.}$$

OPERATING COST

		OPTION A	OPTION B	OPTION C
Operating hours/yr	=	6,468	NA	6,359
Operating cost/operating hour	= \$	497.02	NA	\$ 550.71
Operating cost/yr	=	\$3,214,700.00	NA	\$3,502,000.00
Operating cost/annual depreciation hour	= \$	367.98	NA	\$ 400.87
Total Ownership and Operating Cost/Depreciation Hour	= \$	757.17	NA	\$ 790.06

Notes:

- Power at \$0.03/Kw-hr.
- OPTION C reduced operating hours due to increased productivity (number of swings/operating hour) leading to slightly increased maintenance time.
- Operating costs based on manufacturers studies for machine in similar digging conditions.

DISCOUNTED CASH FLOW ANALYSES
OWNERSHIP AND OPERATING COST ESTIMATE
(Excluding Operator, Maintenance Labor, and Depreciation)
OLD BEN NO.1 COAL MINE BLAST DESIGN PROGRAM

BE 1570-W Dragline

OWNERSHIP COST

Capital Cost: \$22,000,000
Annual Depreciation Hours: 8736

$$\text{Ownership Cost} = \frac{(.20)(.5)(\$22,000,000)}{(8736)} = \$251.83/\text{depr. hr.}$$

OPERATING COST

		OPTION A	OPTION B	OPTION C
Operating hours/yr	=	NA	6,468	NA
Operating cost/operating hour	=	NA	\$ 413.06	NA
Operating cost/yr	=	NA	\$2,671,672.00	NA
Operating cost/annual depreciation hour	=	NA	\$ 305.82	NA
Total Ownership and Operating Cost/Depreciation Hour	=	NA	\$ 557.65	NA

Notes:

- a) Power at \$0.03/Kw-hr.
- b) Operating costs based on manufacturers studies for machine in similar digging conditions.

DISCOUNTED CASH FLOW ANALYSES
OWNERSHIP AND OPERATING COST ESTIMATE
(Excluding Operator, Maintenance labor, and Depreciation)
OLD BEN. NO. 1 COAL MINE BLAST DESIGN PROGRAM

BE 1150-B Dragline

OWNERSHIP COST

Capital Cost: \$8,000,000
Annual Depreciation Hours: 8736

$$\text{Ownership Cost} = \frac{(.20)(.5)(\$8,000,000)}{(8736)} = \$91.58/\text{depr. hr.}$$

OPERATING COST

	OPTION A	OPTION B	OPTION C
Operating hours/yr	= 4,900	4,900	6,000
Operating cost/operating hour	= \$ 167.51	\$ 167.51	\$ 167.51
Operating cost/yr	= \$820,799.00	\$820,799.00	\$1,005,060.00
Operating cost/annual depreciation hour	= \$ 93.96	\$ 93.96	\$ 115.05
Total Ownership and Operating Cost/Depreciation Hour	= \$ 185.54	\$ 185.54	\$ 206.63

Notes:

- Power at \$0.03/Kw-hr.
- Operating costs based on manufacturers studies for machine in similar digging conditions.
- Pullback requirement is not rigidly scheduled, hence, BE 1150 operates on (nominally) 25% of the BE 2570-W or BE 1570-W loose cubic yardage.

MACHINE OPERATING COSTS

DISCOUNTED CASH FLOW ANALYSES
OWNERSHIP AND OPERATING COST ESTIMATE
(Excluding Operator, Maintenance labor, and Depreciation)
OLD BEN NO.1 COAL MINE BLAST DESIGN PROGRAM

BE 6IR Drill

OWNERSHIP COST

Capital Cost: \$1,400,000
Annual Depreciation Hours: 4992

$$\text{Ownership Cost} = \frac{(.20)(.5)(\$1,400,000)}{(4992)} = \$28.04/\text{depr. hr.}$$

OPERATING COST

	OPTION A	OPTION B	OPTION C
Operating hours/yr	= 3,500	3,400	4,300
Operating cost/operating hour	= \$ 75.00	\$ 75.00	\$ 75.00
Operating cost/yr	= \$262,500.00	\$255,000.00	\$322,500.00
Operating cost/annual depreciation hour	= \$ 52.58	\$ 51.08	\$ 64.60
Total Ownership and Operating Cost/Depreciation Hour	= \$ 80.62	\$ 79.12	\$ 92.64

Notes:

- Drill bit consumption from historical records = \$0.15/ft.
- Drill chain replacement historical cost = \$ 0.01/ft.
- Power at \$ 0.03/Kw-hr = \$12.50/op-hour
- Option A = 333,500 ft. drill hole + 5% (lost hole/yr) $\approx$ 350,000 ft. drill hole/yr.
Option B = 326,800 ft. drill hole + 5% (lost hole/yr) $\approx$ 340,000 ft. drill hole/yr.
Option C = 408,500 ft. drill hole + 5% (lost hole/yr) $\approx$ 430,000 ft. drill hole/yr.
- 100 ft. drillhole/operating hour

DISCOUNTED CASH FLOW ANALYSES
OWNERSHIP AND OPERATING COST ESTIMATE
(Excluding Operator, Maintenance labor, and Depreciation)
OLD BEN NO.1 COAL MINE BLAST DESIGN PROGRAM

Terex S-24 Reclamation Scraper

OWNERSHIP COST

Capital Cost: \$400,000,
Annual Depreciation Hours: 4992 (2496 in West Field)

$$\frac{\text{Ownership Cost}}{\text{Depreciation Hr}} = \frac{(.20)(.5)(\$400,000)}{(4992)} = \$8.01/\text{depr. hr}$$

OPERATING COST (West Field Only)

	OPTION A	OPTION B	OPTION C
Operating hours/yr	= 1,872	1,872	2,340
Operating cost/operating hour	= \$ 53.50	53.50	\$ 53.50
Operating cost/yr	= \$100,152.00	\$100,152.00	\$125,190.00
Operating cost/annual depreciation hour	= \$ 40.12	\$ 40.12	\$ 50.16
Total Ownership and Operating Cost/Depreciation Hour	= \$ 48.14	\$ 48.14	\$ 58.17

Notes:

- Machine scheduled for West Field 2496 hrs/yr.
- Tire wear average.
- Tire capital cost of \$30,000 excluded from depreciation (Table II) schedule.

DISCOUNTED CASH FLOW ANALYSES
OWNERSHIP AND OPERATING COST ESTIMATE
(Excluding Operator, Maintenance labor, and Depreciation)
OLD BEN NO.1 COAL MINE BLAST DESIGN PROGRAM

Fiat-Allis HD 31 Ground Dozer

OWNERSHIP COST

Capital Cost: \$360,000
Annual Depreciation Hours: 8736

$$\frac{\text{Ownership Cost}}{\text{West Field Depreciation Hour}} = \frac{(.20)(.5)(\$360,000)}{(8736)} = \$4.12/\text{depr. hr.}$$

OPERATING COST

	OPTION A	OPTION B	OPTION C
Operating hours/yr	= 2,000	2,000	2,500
Operating cost/operating hour	= \$ 35.00	\$ 35.00	\$ 35.00
Operating cost/yr	= \$70,000.00	\$70,000.00	\$87,500.00
Operating cost/annual depreciation hour	= \$ 8.01	\$ 8.01	\$ 10.02
Total Ownership and Operating Cost/Depreciation Hour	= \$ 12.13	\$ 12.13	\$ 14.14

Notes:

- The ground dozer activity is limited to bench smoothing, cable hauling and other light duty around the stripping dragline.
- Fuel at \$1/gallon.

DISCOUNTED CASH FLOW ANALYSES
OWNERSHIP AND OPERATING COST ESTIMATE
(Excluding Operator, Maintenance Labor, and Depreciation)
OLD BEN NO.1 COAL MINE BLAST DESIGN PROGRAM

Cat D-9 Reclamation Dozer

OWNERSHIP COST

Capital Cost: \$450,000
Annual Depreciation Hours: 2496

$$\frac{\text{Ownership Cost}}{\text{West Field Depreciation Hour}} = \frac{(.20)(.5)(\$450,000)}{(2496)} = \$19.03/\text{depr. hr.}$$

OPERATING COST (West Field Only)

	OPTION A	OPTION B	OPTION C
Operating hours/yr	= 1,800	1,800	2,250
Operating cost/operating hour	= \$ 45.00	\$ 49.00	\$ 49.00
Operating cost/yr	= \$81,000.00	\$88,200.00	\$110,250.00
Operating cost/annual depreciation hour	= \$ 32.45	\$ 35.34	\$ 44.17
Total Ownership and Operating Cost/Depreciation Hour	= \$ 50.48	\$ 53.37	\$ 62.20

Notes:

- Fuel at \$1/gallon.
- Higher operating costs for OPTION B & C reflect coarse material cast to top of spoils under new method as opposed to historical (OPTION A) practice of placing clay overburden onto top of spoils.

DISCOUNTED CASH FLOW ANALYSES
OWNERSHIP AND OPERATING COST ESTIMATE
(Excluding Operator, Maintenance Labor, and Depreciation)
OLD BEN NO.1 COAL MINE BLAST DESIGN PROGRAM

Cat. D-9 Bench Preparation Dozer

OWNERSHIP COST

Capital Cost: \$450,000
Annual Depreciation Hours: 8736

$$\text{Ownership Cost} = \frac{(.20)(.5)(\$450,000)}{(8736)} = \$5.15/\text{depr. hr.}$$

OPERATING COST

		OPTION A	OPTION B	OPTION C
Operating hours/yr	=	NA	5,500	6,115
Operating cost/operating hour	=	NA	\$ 53.00	\$ 53.00
Operating cost/yr	=	NA	\$291,500.00	\$324,095.00
Operating cost/annual depreciation hour	=	NA	\$ 33.37	\$ 37.10
Total Ownership and Operating Cost/Depreciation Hour	=	NA	\$ 38.52	\$ 42.25

Notes:

- a) Fuel at \$1/gallon

DISCOUNTED CASH FLOW ANALYSES
OWNERSHIP AND OPERATING COST ESTIMATE
(Excluding Operator, Maintenance Labor, and Depreciation)
OLD BEN NO.1 COAL MINE BLAST DESIGN PROGRAM

BE 150 B Coal Shovel

OWNERSHIP COST (West Field Only)

Capital Cost: \$1,500,000
Annual Depreciation Hours: 4992 (2496 in West Field)

$$\text{Ownership Cost} = \frac{(.20)(.5)(\$1,500,000)}{(4992)} = \$30.05/\text{ depr. hr.}$$

OPERATING COST (West Field Only)

	OPTION A	OPTION B	OPTION C
Operating hours/yr	= 1,640	1,640	2,050
Operating cost/operating hour	= \$ 48.20	\$ 48.20	\$ 48.20
Operating cost/yr	= \$79,048.00	\$79,048.00	\$98,810.00
Operating cost/annual depreciation hour	= \$ 31.67	\$ 31.67	\$ 39.59
Total Ownership and Operating Cost/Depreciation Hour	= \$ 61.72	\$ 61.72	\$ 69.64

Notes:

- Power at \$0.03/Kw-hr.
- Pit time the same for all options; machine activity improvement in Option C.
- West Field 2496 depreciation hours/yr.

DISCOUNTED CASH FLOW ANALYSES
OWNERSHIP AND OPERATING COST ESTIMATE
(Excluding Operator, Maintenance labor, and Depreciation)
OLD BEN NO.1 COAL MINE BLAST DESIGN PROGRAM

Michigan 175 Front-End Loader
(with Coal Shovel)

OWNERSHIP COST

Capital Cost: \$250,000

Annual Depreciation Hours: 4992 (2496 in West Field)

$$\text{Ownership Cost} = \frac{(.20)(.5)(\$250,000)}{(4992)} = \$5.01/\text{depr. hr.}$$

OPERATING COST (West Field Only)

	OPTION A	OPTION B	OPTION C
Operating hours/yr	= 1,640	1,640	2,050
Operating cost/operating hour	= \$ 30.00	\$ 30.00	\$ 30.00
Operating cost/yr	= \$49,200.00	\$49,200.00	\$61,500.00
Operating cost/annual depreciation hour	= \$ 19.71	\$ 19.71	\$ 24.64
Total Ownership and Operating Cost/Depreciation Hour	= \$ 24.73	\$ 24.73	\$ 29.65

Notes:

- Fuel at \$1/gallon.
- Pit time same in all options; machine activity improvement in Option C.
- 2496 depreciation hour in West Field.
- Tire capital cost of \$20,000 excluded from depreciation (Table II) schedule.

DISCOUNTED CASH FLOW ANALYSES
OWNERSHIP AND OPERATING COST ESTIMATE
(Excluding Operator, Maintenance, Labor and Depreciation)
OLD BEN NO.1 COAL MINE BLAST DESIGN PROGRAM

Rimpull 120-Ton Coal Hauler (1)

OWNERSHIP COST

Capital Cost: \$450,000
Annual Depreciation Hours: 4992 (2496 in West Field)

$$\text{Ownership Cost} = \frac{(.20)(.5)(\$450,000)}{(4992)} = \$9.01/\text{depr. hr.}$$

OPERATING COST (West Field Only)

	OPTION A	OPTION B	OPTION C
Operating hours/yr	= 1,800	1,800	2,250
Operating cost/operating hour	= \$ 56.00	\$ 56.00	\$ 58.00
Operating cost/yr	= \$100,800.00	\$100,800.00	\$130,500.00
Operating cost/annual depreciation hour	= \$ 40.38	\$ 40.38	\$ 52.28
Total Ownership and Operating Cost/Depreciation Hour	= \$ 49.39	\$ 49.39	\$ 61.29

Notes:

- Fuel at \$1/hour.
- West Field operation time same for all options; machine activity improvement in Option C.
- 2496 West Field depreciation hours/yr.
- Tire capital cost of \$50,000/unit excluded from depreciation (Table II) schedule.

DISCOUNTED CASH FLOW ANALYSES
OWNERSHIP AND OPERATING COST ESTIMATE
(Excluding Operator, Maintenance labor, and Depreciation)
OLD BEN NO. 1 COAL MINE BLAST DESIGN PROGRAM

Lube, Mechanics, Welding, Electrician, and Service Trucks

OWNERSHIP COST

Capital Cost: \$130,000

Annual Depreciation Hours: 8736 (4368 in West Field)

$$\text{Ownership Cost} = \frac{(.20)(.5)(\$130,000)}{(8736)} = \$1.49/\text{depr. hr.}$$

OPERATING COST (West Field Only)

	OPTION A	OPTION B	OPTION C
Operating hours/yr	= 2,184	2,184	2,184
Operating cost/operating hour	= \$ 30.00	\$ 30.00	\$ 30.00
Operating cost/yr	= \$65,520.00	\$65,520.00	\$65,520.00
Operating cost/annual depreciation hour	= \$ 15.00	\$ 15.00	\$ 15.00
Total Ownership and Operating Cost/Depreciation Hour	= \$ 16.49	\$ 16.49	\$ 16.49

Notes:

- a) Fuel at \$1/gallon.
- b) 50% activity in all options.
- c) West Field depreciation hours = 4368.

DISCOUNTED CASH FLOW ANALYSES
OWNERSHIP AND OPERATING COST ESTIMATE
(Excluding Operator, Maintenance labor, and Depreciation)
OLD BEN NO.1 COAL MINE BLAST DESIGN PROGRAM

Water Truck

OWNERSHIP COST

Capital Cost: \$77,000

Annual Depreciation Hours: 4992 (2496 in West Field)

$$\text{Ownership Cost} = \frac{(.20)(.5)(\$77,000)}{(4992)} = \$1.54/\text{depr. hr.}$$

OPERATING COST (West Field Only)

	OPTION A	OPTION B	OPTION C
Operating hours/yr	= 1,250	1,250	1,250
Operating cost/operating hour	= \$ 18.00	\$ 18.00	\$ 18.00
Operating cost/yr	= \$22,500.00	\$22,500.00	\$22,500.00
Operating cost/annual depreciation hour	= \$ 9.01	\$ 9.01	\$ 9.01
Total Ownership and Operating Cost/Depreciation Hour	= \$ 10.55	\$ 10.55	\$ 10.55

Notes:

- 2496 depreciation hours in West Field activities.
- Tire capital cost of \$5000 excluded from depreciation (Table II) schedule.

DISCOUNTED CASH FLOW ANALYSES
OWNERSHIP AND OPERATING COST ESTIMATE
(Excluding Operator, Maintenance labor, and Depreciation)
OLD BEN NO.1 COAL MINE BLAST DESIGN PROGRAM

Explosives Truck

OWNERSHIP COST

Capital Cost: \$47,000
Annual Depreciation Hours: 4992

$$\text{Ownership Cost} = \frac{(.20)(.5)(\$47,000)}{(4992)} = \$0.94/\text{depr. hr.}$$

OPERATING COST

		OPTION A	OPTION B	OPTION C
Operating hours/yr	=	2,200	2,200	2,200
Operating cost/operating hour	= \$	11.92	\$ 11.92	\$ 11.92
Operating cost/yr	=	\$26,224.00	\$26,224.00	\$26,224.00
Operating cost/annual depreciation hour	= \$	5.25	\$ 5.25	\$ 5.25
Total Ownership and Operating Cost/Depreciation Hour	= \$	6.19	\$ 6.19	\$ 6.19

Notes:

- a) Fuel at \$1/gallon.

DISCOUNTED CASH FLOW ANALYSES
OWNERSHIP AND OPERATING COST ESTIMATE
(Excluding Operator, Maintenance labor, and Depreciation)
OLD BEN NO.1 COAL MINE BLAST DESIGN PROGRAM

Pick-up Trucks (4)

OWNERSHIP COST

Capital Cost: \$44,000
Annual Depreciation Hours: 8736 (4368 in West Field)

$$\text{Ownership Cost} = \frac{(.20)(.5)(\$44,000)}{(8736)} = \$0.50/\text{depr. hr.}$$

OPERATING COST (West Field Only)

	OPTION A	OPTION B	OPTION C
Operating hours/yr	= 3,300	3,300	3,300
Operating cost/operating hour	= \$ 8.97	\$ 8.97	\$ 8.97
Operating cost/yr	= \$29,601.00	\$29,601.00	\$29,601.00
Operating cost/annual depreciation hour	= \$ 6.78	\$ 6.78	\$ 6.78
Total Ownership and Operating Cost/Depreciation Hour	= \$ 7.28	\$ 7.28	\$ 7.28

Notes:

- a) Fuel at \$1/gallon.
- b) 75% activity 1 1/2 shift/day in West Field by supervisory staff.
- c) West Field depreciation hours = 4368.

DISCOUNTED CASH FLOW ANALYSES
OWNERSHIP AND OPERATING COST ESTIMATE
(Excluding Operator, Maintenance labor, and Depreciation)
OLD BEN NO.1 COAL MINE BLAST DESIGN PROGRAM

Cat. 14G Grader

OWNERSHIP COST

Capital Cost: \$175,000

Annual Depreciation Hours: 4992 (2496 in West Field)

$$\text{Ownership Cost} = \frac{(.20)(.5)(\$175,000)}{(4992)} = \$3.50/\text{depr. hr.}$$

OPERATING COST (West Field Only)

	OPTION A	OPTION B	OPTION C
Operating hours/yr	= 1,550	1,550	1,550
Operating cost/operating hour	= \$ 22.00	\$ 22.00	\$ 22.00
Operating cost/yr	= \$34,100.00	\$34,100.00	\$34,100.00
Operating cost/annual depreciation hour	= \$ 13.66	\$ 13.66	\$ 13.66
Total Ownership and Operating Cost/Depreciation Hour	= \$ 17.16	\$ 17.16	\$ 17.16

Notes:

- West Field Depreciation hour = 2496
- Tire capital cost of \$8000 excluded from depreciation (Table II) schedule.

APPENDIX E

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REFERENCES

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